Report to:

Pretium Resources Inc.

Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC

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TETRA TECH

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FEASIBILITY STUDY AND TECHNICAL REPORT ON THE BRUCEJACK PROJECT, STEWART, BC

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GLOSSARY

Units of Measure

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
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 (American)	US$
dollar (Canadian)	Cdn$
dry metric ton	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
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	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	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
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
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
short ton (2,000 lb)	st
short tons per day	st/d
short tons per year	st/y
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
three-dimensional	3D
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt

Abbreviations and Acronyms

acid base accounting	ABA
acid rock drainage	ARD
Air Terminal Building	ATB
ALS Minerals	ALS
AMC Mining Consultants (Canada) Ltd.	AMC
Andestite	ANDX
Alpine Solutions Avalanche Services	Alpine Solutions
Argillite	ARG
Association for the Advancement of Cost Engineering International	AACE
atomic absorption spectrophotometer	AAS
atomic absorption	AA

atomic emission spectroscopy	AES
BC *Environmental Assessment Act*	BCEAA
BGC Engineering Inc.	BGC
Black Hawk Mining Inc.	Black Hawk
Bond abrasion index	Ai
Bond ball mill work index	BWi
Bond crushing mill work index	CWi
Bond rod mill work index	RWi
British Columbia	BC
Canadian Council of Ministers of the Environment	CCME
Canadian development expense	CDE
Canadian *Environmental Assessment Act*	CEAA
Canadian Institue of Mining, Metallurgy and Petroleum	CIM
Canadian Pension Plan	CPP
carbon-in-leach	CIL
central equipment enclosure	CEE
CESL Limited	CESL
closed-circuit television	CCTV
construction management team	CMT
Contaminated Site Regulation	CSR
Corona Corporation	Corona
cumulative expenditures account	CEA
cumulative net cash flow	CNCF
cumulative tax credit account	CTCA
cyanide soluble	CN
Department of Earth, Ocean, and Atmospheric Sciences	EOAS
diesel engine exhaust	DEE
digital terrain model	DTM
distributed control system	DCS
drop weight index	DWi
effective grinding length	EGL
Employment Insurance	EI
engineering, procurement, construction management	EPCM
engineer-procure-contract	EPC
environmental assessment certificate	EAC
Environmental Assessment Office	EAO
environmental assessment	EA
environmental impact assessment	EIA
Environmental Management System	EMS
Esso Minerals Canada Ltd.	Esso
extended gravity recoverable gold	EGRG
fly-ash	FA

general & administrative	G&A
General Purpose	GP
GeoSpark Consulting Inc.	GeoSpark
gold	Au
gold equivalent	AuEq
Granduc Mines Ltd.	Granduc
gravity recoverable gold	GRG
gravity recoverable silver	GRS
gross vehicle limit	GVL
hazard and operability analysis	HAZOP
health, safety and environmental	HSE
heating, ventilation, air conditioning	HVAC
high-density polyethylene	HDPE
Impact Benefit Agreement	IBA
inductively coupled plasma	ICP
input/output	I/O
Instrument Approach Procedures	IAP
Instrument Flight Rules	IFR
intensity duration frequency	IDF
internal rate of return	IRR
International Organization for Standardization	ISO
joint venture	JV
Jurassic Conglomerate	JR
Kerr-Sulphurets-Mitchell	KSM
Lancona Mining Corp.	Lancona
Land and Resource Management Plan	LRMP
life-of-mine	LOM
Light Detection and Ranging	LiDAR
linear low density polyethylene	LLDP
load-haul-dump	LHD
Long Lake Hydro	LLH
longhole open stoping	LHOS
magnetotelluric	MT
mass spectrometer	MS
Master Lower Zone	ML
Master Upper Zone	MU
Material Safety Data Sheets	MSDS
Medical Service Plan	MSP
metal leaching	ML
Metal Mining Effluent Regulations	MMER
Meteorological Service of Canada	MSC
methyl isobutyl carbinol	MIBC

Mineable Shape Optimizer	MSO
MineCem	MC
Mineral Titles Online	MTO
Ministry of Energy, Mines, and Natural Gas	MEMNG
Ministry of Environment	MOE
motor control centre	MCC
National Instrument 43-101	NI 43-101
net cash flow	NCF
net invoice value	NIV
net present value	NPV
net smelter return	NSR
neutralization potential ratio	NPR
Newhawk Gold Mines Ltd.	Newhawk
non-acid generating	NAG
North American Datum	NAD
North American Free Trade Agreement	NAFTA
Northern Transmission Line	NTL
Obstacle Limitation Surfaces	OLS
official community plans	OCP
Omni Directional Approach Lighting System	ODALS
Pacific Centre for Isotopic and Geochemical Research	PCIGR
Paterson & Cooke Canada Inc.	P&C
Placer Dome Inc.	Placer
Porphyry	P1
potassium amyl xanthate	PAX
potentially acid generating	PAG
Precision Approach Path Indicators	PAPI
preliminary economic assessment	PEA
pressure reducing values	PRV
Pretium Resources Inc.	Pretivm
Process Research Associates Ltd.	PRA
programmable computer	PC
programmable logic controller	PLC
project execution plan	PEP
qualified person	QP
quality assurance	QA
quality control	QC
radio frequency identification	RFID
Radiogenic Isotope Facility	RIF
Reference Evapotranspiration	REF-ET
Registered Retirement Savings Plan	RRSP
remote avalanche control system	RACS

Rescan Environmental Services Ltd.	Rescan
return air raise	RAR
right-of-way	ROW
rock quality designation	RQD
run-of-mine	ROM
Runway End Identifier Lights	REIL
SAG mill/ball mill	SAB
SAG mill/ball/pebble crusher	SABC
Seabridge Gold Inc.	Seabridge Gold
semi-autogeneous grinding	SAG
SGS Canada	SGS
Silicified Rock	RHY
silver	Ag
Silver Standard Resources Inc.	Silver Standard
Skeena Fold Belt	SFB
Snowden Mining Industry Consultants Inc.	Snowden
Social and Community Management Systems	SCMS
Standards Council of Canada	SCC
Stewart Bulk Terminal	SBT
Sunstate Slag Blend	SS
Sustainable Resource Management Plan	SRMP
Teuton Resources Corporation	Teuton
the Brucejack Project	the Project or the Property
total suspended solids	TSS
Traditional Knowledge/Traditional Use	TK/TU
Transportation Association of Canada	TAC
Triassic Sediment	TRS
twenty foot equivalent unit	TEU
ultra-high frequency	UHF
unconfined compressive strength	UCS
unconfined compressive	UC
underground distribution system	UDS
uninterruptable power supply	UPS
Universal Transverse Mercator	UTM
Valard Construction	Valard
Valley of the Kings	VOK
variable frequency drive	VFD
very high frequency	VHF
Visual Climb Area	VCA
Visual Flight Rules	VFR
Voice over Internet Protocol	VoIP
VOK Domain 1	VOK D1

VOK Domain 2	VOK D2
VOK Domain 3	VOK D3
VOK Fault Zone	VOK FZ
VOK Weathered Rock Zone	VOK WRZ
volcanogenic massive sulphide	VMS
water balance model	WBM
West Zone Fault Zone	WZ FZ
West Zone Fresh Rock	WZ FR
West Zone Weathered Rock Zone	WZ WRZ
work breakdown structure	WBS
Workers' Compensation Board	WCB
Workplace Hazardous Materials Information System	WHMIS

1.0 SUMMARY

1.1 INTRODUCTION

The Brucejack Project (the Project or the Property), located in northwestern British Columbia (BC), will be a 2,700 t/d underground mining operation over a 22-year life-of-mine (LOM). Ore will be processed using a combination of conventional sulphide flotation and gravity concentration to recover gold and silver. The Property is 100% owned by Pretium Resources Inc. (Pretivm).

In 2012, Pretivm commissioned a team of consultants to complete a feasibility study in accordance with National Instrument 43-101 (NI 43-101) for the Project. The following consultants were commissioned to complete the component studies for the purpose of the feasibility study:

- Tetra Tech: overall project management; mineral processing and metallurgical testing; recovery methods; access infrastructure; internal site roads and pad areas; grading and drainage; ancillary facilities; water supply and distribution; water treatment plant; communications; power supply and distribution; fuel supply and distribution; off-site infrastructure; market studies and contracts; capital cost estimate; processing operating cost estimate; financial analysis; and project execution plan
- Snowden Mining Industry Consultants Inc. (Snowden): property description and location, accessibility, climate, and physiology, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation, data verification, adjacent properties, and mineral resource estimates
- AMC Mining Consultants (Canada) Ltd. (AMC): mining including mine capital and operating cost estimates, mineral reserve estimates
- Rescan Environmental Services Ltd. (Rescan): environmental studies, permits, and social or community impacts; and tailings delivery system
- BGC Engineering Inc. (BGC): geotechnical design, mine hydrogeological/groundwater; waste disposal; Brucejack outlet control; environmental water management and water quality, acid rock drainage (ARD) and metal leaching (ML)
- Alpine Solutions Avalanche Services (Alpine Solutions): avalanche hazard assessment
- Valard Construction (Valard): transmission line
- Paterson & Cooke Canada Inc. (P&C): paste fill distribution.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Property is situated approximately at 56°28′20″N Latitude by 130°11′31″W Longitude, a position approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine in the Province of BC. The Property consists of six mineral claims totalling 3,199.28 ha in area and all claims are in good standing until January 31, 2024.

The Property and the surrounding region have a history rich in exploration for precious and base metals dating back to the late 1800s. More recently in 2009 Silver Standard Resources Inc. (Silver Standard) began work on the Property. The 2009 program included drilling, rock-chip and channel sampling, and re-sampling of historical drill core. In 2010, pursuant to a purchase and sale agreement between Silver Standard Resources Inc. (Silver Standard), (as the seller) and Pretivm (as the buyer), Silver Standard sold to Pretivm all of the issued shares of 0890693 BC Ltd., the owner of the Project and the adjacent Snowfield Project.

1.3 GEOLOGY AND MINERALIZATION

The Property is largely underlain by volcano-sedimentary rocks of the Lower Jurassic Hazelton Group. These rocks unconformably overlie volcanic arc sedimentary rocks of the Upper Triassic Stuhini Group along the westernmost part of the Property. Hazelton Group rocks on the Property include hornblende and/or feldspar-phyric volcanic (latite) flows, and pyroclastic fragmental rocks, locally derived heterolithic volcanic pebble to boulder conglomerate, volcanic sandstone, siltstone and mudstone.

The Hazelton Group volcano-sedimentary rocks are interpreted as having been deposited in an extensional structural regime, proximal to a basin margin growth fault (or series of growth faults), owing to the presence of complicated lateral facies variations within the volcanic pile and the diachronous and immature nature of the rock units. The Brucejack fault, which forms a distinct topographical feature across the Property, is currently interpreted as being the reactivated expression of one of these basin margin structures.

Alteration on the Property is characterized by variable, but generally intensely quartz-sericite-pyrite altered rocks that define a distinctive and continuous north-south arcuate (west-concave) band of gossanous rocks that is up to several hundreds of metres wide, and approximately 5 km in strike (north-south) extent. Quartz-sericite-pyrite alteration is broadly spatially associated with the Brucejack Fault and the unconformity between the Stuhini and Hazelton Group rocks, which suggests that these structures (the palaeo-growth fault zone in the case of the Brucejack Fault) may have acted as important fluid conduits during hydrothermal alteration and mineralization.

Gold (± silver) mineralization is hosted in predominantly sub-vertical vein stockwork and subordinate vein breccia systems of variable intensity, throughout the alteration band. The stockwork systems are relatively continuous along strike (several tens of metres to several hundreds of metres) and are characterized by the presence of millimetre- to decimetre-scale transitional meso- to epithermal veins of quartz-adularia, quartz-

carbonate, quartz, and pyrite that form intense cross-cutting networks within the stockworks. Minerals recognized in different stockwork vein generations across the Property include: pyrite, tetrahedrite, tennantite, chalcopyrite, galena, sphalerite, molybdenite, arsenopyrite, pyrrhotite, pyrargyrite, polybasite, acanthite, native silver, native gold, and electrum. Alteration, mineralization and vein texture variations across the Property suggest down-temperature thermal gradients towards the east (i.e. up stratigraphy) and north.

Several mineralization zones have been explored to varying degrees, including (from south to north): Bridge Zone, Valley of the Kings (VOK), West Zone, Gossan Hill, Shore Zone, and SG Zone. There are numerous relatively unexplored mineralization showings within the alteration band across the property that are between the main mineralization zones, highlighting the exceptional exploration potential of the Property.

Hazelton Group rocks and mineralized vein stockworks on the Property display significant multi-phase post-mineralization deformation. A series of variably and locally doubly-plunging tight to open map-scale folds, host tight south-vergent meso- and parasitic scale folds coincident with a regional penetrative east-west trending foliation. The plunge of the minor folds varies and several lines of evidence suggest that they may reflect refolding of northerly-trending tight, upright “early” mid-Cretaceous Skeena Fold Belt-age folds across roughly east-west axes. These later folds and the east-west foliation are spatially associated with the area of the footwall (and immediate hanging wall) of the regional-scale Mitchell thrust system, suggesting that they may have developed in the latter stages of Skeena Fold Belt development. Late stage north-south trending brittle faults are indicated by pronounced surface lineations.

High-grade gold mineralization at VOK, the current focus of the Project, occurs in a series of west-northwest trending sub-vertical corridors of structurally reoriented vein stockworks and subordinate vein breccias, hosted in a folded sequence of fine grained volcanic siltstone and sandstone, variably silicified polylithic volcanic conglomerate, and fragmental latite volcanic rocks. Relatively massive latite flows are present to the immediate north and south of this mineralization host rock sequence. Gold is typically present as gold-rich electrum within deformed quartz-carbonate (±adularia?) vein stockworks, veins, and subordinate vein breccias, with grades ranging up to 41,582 ppm Au and 27,725 ppm Ag over 0.5 m. The gold occurs both as inclusions in euhedral pyrite, as well as interstitially in textural equilibrium with strained quartz (exhibiting primary recrystallization textures), carbonate, several generations of pyrite, and, to a lesser extent, vein-hosted sericite. Gold has also been found in the central parts of deformed quartz veins, as dendritic lattice works pervasive throughout deformed quartz, and quartz-carbonate veins, as well as in the matrix of hydrothermal breccias. The various textural associations of gold suggest a multi-stage paragenesis for gold mineralization. The VOK deposit is currently defined over 1,200 m in east-west extent, 600 m in north-south extent, and 650 m in depth.

1.4 MINERAL RESOURCE ESTIMATES

Snowden, in November 2012,completed a mineral resource estimate for VOK within the Project. The West Zone estimate remains unchanged from the April 2012 mineral resource estimate (Olssen and Jones 2012a). The new estimate is reported at a high-grade cut-off for potential underground extraction.

A threshold grade of 0.3 g/t Au was found to generally identify the broad zones of mineralization in the drill cores at West Zone and VOK. At VOK, a 1 g/t Au to 3 g/t Au threshold grade was used together with Pretivm's interpretation of the lithological domains, to interpret high-grade corridors within the broader mineralized zones, and define a series of mineralized domains for estimation.

High-grade gold mineralization at VOK occurs in a series of west-northwest trending sub-vertical corridors of structurally reoriented vein stockworks and subordinate vein breccias, hosted in a folded sequence of fine grained volcanic siltstone and sandstone, variably silicified polylithic volcanic conglomerate rocks, and fragmental volcanic rocks of latitic composition, bounded by relatively massive porphyritic latite flows. Gold mineralization also appears to form a series of seemingly stratigraphy-parallel pods within the steeply-dipping polylithic conglomerate and immediately overlying fragmental volcanic rocks in the core of the syncline. These pods appear to be spatially associated with intensely silicified zones that appear to have acted as pressure seals which generated local overpressure conditions and subsequent depressurization by hydraulic and/or tectonic brecciation.

All data was composited to the dominant sample length of 1.5 m prior to analysis and estimation. Statistical analysis of the gold and silver data was carried out by lithological domain (at VOK) and mineralized domain. Review of the statistics indicated that the grade distributions for the mineralization within the various volcanic and volcano-sedimentary lithologies are very similar and as a result these were combined for analysis. All domains exhibit a strong positive skewness with high coefficient of variation and extreme outliers.

Because of the extreme positive skew in the histograms of the gold and silver grades within the high-grade domains, Snowden elected to use an indicator approach whereby the proportion of high grade in a block was modelled, the grade of the high-grade portion, and the grade of the low-grade portion.

The high-grade population, which contains a significant number of samples with extreme grades, required indicator kriging methods for grade estimation. The low-grade estimation was estimated with ordinary kriging combined with top cut data.

Density was estimated using simple kriging of specific gravity measurements provided by ALS Minerals (ALS). As part of the 2012 drilling program, Pretivm selected a portion of the samples (207 samples) for core density measurements in addition to pulp specific gravity measurements to determine whether there is any impact on the density as a result of porosity. The results of the comparison indicate that the core density is on

average 6% lower than the pulp specific gravity within the siliceous zone and 9% lower on average in all other rock types. As a result the pulp specific gravity measurements, which are used to estimate density in the model, were factored by 6% (siliceous zone) or 9% (other rock types) prior to statistical analysis and estimation.

Grade estimates and models were validated by: undertaking global grade comparisons with the input drillhole composites; visual validation of block model cross sections; and by grade trend plots.

The resource classification definitions (Measured, Indicated, Inferred) used for this estimate are those published by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) in their document "CIM Definition Standards".

In order to identify those blocks in the block model that could reasonably be considered as a mineral resource, the block model was filtered by a cut-off grade of 5 g/t gold equivalent (AuEq). The gold-equivalent calculation used is: AuEq = Au + (Ag/53). These blocks were then classified as Measured, Indicated or Inferred and reported (Table 1.1 and Table 1.2).

Classification was applied based on geological confidence, data quality and grade variability. Areas classified as Measured Resources were those within the well informed portion of West Zone where the resource is informed by 5 m by 5 m or 5 m by 10 m spaced drilling. Areas classified as Indicated Resources are informed by 20 m by 20 m to 20 m by 40 m drilling within West Zone and VOK. The remainder of the mineral resource is classified as Inferred Resource where there is some drilling information and the blocks lie within the mineralized interpretation. Areas where there is no informing data and/or the lower grade material is outside of the mineralized interpretation are not classified as a part of the mineral resource.

Table 1.1 VOK Mineral Resource Estimate Based on a Cut-off Grade of 5 g/t AuEq – November 2012(1)(4)

Category	Tonnes (Mt)	Gold (g/t)	Silver (g/t)	Contained(3) Gold (Moz)	Contained(3) Silver (Moz)
Indicated	16.1	16.4	14.1	8.5	7.3
Inferred(2)	5.4	17.0	15.7	2.9	2.7

Notes: (1) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues. The mineral resources in this news release were classified using the CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
(2) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.
(3) Contained metal and tonnes figures in totals may differ due to rounding.
(4) The gold equivalent value is defined as AuEq = Au + Ag/53.

Table 1.2 West Zone Mineral Resource Estimated Based on a Cut-off Grade of 5 g/t AuEq – April 2012[(1)(4)(5)]

Category	Tonnes (Mt)	Gold (g/t)	Silver (g/t)	Contained[(3)] Gold (Moz)	Contained[(3)] Silver (Moz)
Measured	2.4	5.85	347	0.5	26.8
Indicated	2.5	5.86	190	0.5	15.1
M+I	4.9	5.85	267	0.9	41.9
Inferred[(2)]	4.0	6.44	82	0.8	10.6

Note: (1), (2), (3), and (4) See footnotes to Table 1.1.

1.5 MINERAL PROCESSING AND METALLURGICAL TESTING

1.5.1 METALLURGICAL TESTING

Several metallurgical test programs were carried out to investigate the metallurgical performance of the mineralization. The main test work was completed from 2009 to early 2013. The samples tested were generated from various drilling programs. The metallurgical test programs conducted on the Brucejack mineralization included head sample characteristics, gravity concentration, gold/silver bulk flotation, cyanidation and the determination of various process related parameters. The early test work focused on developing the flowsheet for gravity concentration, bulk flotation, and flotation concentrate cyanidation. The test work also studied the metallurgical responses of the samples to the gravity concentration flowsheet for gravity concentration followed by whole ore leaching. The later test work concentrated on the gravity-flotation concentration flowsheet.

In general, the VOK Zone and West Zone mineralization is moderately hard. The mineral samples tested responded well to the conventional combined gravity and flotation flowsheet. The gold in the mineralization was amenable to centrifugal gravity concentration. On average, 40 to 50% of the gold in the samples were recovered by the gravity concentration. The flotation tests results indicated that bulk flotation can effectively recover the gold remained in the gravity concentration tailings using potassium amyl xanthate (PAX) as a collector at the natural pH. Two stages of cleaner flotation would significantly upgrade rougher flotation concentrate. The gold in the mineralization showed better metallurgical performance, compared to silver. On average, approximately 96 to 97% of the gold and 91 to 92% of the silver were recovered to the gravity concentrate and bulk flotation concentrate at the grind size of 80% passing approximately 70 to 80 µm. There was a significant variation in metallurgical performances among the samples tested. This may be a result of the nugget gold effect.

Cyanide leach tests were also conducted to investigate the gold and silver extractions from various samples, including head samples, flotation concentrates, flotation tailings and gravity concentrates. In general, most of the sample responded reasonably well to direct cyanidation, excluding a few of samples containing higher contents of graphite

(carbon), arsenic, or electrum. Cyanide leach process has not been recommended for the study. Further tests are required to evaluate the responses of the mineralization to cyanidation.

The test results suggest that the gold and silver recovery flowsheet for the mineralization should include gravity concentration, bulk rougher and scavenger flotation, rougher and scavenger concentrate regrinding, followed by cleaner flotation.

1.5.2 Mineral Processing

The process flowsheet developed for the Brucejack mineralization is a combination of conventional bulk sulphide flotation and gravity concentration to recover gold and silver. The processing plant will produce a gold-silver bearing flotation concentrate and gold-silver doré that will be produced by melting the gravity concentrate produced from the gravity concentration circuits. Based on the LOM average, the recovery process is estimated to produce approximately 4,300 kg of gold and 1,500 kg of silver as doré per year and 42,000 t of gold-silver bearing flotation concentrate per year from the mill feed, grading 12.0 g/t gold and 57.9 g/t silver. The estimated gold recoveries to the doré and flotation concentrate are 41.6% and 54.9%, respectively, totalling 96.5%. The estimated silver recoveries reporting to the doré and flotation concentrate are 3.0% and 86.6%, respectively, totalling 89.6%. The LOM average gold and silver contents of the flotation concentrate are anticipated to be approximately 130 g/t gold and 1,000 g/t silver. The flotation concentrate will be shipped off site to a smelter for further treatment to recover the gold and silver.

The process plant will consist of:

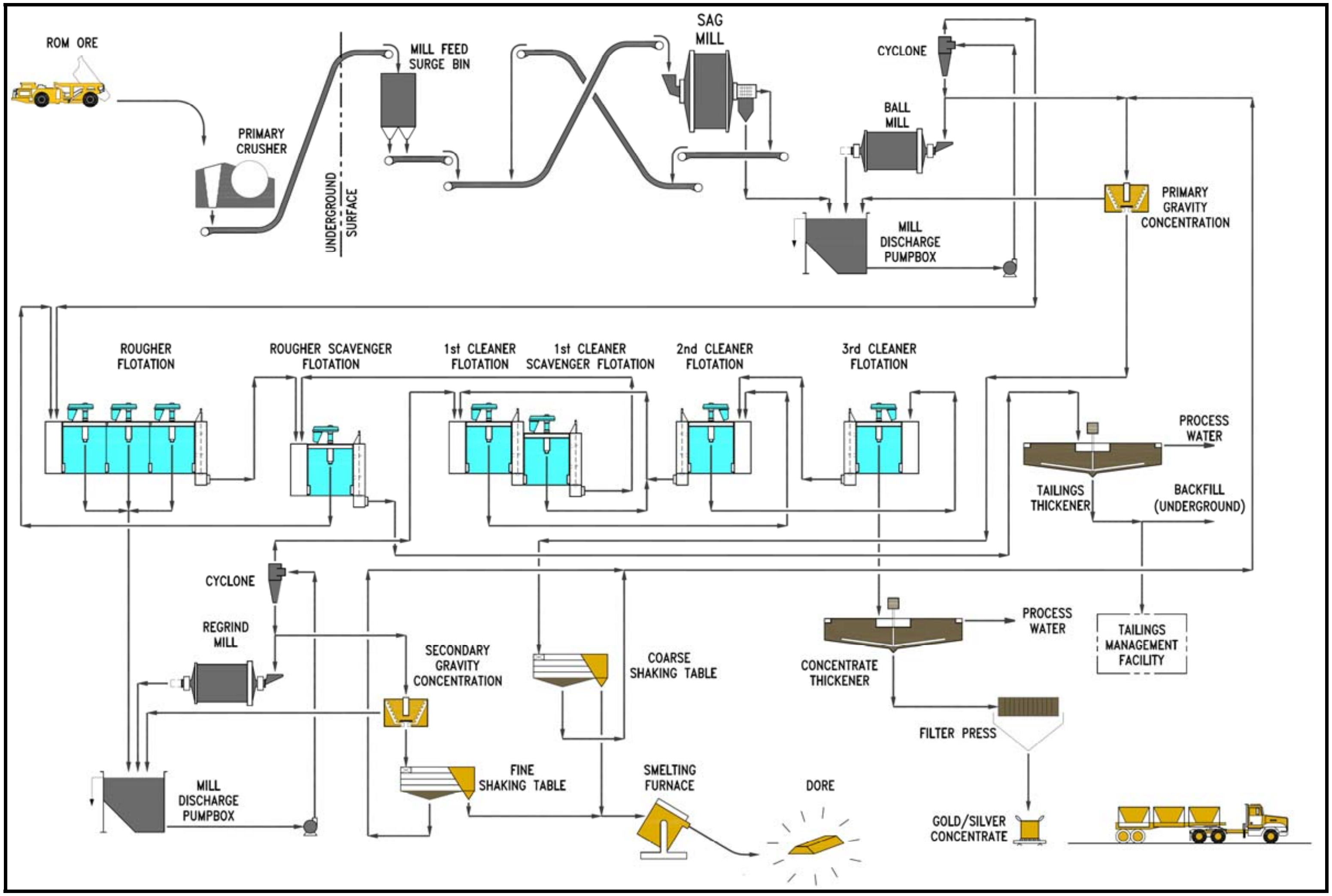
- one stage of crushing (located underground)
- a surge bin with a live capacity of 2,500 t on surface
- a semi-autogeneous grinding (SAG) mill and ball mill primary grinding circuit integrated with gravity concentration
- rougher flotation followed by rougher flotation concentrate regrinding
- cleaner flotation processes.

A gravity concentration circuit will also be incorporated in the bulk concentrate regrinding circuit. The final flotation concentrate will be dewatered, bagged, and trucked to the transload facility in Terrace, BC. It will be loaded in bulk form into rail cars for shipping to a smelter located in eastern Canada. The gravity concentrate will be refined in the gold room on site to produce gold-silver doré.

A portion of the flotation tailings will be used to make paste for backfilling the excavated stopes in the underground mine, and the balance will be stored in Brucejack Lake. The water from the thickener overflows will be recycled as process make-up water. Treated water from the water treatment plant will be used for mill cooling, gland seal service, reagent preparation, and make-up water.

The simplified flowsheet for the operation is shown in Figure 1.1.

Figure 1.1 Simplified Process Flowsheet



1.6 MINERAL RESERVE ESTIMATES

A net smelter return (NSR) cut-off grade of $180/t of ore was used to define the mineral reserves (as used in previous studies). For the feasibility study, the average site operating costs over the LOM are calculated as $157.7/t, providing a minimum $22.3/t operating margin on ore mined.

The NSR for each block in the resource model was calculated as the payable revenue for gold and silver less the costs of refining, concentrate treatment, transportation and insurance. The metal price assumptions are US$1,350/oz gold and US$22/oz silver.

Table 1.3 shows the dilution and recovery factors used in the mineral reserve estimation.

Table 1.3 Dilution Factors and Recovery Factors by Type of Excavation

Type of Excavation	Dilution Factor* (%)	Recovery Factor* (%)
Primary Stopes	6.8	97.5
Secondary Stopes	15.2	92.5
Sill Pillar Stopes**	15.2	75.0
Ore Cross-cuts	4.0	100.0
Production Slashing	7.5	100.0

Notes: *Expressed on a weight basis.
**Includes stope ore to 30 m beneath the surface crown pillar.

The mineral reserves are delineated in an orebody consisting of numerous independent lenses in the VOK Zone and two distinct lenses in the West Zone (Table 1.4). The mineral reserves were developed from the resource model, "bjbm_1211_v2_cut" provided by Snowden–on behalf of Pretivm–to AMC in November 2012.

Table 1.4 Brucejack Mineral Reserves*, by Zone and by Reserve Category

Zone		Ore Tonnes (Mt)	Grade		Metal	
			Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
VOK Zone	Proven	-	-	-	-	-
	Probable	15.1	13.6	11	6.6	5.3
	Total	**15.1**	**13.6**	**11**	**6.6**	**5.3**
West Zone	Proven	2.0	5.7	309	0.4	19.9
	Probable	1.8	5.8	172	0.3	10.1
	Total	**3.8**	**5.8**	**243**	**0.7**	**30.0**
Total Mine	Proven	2.0	5.7	309	0.4	19.9
	Probable	17.0	12.8	28	7.0	15.4
	Total	**19.0**	**12.0**	**58**	**7.3**	**35.3**

Notes: *Rounding of some figures may lead to minor discrepancies in totals.
*Based on Cdn$180/t cut-off grade, US$1,350/oz gold price, US$22/oz silver price, Cdn$/US$ exchange rate = 1.0

1.7 MINING METHODS

The underground mine design supports the extraction of 2,700 t/d of ore via transverse longhole open stoping (LHOS) and longitudinal LHOS. Paste backfill and modern trackless mobile equipment will be used. Mine access will be by a main decline from a surface portal close to the concentrator. A parallel decline will be dedicated to conveying crushed ore directly to the concentrator via a 650 m long conveyor. There will be a two-year pre-production development period, with steady-state production being reached in Year 2 of a 22-year LOM. The highest value ore will be targeted in the early production years. Steady-state production from years 2 through 18 will average about 980,000 t/a.

Geotechnical designs and recommendations are based on the results of site investigations, and geotechnical assessments that include rock mass characterization, structural geology interpretations, excavation and pillar stability analyses, and ground support design.

The groundwater flow system was conceptualized to provide inflow estimates to mine workings. These estimates referenced results of site investigations and hydrogeologic testing and were used to size dewatering equipment and as input to the process water balance.

During the pre-production period, most of the mobile equipment for development and stoping work will be supplied by the Owner and operated by a contractor. Key equipment requirements will include jumbos, load-haul-dumps (LHDs), haulage trucks, bolters, shotcrete sprayers, a long hole drill and a cable bolter. Raise development will be contracted out.

Manpower will consist of technical staff, mining crews, mechanics, electricians, and other support personnel. Pre-production manpower will be supplied by contractor, except for technical support. Manpower attains 349 personnel at full production, with up to 163 personnel on site at any given time.

Infrastructure design is for a mine life over 20 years. Electric power use will be maximized.

The ventilation system is designed to meet BC regulations. Permanent surface fans will be located at the portals of the twin, intake declines. All intake air entering the mine will be heated above freezing point.

Paste fill distribution design is based on a dual pumping system. A positive displacement pump in the paste fill plant will provide paste to all of the West Zone and the lower zones of the VOK. The paste plant pump will also feed a booster pump located near the crusher station at the bottom of the conveyor ramp and near to the main entrance to the VOK area on the 1,330 Level.

Ore will be trucked from working areas to an underground crusher and then transferred to surface via transfer belt to a 42 in main conveyor. Waste will be trucked to surface waste piles.

The mine will be dewatered using a dirty water system of sumps and pumps. Submersible and centrifugal pumps will be used for development and permanent mine operations. For underground worker safety, both permanent and portable refuge stations are planned. The emergency warning system will include phones, cap lamp warning system, and stench gas.

The total project mining capital, including a 10% contingency, is estimated at $210 million. Sustaining mining capital of $265 million has been estimated for the production period. The total underground operating cost over the LOM is estimated to be $1,769 million, at an average LOM cost of $94.40/t.

1.8 PROJECT INFRASTRUCTURE

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 ensure efficient and convenient operation of the mine haul fleet. Figure 1.2 illustrates the overall site layout for the Project. Figure 1.3 illustrates the mill site layout and Figure 1.4 illustrates the Knipple Transfer Station facility layout.

Project infrastructure will include:

- a 79 km access road at Highway 37 and travelling westward to Brucejack Lake with the last 12 km of access road to the mine site traversing the main arm of the Knipple Glacier
- internal site roads and pad areas
- grading and drainage
- avalanche hazard assessment
- transmission line
- ancillary facilities
- water supply and distribution
- water treatment plant
- waste disposal
- tailings delivery system
- Brucejack outlet control
- communications
- power supply and distribution
- fuel supply and distribution
- off-site infrastructure including the Bowser Airstrip and Camp and the Knipple Transfer Station facilities.

Figure 1.2 Overall Site Layout

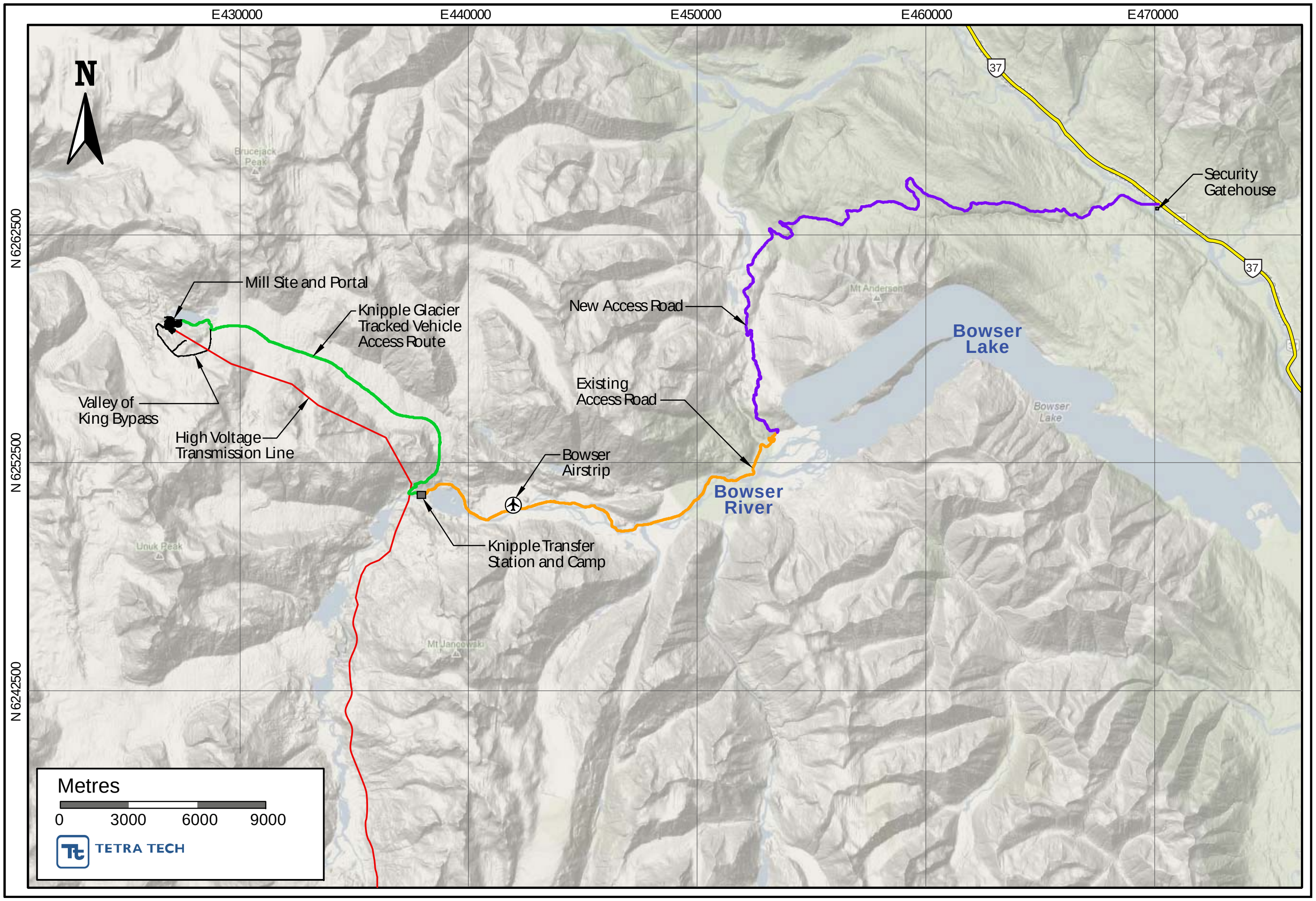


Figure 1.3 Mill Site Layout

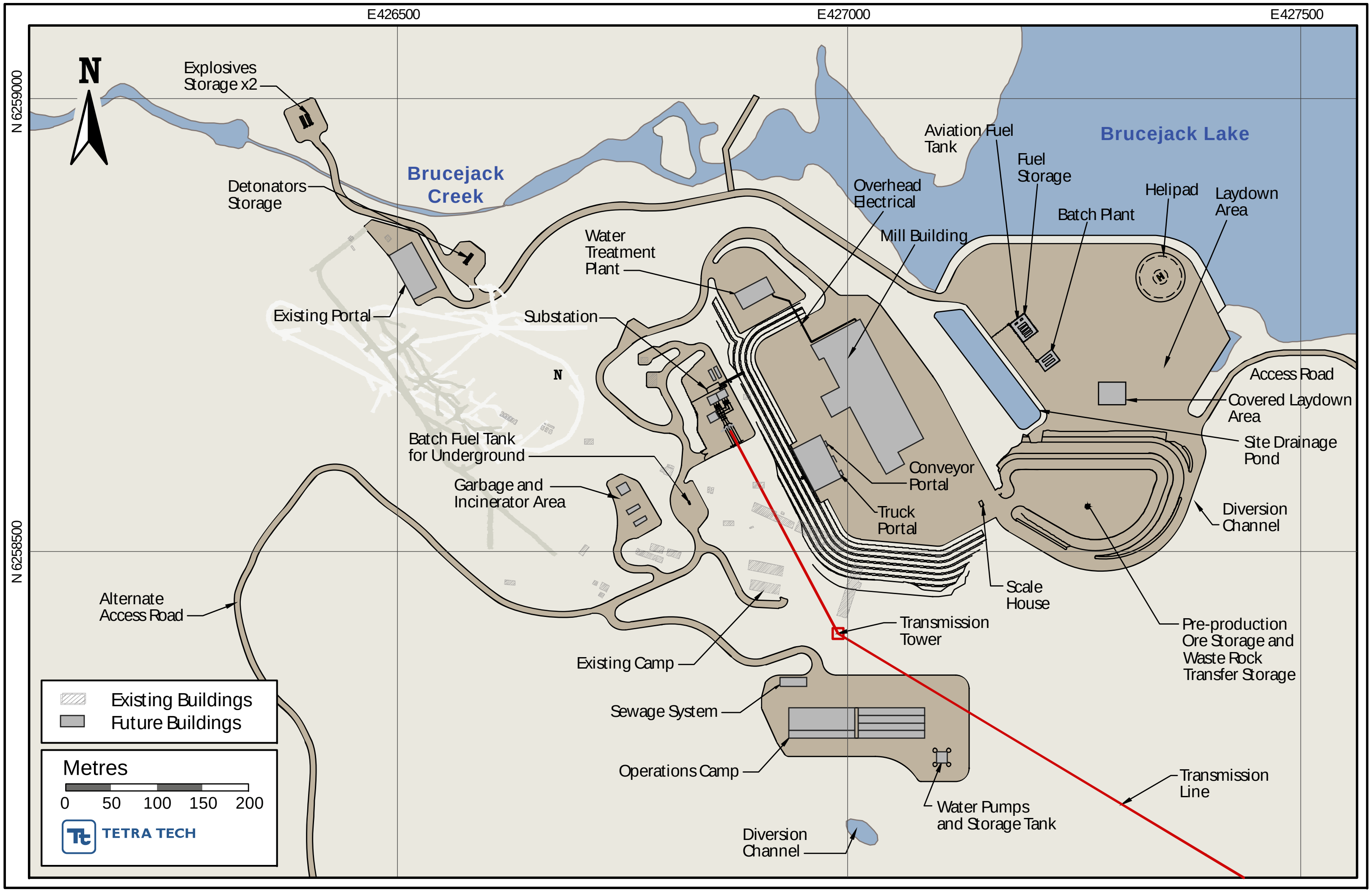
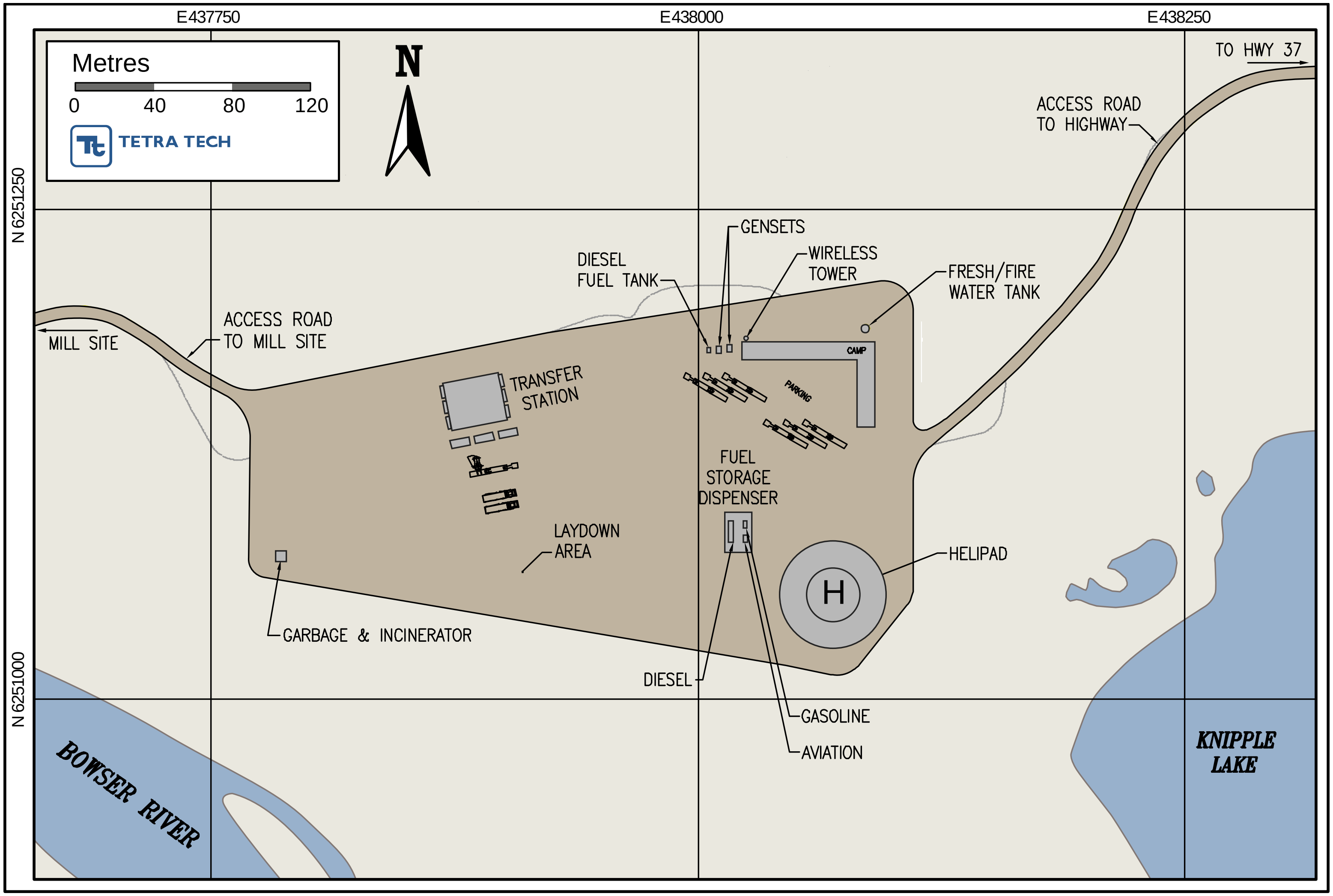


Figure 1.4 Knipple Transfer Station Facility Layout



1.8.1 AVALANCHE HAZARD ASSESSMENT

An avalanche hazard assessment has been completed for the Project. Facilities and access routes are exposed to approximately 15 avalanche paths or areas. Avalanche magnitude varies between Size 2 and 4. Avalanche frequency varies between annual and 1:100 years. Potential consequences of avalanches reaching the Brucejack mine facilities, transmission line, worksites, and roads include damage to infrastructure, worker injury (or fatality), and project delays. Potential consequences of static snow loads on transmission towers include damage to towers and foundations, and potential loss of electrical service to the mine. Without mitigation to the effects of avalanches and static snow loading, there is a high likelihood of some of the above consequences affecting operations on an annual basis.

Avalanche mitigation for the Project includes location planning, in order to avoid placement of facilities in avalanche hazard areas. For areas where personnel and infrastructure may be exposed, an avalanche management program will be implemented for mine operations during avalanche season (October through June). The program will utilize an Avalanche Technician team to determine periods of elevated avalanche hazard and provide recommendations for closures of hazard areas. The options for reducing control include explosive control, or waiting for natural settlement. Areas that are expected to have increased frequency of hazard and consequences will be evaluated for the installation of the remote avalanche control system (RACS) in order to allow for avalanche explosive control during reduced visibility (darkness and during storms). An allowance has been made in the capital and operating cost estimates for six RACSs.

1.8.2 TRANSMISSION LINE

For the Brucejack transmission line, Pretivm retained Valard to review potential routes and develop an initial design for the transmission line to the Project site, based on Valard's current experience in the area. To this end, Valard reviewed potential routes and determined the preferred route to be an extension from an existing transmission line from a hydro generation facility to the south (near Stewart, BC) to the Project site. Based on the terrain and the expected construction conditions, single metal monopole towers are recommended for the design. Site review indicates that the hazards in the area can be avoided through diligent siting of the tower structures as well as through an active snow avalanche program.

1.8.3 TAILINGS DELIVERY SYSTEM

Approximately one half of the tailings produced by mine operations will be stored underground as paste backfill and approximately one half will be placed on the bottom of Brucejack Lake. Tailings will be pumped from the tailings thickener at the process plant by slurry pipeline to the deepest location in the lake (80 m depth). A deposit of solids will intentionally be allowed to build over the end of the outfall. The solids will act as a filter to minimize the transport of fine particulate solids to the surface layer of Brucejack Lake with the goal of minimizing total suspended solids (TSS) concentrations in the lake

outflow. This approach–discharge through a deposit of tailings–has successfully reduced suspended solids concentrations at other projects.

Fine particulate solids may also be suspended in the lake surface layer if fine waste rock is placed in the lake. Investigations on minimizing or eliminating this source of suspended solids in the lake outflow are underway.

1.9 ENVIRONMENTAL

Pretivm is committed to operating the mine in a sustainable manner and according to their guiding principles. Every reasonable effort will be made to minimize long-term environmental impacts and to ensure that the Project provides lasting benefits to local communities while generating substantial economic and social advantages for shareholders, employees, and the broader community. Pretivm respects the traditional knowledge of the Aboriginal peoples who have historically occupied or used the Project area. The Project area ecosystem is relatively undisturbed by human activities. Pretivm's objective is to retain the current ecosystem integrity as much as possible during the construction and operation of the Project. Upon closure and reclamation of the Project, the intent will be to return the disturbed areas to a level of productivity equal to or better than that which existed prior to project development, and for the end configuration to be consistent with pre-existing ecosystems to the extent possible.

1.10 CAPITAL COSTS

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $663.5 million. A summary breakdown of the initial capital cost, including direct costs, indirect costs, Owner's costs, and contingency is provided in Table 1.5.

Table 1.5 Summary of Initial Capital Cost

Major Area	Area Description	Capital Cost ($ million)
Direct Costs		
11	Mine Site	32.7
21	Mine Underground	174.5
31	Mine Site Process	80.1
32	Mine Site Utilities	23.7
33	Mine Site Facilities	43.7
34	Mine Site Tailings	3.5
35	Mine Site Temporary Facilities	10.2
36	Mine Site (Surface) Mobile Equipment	14.3
84	Off Site Infrastructure	69.1
Subtotal Direct Costs		451.8
91	Indirect Costs	125.0

table continues...

Major Area	Area Description	Capital Cost ($ million)
98	Owner's Costs	22.3
99	Contingencies	64.4
Total Initial Capital Cost		663.5

Note: Numbers may not add due to rounding.

This estimate is a Class 4 feasibility cost estimate prepared in accordance with the standards of the Association for the Advancement of Cost Engineering International (AACE). There was no deviation from the AACE's recommended practices in the preparation of this estimate.

This feasibility estimate was prepared with a base date of Q2 2013 and does not include any escalation beyond this date. The quotations used for this feasibility study estimate were obtained in Q2 2013, and have a validity period of 90 days.

The capital cost estimate uses Canadian dollars as the base currency. Foreign exchange rates were applied as required. Duties and taxes and taxes are not included in the estimate. This estimate was developed based largely on first principles and was prepared from a design, planning, and cost basis.

1.11 OPERATING COSTS

The total LOM average operating cost for the Project is estimated at $156.46/t ore milled which includes for:

- mining
- process
- material re-handling in Year 1 for the stockpiled ore produced during pre-production
- general and administration (G&A)
- surface services
- backfill, including paste preparation
- water treatment.

The operating costs exclude sustaining capital costs, off-site costs (such as shipping and smelting costs), taxes, permitting costs, or other government imposed costs, unless otherwise noted.

A total of 542 personnel are projected to be required for the Project. The unit cost estimates are based on the LOM ore production and a mine life of 22 years. The currency exchange rate used for the estimate is 1:1 (Cdn$:US$). The operating cost for the Project has been estimated in Canadian dollars within an accuracy range of ±15%. A

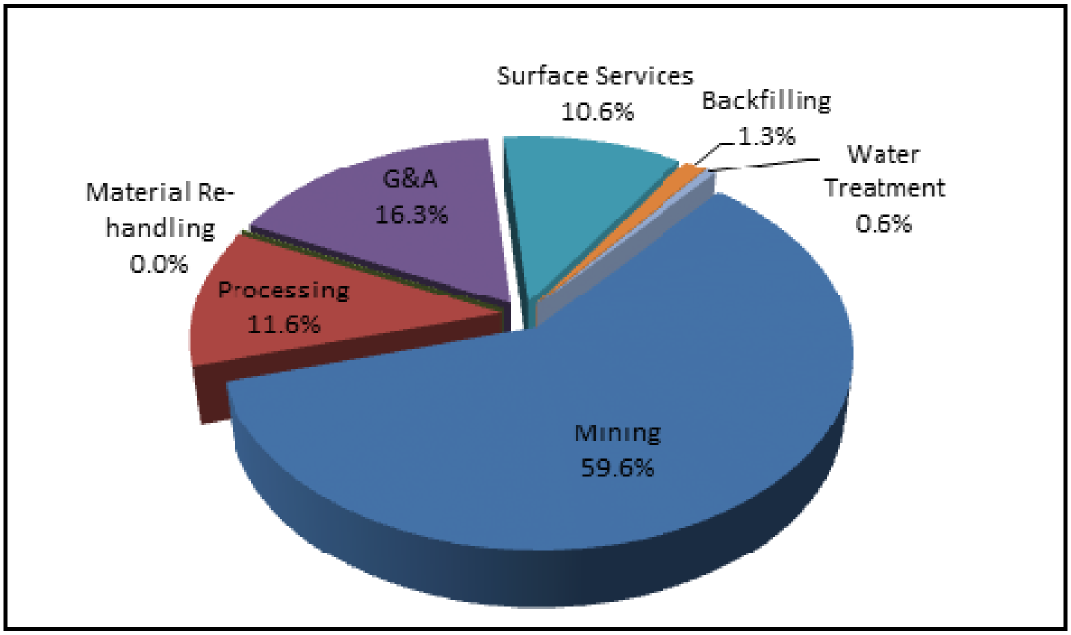
summary of the overall operating cost is presented in Table 1.6. The cost distribution is illustrated in Figure 1.5.

Table 1.6 Overall Operating Cost

Area	Personnel	Unit Operating Cost ($/t milled)
Mining*	316**	93.18
Processing	95	18.16
Material Re-handling***	Contract	0.07
G&A	43	25.47
Surface Services	78	16.53
Backfilling	6	2.10
Water Treatment	4	0.95
Total	**542**	**156.46**

Notes: *Average LOM mining cost including crushing cost and cement cost for backfill; if excluding the ore mined during preproduction, the estimated unit cost is $94.40/t
**316 workers during Year 1 to 14 and then reduce to 167 workers at the end of the mine life.
***Material re-handling cost is the LOM average cost, which will occur in Year 1 only. The operation is assumed to be contracted with approximately eight workers required.

Figure 1.5 Overall Operating Cost Distribution



1.12 ECONOMIC ANALYSIS

Tetra Tech prepared an economic evaluation of the Project based on a pre-tax financial model. For the 22-year LOM and 18.99 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 42.9% internal rate of return (IRR)
- 2.1-year payback on the US$663.5 million initial capital
- US$2,687 million net present value (NPV) at 5% discount rate.

A post-tax economic evaluation of the Project was prepared with the inclusion of applicable taxes (Section 22.0).

The following post-tax financial parameters were calculated:

- 35.7% IRR
- 2.2-year payback on the US$663.5 million initial capital
- US$1,763 million NPV at 5% discount rate.

The base case metal prices used for this study are as follows:

- gold – US$1,350/oz
- silver – US$20.00/oz
- exchange rate – 1.00:1.00 (US$:Cdn$).

1.13 PROJECT EXECUTION PLAN

The Project will take approximately 37 months to complete from the start of basic engineering, through construction, to introduction of first material into the mill. A further three to four months is planned for commissioning and production ramp-up. The Project execution schedule was developed to a Level 2 detail of all activities required to complete the Project.

The Project will transition from the study phase to basic engineering in Q3 2013 and will move forward in the following phases:

- Stage I – early works including mine development, the environmental assessment certificate (EAC) application, permitting, access road upgrades, preliminary power transmission line right-of-way (ROW), basic engineering, and the procurement of long-lead equipment.
- Stage II – full project execution (following permit approval), including detailed engineering, procurement, construction team mobilization, construction, and commissioning.

The Project schedule identifies the following significant key milestone dates (Table 1.7) from feasibility completion to project handover.

Table 1.7 Key Milestone Dates

Year	Quarter	Activity
2013	2	Feasibility Study Completion
2013	3	Start of Basic Engineering
2014	1	EPCM Award
2015	1	Detailed Engineering Completion
2014	3	Start of Stage I Early Infrastructure Construction Works
2015	1	Start of Stage II Mine Site Surface Construction
2015	4	Mechanical Completion Stage I Works
2016	2	Mechanical Completion Stage II Works
2016	3	Underground Development Completion
2016	3	Mine Site Commissioning Completion
2016	3	Project Handover

Note: EPCM = engineering, procurement, construction management

1.14 Conclusions and Recommendations

Pretivm will continue to advance engineering at the Project in support of the ongoing permitting process, and anticipates filing its application for an Environmental Assessment Certificate later this year. After obtaining permits, and subject to a production decision, Pretivm anticipates commencing construction of the mine in the second half of 2014. Detailed recommendations for the Project can be found in Section 26.0

2.0 INTRODUCTION

Pretivm commissioned Tetra Tech to complete a feasibility study on the Project in accordance with CIM Best Practices, and to disclose them in a technical report prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects, Companion Policy 43-101CP, and Form 43-101F1.

All mines acts regulations with respect to health, safety, and environmental considerations have been taken into account and incorporated into the feasibility designs and relevant cost estimates. In addition, the designs take into account the geological location of the Project.

The following consultants were commissioned to complete the component reports for the purposes of the feasibility study:

- Tetra Tech: overall project management; mineral processing and metallurgical testing; recovery methods; access infrastructure; internal site roads and pad areas; grading and drainage; ancillary facilities; water supply and distribution; water treatment plant; communications; power supply and distribution; fuel supply and distribution; off-site infrastructure; market studies and contracts; capital cost estimate; processing operating cost estimate; financial analysis; and project execution plan
- Snowden: property description and location; accessibility, climate, and physiology; history; geological setting and mineralization; deposit types; exploration; drilling; sample preparation and analysis; data verification; adjacent properties; and mineral resource estimates
- AMC: mining including mine capital and operating cost estimates; and mineral reserve estimates
- Rescan: environmental studies, permits, and social or community impacts; and tailings delivery system
- BGC: geotechnical design, mine hydrogeological/groundwater; waste disposal; Brucejack outlet control; environmental water management and water quality; and ARD and ML
- Alpine Solutions: avalanche hazard assessment
- Valard: transmission line
- P&C: paste fill distribution.

2.1 QUALIFIED PERSONS

The qualified persons (QPs) responsible for this report are listed in Table 2.1. The following QPs completed a site visit of the Property:

- Ivor W.O. Jones, M.Sc., CP, FAusIMM, completed a site visit on February 15, 2012 for two days.
- Dave Ireland, C.Eng., P.Eng., completed a site visit on August 7, 2012 for one day.
- John Huang, Ph.D., P.Eng., completed a site visit on August 7, 2012 for one day.
- Pierre Pelletier, P.Eng., completed a site visit on August 7, 2012 for one day.
- Paul Greisman, Ph.D., P.Eng. complete a site visit on August 17, 2010 for one day.
- Michael Wise, P.Eng., completed a site visit on August 5, 2012 for one day.
- Brian Gould, P.Eng., completed a site visit on April 29, 2013 for two days.
- Hamish Weatherly, P.Geo., completed a site visit on August 7, 2012 for one day.
- Colm Keogh, P.Eng., completed a site visit on October 24, 2012 for one day.
- Catherine Schmid, M.Sc., P.Eng., completed a site visit in February 2012 for seven days.
- Brent McAfee, P.Eng., completed a site visit from June 6 to 12, 2012 for seven days.
- Virginia Cullen, M.Eng., P.Eng., completed a site visit on July 6, 2012 for three days.

Table 2.1 Summary of QPs

Report Section		Company	QP
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
4.0	Property Description and Location	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
6.0	History	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
7.0	Geological Setting and Mineralization	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
8.0	Deposit Types	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
9.0	Exploration	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
10.0	Drilling	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
11.0	Sample Preparation, Analyses and Security	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
12.0	Data Verification	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM

table continues...

Report Section			Company	QP
13.0	Mineral Processing and Metallurgical Testing		Tetra Tech	John Huang, Ph.D., P.Eng.
14.0	Mineral Resource Estimate		Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
15.0	Mineral Reserve Estimate		AMC	Colm Keogh, P.Eng.
16.0	Mining Methods		AMC/ BGC P&C	Colm Keogh, P.Eng./ Mo Molavi, P.Eng./ Catherine Schmid, M.Sc., P.Eng./ Virginia Cullen, M.Eng., P.Eng./ Maureen McGuiness, P.Eng.
17.0	Recovery Methods		Tetra Tech	John Huang, Ph.D., P.Eng.
18.0	Project Infrastructure		-	-
	18.1	Overview	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
	18.2	Site Geotechnical	BGC	Brent McAfee, P.Eng.
	18.3	Access	Tetra Tech	Dave Ireland, P.Eng.
	18.4	Internal Site Roads and Pad Areas	Tetra Tech	Mike Chin, P.Eng.
	18.5	Grading and Drainage	Tetra Tech	Mike Chin, P.Eng.
	18.6	Avalanche Hazard Assessment	Alpine Solutions	Brian Gould, P.Eng.
	18.7	Transmission Line	Valard	Michael Wise, P.Eng.
	18.8	Ancillary Facilities	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
	18.9	Water Supply and Distribution	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
	18.10	Water Treatment Plant	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
	18.11	Waste Rock Disposal	BGC	Brent McAfee, P.Eng.
	18.12	Tailings Delivery System	Rescan	Paul Greisman, Ph.D., P.Eng.
	18.13	Brucejack Lake Suspended Solids Outflow Control	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
	18.14	Communications	Tetra Tech	Clayton Richards, P.Eng.
	18.15	Power Supply and Distribution	Tetra Tech	Wayne E. Scott, P.Eng.
	18.16	Fuel Supply and Distsribution	Tetra Tech	Ali Farah, P.Eng.
	18.17	Off-site Infrastructure	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
19.0	Market Studies and Contracts		Tetra Tech	John Huang, Ph.D., P.Eng.
20.0	Environmental Studies, Permitting, and Social or Community Impact		Rescan/ BGC	Pierre Pelletier, P.Eng./ Hamish Weatherly, M.Sc., P.Geo./ Virginia Cullen, M.Eng., P.Eng.
21.0	Capital and Operating Costs		-	-
	21.1	Capital Cost Estimate	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
	21.2	Operating Cost Estimate	-	-
	21.2.1	Summary	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.2.2	Mining Operating Costs	AMC	Colm Keogh, P.Eng.
	21.2.3	Process Operating Costs	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.2.4	Backfilling Operating Costs	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.2.5	Water Treatment Operating Costs		John Huang, Ph.D., P.Eng.
	21.2.6	General and Administrative and Suface Services		John Huang, Ph.D., P.Eng.
22.0	Economic Analysis		Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.

table continues…

Report Section		Company	QP
23.0	Adjacent Properties	Snowden	Ivor W.O. Jones, M.Sc., CP, FAusIMM
24.0	Other Relevant Data	Tetra Tech	Dave Ireland, C.Eng., P.Eng.
25.0	Interpretations and Conclusions	All	Sign-off by Section
26.0	Recommendations	All	Sign-off by Section
27.0	References	All	Sign-off by Section
28.0	Certificates of Qualified Persons	All	Sign-off by Section

2.2 INFORMATON AND DATA SOURCES

A complete list of references is provided in Section 27.0.

2.2.1 SNOWDEN

Pretivm has provided to Snowden the data used as the basis of this report from geological mapping, sampling and various drilling campaigns.

This report is based, in part, on internal company technical reports, and maps, published government reports, company letters and memoranda, and public information as listed in Section 27.0. Several sections from reports authored by other consultants have been directly quoted in this report, and are so indicated in the appropriate sections. Snowden has not conducted detailed land status evaluations, and has relied upon previous qualified reports, public documents and statements by Pretivm regarding property status and legal title to the Property.

3.0 RELIANCE ON OTHER EXPERTS

3.1 INTRODUCTION

The QPs who prepared this report relied on information provided by experts who are not QPs. The relevant QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

3.2 IVOR W. O. JONES, M.SC., CP, FAUSIMM

Snowden has only verified information relating to tenure in Section 4.0 through review of public information available through the Mineral Titles Branch of the BC Ministry of Energy, Mines, and Natural Gas (MEMNG) Mineral Titles Online (MTO) land tenure database. Snowden has relied upon this public information, as well as information from Pretivm, and has not undertaken an independent verification of title and ownership of the Property claims.

3.3 SABRY ABDEL HAFEZ, PH.D., P.ENG.

Sabry Abdel Hafez, Ph.D., P.Eng., relied on Sadhra & Chow LLP, concerning tax matters relevant to this report. The reliance is based on a letter to Pretivm titled "Insert and review of the income and mineral tax portions of the economic analysis prepared by Tetra Tech WEI Inc. ("Tetra Tech") in connection with the Feasibility Study Report (the "Report")" and dated June 10, 2013.

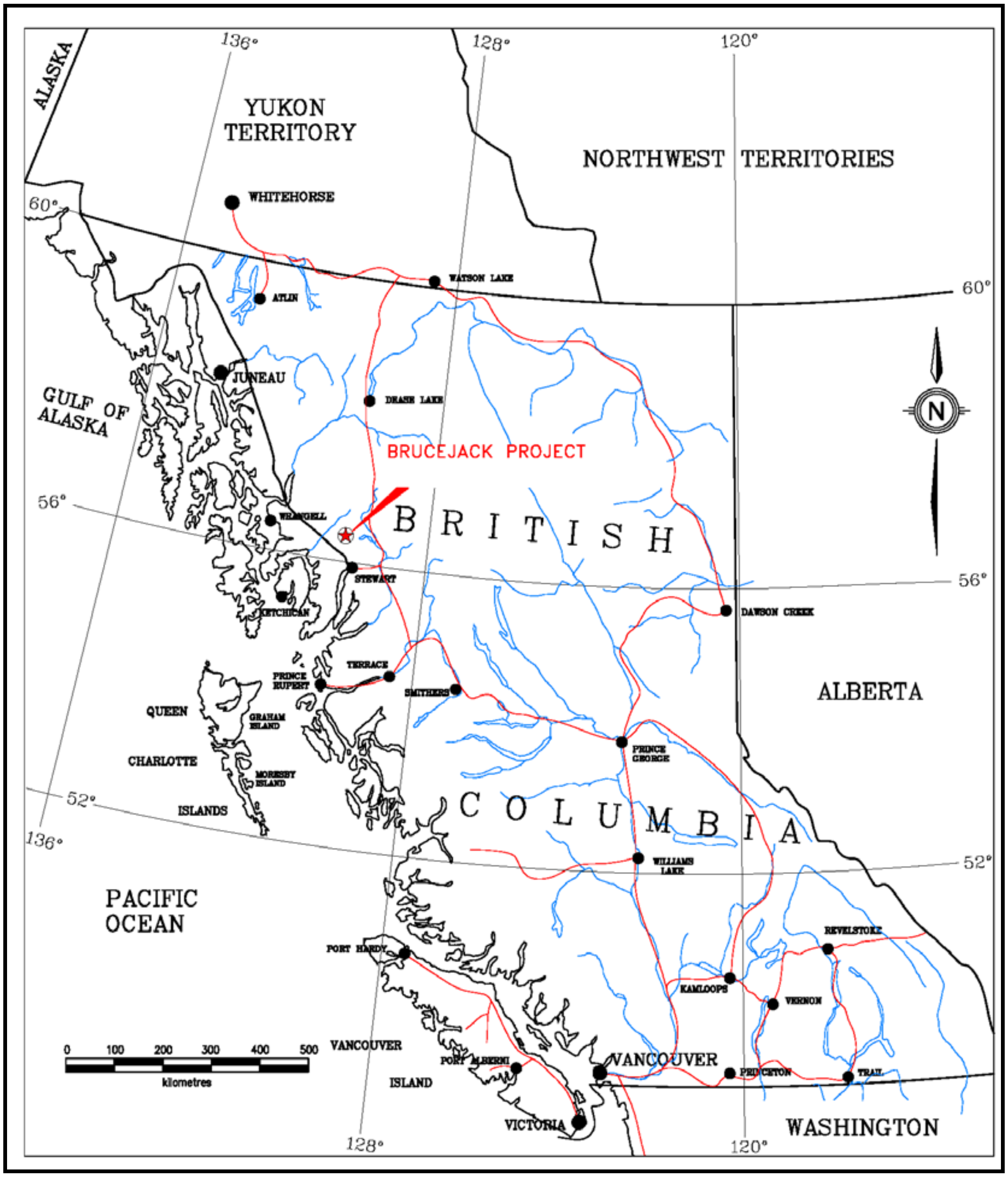
4.0 PROPERTY DESCRIPTION AND LOCATION

Information in this section has been excerpted and updated from Olssen and Jones (2012c). Olssen and Jones (2012a, b and c) excerpted and updated the information for this section in its report from Ghaffari et al. (2012).

4.1 LOCATION

The Property is situated approximately at 56°28′20″N Latitude by 130°11′31″W Longitude (Universal Transverse Mercator (UTM) 426,967E 6,258,719N North American Datum (NAD) 83 Zone 9), a position approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine (Figure 4.1). The Property coordinates used in this report are located relative to the NAD83 UTM coordinate system.

Figure 4.1 Property Location Map



4.2 TENURE

In 2010, pursuant to a purchase and sale agreement between Silver Standard (as the seller) and Pretivm (as the buyer), Silver Standard sold to Pretivm all of the issued shares of 0890693 BC Ltd., the owner of the Brucejack Project and the Snowfield Project. Subsequently, the name of 0890693 BC Ltd. changed to Pretivm Exploration Inc.

4.3 STATUS OF MINING TITLES

The Property consists of six mineral claims totalling 3,199.28 ha in area (Table 4.1 and Figure 4.1) and all claims are in good standing until January 31, 2024.

Table 4.1 List of Mineral Claims

Tenure No.	Tenure Type	Map No.	Owner	Pretivm Interest	Status	In Good Standing To	Area (ha)
509223	Mineral	104B	Pretivm Exploration Inc.	100%	Good	January 31, 2024	428.62
509397	Mineral	104B	Pretivm Exploration Inc.	100%	Good	January 31, 2024	375.15
509400	Mineral	104B	Pretivm Exploration Inc.	100%	Good	January 31, 2024	178.63
509463	Mineral	104B	Pretivm Exploration Inc.	100%	Good	January 31, 2024	482.57
509464	Mineral	104B	Pretivm Exploration Inc.	100%	Good	January 31, 2024	1,144.53
509506	Mineral	104B	Pretivm Exploration Inc.	100%	Good	January 31, 2024	589.78
Total	-	-	-	-	-	-	**3,199.28**

Information relating to tenure was verified by P&E by means of the public information available through the Mineral Titles Branch of the BC MEMNG MTO land tenure database. In 2005, the six mineral claims that comprise the Property were converted from 28 older legacy claims to BC's new MTO system. P&E and Snowden have relied upon this public information, as well as information from Pretivm, and have not undertaken an independent verification of title and ownership of the Property claims.

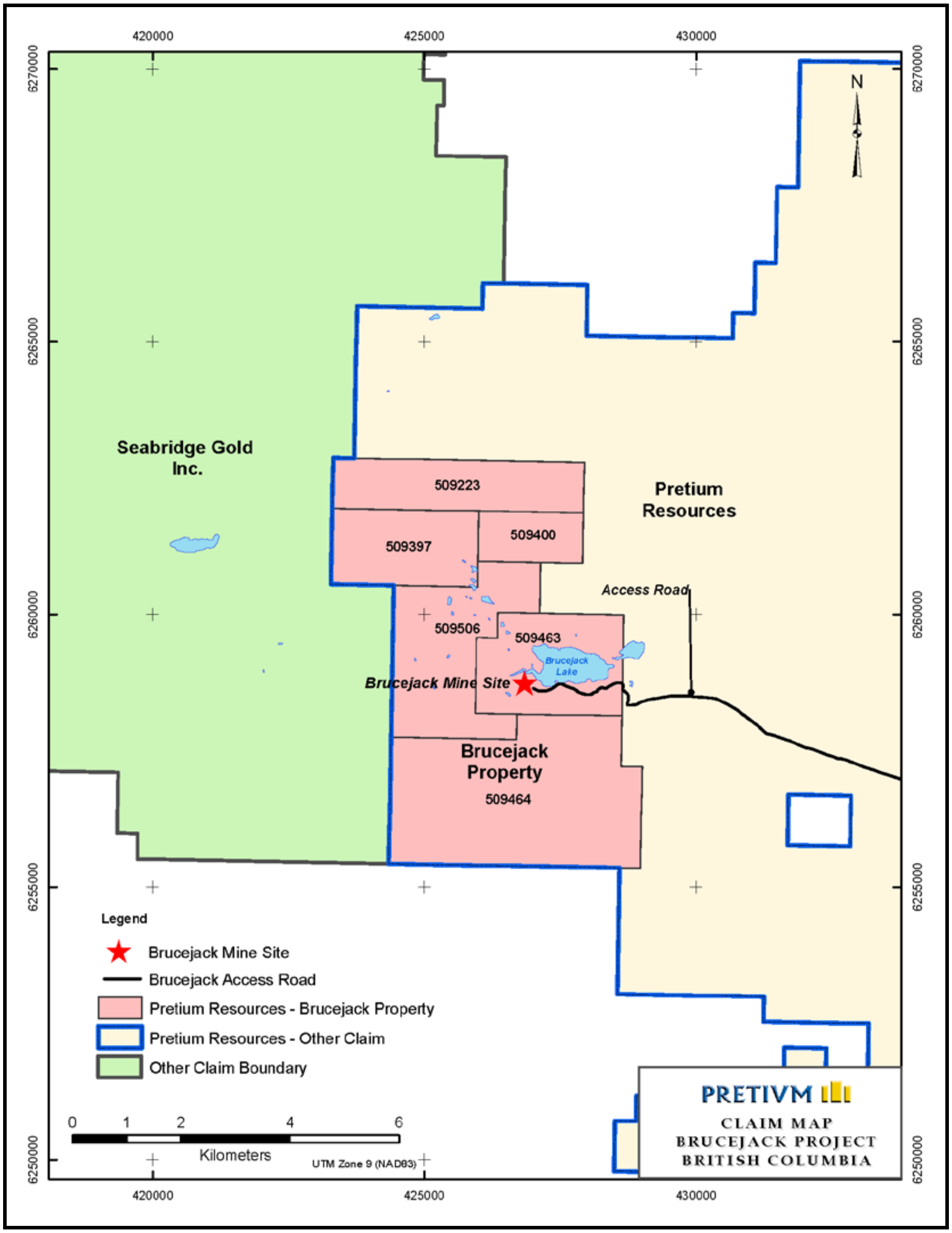
A legal land survey of the claims has not been undertaken.

There are no annual holding costs for any of the six mineral claims at this time, as the claims are paid up until January 31, 2024.

The majority of the Property falls within the boundaries of the Cassiar-Iskut-Stikine Land and Resource Management Plan (LRMP) area, with only a minor south-eastern segment of Mineral Claim No. 509506 falling outside this area. All claims located within the boundaries of the LRMP are considered areas of General Management Direction, with none of the claims falling inside any Protected or Special Management Areas.

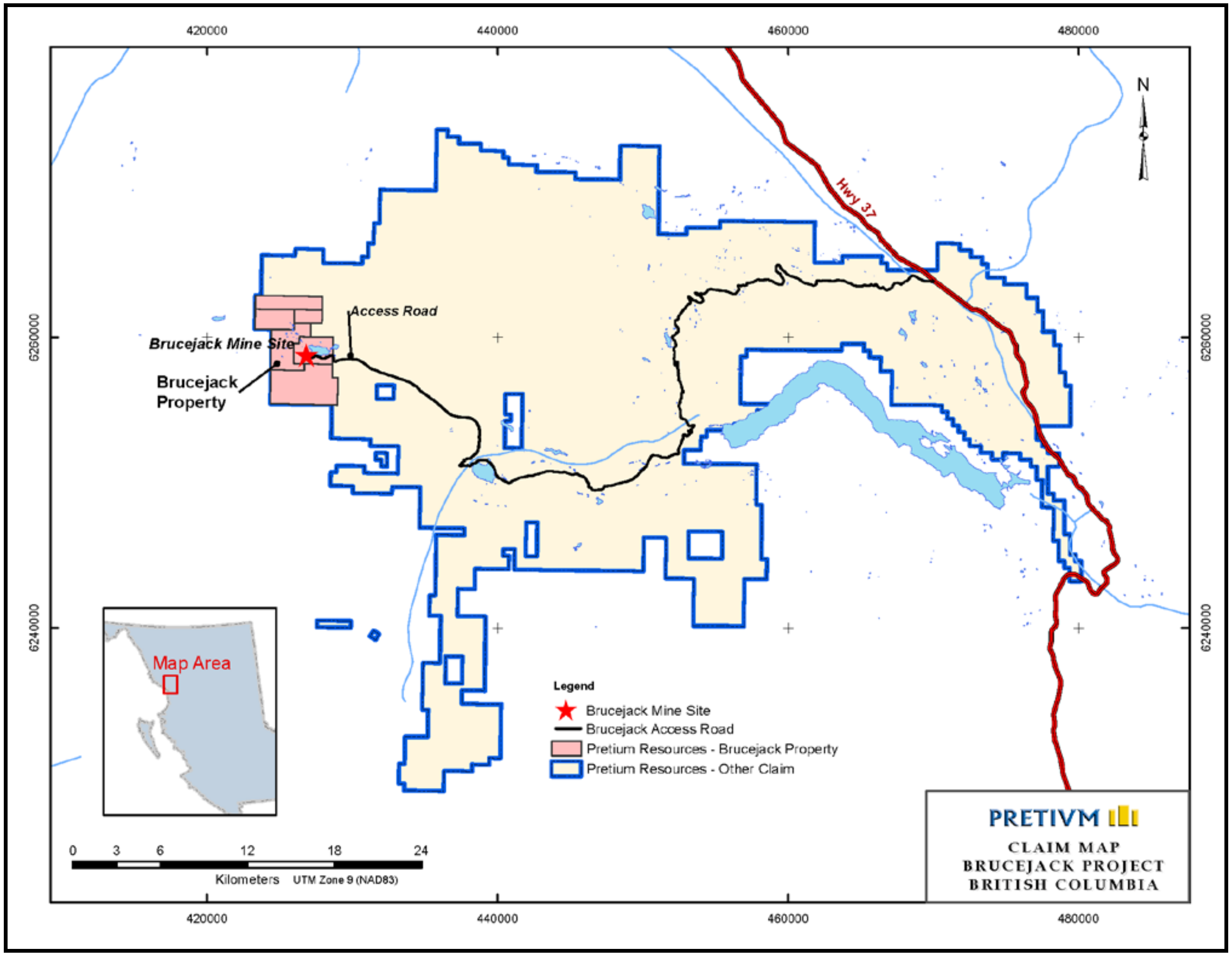
At the time of this report, the land claims in the area are in review and subject to ongoing discussions between various First Nations and the Government of BC.

Figure 4.2 Brucejack Property Mineral Claims



Source: Pretivm

Figure 4.3 Pretivm Mineral Claims



Source: Pretivm

4.4 CONFIRMATION OF TENURE

Snowden is not qualified to provide legal comment on the mineral title to the Property, and has relied on the provided information. No warranty or guarantee, be it expressed or implied, is made by Snowden with respect to the completeness or accuracy of the tenement description referred to in this document.

4.5 ROYALTIES, FEES AND TAXES

The royalties applicable to the Project are as follows:

- "Royalty" means the amount payable by the Owner, calculated as 1.2% of the NSR, with the following exemptions:
  - gold: the first 503,386 oz produced from the Project
  - silver: the first 17,907,080 oz produced from the Project.

Snowden understands that the 1.2% NSR royalty is, at the time of this report, in favour of Franco-Nevada Corporation.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

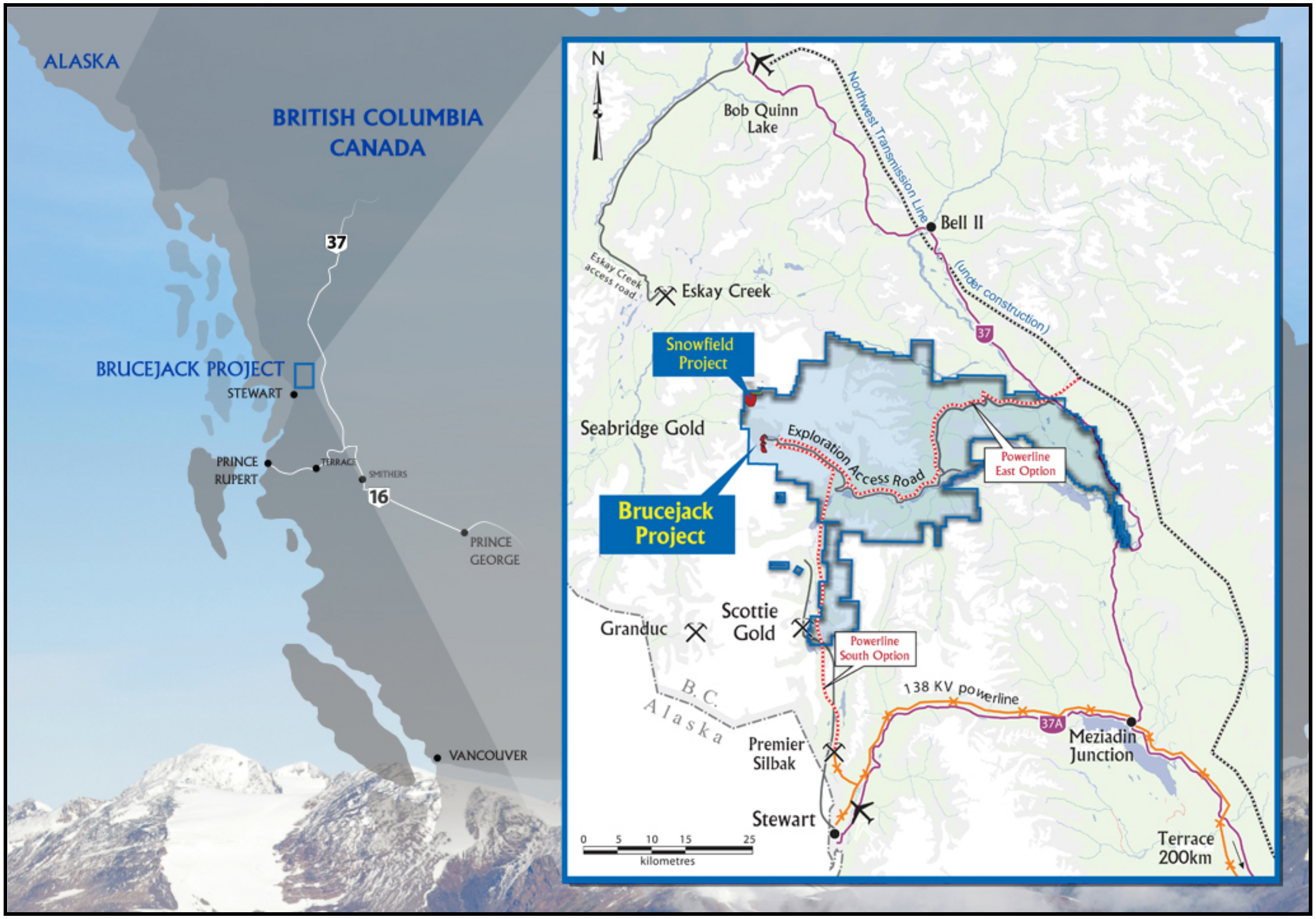
Information in this section has been excerpted and updated from Olssen and Jones (2012c). Olssen and Jones (2012a, b and c) excerpted and updated the information for this section in its report from Ghaffari et al. (2012).

5.1 ACCESSIBILITY

The Property is located in the Boundary Range of the Coast Mountain Physiographic Belt, along the western margin of the Intermontane Tectonic Belt. The terrain is generally steep with local reliefs of 1,000 m from valleys occupied by receding glaciers, to ridges at elevations of 1,200 masl. Elevations within the Property area range from 1,366 masl along Brucejack Lake to 1,650 masl at the Bridge Zone. However, within several areas of the Property, the relief is relatively low to moderate.

The Property area is easily accessible with the use of a chartered helicopter from the town of Stewart, or seasonally from the settlement of Bell II. The flight time from Stewart is approximately 30 minutes and slightly less from Bell II; however, Stewart has the advantage of having an established year-round helicopter base. Pretivm has completed construction of its 74 km access road that links the Brucejack Camp to Highway 37 via the Knipple Glacier, Bowser Camp, Scott Creek Camp, and Wildfire Camp (Figure 5.1). Equipment, fuel, and camp provisions are trucked to a staging area on the Knipple Glacier (at km 60), before being taken over the glacier to the Brucejack camp. Personnel are driven to the Knipple Glacier staging area from the Wildfire Camp at Highway 37, before being transported by tracked vehicle to the Brucejack camp over the glacier. This has significantly reduced transportation costs.

Figure 5.1 Project Access



Source: Pretivm

5.2 CLIMATE AND PHYSIOGRAPHY

The climate is typical of northwestern BC with cool, wet summers, and relatively moderate but wet winters. Annual temperatures range from +20°C to -20°C. Precipitation is high with heavy snowfall accumulations ranging from 10 to 15 m at higher elevations and 2 to 3 m along the lower river valleys. Snow packs cover the higher elevations from October to May. The optimum field season is from late June to mid-October.

5.2.1 VEGETATION

The tree line is at an elevation of approximately 1,200 m. Sparse fir, spruce, and alder grow along the valley bottoms with only scrub alpine spruce, juniper, alpine grass, moss, and heather covering the steep valley walls. The Property, at an elevation above 1,300 m, has only sparse mosses along drainages. Rocky glacial moraine and polished glacial-striated outcrops dominate the terrain above tree line.

5.3 INFRASTRUCTURE AND LOCAL RESOURCES

The exploration access road from Highway 37 is complete and in use (Figure 5.1).

There are no local resources other than abundant water for any drilling work. The nearest infrastructure is the town of Stewart, approximately 65 km to the south, which has a minimum of supplies and personnel. The towns of Terrace and Smithers are also located in the same general region as the Property. Both are directly accessible by daily air service from Vancouver.

The nearest railway is the Canadian National Railway Yellowhead route, which is located approximately 220 km to the southeast. This line runs east-west and terminates at the deep water port of Prince Rupert on the west coast of BC.

Stewart, BC, the most northerly ice-free shipping port in North America, is accessible to store and ship concentrates. At the time of this report, the Wolverine and Huckleberry mines are shipping material via this terminal.

A high-voltage, 138 kV power line currently services Stewart, BC, and has sufficient capacity to provide power to the Project. Also, a high-voltage power line is currently being constructed that runs parallel to Highway 37 (www.highway37.com). The plan calls for the new 287 kV line to extend from the community of Terrace to the beginning of the Galore Creek access road at Bob Quinn Lake, providing the Property access to the BC Hydro electric grid (Figure 5.2). The final capacity of this transmission line has yet to be determined and may be increased due to demand.

Figure 5.2 Proposed High-voltage Northwest Transmission Line

Source: www.highway37.com

6.0 HISTORY

Information in this section has been excerpted and updated from Olssen and Jones (2012c). Olssen and Jones (2012a, b and c) excerpted and updated the information for this section in its report from Ghaffari et al. (2012).

The Property and the surrounding region have a history rich in exploration for precious and base metals, dating back to the late 1800s. This section of the report describes the mineral exploration, including the historical drilling, carried out prior to Pretivm's acquisition of the Property. The historical data have been summarized mostly from various assessment reports available through the BC MEMNG.

In 1935, prospectors discovered copper-molybdenum mineralization on the Sulphurets Property in the vicinity of the Main Copper Zone, approximately six km northwest of Brucejack Lake; however, these claims were not staked until 1960.

From 1935 to 1959, the area was relatively inactive with respect to prospecting; however, it was intermittently evaluated by a number of different parties, and several small copper and gold-silver occurrences were discovered in the Sulphurets-Mitchell Creek area.

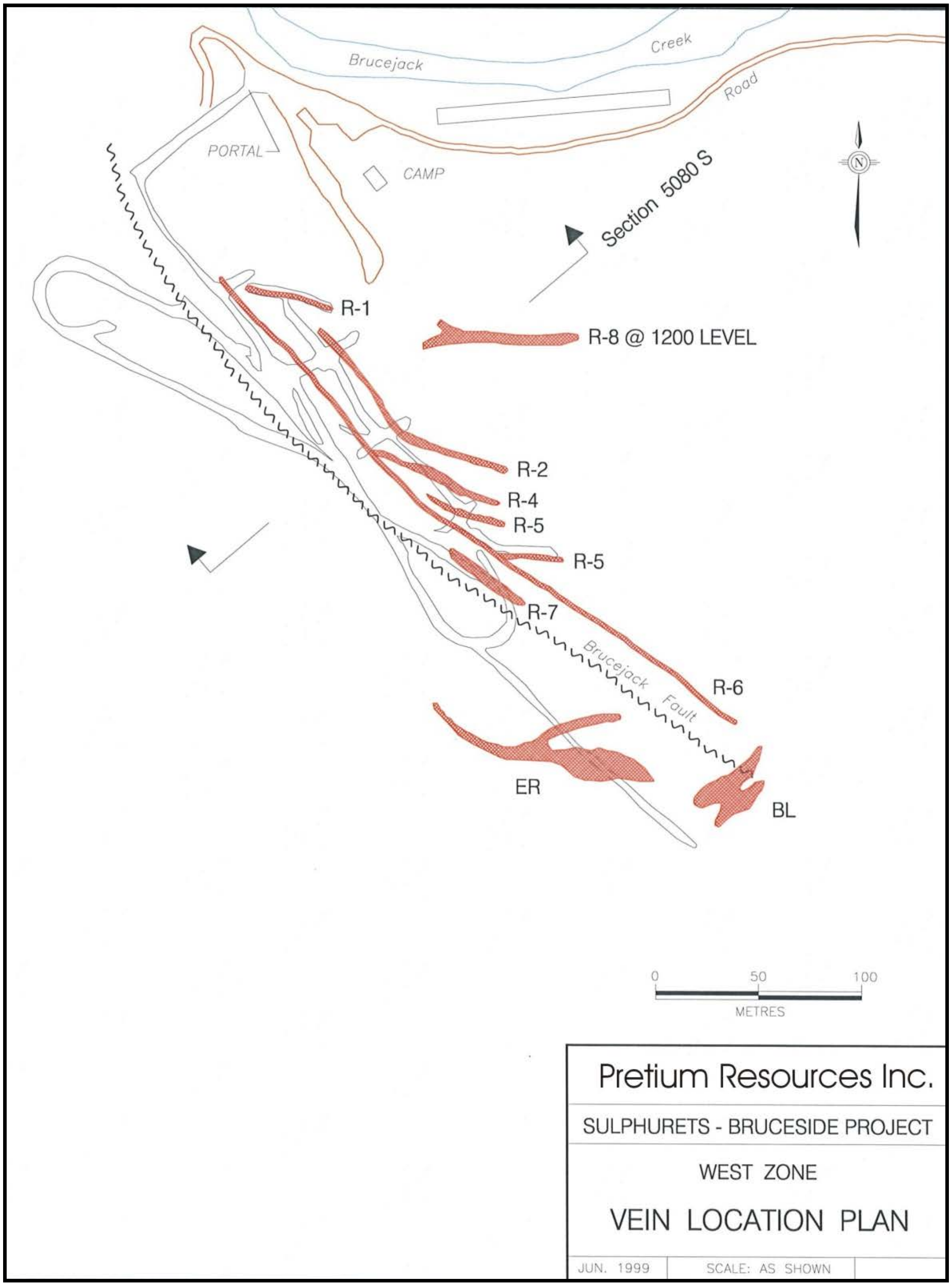
In 1960, Granduc Mines Ltd. (Granduc) and Alaskan prospectors staked the main claim group, covering the known copper and gold-silver occurrences, which collectively became known as the Sulphurets Property, starting the era of modern exploration, outlined as follows:

1960 to 1979	Granduc continued exploration, conducting further geological mapping, lithogeochemical sampling, trenching, and diamond drilling on known base and precious metal targets north and northwest of Brucejack Lake. This resulted in the discovery of gold-silver mineralization in the Hanging Glacier area and molybdenum on the south side of the Mitchell Zone.
1980	Esso Minerals Canada Ltd. (Esso) optioned the Property from Granduc and subsequently completed an extensive program consisting of mapping, trenching, and geochemical sampling that resulted in the discovery of several showings including the Snowfield, Shore, West, and Galena zones. Gold was discovered on the peninsula at Brucejack Lake near the Shore Zone.
1982 and 1983	Exploration was confined to gold- and silver-bearing vein systems in the Brucejack Lake area at the southern end of the property from 1982 to 1983. Drilling was concentrated in 12 silver- and gold-bearing structures, including the Near Shore and West zones, located 800 m apart near Brucejack Lake. Drilling commenced on the Shore Zone.

1983 and 1984	Esso continued work on the property and (in 1984) outlined a deposit on the west Brucejack Zone.
1985	Esso dropped the option on the Sulphurets property.
1985	The property was optioned by Newhawk Gold Mines Ltd. (Newhawk) and Lacana Mining Corp. (Lacana) from Granduc under a three-way joint venture (JV) (the Newcana JV). The Newcana JV completed work on the Snowfield, Mitchell, Golden Marmot, Sulphurets Gold, and Main Copper zones, along with lesser known targets.
1986 to 1991	Between 1986 and 1991, the Newcana JV spent approximately $21 million developing the West Zone and other smaller precious metal veins, on what would later become the Bruceside Property.
1991 and 1992	Newhawk officially subdivided the Suphurets claim group into the Sulphside and Bruceside properties and optioned the Sulphside property (including the Sulphurets and Mitchell Zones) to Placer Dome Inc. (Placer Dome). From 1991 to 1994, joint venture exploration continued on the Sulphurets-Bruceside property, including property-wide trenching; mapping; airborne surveys; and surface drilling, evaluating various surface targets including the Shore; Gossan Hill; Galena Hill; Maddux; and SG zones. Newhawk purchased Granduc's interest in the Snowfield Property in early 1992.
1991	Six holes were drilled at the Shore Zone, totalling 1,200 m, to test its continuity and to determine its relationship to the West and R-8 zones. Results varied from 37 g/t Au over 1.5 m, to 13 g/t Au over 4.9 m (www.infomine.com).
1994	Exploration in the Brucejack area consisted of detailed mapping and sampling in the vicinity of the Gossan Hill Zone, and 7,352 m of diamond drilling (over 20 holes) primarily on the West, R-8, Shore, and Gossan Hill zones. Mapping, trenching, and drilling of the highest priority targets were conducted on 10 of the best deposits (including the West Zone).
1996	Granduc merged with Black Hawk to form Black Hawk Mining Inc.
1997and1998	No exploration or development work was carried out on the Property (Budinski et al. 2001).
1999	Silver Standard acquired Newhawk and with it, Newhawk's 60% interest and control of the Property (www.infomine.com).
2001	Silver Standard entered into an agreement with Black Hawk whereby Silver Standard acquired Black Hawk's 40% direct interest in the Property, resulting in Silver Standard's 100% interest in the Property.
1999 to 2008	No exploration or development work was carried out on the Property during the period from 1999 to 2008.

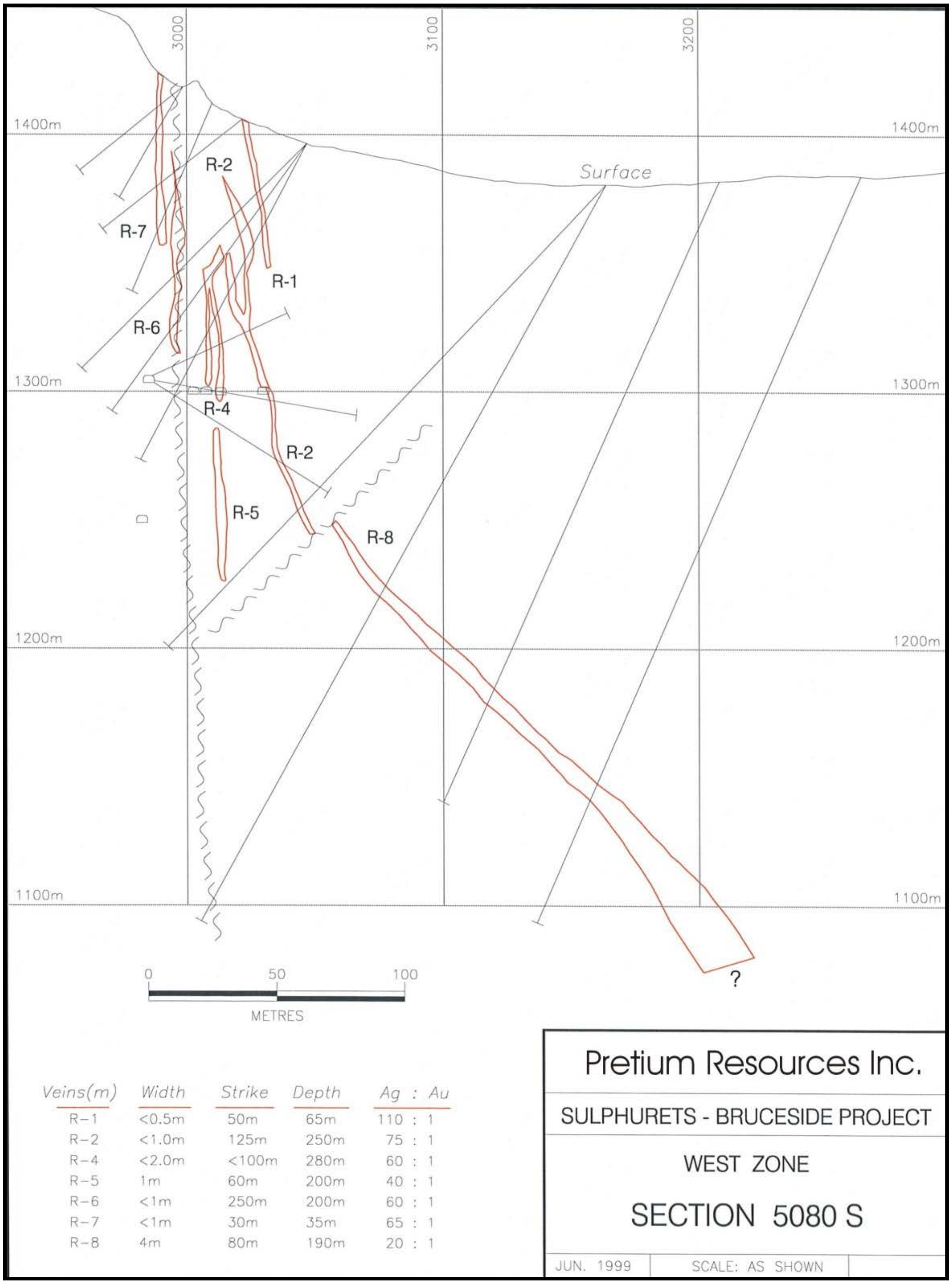
The historical interpretation (Budinski et al. 2001) of mineralized zones in the West Zone, prior to Silver Standard undertaking their exploration work in 2009, is shown in Figure 6.1 (underground vein location plan map) and Figure 6.2 (cross-section map).

Figure 6.1 West Zone Underground Vein Location Plan



Source: Pretivm

Figure 6.2 West Zone Section 5080S



Source: Pretivm

6.1 Work Completed by Silver Standard

In 2009, Silver Standard began their first work on the Property following its acquisition. The 2009 program included drilling, rock-chip and channel sampling, and re-sampling of historical drill core.

During the 2009 field program, Silver Standard collected a total of 1,940 drill core samples from 25 historical drillholes stored on site, and sent them to ALS for analysis. The samples were sent to the ALS assay laboratory in Terrace for preparation, and then forwarded to the ALS facility in Vancouver for analysis. Samples were analyzed for gold (30 g fire assay with atomic absorption (AA) finish) as well as 33 other elements by using four-acid digestion with inductively coupled plasma (ICP) analysis. The 2009 program also included re-analysis of 941 pulp samples derived from historical drill core samples. These samples were also analyzed for gold and 33 other elements at the ALS facility in Vancouver.

During the 2009 field work program, 2,739 rock-chip and channel samples were collected from surface outcrops. This sampling work was mostly done in target areas that were drilled by Silver Standard in 2009, with samples generally collected along north-south oriented lines that corresponded to the surface traces of some of the 2009 drillholes. Specifically, rock-chip and channel sampling were completed at the Galena Hill, Bridge, SG, and Mammoth zones (where drilling was carried out in 2009), as well as at the Hanging Glacier Zone (where historical surface sampling had identified rocks enriched in gold and silver). The surface samples were analyzed for gold and 33 other elements.

A total of 17,846 m of diamond drilling was completed in 37 holes during the 2009 field season.

In 2010, a total of 33,400 m of diamond drilling was completed in 72 holes.

6.2 Previous Feasibility Studies at the Property

Corona Corporation (Corona) completed a feasibility study on a proposed underground mine with decline access for the Sulphurets Project (West and R-8 Zones only) in 1990. A total operating cost of $145 per ton was estimated based on a 350 ton per day mill facility for processing, along with a capital cost of $42.7 million and a 6.7% pre-tax return at a price of US$400/oz gold and US$5/oz silver. The study concluded that higher metal prices must be realized before a production decision could be taken.

The reader is cautioned that the information above, mentioned in the 1990 Corona Sulphurets Project Feasibility Study is no longer relevant, is not NI 43-101 compliant, and should not be relied upon.

6.3 PRIOR MINERAL PRODUCTION

In excess of 5 km of underground ramps, level development, and raises were completed on the West Zone down to the 1100 Level. In 1993, a Project Approval Certificate was issued in respect of the Project by the Minister of Sustainable Resource Management and Minister of Energy and Mines for the Province of BC. The mine was not developed and the certificate as amended expired in 2006. No ore has been mined or processed from the Property, including the West Zone.

6.4 PRELIMINARY ECONOMIC ASSESSMENT 2010

Silver Standard commissioned Wardrop to complete a preliminary economic assessment (PEA) on the combined resources of the Brucejack Project and Snowfield Project in 2010 (Wardrop 2010).

The following consultants were commissioned to complete the component studies for the NI 43-101 technical report:

- Wardrop: processing, infrastructure, capital and operating cost estimates, and financial analysis
- AMC: mining
- P&E: mineral resource estimate
- Rescan: environmental aspects, waste and water treatment
- BGC: tailings impoundment facility, waste rock and water management, and geotechnical design for the open pit slopes.

Based on the results of the PEA, it was recommended that Silver Standard continue with the next phase - a Pre-feasibility Study, in order to identify opportunities and further assess viability of the Property. This report was reissued for Pretivm in October 2010; however, the report is no longer current.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

Information in this section has been partly excerpted, condensed, and updated from sections within Ghaffari et al. (2012), and from relevant sections in Olssen and Jones (2012) by Pretivm's Chief Geologist Dr. Warwick Board, P.Geo.

7.1 REGIONAL GEOLOGICAL SETTING

The Property is located in the western Stikine terrane (or Stikinia), the largest of several allocthonous terranes in the Intermontane Belt of the Canadian Cordillera. Stikinia, which is considered to be a multistage mid-Palaeozoic to Middle Jurassic island arc terrane that developed in an intra-oceanic setting isolated from the North American continental margin (Gagnon et al. 2012), underlies much of western BC (Figure 7.1). Stikinia appears to have been accreted to the North American continental margin as early as the late Middle Jurassic (c. 173 Ma).

The Stikine terrane in northwestern BC (MacDonald et al. 1996) consists of a series of unconformity-bound tectonostratigraphic elements, including:

- Palaeozoic island-arc rocks of the Stikine assemblage
- Mesozoic island-arc rocks of the Upper Triassic Stuhini Group and Lower to Middle Jurassic lower Hazelton Group
- Middle to Upper Jurassic overlap assemblage sedimentary rocks of the Bowser Lake Group.

Tertiary igneous and metamorphic rocks of the Coast Plutonic Complex occur to the west of the Stikine terrane in this area.

At least four magmatic episodes and three mineralizing events have been recognized in northwestern Stikinia (Anderson et al. 2003):

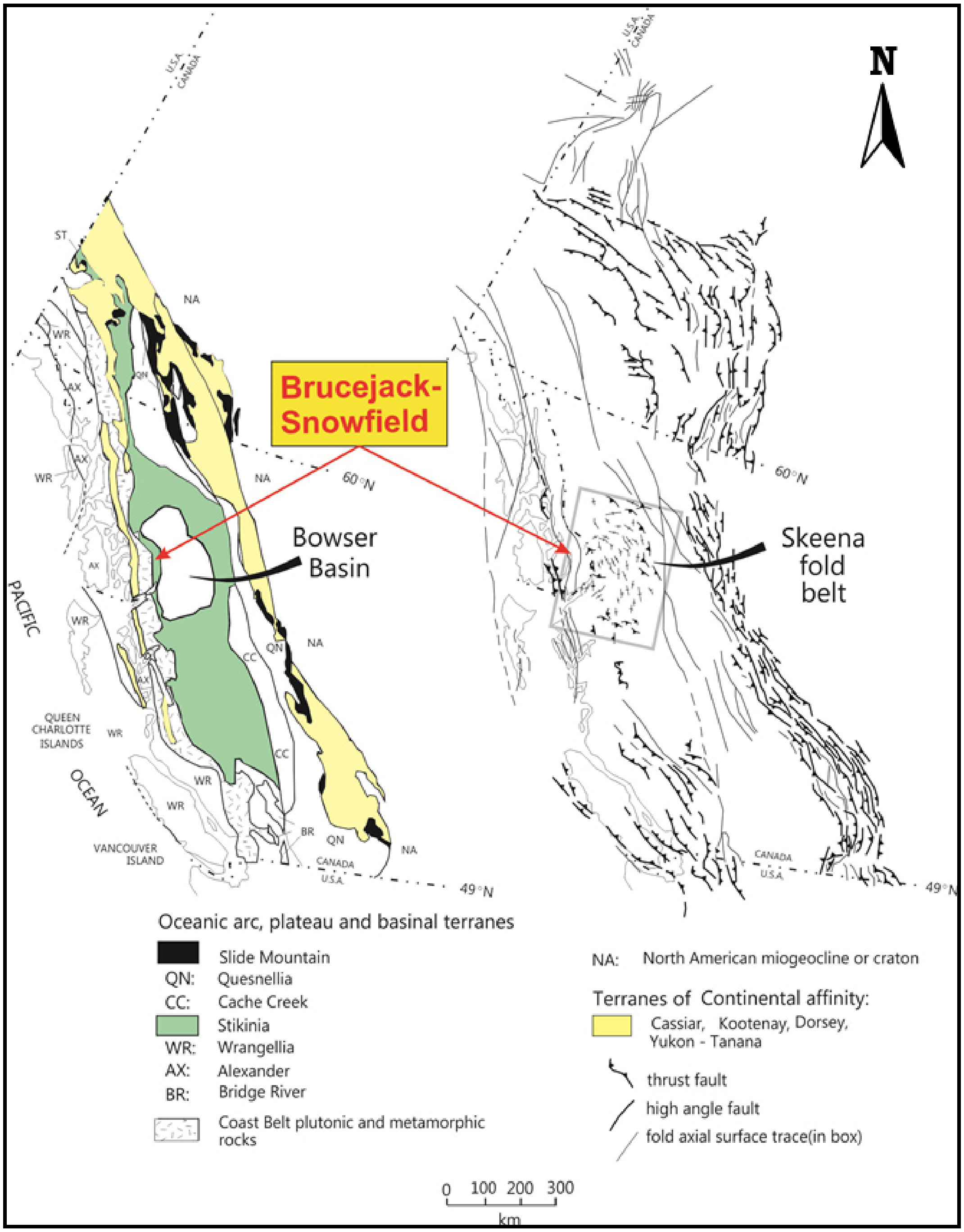
- Late Triassic to Early Jurassic (205 to 196 Ma) alkaline porphyry-related magmatism and associated deformed mesothermal silver-gold veins (e.g. Red Mountain)
- Early Jurassic (196 to 187 Ma) alkaline porphyry-related epithermal and mesothermal gold-silver veins and base and precious metal deposits (e.g. Premier, Sulphurets, and Bronson Creek)
- Early to Middle Jurassic (184 to 183 Ma) small and poorly mineralized porphyry intrusions

- Middle Jurassic (175 to 172 Ma) calc-alkaline and tholeiitic back-arc magmatism and syn- to epigenetic back-arc basin-related stratabound base and precious metal deposits (e.g. Eskay Creek, RDN).

The northwest part of Stikinia (in particular the volcanic and sedimentary rocks of the Hazelton Group) and related Early Jurassic plutons, represent perhaps the most well-endowed metallogenetic assemblage in BC. In addition to the Brucejack and Snowfield Properties, this area also includes nearby former producers such as Eskay Creek, Silbak-Premier, Big Missouri, Dolly Varden, Torbrit Silver, Granduc, and Anyox (Figure 7.2). Furthermore, adjacent properties host significant precious and base metal resources (e.g. Kerr-Sulphurets-Mitchell (KSM) and Red Mountain deposits) as well as a number of high-potential mineral occurrences (e.g. Homestake Ridge, Silver Coin, Big Missouri, Clone, and Tennyson Properties). The KSM deposits, along with the Snowfield and Brucejack Deposits, comprise what is commonly referred to as the Sulphurets Mining Camp.

Several major compressional tectonic events affected rocks of the Stikine terrane in northwestern BC throughout the Mesozoic. The earliest event in the Late Triassic to Early Jurassic affected Palaeozoic and Triassic rocks of the Stikine assemblage and Stuhini Group. A second, younger event in the Late Jurassic through Late Cretaceous, which has been associated with accretion of the outboard Insular terranes west of the Coastal Plutonic Complex and the formation of the Skeena Fold Belt (SFB), resulted in widespread predominantly east-verging fold and thrust deformation of rocks in western Stikinia (Figure 7.1 and Figure 7.2). Deformation associated with the Middle Jurassic accretion of Stikinia to the North American continent appears to have mainly affected rocks of eastern Stikinia (e.g. Evenchick et al. 2007).

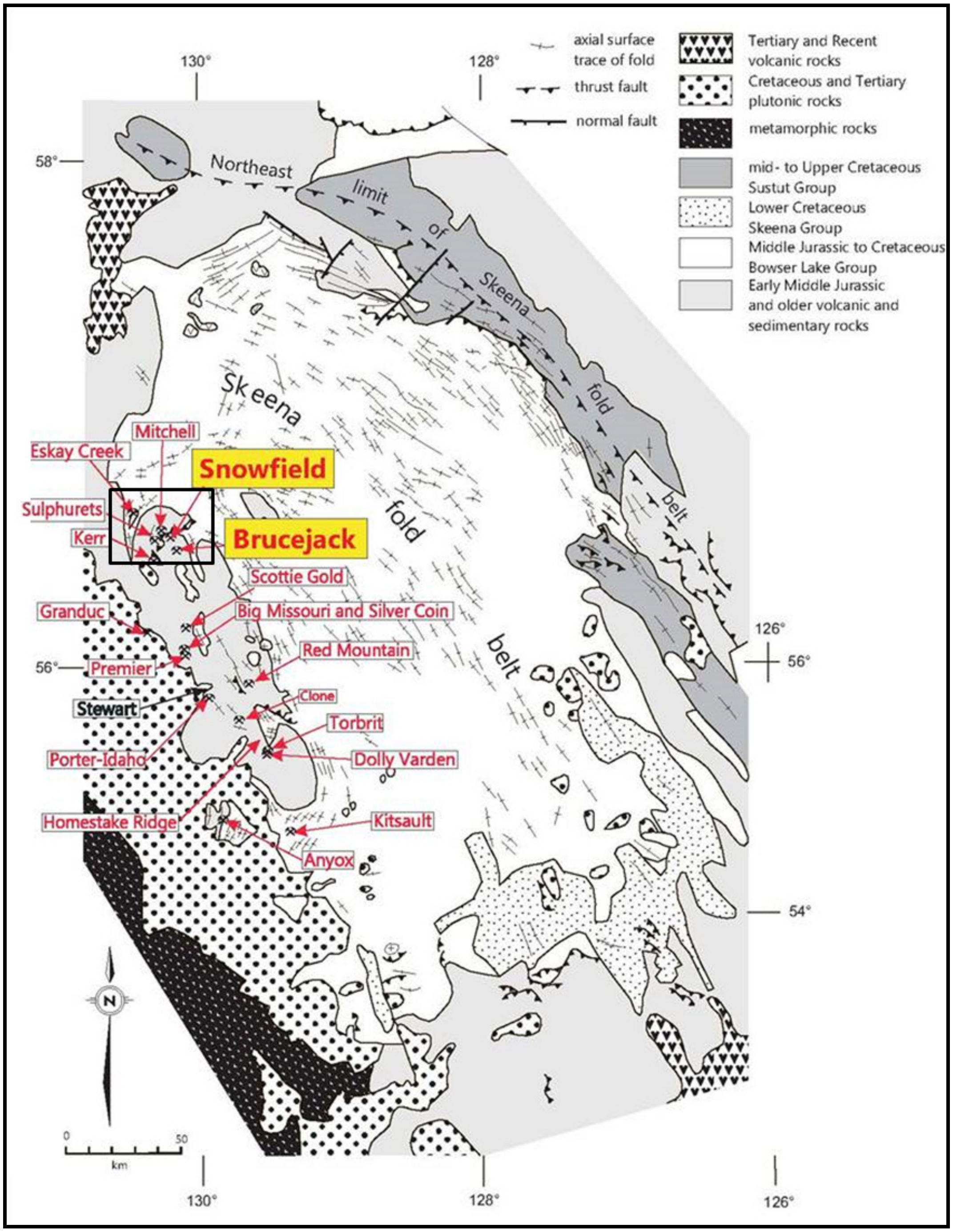
Figure 7.1 Tectonic Setting of the Brucejack and Snowfield Properties in the Northwest Cordillera



Note: Shows location of the Project area in north-central Stikine terrane, as well as nearby Bowser Basin (left) and latest Jurassic to mid-Cretaceous SFB (right). Rectangle on right hand image represents location of Figure 7.2.

Source: Ghaffari et al. (2012)

Figure 7.2 Regional Structural and Stratigraphic Setting of the Brucejack Property and Sulphurets Mining Camp in Northwest BC



Note: Shows significant past-producing mines, as well as selected advanced exploration projects. Rectangle represents location of Figure 7.3.

Source: Ghaffari et al. (2012)

7.2 LOCAL GEOLOGY - SULPHURETS MINING CAMP

The Sulphurets Mining Camp, which includes the Snowfield and Brucejack Properties, is located on the eastern limb of the broad McTagg anticlinorium, a major north-trending mid-Cretaceous structural culmination in the western SFB (Figure 7.3). Sedimentary and volcanic rocks of the Upper Triassic Stuhini Group form the core of the anticlinorium, and are successively replaced outwards towards the west, north, and east of the core by progressively younger rocks of the Lower to Middle Jurassic volcanic and lesser sedimentary rocks of the Hazelton Group followed by sedimentary rocks of the Bowser Lake Group.

Details of the local geology of the Sulphurets Mining Camp in northwestern BC presented in this section are partly drawn from existing literature, including Kirkham (1963), Grove (1986), Britton and Alldrick (1988), Alldrick and Britton (1988, 1991), Anderson (1989), Anderson and Thorkelson (1990), Kirkham (1991, 1992), Henderson et al. (1992), Roach and MacDonald (1992), Margolis (1993), Davies et al. (1994), Kirkham and Margolis (1995), Childe (1996), Macdonald et al. (1996), Lewis et al. (2001), Evenchick et al. (2007), Armstrong et al. (2011), Gagnon et al. (2012), and the Seabridge Gold Inc. (Seabridge Gold) website (accessed September 2012), and from work conducted by Pretivm's geologists.

7.2.1 STRATIGRAPHIC SETTING AND MAJOR MINERAL DEPOSITS

The Stuhini Group, which generally underlies the western and northern parts of the Sulphurets Mining Camp (Figure 7.4), is characterized by fine-grained and well-stratified sedimentary rocks and subordinate mafic volcanic arc-related rocks. The sedimentary package includes dark grey turbiditic siltstone, minor interbedded micritic limestone, and thick sequences of immature conglomerate and sedimentary breccia. Mafic volcanic rocks in this unit include alkalic pyroxene- and hornblende-phyric massive and pillowed basaltic flows, flow breccia, and tuff.

The central and eastern parts of the Sulphurets Mining Camp are largely underlain by subaqueous to locally sub-aerial, arc-related volcanic and subordinate sedimentary rocks of the lower Hazelton Group, which unconformably overlie the Stuhini Group. The lowermost unit of the lower Hazelton Group, the Lower Jurassic Jack Formation, is characterized by polylithic (granitoid and volcanic) pebble to boulder conglomerate and limy fossiliferous sandstone and siltstone. The Jack Formation appears to be conformably overlain by the volcanic rocks of the lower Hazelton Group, which generally consist of thick massive plagioclase (± hornblende, K-feldspar, and pyroxene)-phyric andesitic and dacitic flows, breccias, and related predominantly pyroclastic fragmental rocks, with subordinate mafic and felsic rocks and minor siltstone and mudstone layers of the Unuk River Formation. These rocks are overlain by well-bedded green, maroon, and grey andesitic to dacitic pyroclastic and epiclastic rocks, mafic flows, and minor carbonaceous mudstone, chert and limestone of the Betty Creek Formation, which are in turn disconformably overlain by the Mount Dilworth Formation. The Mount Dilworth Formation is characterized by dacitic and rhyolitic tuff, welded ash-flow tuff, volcanic

breccia, flow-layered lava domes, and subordinate interbedded limy fossiliferous sandstone.

The upper Hazelton Group, which is limited to the northern and extreme eastern parts of the Sulphurets Mining Camp, is characterized by distinctive black carbonaceous pyritic mudstone, light and dark banded tuffaceous siltstone, and local amygdaloidal basalt of the Salmon River Formation. These rocks display unconformable relationships to the rocks of the lower Hazelton Group. Recent re-examination of stratigraphic sections through the Hazelton Group (Gagnon et al. 2012) has suggested that the term "Salmon River Formation" be replaced by "Iskut River Formation". These rocks clearly delineate the outline of the anticlinorium in the broader Sulphurets area (Figure 7.3).

Rocks of the Middle to Upper Jurassic Bowser Lake Group, which are generally characterized by clastic basin-fill sediments including submarine fan, prodelta slope, shelf, and fan delta sedimentary assemblages, are limited to the extreme north and northeast of the Sulphurets Mining Camp (see Figure 7.4). These rocks display conformable to disconformable relationships to the underlying Hazelton Group rocks.

Plutonic rocks are located in the western and northern parts of the Sulphurets Mining Camp, and occur as dykes, sills, and plugs, which generally intrude Stuhini Group rocks. These rocks, the so-called "Mitchell intrusions" of Kirkham and Margolis (1995), include diorite, monzodiorite, monzonite porphyry, syenite porphyry, quartz syenite porphyry, porphyritic aplitic low-silica granite, sodic albite-hornblende porphyry, and K-feldspar megacrystic porphyry. Monzonitic, syenitic, and granitic intrusions display a close spatial and temporal relationship to the porphyry-style copper ± gold ± molybdenum mineralization in the KSM deposits.

A number of internally consistent uranium-lead zircon dates from pre-, syn- and post-mineral intrusive phases of the Mitchell intrusions at the KSM deposits suggest that porphyry-style mineralization was emplaced between 192 and 195 Ma. This Early Jurassic age is consistent with a number of galena lead dates for mineralization from these deposits, as well as for mineralization from the Snowfield Deposit and from the West Zone on the Property. All of these dates plot in the “Jurassic cluster” of galena lead dates defined by Alldrick, Gabites, and Godwin (1987) and Alldrick et al. (1990) for the so-called “Stewart Mining Camp.” Other regionally close mineral deposits that have similar age dates include Silbak-Premier and Big Missouri.

7.2.2 Alteration and Mineralization

Large, coalescing hydrothermal alteration haloes are developed around the intrusive complexes. Potassic K-feldspar alteration associated with copper and gold mineralization is widespread in the Mitchell Intrusions and adjacent Stuhini Group rocks. Propylitic and chlorite-sericite alteration is also developed around the KSM intrusions and the Snowfield Deposit, often overprinting earlier potassic alteration at KSM. Quartz-sericite-pyrite alteration is widely developed in the Stuhini Group and lower Hazelton Group rocks further to the east of the intrusions in the Sulphurets Mining Camp (Figure 7.4), and also occurs as a pervasive overprint to earlier alteration in the intrusive rocks.

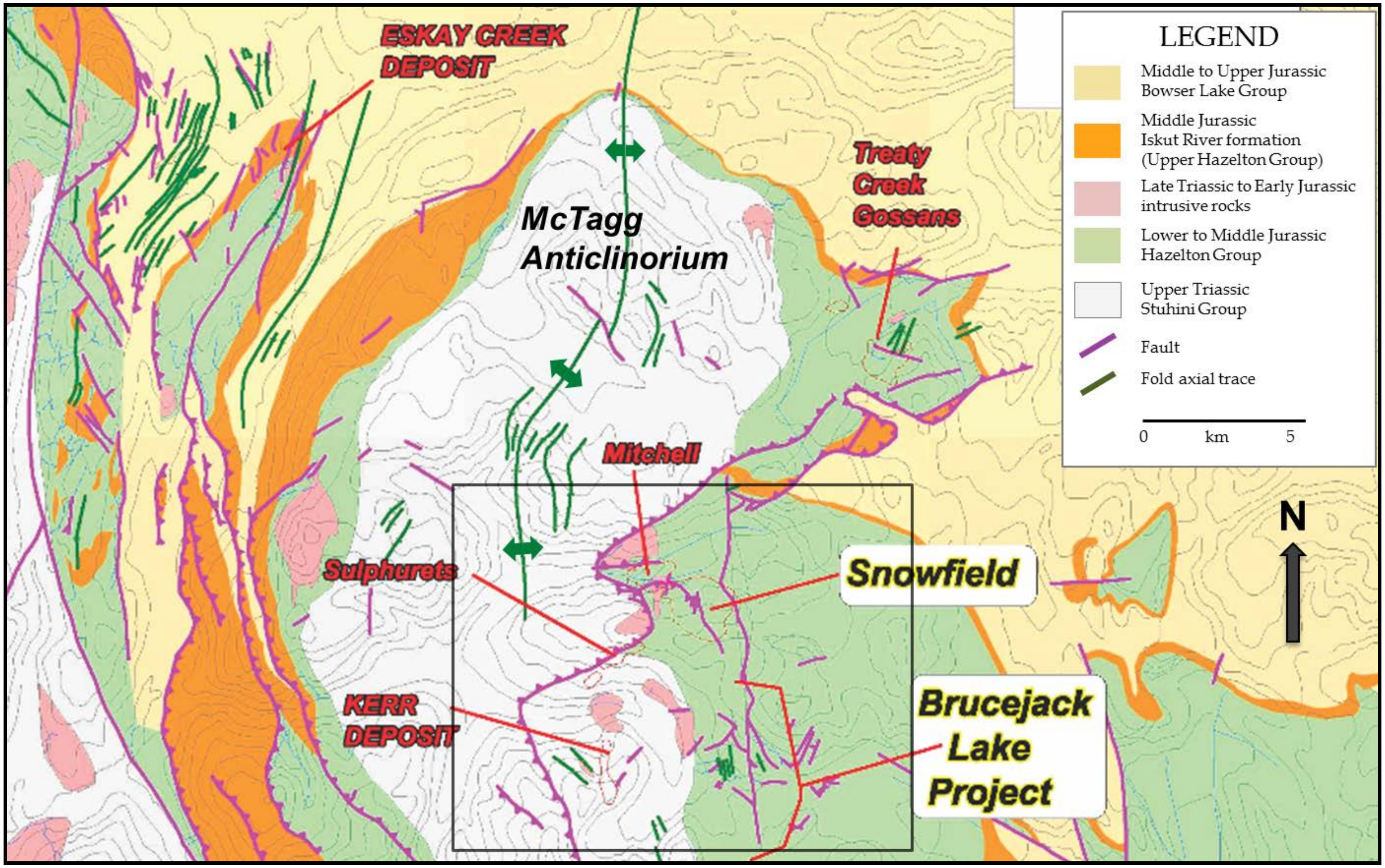
Altered Stuhini Group rocks and Mitchell intrusions are the main host rocks to porphyry-style mineralization at the Kerr (copper-gold), Sulphurets (gold-copper), and Mitchell (gold-copper-molybdenum) deposits. Mineralization at the KSM deposits occurs within a gold-enriched copper porphyry system controlled by a series of dikes, sills and plugs, and is associated with quartz veinlet stockworks and sheeted quartz veinlet arrays mainly in the altered host rocks adjacent to the intrusions. Pyrite and chalcopyrite are the dominant sulphide minerals, with minor molybdenite, and trace amounts of tennantite, bornite, sphalerite, and galena. All mineralization is hypogene, except for small remnants of preserved weak supergene at higher elevations.

7.2.3 STRUCTURAL SETTING AND METAMORPHISM

Rocks in the Sulphurets Mining Camp have been affected by folding, faulting, penetrative cleavage formation, late stage quartz vein formation, and low-grade lower greenschist facies (or lower) regional metamorphism. Rocks of the Stuhini Group were subjected to intense ductile deformation during the Late Triassic to Early Jurassic prior to the deposition of the Hazelton Group rocks. Ductile deformation during the Late Jurassic to Late Cretaceous development of the SFB resulted in the formation of the major structural culmination of the McTagg anticlinorium and associated fold and thrust structures that affected the Stuhini Group through Bowser Lake Group rocks in the Sulphurets Mining Camp.

Penetrative cleavage (foliation) development was associated with the Late Jurassic to Late Cretaceous event and affected most of the altered and unaltered rocks in the area, where host rock mineral assemblage (i.e. the presence and concentration of phyllosilicates in the rock) permitted its development. Age dating (argon-argon) of sericite within pressure shadows about pyrite provide a minimum age for this deformation at 110 ±2 Ma. Foliation orientations, while variable within the Sulphurets Mining Camp, are dominantly east-west trending on the Property (Figure 7.4). This is in contrast to the more regional scale north-northwest to north-east orientation of structural fabrics (particularly foliation) that is more in accordance with typical Cordilleran structural trends. The east-west trending foliation orientations coincide with a distinct westward-oriented "kink" in the McTagg anticlinorium (Figure 7.3), which is possibly a reflection of the warping of this structure about a protrusion of more strain-resistant rocks inboard of the culmination, that resulted in apparent north-south compression in this part of the area. The existence of such a strain resistor might also have resulted in the development of the southeast-vergent fold-induced thrusts during strain accommodation to the east of the anticlinorium axis. The Snowfield Deposit is considered to be part of the upper Mitchell Deposit that was thrust southeastward over lower Hazelton Group rocks during the Late Jurassic to Late Cretaceous deformation.

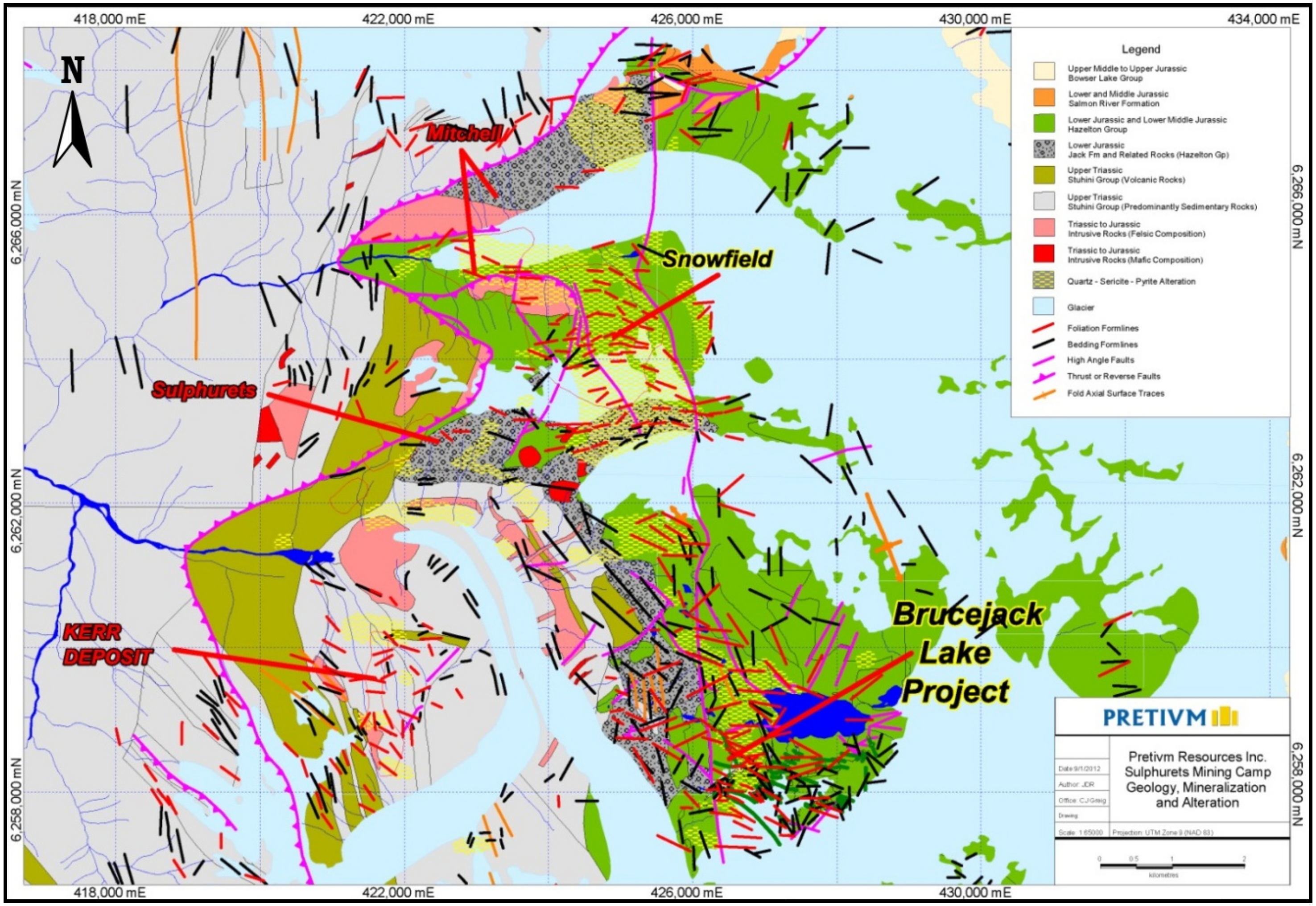
Figure 7.3 Local Structural and Stratigraphic Setting of the Brucejack Property and Sulphurets Mining Camp



Note: Rectangle represents location of Figure 7.4

Source: Pretivm

Figure 7.4 Sulphurets Mining Camp Geology and Mineralization



Source: Ghaffari et al. (2012)

Mineralized quartz (±carbonate, ±adularia) veins, vein networks, and vein stockworks in and around the various mineral deposits in the Sulphurets Mining Camp display clear and abundant evidence for significant post-mineralization deformation, including:

- tight through isoclinally folded veins
- rootless intrafolial folded veins
- apparently ptygmatically folded veins in less competent and deformed host rocks
- boudinaged veins
- rootless boudinaged veins hosting gold mineralization tracing out tight folds and terminating at the vein contacts
- transposition of veins into foliation planes with extension cracks perpendicular to the foliation
- pinch-and-swell deformed veins with cuspate and lobate margins wrapped by the foliation
- mesoscale folded stockworks
- brecciated veins
- fracture offset veins
- other small-scale post-mineral deformation features.

These features are visible on the microscope (including strained and partially re-crystallized quartz in veins and vein stockworks), hand-specimen, drill core, and outcrop scales.

Development of the McTagg anticlinorium effectively exposed older pre-Salmon River Formation rocks in the Sulphurets Mining Camp. Rocks of the Hazelton Group and Bowser Lake Group, which are located on the eastern limb of the north-plunging anticlinorium, display moderate to steep dips towards the southeast, east, and northeast, indicative of an overall eastward tilting of the original strata and porphyry-associated mineralization in this area as a result of the Late Jurassic to Late Cretaceous deformation event.

En echelon arrays of late shallow southeast-dipping (25° to 40°) veins with vertical or steeply-oriented crystal fibres in thin crack-seal textures cut across foliated and unfoliated altered and mineralized rocks, mineralized veins, and unaltered rocks throughout the region. Arrays of similarly late sigmoidally-folded veins with a top-to-the-southeast sense of shear are also present. These late quartz veins have been interpreted as having formed during the southeast-vergent thrusting that produced the Sulphurets and Mitchell thrusts in the eastern part of the anticlinorium.

The Brucejack Fault is a late, steeply dipping, northerly striking brittle structure which forms a distinct topographical feature in the centre of the Sulphurets Mining Camp (Figure 7.4). Pre-existing folds, thrust faults, alteration, and mineralization zones are cut

by the Brucejack Fault as well as by many other similar late northerly-striking faults. Movement on the fault is probably complex and has been difficult to determine. A much thicker section of Hazelton Group rocks on the east side of the fault suggests considerable east-side-down displacement, which may be interpreted as reflecting post-depositional displacement. Kirkham and Margolis (1995) indicate that the Brucejack Fault appears to have an east-side down dip-slip displacement of greater than 500 m with a dextral strike-slip component in the area north of the Snowfield Deposit, and approximately 100 m of dextral strike-slip with uncertain dip-slip on the Property. Davies et al. (1994) noted that, northwest of Brucejack Lake, preserved slickenside and cast elongation lineations on a steeply west-dipping surface indicate dip-slip offset of potentially between 700 m and 800 m with a reverse fault sense of movement (i.e. west-side up). Stratigraphic contacts a short distance northwest of the Brucejack Lake have been interpreted as indicating a possible strike separation of between 200 m to 300 m, and dip-slip displacement is likely less than in the north. In contrast, Britton and Alldrick (1988) considered that displacement on the Brucejack Fault was on the order of tens of metres. Elsewhere in the Sulphurets Mining Camp the northerly, north-easterly and north-westerly striking brittle faults, and rare east-west striking faults, display typically steep dips, steeply-plunging fault fabrics, and locally normal-dextral oblique displacements of up to tens of metres.

The possibility that the Brucejack Fault structure was formed as a result of late brittle deformation re-activation of a pre-existing syn-depositional fault developed at or near a volcanic sub-basin margin during deposition of the lower Hazelton Group is being considered based on recent fieldwork. If this hypothesis is correct then, given the spatial proximity of the fault to the alteration and contained mineralization zones on the Brucejack and Snowfield Properties (Figure 7.4), it may have partly controlled the hydrothermal alteration and mineralization in this part of the Sulphurets Mining Camp.

Rocks of the Sulphurets Mining Camp were subjected to, at most, lower greenschist facies metamorphism characterized by epidote, calcite, quartz, and chlorite, and the absence of biotite, hornblende, and actinolite in andesitic volcanic rocks and sedimentary rocks outside of the areas of hydrothermal alteration. The peak metamorphic temperature probably did not exceed 275°C (assuming a 3 km depth of burial).

7.3 PROPERTY GEOLOGY

The information in this section on the Property geology is summarized and updated from the work of Mr. Charles Greig, Senior Geologist for Pretivm, as presented in Olssen and Jones (2012a).

Geology on the Property can generally be characterized as a northerly trending, broadly arcuate, concave-westward structural-stratigraphic belt of variably altered rocks. This belt is bisected on the western side of the Property by a prominent topographic lineament, the Brucejack Fault (Figure 7.3 and Figure 7.4). To the south of Brucejack Lake, the belt generally displays a north-easterly trend, rotating towards the northwest north of the lake. The arcuate trend is outlined by the stratified rocks and the intensely

quartz-sericite-pyrite altered rocks. Most of the defined mineral resources on the Property are located within the intensely altered zone.

7.3.1 LITHOLOGY AND STRATIGRAPHY

TRIASSIC STUHINI GROUP

Rocks of the Upper Triassic Stuhini Group, which are typically fine-grained and well bedded siltstone and mudstones with minor micritic limestone, conglomerate, and sedimentary breccia, are limited to the western parts of the Property, west of the Brucejack Fault. These rocks are intruded by a number of mafic to felsic predominantly alkalic intrusions, a number of which have been dated as Early Jurassic in age. The Upper Triassic clastic rocks have been folded across steep northerly-trending folds and related faults and were deformed and eroded prior to deposition of the lowermost rocks of the Hazelton Group.

LOWER JURASSIC LOWER HAZELTON GROUP

The majority of the lithological units mapped on the Property appear to correlate reasonably well with those of the Unuk River and Betty Creek Formations of the Early Jurassic lower Hazelton Group, as described by Britton and Alldrick (1988). However, Pretivm has elected not to assign formation-level regional stratigraphic names to these rocks until the current detailed field mapping is complete, due to the existence of complicated lateral facies variations and the diachronous nature of many of the units.

Unconformably overlying the Triassic rocks are rocks of the Lower Jurassic Hazelton Group, which comprise five principal intercalated rock types:

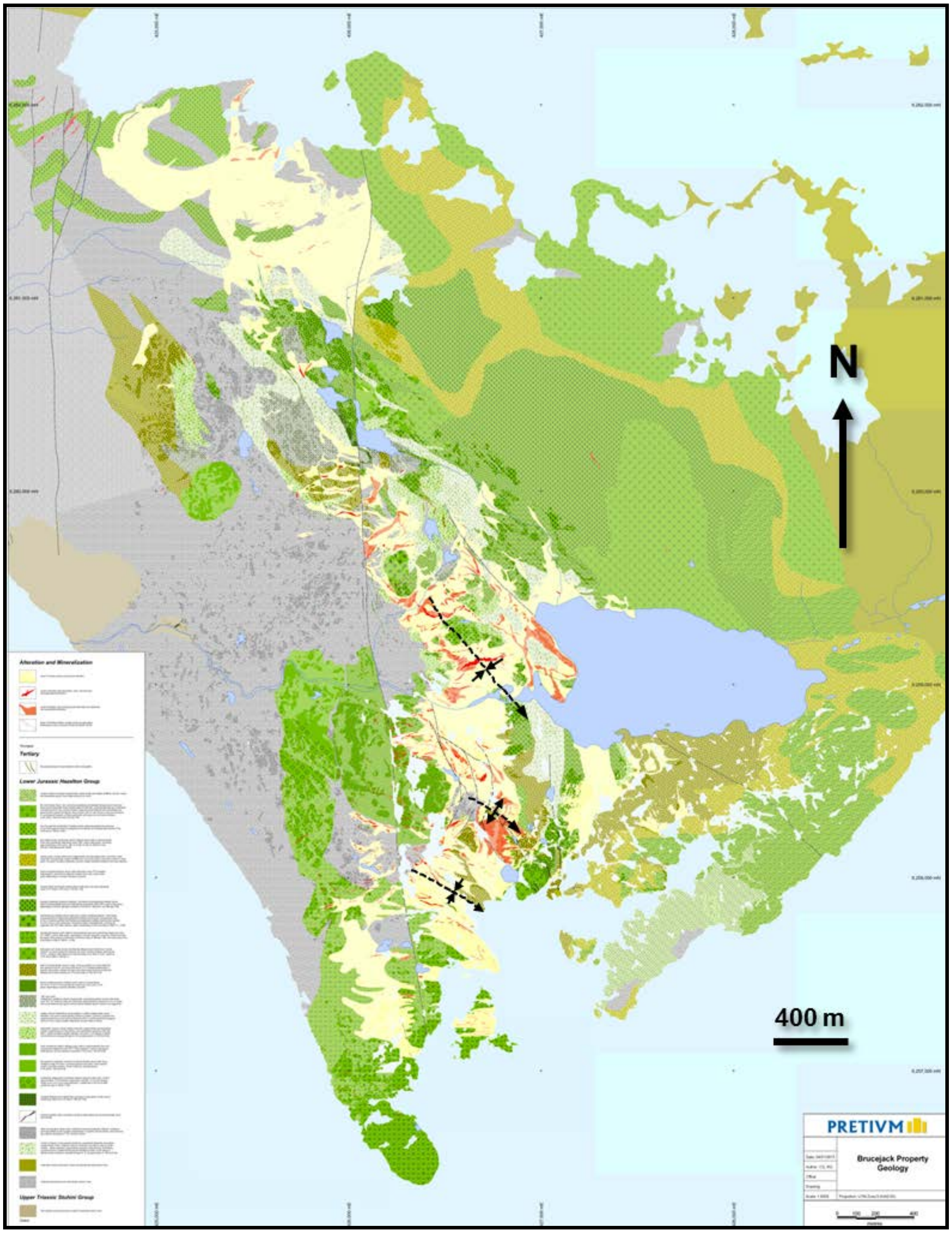
1. heterolithic volcanic conglomerate, most common at the base and typically coarse-grained (Jack formation)
2. massive and locally well-layered medium to dark green volcanic siltstone containing common carbonate concretions, and subordinate litharenite and locally-derived pebble conglomerate
3. hornblende and/or feldspar-phyric volcanic rocks, principally flows and related coarse fragmental rocks
4. weakly stratified heterolithic green to dark green volcanic pebble to boulder conglomerate, sandstone and local mudstone, containing zones of intensely silicified conglomerate
5. pyroclastic rocks, including medium- and coarse lapilli tuff and tuff-breccia, with minor intercalated intensely silicified zones.

The flows include several subtypes that can be distinguished by the grain size and compositions of their phenocryst assemblages (generally fine- to medium-grained hornblende and plagioclase feldspar, ±medium- and locally coarse-grained potassium feldspar) the flows are typically rich in groundmass potassium feldspar, and are essentially latites to trachydacites or trachyandesites.

Previously, some of the rocks straddling the Brucejack Fault were mapped as intrusive (e.g. Davies et al. 1994). Based on drilling and detailed mapping across the Property over the last three years, Pretivm now interprets the majority of the rocks on the Property as being extrusive. Most of the bodies of massive fine-grained rocks contain local fragmental layers, which are interpreted to represent interflow block tuff or flow-breccia. In addition, there is little or no evidence in the vicinity of the larger masses for associated dykes, and little evidence for contact aureoles. In a number of outcrops, there is clear evidence for the incorporation of large, angular fragments of these bodies, which are texturally distinctive (they typically contain abundant fine- to medium-grained hornblende and/or feldspar phenocrysts within an aphanitic groundmass) within marginal and/or overlying fragmental units. Furthermore, the relatively massive rocks are commonly interlayered with clastic sedimentary rocks near their basal contacts, and locally they contain fragments of lithologies known to be Upper Triassic in age. The various Early Jurassic hornblende feldspar-phyric rocks display variable ages over a range of 15 Ma, as determined from preliminary uranium-lead dating, which is more consistent with an extrusive interpretation.

The polylithic conglomerate and overlying pyroclastic fragmental units appear to have been favourable sites for channelling hydrothermal fluids due to the presence of numerous stratiform intensely silicified zones within these relatively porous and permeable rocks. Intense silica flooding with an associated cross-cutting network of crack-seal and hydraulic fractures filled with cryptocrystalline quartz indicates that these zones may have acted as local pressure caps during fluid infiltration that induced local overpressure conditions and subsequent hydraulic fracturing. The presence of multiple zones of intense silicification that are effectively stratiform and that are present at different stratigraphic levels within the conglomerate and younger fragmental units suggests a continuum of fluid infiltration, silica ponding, over-pressurization, and hydraulic fracturing. A Late Triassic uranium-lead age obtained by Pretivm on one of these silicified zones hosted within these Early Jurassic volcano-sediments therefore reflects the detrital age of one of the pebble or cobble clasts (most likely rhyolite). The intensely silicified zones have been intersected in VOK (particularly on the southern limb of the VOK syncline), as well as in West Zone, Gossan Hill, and Golden Marmot. These units were previously variably interpreted as submarine rhyolite flows (which is irreconcilable with the geochronology), dykes, sills, or chert horizons (McPhearson et al. 1994).

Figure 7.5 Brucejack Property Geology



Note: Enlarged legend provided in Figure 7.6

Source: Pretivm

Figure 7.6 Brucejack Property Geology Legend

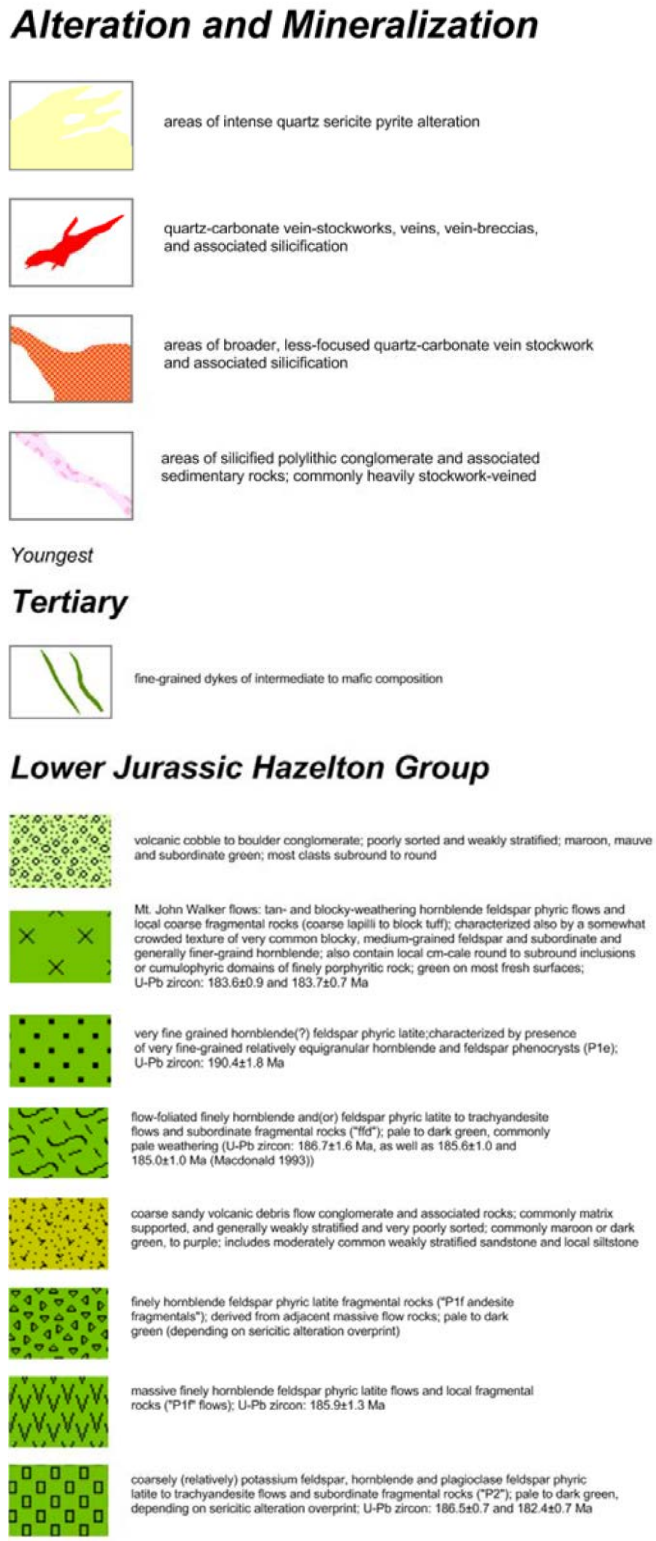


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Figure 7.6 *(con't)* Brucejack Property Geology Legend

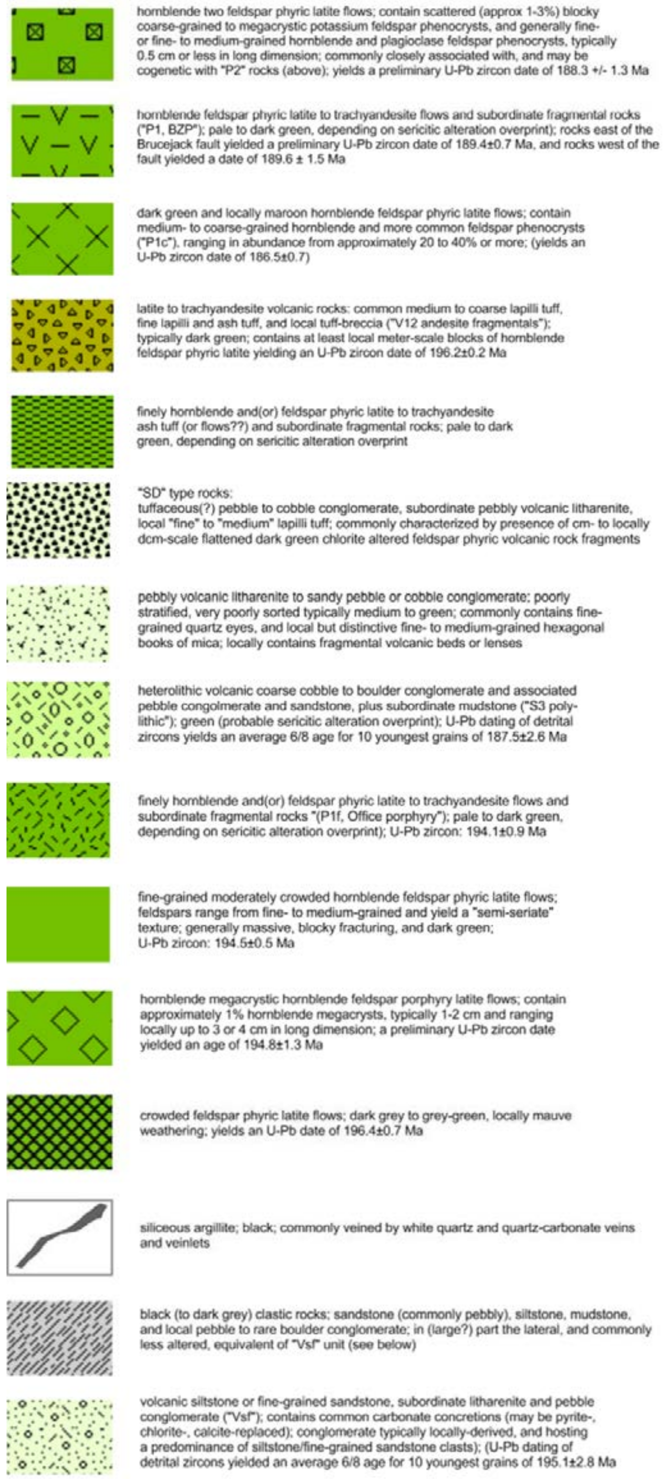


figure continues...

Figure 7.6 *(con't)* **Brucejack Property Geology (Figure 7.5) Legend**

Well-bedded green, maroon, and grey andesitic to dacitic pyroclastic and epiclastic rocks comparable to rocks of the Betty Creek Formation, are present to the northeast and southeast of Brucejack Lake, off the eastern edges of the Brucejack geological map in Figure 7.5.

There are almost no intrusive rocks east of the Brucejack Fault, other than a limited number of narrow post-mineral and post-tectonic amygdaloidal mafic dykes. This is in contrast to the increased abundance in intrusives noted west of the Property (Kirkham and Margolis 1995).

7.3.2 Alteration and Mineralization

The prominent gossanous alteration features on the Property define a distinctive north-south trending and west-concave arcuate belt that is generally located on the east of the Brucejack Fault. This belt is characterized by a broad band of variably but generally intensely quartz-sericite-pyrite altered rocks of up to several hundred metres or more across, and approximately 5 km in strike extent. The quartz-sericite-pyrite alteration typically contains between 2 and 20% pyrite, and, depending on the alteration intensity, can preclude protolith recognition.

High-grade gold (± silver) mineralization is predominantly located within this alteration band and is generally associated with vein-stockwork systems of varying intensity. These stockwork systems display good continuity along-strike (several tens of metres to several hundreds of metres) and are characterized by the presence of mesothermal to epithermal veins of quartz, quartz-carbonate, quartz-adularia, and pyrite that are typically on the order of millimetres to tens of centimetres in thickness and form intense crosscutting networks within the stockworks. In rare cases, the veins may range up to nearly 10 m in thickness. Most high-grade zones are either on the margins of, or contained within, a zone of bulk mineralization. Bulk low-grade mineralization zones (locally up to several grams per tonne gold) tend to be associated with disseminated anhedral pyrite. Zones and veins of euhedral pyrite are barren.

Vein mineralization includes trace to 10% combined disseminated pyrite, tetrahedrite, tennantite, arsenopyrite, chalcopyrite, galena, sphalerite, pyrargyrite, polybasite, acanthite, and rare native gold and electrum. The presence of base metals and/or arsenopyrite in veins and vein stockworks is only weakly correlated to gold mineralization, and therefore are not considered an indicator thereof. Where visible, gold in the form of electrum typically occurs as coarse aggregates or late stage fracture fillings, as rims on subhedral quartz crystals, or as lace-like networks formed interstitially to quartz and quartz-carbonate, as well as around coarse grains of adularia, where the latter is present. Seams of electrum up to a centimetre in thickness have also been observed in sericitized country rocks, with little obvious association to veins. Appreciable silver grades are generally present in mineralized zones where gold to silver ratios are less than 1:10; all of the known bonanza grade intersections on the Property have gold to silver ratios of roughly 2:1. Silver-dominant veins tend to be restricted in extent. High-grade silver mineralization occurs as silver sulphides and sulphosalts that are related to adularia-rich veins in which adularia pseudomorphs bladed calcite, indicative of epithermal conditions. Preliminary mineral zonation patterns (e.g. Au:Ag ratios, absence/presence of adularia and/or base metal veining, and meso- versus epithermal vein textures) on the Property suggest down temperature thermal gradients towards the east (i.e. up stratigraphy) and north. However, it is likely that a complex combination of temporal (overprinting by later stage lower temperature hydrothermal fluids) and spatial (distal, lower-temperature parts of the system) controls on mineralization probably occurred in response to the telescoping porphyry system, which did not necessarily affect all previously mineralized parts of the system equally.

Mineralized quartz veins, vein networks, and vein stockworks in and around the various mineral deposits on the Property have been affected by significant post-mineralization deformation. The various deformation features noted from the broader Sulphurets Mining Camp (Section 7.2.3) are developed throughout the mineralized zones on the Property. The nature, spatial, and geological associations of the mineralization within a zoned alteration environment proximal to known porphyry bodies with associated porphyry-style mineralization, coupled with the intense deformation of the bulk mineralized host-rocks, veins, and vein stockworks argue for the high-grade mineralization on the Property being formed in a porphyry-driven hydrothermal system, which was deformed during subsequent post-mineral tectonism. Vein development in response to the later tectonic event appears to be limited to late unmineralized shallow southeast-dipping veins and sigmoidal shear veins (Section 7.2.3).

The currently hypothesis for the mineralization on the Property is that it represents a deformed transitional meso- to epithermal porphyry-associated quartz stockwork in pervasively altered (quartz-sericite-pyrite; i.e. within the sericite alteration zone of Sillitoe 2010) lower Hazelton Group rocks. The bulk mineralization may have been formed shortly after consolidation of the volcanic pile due to reactions between these rocks and seawater (Margolis 1993), possibly as a result of hydrothermal fluid circulation driven by a distal and developing porphyry system. Progressive development and telescoping of the porphyry system in rocks of the volcanic pile would then have resulted in widespread and zoned porphyry-style alteration and mineralization, with the high-grade gold and gold-silver mineralization generated in a transitional meso- to epithermal environment slightly

more distal (i.e. down temperature) from the intrusive stock/dyke/sill body than the KSM and Snowfield deposits.

Thermal perturbations associated with pulsing in the porphyry system would likely have resulted in a succession of alteration and mineralization imprints and overprints within such a transitional and active environment and possibly induced upgrading and zonation of the precious metal mineralization, especially considering constraints on gold mobility in this environment (Gammons and Williams-Jones 1997).

More than 40 gossanous zones of gold, silver, copper, and molybdenum mineralization have been identified along the length of the arcuate band of altered rocks as a result of periodic exploration over the past several decades (Figure 7.7). A subset of these zones was selected for additional follow-up exploration in 2011 by Pretivm (Figure 7.8). A total of five zones were modelled for mineral resource estimation: Bridge Zone, VOK, West Zone, Gossan Hill, and Shore Zone. These zones range from deformed high-grade gold-rich, silver-poor zones such as VOK, through deformed high-grade gold-silver zones like West Zone, to high-tonnage but relatively low-grade zones like the Bridge Zone. Recent geological interpretation has shown that the area previously covered by Galena Hill is actually an extension of VOK and this area has now been incorporated into VOK.

Bridge Zone

The Bridge Zone is located about 1,500 m south of the West Zone and is centred on a 3 ha nunatak surrounded by ice of the eastern arm of the Sulphurets Glacier. Gold mineralization at the Bridge Zone is hosted by plagioclase-hornblende phyric volcanic rocks that are moderately to strongly sericite-chlorite altered, with disseminated and stringer pyrite making up a few percent of the rock by volume. Mineralization occurs both in low-grade bulk tonnage associated with the altered host rocks, and in deformed quartz vein and vein stockwork associated moderate to high-grade styles. The moderate- to high-grade mineralization is generally associated with east-west trending quartz vein and vein stockworks that dip steeply to the north.

Quartz ± chlorite ± sericite veins, ranging from 20 cm to 200 cm in thickness, are fairly common and contain minor to trace amounts of pyrite, sphalerite, galena, molybdenite, and an unknown dark grey, possibly silver-bearing sulphosalt (or salts). Some of the veins contain appreciable concentrations of molybdenum and rhenium, with the molybdenum/rhenium ratio similar to that at Pretivm's Snowfield Deposit.

The pervasive nature of the mineralization in the Bridge Zone, the association of this mineralization with molybdenum and rhenium, and the spatial proximity of the intensely hydrothermally altered and mineralized southern limb of the VOK syncline (immediately to the north of the Bridge Zone), suggests that the Bridge Zone rocks were relatively proximal to an intrusive body (stock/sill/dyke/other narrow apophysis) of the mineralizing porphyry system.

VALLEY OF THE KINGS

High-grade gold mineralization in VOK, the current focus of the Project, occurs in a series of west-northwest trending sub-vertical corridors of structurally reoriented vein stockworks and subordinate vein breccias, hosted in a folded sequence of fine grained volcanic siltstone and sandstone, variably silicified polylithic volcanic conglomerate, and fragmental latite volcanic rocks (Figure 7.9, Figure 7.10, and Figure 7.11). Relatively massive latite flows are present to the immediate north and south of this mineralization host rock sequence.

Gold is typically present as gold-rich electrum within deformed quartz, and quartz-carbonate (±adularia?) vein stockworks, veins, and subordinate vein breccias, with grades ranging up to 41,582 ppm Au and 27,725 ppm Ag over 0.5 m. The gold occurs both as inclusions in euhedral pyrite, as well as interstitially in textural equilibrium with strained quartz (exhibiting primary recrystallization textures), carbonate, several generations of pyrite, and, to a lesser extent, vein-hosted sericite. Gold has also been found in the central parts of deformed quartz veins, as dendritic lattice works pervasive throughout deformed quartz, and quartz-carbonate veins, as well as in the matrix of hydrothermal breccias. The various textural associations of gold suggest a multi-stage paragenesis for gold mineralization.

The relatively common association of gold and pyrite suggests a link in their precipitation. The presence of pyrite in a porphyry-epithermal environment indicates elevated levels of hydrogen sulphide in the magmatic-hydrothermal fluids, which suggests gold solubility and mobility as bisulphide complexes (Gammon and Williams-Jones 1997). Destabilization of the bisulphide complex during pyrite precipitation, brought on by changes to one or more physicochemical parameters (e.g. change in pH, oxidation state, fluid mixing, depressurization, cooling, sulphur activity), would result in gold precipitation. Gold is also present in veins and silica-flooded host rocks that do not contain pyrite. This indicates that gold precipitation is not solely linked to pyrite, and that other factors have also played a role in controlling gold mineralization during the genesis of the VOK deposit. Early pyrite precipitation during initial porphyry development is ubiquitous in the host rocks to the VOK mineralization. This would have created a favourable environment for gold remobilization, enrichment, and transportation as bisulphide compounds above the telescoping porphyry system, as the hydrothermal fluids would effectively be unbuffered by the already altered host rocks (Gammon and Williams-Jones 1997). An increase in the residency time of the cooling hydrothermal fluid in previously mineralized rocks (mineralized during the earlier porphyry phase) beneath siliceous caprocks, where the gold is transported as bisulphides is considered a key factor in the generation of the elevated grades in the VOK deposit.

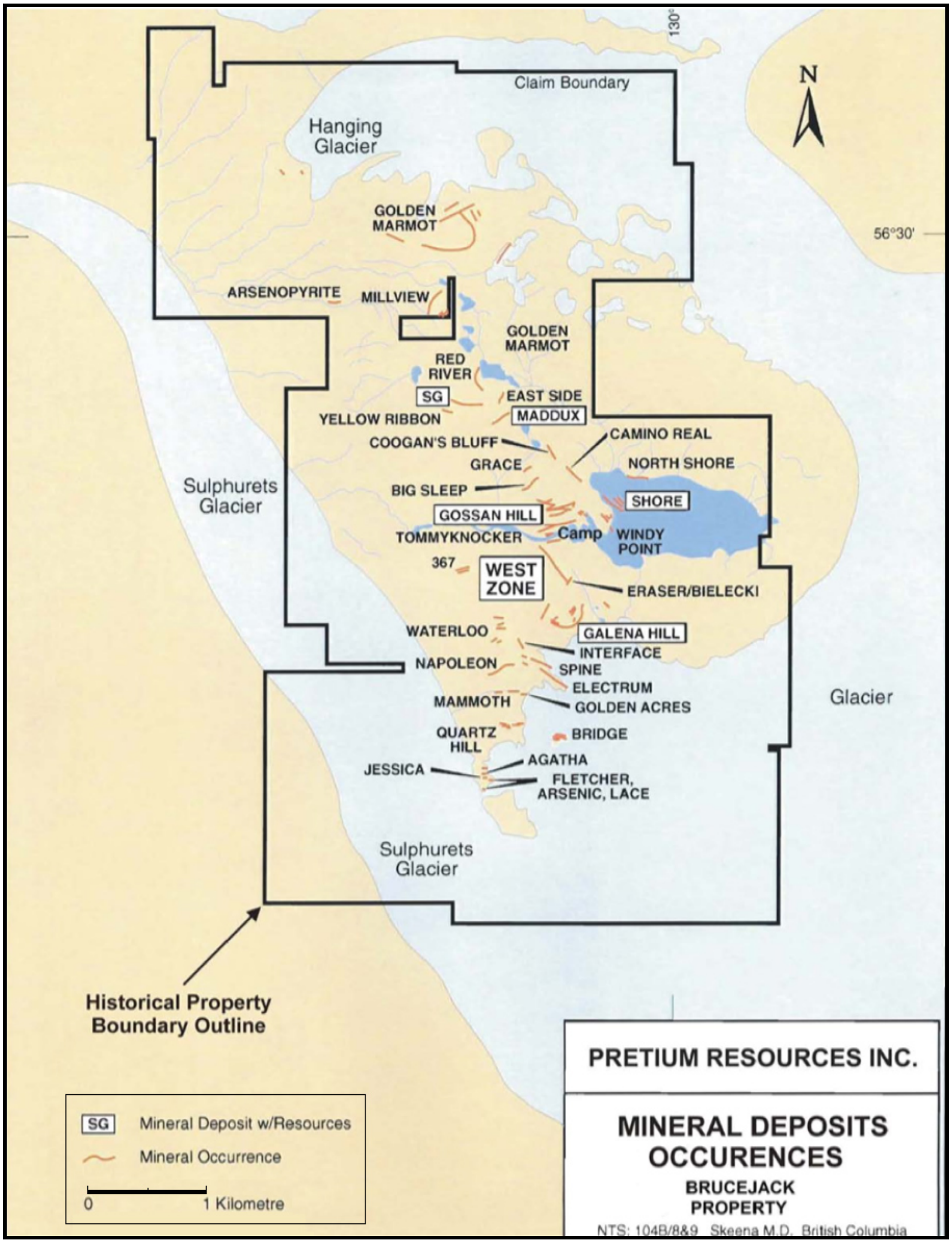
While quartz veining and stockworks are common throughout the zone, the majority of significant gold intersections are confined to corridors within a 100 m to 125 m wide zone on the southern limb of the syncline. The orientation of these corridors is sub-parallel to the fold axis, with some of the stockworks being spatially associated with intensely silicified zones developed in and at the contacts of the polylithic conglomerate, as well as in the overlying fragmental volcanic rocks. Asymmetrical alteration is present on either side of the intensely silicified conglomerate zones, with (the underlying rocks

more heavily altered than those above, being almost completely metasomatized to green sericite in places. Coupled with evidence for hydraulic and tectonic fracturing and brecciation of the siliceous zones and previously-formed stockwork and veins, these features suggest that the siliceous zones formed in response to silica flooding of the preferentially permeable conglomerate, and then acted as local pressure seals that were eventually brecciated as a response to local overpressure conditions. The intensely silicified zones generally appear to be sub-parallel to the stratigraphy.

Au:Ag ratios within the VOK are typically 2:1 or higher. Variations in this ratio, which could be a function of thermal gradients developed at the time of mineralization, are suggested by a visible increase in the proportion of silvery electrum (at the expense of more gold-coloured electrum) with a concomitant increase in the proportion of vein-hosted adularia towards the eastern parts of the zone. Additional precious metal-bearing minerals found in the VOK, typically in trace quantities, include silver sulphides, acanthite, pyrargyrite and tetrahedrite, and associated base metal-bearing sulphides include sphalerite and galena. Low-grade bulk tonnage mineralization, associated with disseminated anhedral pyrite, forms a halo within the altered rocks, surrounding the high-grade mineralization corridors.

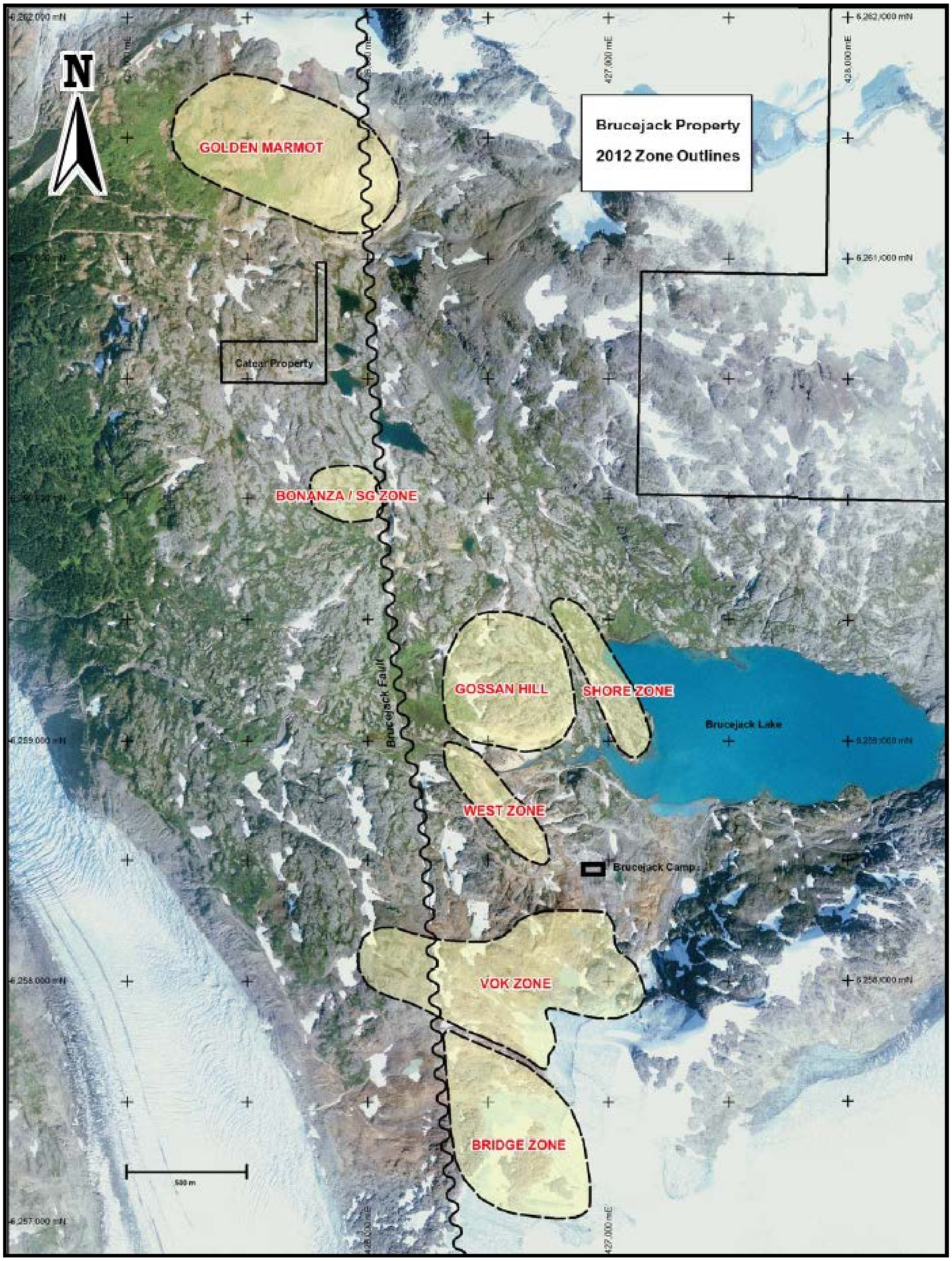
The VOK Deposit is currently defined over 1,200 m in east-west extent, 600 m in north-south extent, and 650 m in depth. Drilling during the 2012 drilling program resulted in the extension of the VOK west across the Brucejack Fault, thereby making the zone open to the west, as well as to the east and at depth.

Figure 7.7 Historical Map with Mineral Deposits and Occurrences



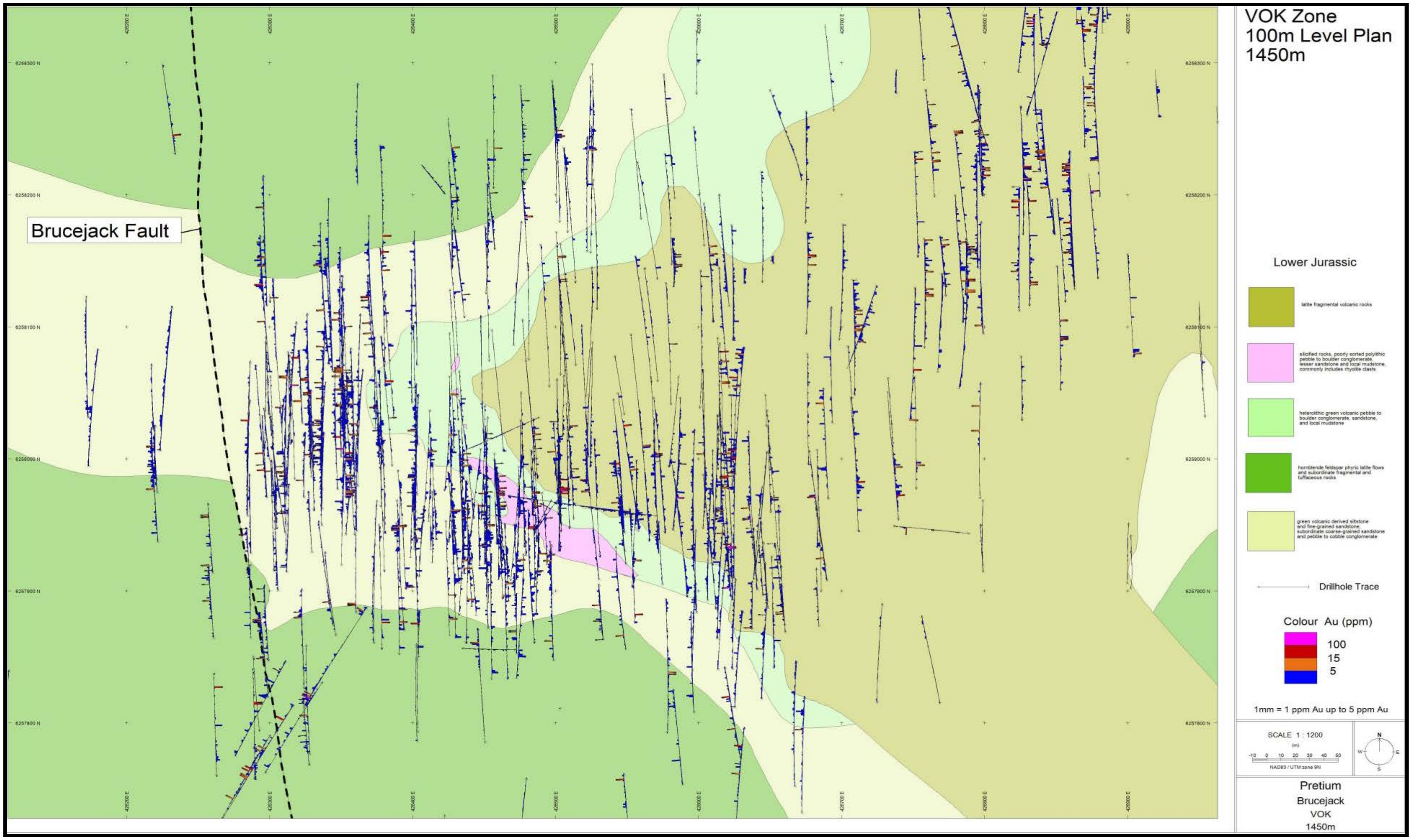
Note: Modified after Budinski (1995)
Source: Ghaffari et al. (2012)

Figure 7.8 Brucejack Property Mineralization Zones



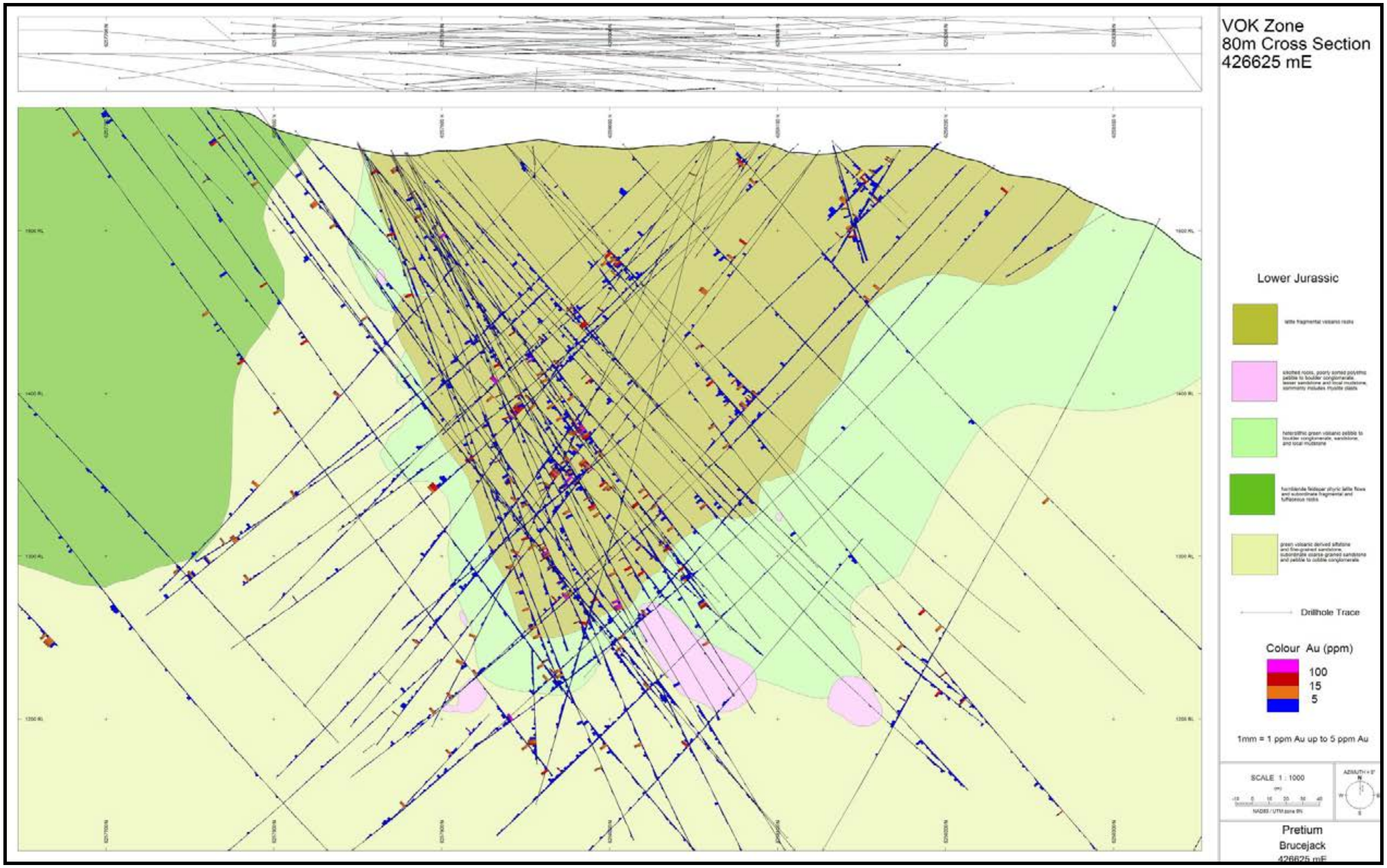
Source: Pretivm

Figure 7.9 1,450 m Level Plan Geology of the VOK



Source: Pretivm

Figure 7.10 South-north Cross-section Along Easting 426625 E, VOK



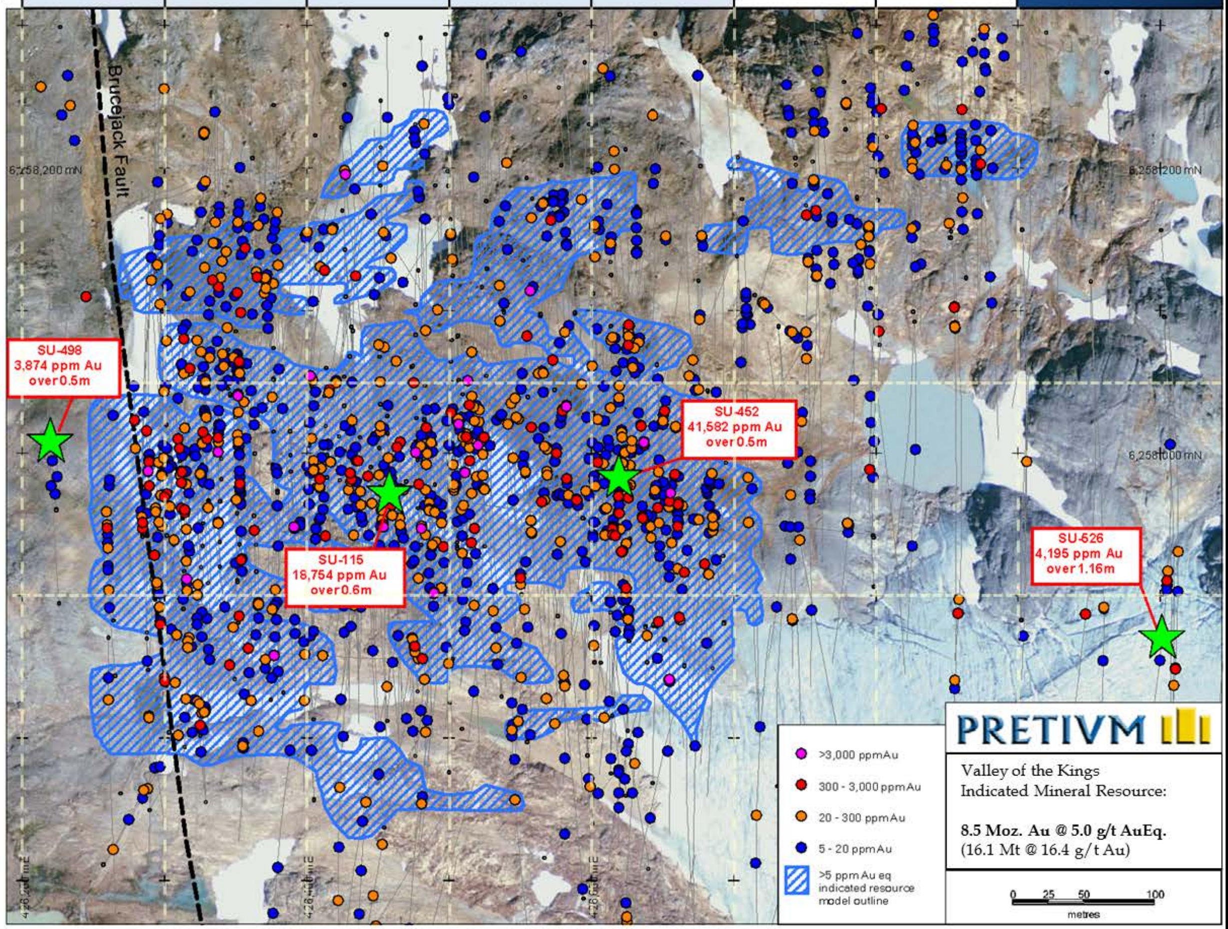
Source: Pretivm

Figure 7.11 Plan View of VOK Showing Concentration of High-grade Intersections on the Southern Limb of Syncline

1,092,100	1,776,900	1,467,600	2,185,800	1,391,000	380,100	192,900	Ind. Au* (Oz.)

Northing (± 25 m)	Ind.Gold* (Oz.)
6258225	230,900
6258175	484,000
6258125	703,100
6258075	336,600
6258025	1,951,100
6257975	2,365,900
6257925	1,021,000
6257875	513,600
6257825	687,500
6257775	176,400

*Nov. 2012 Indicated Mineral Resource; ≥5.0 g/t AuEq.



Total Ounces per 100 m Easting and 50 m Northing strips

WEST ZONE

The West Zone gold-silver deposit (Figure 7.12 and Figure 7.13) is hosted by a north-westerly trending band of intensely altered Lower Jurassic latitic to trachyandesitic volcanic and subordinate sedimentary rocks, as much as 400 m to 500 m thick, which passes between two more competent bodies of hornblende plagioclase hornblende phyric flows (Figure 7.12). The stratified rocks dip moderately to steeply to the northeast and are intensely altered, particularly in the immediate area of the precious metals mineralization. The West Zone appears to form the northern limb of an anticline that links up with the VOK to the south (Figure 7.12), and the southern limb of a syncline that extends further to the north.

The West Zone deposit itself comprises at least 10 quartz veins and mineralized quartz stockwork ore shoots, the longest of which has a strike length of approximately 250 m and a maximum thickness of about around 6 m. Most mineralized shoots have vertical extents that are greater than their strike lengths. Veins and stockworks in this zone display clear evidence of post-mineral ductile and brittle deformation. The West Zone is open along strike to the southeast, and at depth to the northeast.

In terms of hydrothermal alteration, the West Zone is marked by a central silicified zone that passes outwards to a zone of sericite ± quartz ± carbonate and then an outer zone of chlorite ± sericite ± carbonate. The combined thickness of the alteration zones across the central part of the deposit is between 100 m and 150 m.

Gold in West Zone occurs principally as electrum in quartz veins and is associated with, in decreasing order of abundance, pyrite, sphalerite, chalcopyrite, and galena. Besides being found with gold in electrum, silver occurs in tetrahedrite, pyrargyrite, polybasite and, rarely, stephanite and acanthite. Gangue mineralogy of the veins is dominated by quartz, with accessory adularia, albite, sericite, and minor carbonate and barite. The increased abundance of silver in West Zone may suggest that this zone was formed down temperature gradient from the VOK (either spatially or temporally). The West Zone is open to the southeast, northwest, and northeast (i.e. towards Gossan Hill).

GOSSAN HILL

The mineralized zone known as Gossan Hill is a circular area, about 400 m in diameter, of intense quartz-sericite-pyrite alteration developed in Lower Jurassic volcanic rocks (Figure 7.7 and Figure 7.8). The visually impressive alteration zone at Gossan Hill is host to at least 11 deformed quartz vein and quartz vein stockwork structures, most of which trend east-west and dip steeply to the north. Individual structures are up to 250 m long and 20 m thick.

Precious metal mineralization at Gossan Hill occurs both as low-grade bulk tonnage and high-grade styles. The low-grade bulk tonnage style is associated with fine quartz stockworks and anhedral pyrite. Higher-grade gold mineralization at Gossan Hill differs somewhat from other zones on the Property in that it is associated with the larger quartz lenses, particularly where they contain local aggregates of pyrite, tetrahedrite, sphalerite, and galena. Electrum is observed in the bonanza grade intersections, while silver also

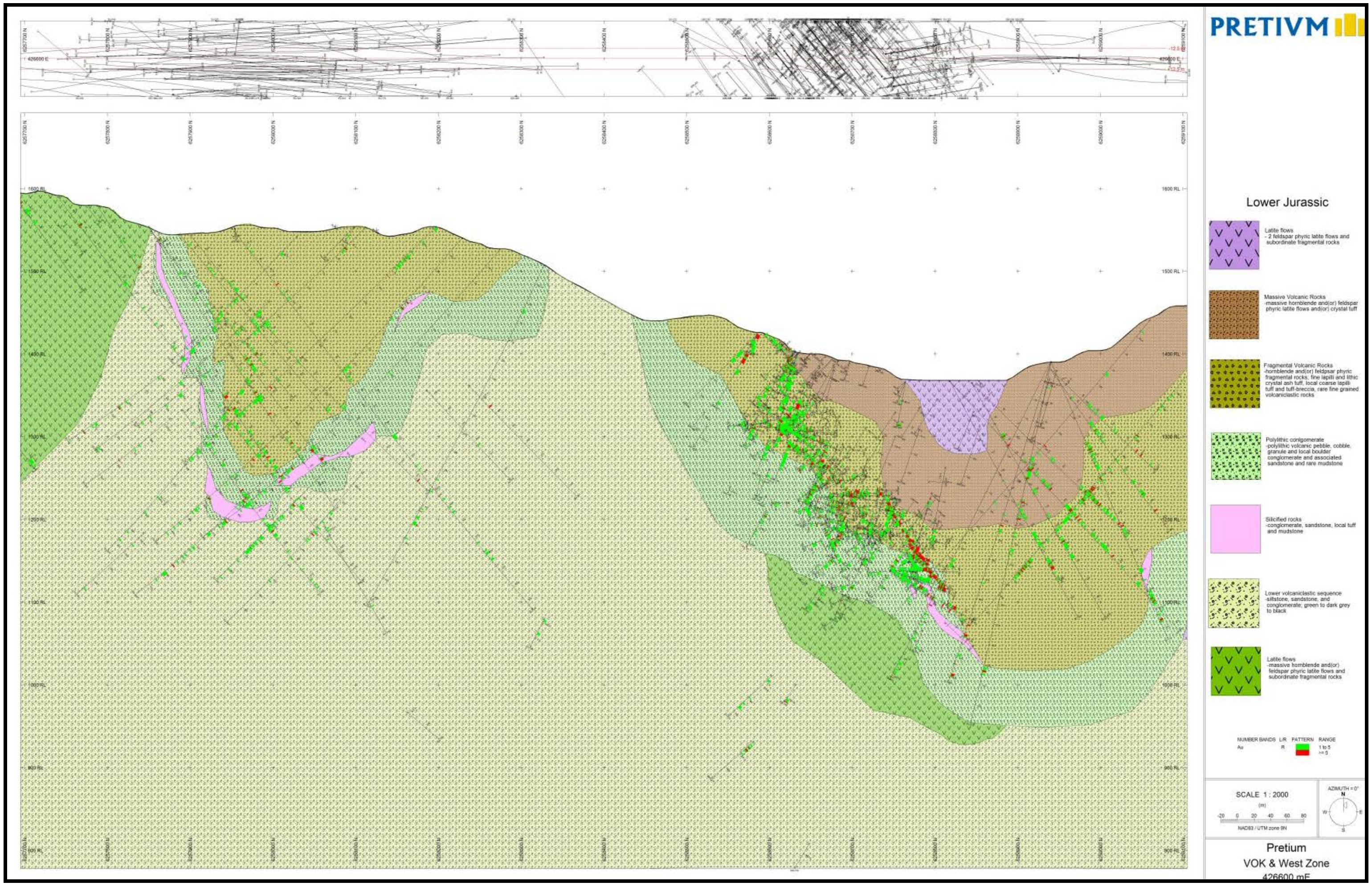
occurs in tetrahedrite, pyrargyrite, and polybasite. The Gossan Hill deposit remains open along strike, across strike, and at depth.

SHORE ZONE

The Shore Zone is a relatively small gold-silver deposit located along the northeastern shore of the peninsula that extends into the west end of Brucejack Lake (Figure 7.7 and Figure 7.8).

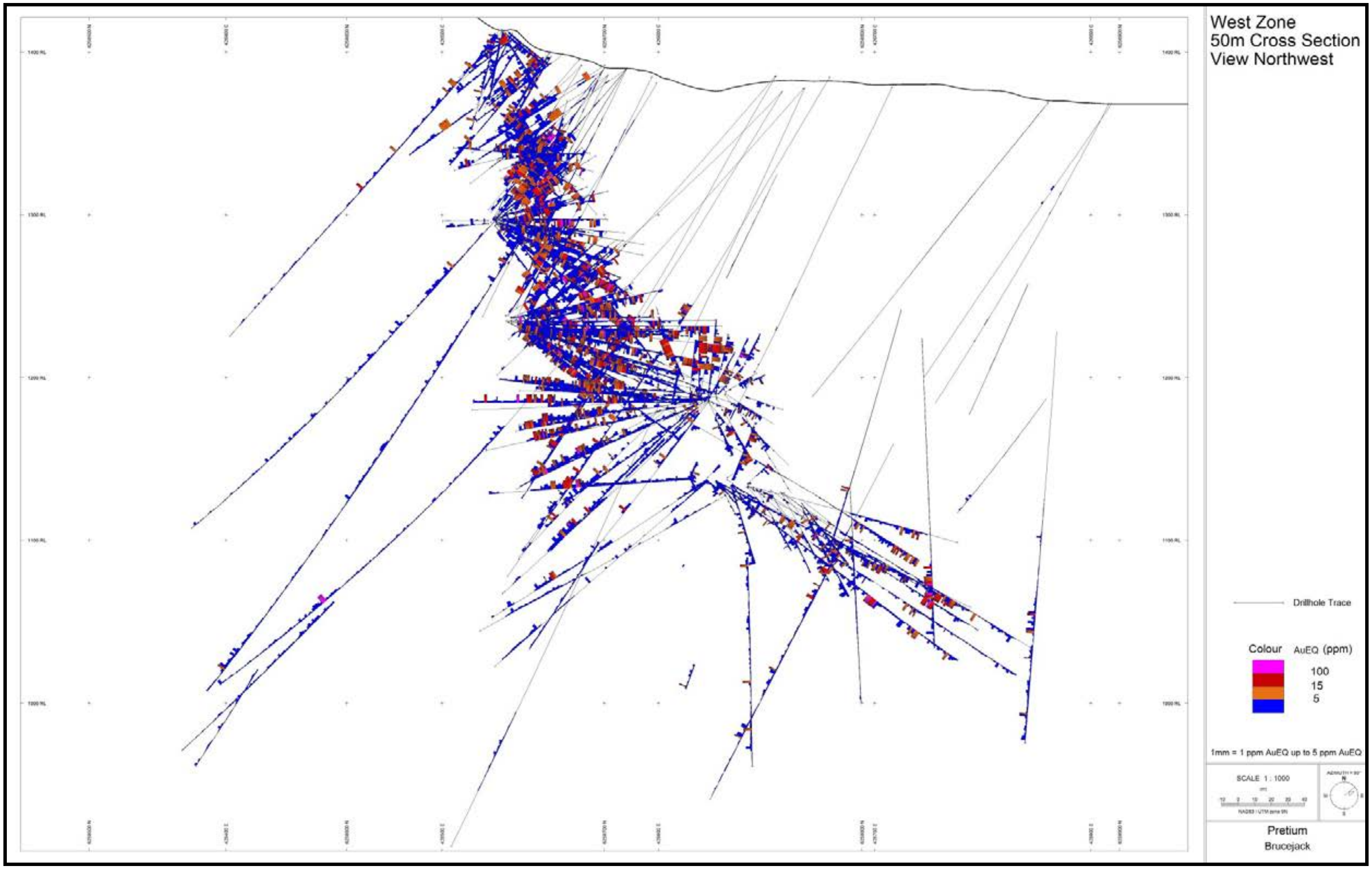
This zone is characterized by deformed quartz veins and quartz vein stockworks up to 100 m in length, which are hosted in deformed and quartz-sericite-pyrite altered trachyandesite, sandstone and pebble conglomerates of the lower Hazelton Group. The zone has a strike length of approximately 530 m and a maximum width of 50 m. The northwest-southeast trend of the zone is coincident with a pronounced lineament that extends south-eastward from the Brucejack Fault beneath Brucejack Lake, and which is likely a fault. It is likely that the Shore Zone forms the eastern limb of the meso-scale syncline, linking up with the West Zone underneath the Gossan Hill.

Figure 7.12 VOK to West Zone Geological Section 426600 E - Looking West



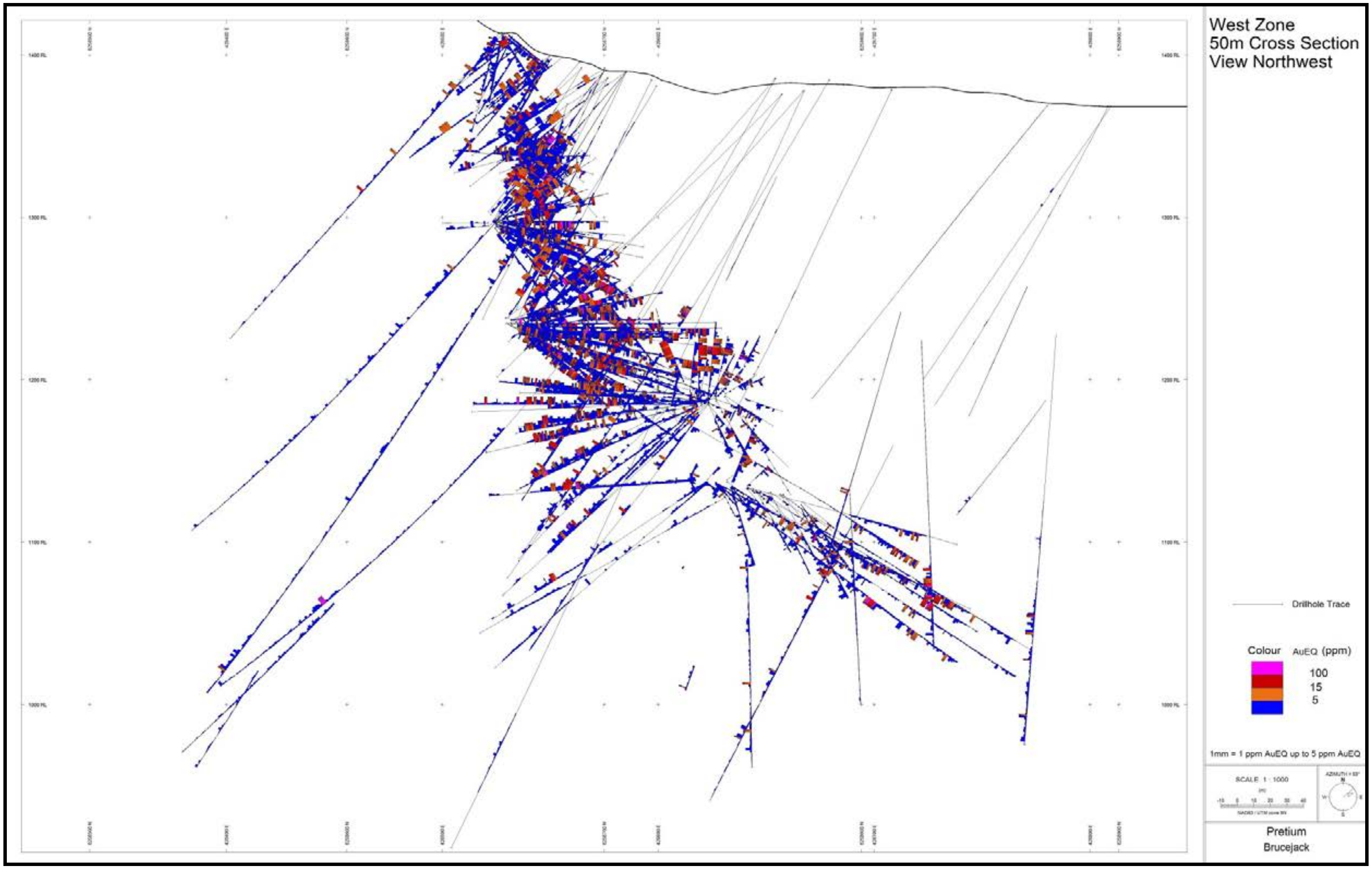
Source: Pretivm

Figure 7.13 West Zone Drillholes and Assay Cross-section



Source: Pretivm

Figure 7.14 West Zone Drillholes and Assay Cross-section



Source: Pretivm

Given the intense folding displayed by the lower Hazelton Group rocks in this area, as well as the variable competency of the rocks in this part of the Property, it is likely that a combination of ductile deformation and rock competency differences controlled the orientation of this zone. Way-up structures in steeply northeast dipping sedimentary units in the Shore Zone indicate that it likely forms the northern limb of a parasitic southeast plunging anticline. The aforementioned lineament likely utilized the ductile deformation-prepared northwest-trending zone of structural weakness for propagation during late brittle deformation.

The veins and vein stockworks consist predominantly of quartz with minor carbonate and barite, with patchy sulphide mineralization consisting of variable quantities of pyrite, tetrahedrite, sphalerite, galena, and arsenopyrite. Electrum has been observed in trace amounts. Silver is present in some of the highest concentrations observed in the Property. This observation, together with the stratigraphic (up stratigraphy) and spatial position (i.e. relatively far northeast of the Bridge Zone) of the Shore Zone, provides further evidence for thermal gradient-induced mineral zonation across the Property.

7.3.3 Structure and Metamorphism

Hazelton Group rocks and mineralized vein stockworks on the Property display significant multi-phase post-mineralization deformation. The Brucejack is characterized by the presence of steep structural elements, including steeply dipping planar features such as bedding, foliation, and brittle faults, and steeply plunging linear features, such as fold hinges, pencil cleavage, or mineral lineation. These structural elements are associated with the Late Jurassic to Late Cretaceous SFB deformation, and deform unaltered and altered rocks, as well as mineralized veins and stockworks.

Foliation

Foliation on the Property is pervasive, although it is best developed in the most intensely altered rocks. The foliation is defined by muscovite in altered rocks, and by sericite and chlorite in less altered rocks. The foliation displays a dominantly east-west trend across the Property, with a sub-vertical dip that is generally to the north, but which does vary about the vertical.

Foliation orientation appears to be locally controlled by the presence of proximal competent rock masses that acted as strain resistors during deformation (e.g. the relatively competent hornblende feldspar phyric volcanic flow that forms the southern and south-western margin of the West Zone probably controlled the northwest trend of the foliation in this area).

Within the broader zones of veining, individual veins, veinlets, and narrower stockwork zones have been partially to completely reoriented sub-parallel to the foliation in the host rocks. The most intensely foliated rocks tend to be altered rocks immediately adjacent to the veins and stockworks.

A second, locally developed foliation has been observed in the footwall to West Zone, Gossan Hill, and Golden Marmot. The development of this foliation, which is also typically

steep, is generally associated with the most intensely foliated and altered lithologies. A steeply southeast plunging intersection cleavage lineation (or pencil structure) is often associated with the presence of the second foliation, and is sub-parallel to mesoscale fold axes. Pencil cleavage is also developed at the intersection of steep (and commonly curvilinear) joint sets with the steep foliation.

Folding

Rocks on the Property have been folded into a series of tight, moderate to steeply east- to southeast-plunging south-vergent synclines and anticlines, with wavelengths on the order of approximately 100 m. Smaller-scale parasitic folds (few metres scale) with similar orientations to these folds are locally developed. These folds are generally quite difficult to recognize in the field, and have been delineated using both field observations and lithological domain trace element analyses (Figure 7.7 and Figure 7.15). They tend to be best delineated in the field by following the contacts between older, predominantly clastic rocks and the younger, predominantly volcanic flows and associated coarse volcanic fragmental rocks of the lower Hazelton Group. The east-west trending fabric, which generally tends to be axial planar to these folds is interpreted as having formed at the same time as this folding event. The variation in intensity of folding between less competent clastic rocks and more competent volcanic flows suggests preferential strain partitioning in the less competent rocks during deformation. The more competent rocks possibly acted as strain resistors, which locally controlled folding of the less competent rocks (e.g. the westward tightening up of the syncline in the VOK between the two porphyritic flows in this part of the Property).

The plunge of the minor folds varies and several lines of evidence suggest that they may reflect refolding of northerly-trending tight, upright “early” mid-Cretaceous SFB-age folds across roughly east-west axes. These later folds and the east-west foliation are spatially associated with the area of the footwall (and immediate hanging wall) of the regional-scale Mitchell thrust system, suggesting that they may have developed in the latter stages of SFB development.

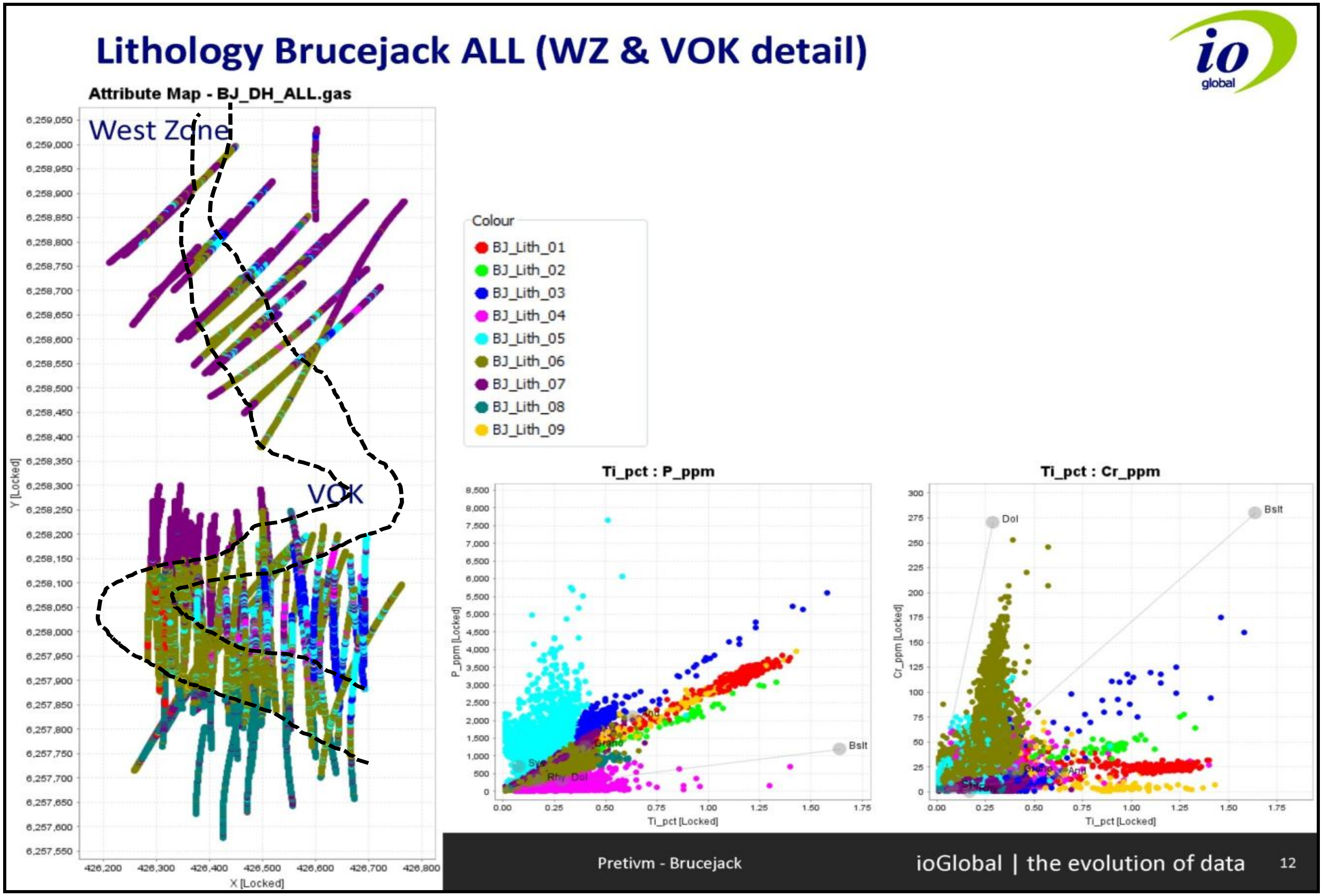
Brittle Faulting

Across the Property, steep post-mineral late brittle faults are present, which cut deformed unaltered and altered rocks as well as deformed mineralized veins and vein stockworks. Many form well-defined lineaments, with that of the Brucejack Fault being the most prominent. Few have any well-defined offsets, and most offsets appear relatively minor (less than several tens of metres). An exception is noted in the area near the western margin of the Sulphurets Glacier, to the northwest of Brucejack Creek, where clean and well-exposed outcrops show apparent dextral offsets of up to 20 m or more along north-trending faults occupying lineaments that are sub-parallel to the Brucejack Fault. Numerous smaller-scale north-trending brittle features on the Property show similar apparent displacements. Discussion of the offset along the Brucejack Fault is presented in Section 7.2.3.

METAMORPHISM

The rocks on the Property appear to have experienced low-grade metamorphism (lower greenschist facies, or lower) around 110 Ma associated with the SFB deformation. Further discussion on the metamorphism of the rocks of the Sulphurets Mining Camp is provided in Section 7.2.3.

Figure 7.15 Trace Element Analysis by Lithology for VOK and West Zone



7.3.4 GEOCHRONOLOGY

Uranium-lead zircon and rhenium-osmium molybdenite age dates have been obtained from suitable geologically-constrained surface and drill core samples collected from across the Property (Figure 7.16). Zircon age dating was conducted at the Pacific Centre for Isotopic and Geochemical Research (PCIGR) analytical facility, Department of Earth, Ocean, and Atmospheric Sciences (EOAS), University of British Columbia. Molybdenite age dates were determined at the Radiogenic Isotope Facility (RIF), Department of Earth and Atmospheric Sciences, University of Alberta, having been contracted through ALS in North Vancouver.

Magmatic zircons constrain the volcano-sedimentary rock sequence underlying the Property to between c.196 Ma and c.182 Ma, consistent with previous stratigraphic interpretations that placed these rocks in the Lower Jurassic Lower Hazelton Group. Detrital zircons hosted in immature volcaniclastic conglomeratic rocks display a range of Triassic and Jurassic ages (from c.222 Ma to c.183 Ma), which are interpreted as indicating uplift and erosion of an earlier island arc assemblage (Stuhini Group) during the formation of the Lower Jurassic Hazelton Group. This is consistent with the presence of a regional-scale angular unconformity between rocks of the Stuhini and Hazelton Groups, as well as the volcano-sedimentary growth basin interpretation for the Lower Hazelton Group rocks on the Property. Both magmatic and detrital zircons display a decrease in age towards the east across the Property, consistent with the regional geological way up on this (the eastern) side of the McTagg Anticlinorium.

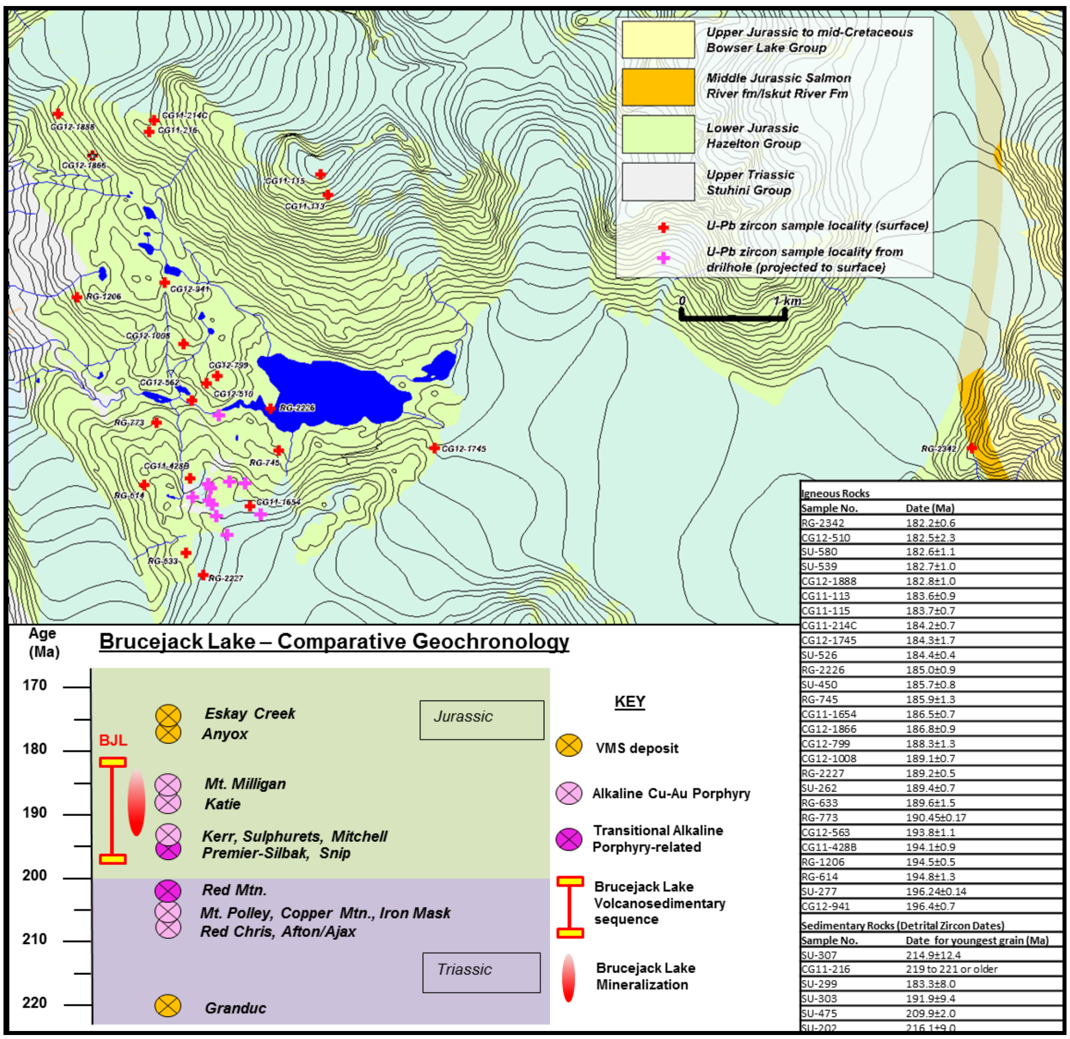
Rhenium-osmium ages obtained from two vein-hosted molybdenite samples yielded age dates of 191.5 ±0.8 Ma and 190.2 ±0.8 Ma. These samples were collected from drillhole SU-151 in the Bridge Zone, which is directly south of VOK, and represent the oldest mineralization age reported from the Property. Rhenium-osmium age dates collected from Pretivm's Snowfield porphyry gold-copper deposit indicate an age of 191.1 ±0.8 Ma for porphyry mineralization, which is statistically indistinguishable from the Brucejack molybdenite age dates. This indicates that the hydrothermal system responsible for the molybdenum mineralization in the porphyritic flows in the Bridge Zone was contemporaneous to the porphyry system developed in the upper (i.e. Snowfield) parts of the Mitchell stock. This provides further evidence for the suggested link between at least the onset of magmatic-hydrothermal mineralization on the Property and a subjacent porphyritic intrusive. The similarity in molybdenum/rhenium ratios in rocks from the Bridge Zone to those in the Snowfield Deposit further supports this contention.

Uranium-lead zircon age date of 182.7 ±1.0 Ma obtained for a deformed post-mineral mafic dyke provides a minimum age for the mineralization, with the youngest altered and mineralized flows dated at c.185 Ma. These data indicate that the mineralizing systems were relatively long-lived, spanning approximately 8 Ma, and coeval with island arc volcanism. Sillitoe (2010) notes that porphyry deposit clusters, like those in the vicinity of the Property, may remain active for 10 Ma or longer. The Brucejack geochronological data are consistent with field observations that the volcanic basin formation and mineralization were pre-tectonic with respect to the pervasive Cretaceous deformation,

for which an age of c.110 Ma has been reported in the literature (e.g. Kirkham and Margolis 1995).

The host rock and mineralization ages from the Property overlap with dates determined for known alkaline porphyry copper-gold deposits in the Intermontane Belt (Figure 7.16), particularly those of the nearby KSM deposits. The spatial, stratigraphic, and geochronological association between the Brucejack deposits and the KSM porphyry intrusive rocks suggest a genetic link between the high-grade gold mineralization at Brucejack and the KSM deposits. However, the age constraints on the Brucejack hydrothermal system indicate it may have been driven, in part, by a somewhat younger and relatively long-lived porphyritic stock, or a series of successive porphyritic stocks, emanating from the same deep-seated arc-related magmatism that led to the formation of the slightly older KSM deposits.

Figure 7.16 Brucejack Property Geochronology

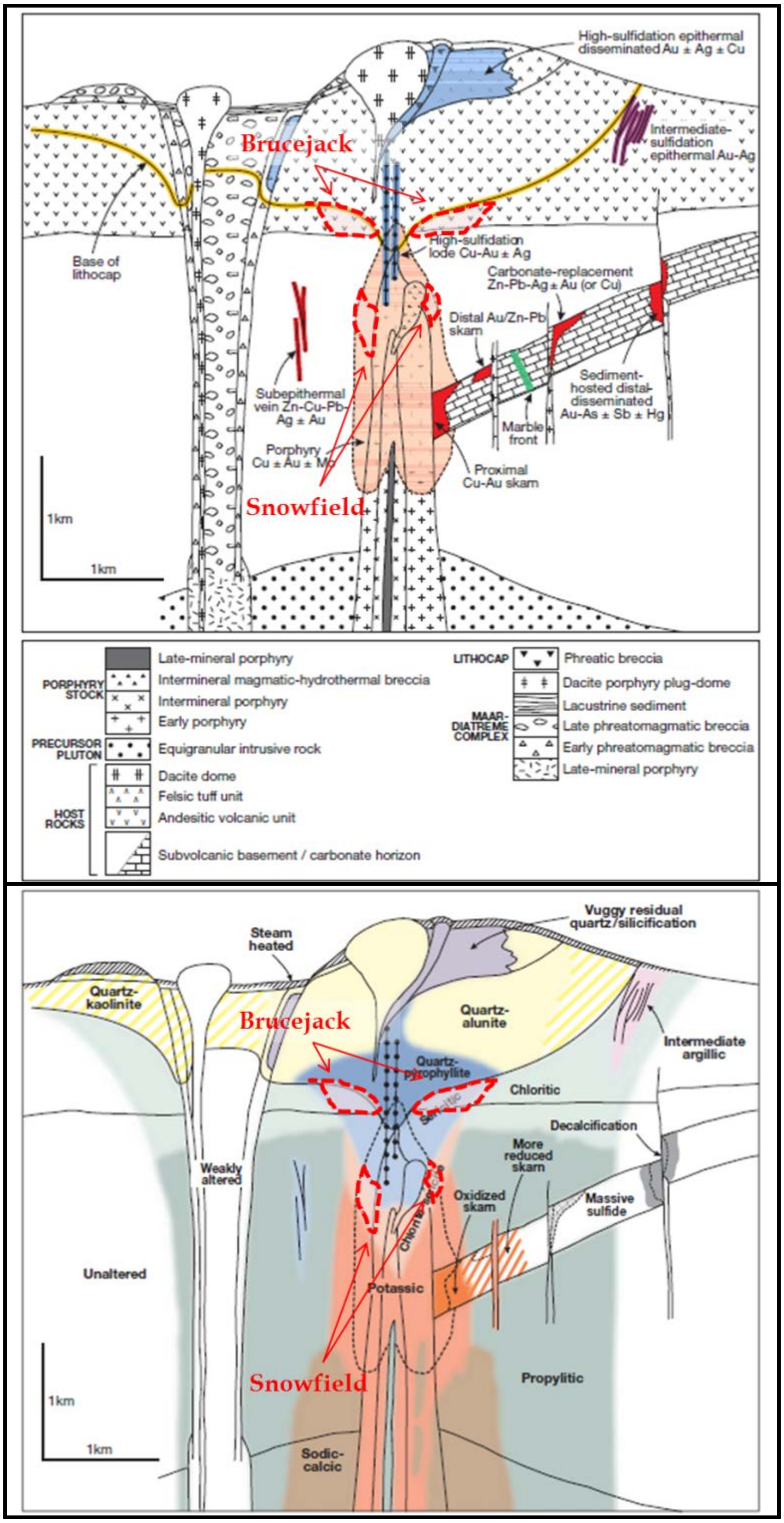


Igneous Rocks	
Sample No.	Date (Ma)
RG-2342	182.2±0.6
CG12-510	182.5±2.3
SU-580	182.6±1.1
SU-539	182.7±1.0
CG12-1888	182.8±1.0
CG11-113	183.6±0.9
CG11-115	183.7±0.7
CG11-214C	184.2±0.7
CG12-1745	184.3±1.7
SU-526	184.4±0.4
RG-2226	185.0±0.9
SU-450	185.7±0.8
RG-745	185.9±1.3
CG11-1654	186.5±0.7
CG12-1866	186.8±0.9
CG12-799	188.3±1.3
CG12-1008	189.1±0.7
RG-2227	189.2±0.5
SU-262	189.4±0.7
RG-633	189.6±1.5
RG-773	190.45±0.17
CG12-563	193.8±1.1
CG11-428B	194.1±0.9
RG-1206	194.5±0.5
RG-614	194.8±1.3
SU-277	196.24±0.14
CG12-941	196.4±0.7
Sedimentary Rocks (Detrital Zircon Dates)	
Sample No.	Date for youngest grain (Ma)
SU-307	214.9±12.4
CG11-216	219 to 221 or older
SU-299	183.3±8.0
SU-303	191.9±9.4
SU-475	209.9±2.0
SU-202	216.1±9.0

8.0 DEPOSIT TYPES

Based on the geological discussions presented in Section 7.0, the gold-silver quartz (± carbonate, barite, adularia) and minor base metal (galena, sphalerite, and rare chalcopyrite) veins and vein stockworks of the Brucejack Deposit are considered as having been formed in a transitional meso- to epithermal porphyry-associated quartz stockwork system in pervasively altered lower Hazelton Group rocks (quartz-sericite-pyrite alteration as per the alteration zone of Sillitoe, 2010; Figure 8.1) between 192 Ma and 184 Ma. Progressive development and telescoping of a porphyry system in the volcanic pile resulted in a widespread zonation of porphyry-style alteration and mineralization, and multiple stages of vein and alteration overprinting. Gold concentration and subsequent deposition probably occurred as a result of complex interactions between various physicochemical parameters (e.g. pressure, temperature, pH, activities of oxygen, sulphur and other volatiles, concentration of dissolved salts, differential permeability of the volcanic pile) in the magmatic-heated seawater hydrothermal system developed above the pulsing porphyry system. Metal deposition was likely triggered by a combination of structural preparation, depressurization, cooling, phase separation, solution mixing, and fluid-host rock interactions. Geological, mineralization, and alteration features of the Snowfield Deposit (Armstrong et al. 2011) suggest that this deposit is more proximal to the porphyry apophysis, most likely in the chlorite-sericite alteration zone of Sillitoe (2010) (Figure 8.1).

Figure 8.1 Brucejack Deposit Mineralization within Context of Porphyry Systems



Source: Modified after Sillitoe (2010)

9.0 EXPLORATION

Information in this section has been excerpted from the Olssen and Jones (2012c), which was sourced from sections within Ghaffari et al. (2012) and updated.

In September 2011, Quantec Geoscience was contracted to undertake a Spartan magnetotelluric (MT) survey of the Snowfield and Brucejack Properties. The exploration objectives were to map and detect porphyry mineralization to depth within the Snowfield and Brucejack Projects, and to establish an understanding of the geological system and fluid pathways to great depth within the Snowfield and Brucejack survey area (Gharibi et al. 2011).

Approximately 57 lines were surveyed at a spacing of 500 m.

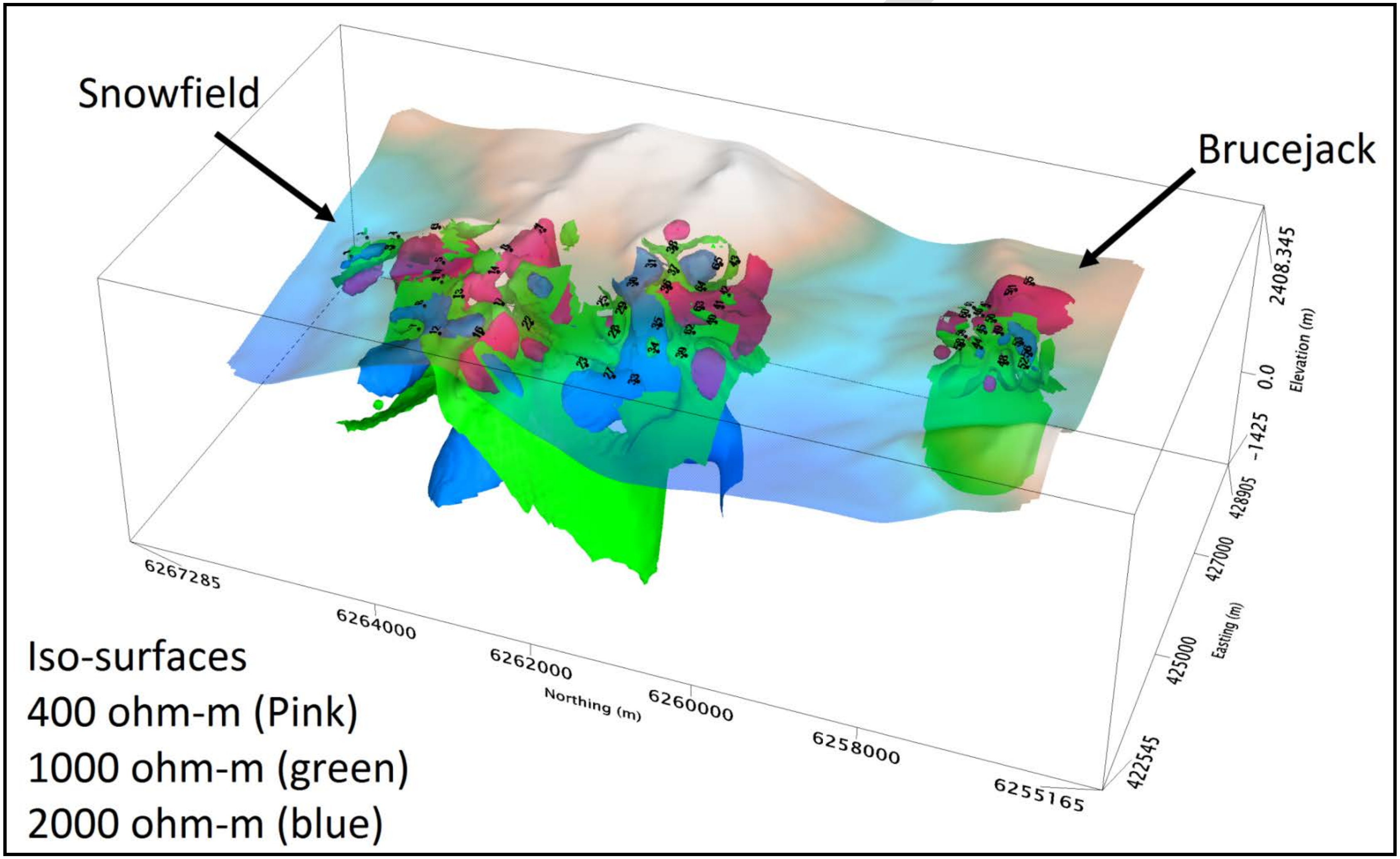
The following definition of MT is taken from Quantec Geoscience's report (Gharibi et al. 2011):

> *The (MT) method is a passive/inductive method which measures the time-variations in the Earth's natural electric (E) and magnetic (H) fields to image the subsurface resistivity below the sounding site. No source or transmitter is used. The E and H fields are measured over a broad range of frequencies from 10 kHz to 1Hz (worldwide lightning activity), and from 1Hz to 0.001Hz (oscillations of the Earth's ionosphere as it interacts with the solar wind). While the E and H fields are random, (solar wind and lightning activity) the ratio of the fields depends on the subsurface resistivity structure.*

The 16 MT stations scattered over the southern part of the Brucejack area provided a preliminary understanding of the resistivity contrasts. There are clear contrasts between rock types and a significant interpreted northerly structure that appears to correlate with indicated mineralized zones from Bridge Zone to West Zone. A second interpreted east-northeast structure that may off-set the northerly structure is also apparent. The 16 stations sparsely cover more than 2 km^2 with significant topography, and show differences in resistivity responses between individual stations. While the MT data may suggest geologic features, it should be used with caution.

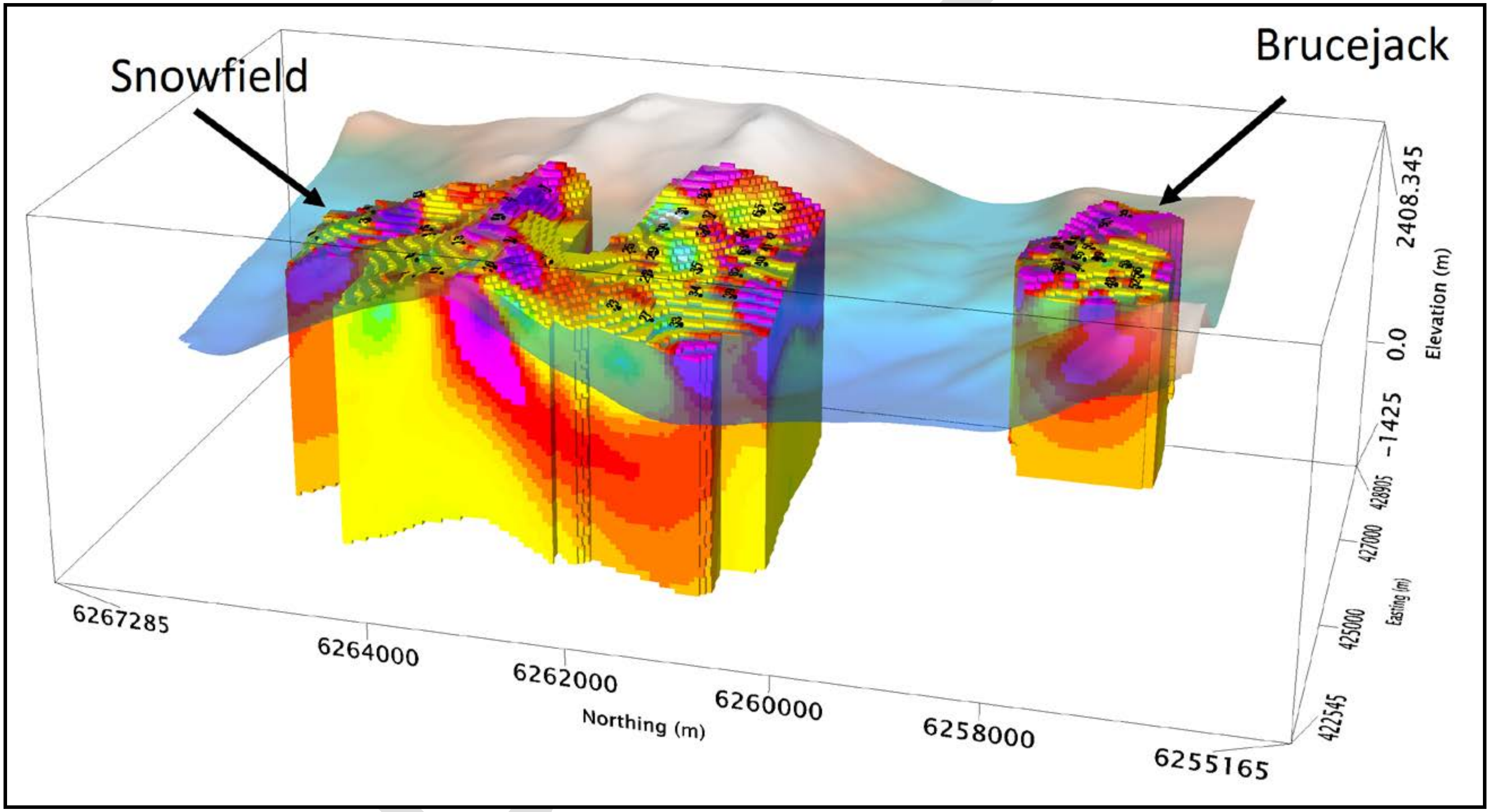
Three-dimensional figures from the Quantec Geoscience report (Gharibi et al. 2011) are presented in Figure 9.1 and Figure 9.2.

Figure 9.1 3D Geophysical Model Schematic According to Quantec Geoscience MT Survey



Source: Ghaffari et al. (2012)

Figure 9.2 3D Geophysical Model Schematic According to Quantec Geoscience MT Survey



Source: Ghaffari et al. (2012)

10.0 DRILLING

Information in this section has been excerpted from Olssen and Jones (2012c) which was sourced from sections within Ghaffari et al. (2012). Snowden has updated and verified this information.

10.1 HISTORICAL DRILLING

Drilling on the Property dates back to the 1960s, although most of the historical drilling was completed in the late 1980s and early 1990s. Up to this time, 452 surface diamond drillholes were completed, which totalled 60,854 m. These holes were relatively short, averaging 135 m per hole, and were mostly concentrated on West Zone, followed by Shore Zone, Galena Hill and Gossan Hill. As part of this exploration program, an exploration ramp was driven on the West Zone from which an additional 442 underground diamond drillholes were completed, which totalled 33,750 m. This drilling was focused exclusively on West Zone and increased the drill density to approximately 5 m centres between 5 m and 10 m sections. Historical drill core sizes for surface drillholes were NQ (47.6 mm diameter) and BQ (36.5 mm diameter). Core size for drillholes collared from the underground exploration ramp was AQ (27 mm diameter).

10.2 SILVER STANDARD DRILLING

Using the historical drill and trench data as a baseline, and following on the success of the Snowfield bulk tonnage drilling to the north, the 2009 Brucejack drill program was designed to test for additional bulk tonnage resources on the Property. This program successfully discovered several areas with bulk tonnage mineralization. Within the broader bulk tonnage mineralization were locally discreet high-grade intersections. These included holes SU-5 and SU-12, which were drilled to test for the western extension of the previously defined Galena Hill. These two holes intersected 1.5 m of 215 g/t Au and 1.5 m of 16,949 g/t Au, respectively, in what would eventually be called VOK. Drilling in 2009 totalled 17,846 m in 37 holes, of which 2,913 m in 6 holes were targeted at VOK.

In 2010, the drill program was designed to further define the bulk tonnage mineralization found the previous year, as well as to attempt to define a high-grade resource at VOK. In this year, a total of 73 diamond drillholes was completed which totalled 33,400 m. Of this, 11 holes comprising 3,693 m were targeted at VOK, and 2 holes totalling 1,119 m at the footwall of West Zone. In VOK, wide-spaced drilling intersected enough high-grade mineralization to confirm the exploration potential of the zone. The exploration potential included the preliminary definition of some of the ore controls, which put the intersections into a geologic context. The West Zone drilling intersected a broad zone of bulk tonnage mineralization, within which were several high-grade intersections.

10.3 PRETIVM DRILLING

The 2011 diamond drill program was the first in almost 20 years that was focused specifically on defining high-grade resources. In 2011, a total of 178 holes were completed totalling 72,805 m in holes SU-110 to SU-288. Included in this were 97 holes (41,219 m) targeted at VOK, 16 holes (7,471 m) at West Zone, and 21 holes (7,220 m) targeting the surrounding areas. The remaining drilling was focused on expansion of Shore Zone, testing for structurally controlled high-grade mineralization in Galena Hill (now part of VOK) and Bridge Zones, and testing new target areas. Drill collar coordinates, and results of the drilling in 2011 are described by P&E in Ghaffari et al. (2012).

The 2012 diamond drill program was focused on defining the high-grade resource at VOK, specifically targeting geological and structural features believed to be associated with gold mineralization. Diamond drilling was also focused on expanding VOK, both west of the Brucejack Fault and along trend to the east of the main mineralized zone. A total of 301 drillholes (105,571 m) were competed in holes SU-289 to SU-589 as part of the 2012 drilling program.

In 2011 and 2012, the drilling contractors were Radius Drilling from Prince George BC, and Matrix Drilling from Kamloops BC. They both used Hydracore drill rigs.

Core sizes for Pretivm's surface collared drillholes are PQ (85 mm diameter), HQ (63.5 mm diameter) and NQ (47.6 mm diameter). Approximately 50 to 60% of core is HQ size. For drillholes less than 600 m length, core size is commenced at HQ and is reduced to NQ when conditions require. For drillholes greater than 600 m length, the commencing core size is PQ, which is run down to approximately between 200 and 300 m in order to minimize drill path deviation.

Drill collars were surveyed by McElhanney Surveying from Terrace, BC. McElhanney Surveying use a total station instrument and permanent ground control stations for reference and have completed all the surveying on the project since 2009. Drillhole paths are surveyed at a nominal 50 m interval using a Reflex EZ single shot instrument.

Drill core logging and handling procedures were the same for all three programs. At the end of each drill shift, all core were placed in wooden boxes (Figure 10.1), labelled by drillhole and interval and transported to the core logging and core splitting facility on site by snowcoach (Figure 10.2) or helicopter. The sample boxes were covered to avoid sample loss or contamination during transportation (Figure 10.2).

Figure 10.1 Core in Wooden Core Boxes Ready for Transport

Figure 10.2 Sample Transportation by Snowcoach

Prior to any geotechnical and geological logging, the entire drill core was photographed in detail using a digital camera. These images were stored in individual files per drillhole.

A trained geotechnician recorded the core recovery and rock quality data for each measured drill run. All lithological, structural, alteration, and mineralogical features of the drill core were observed and recorded during the geological logging.

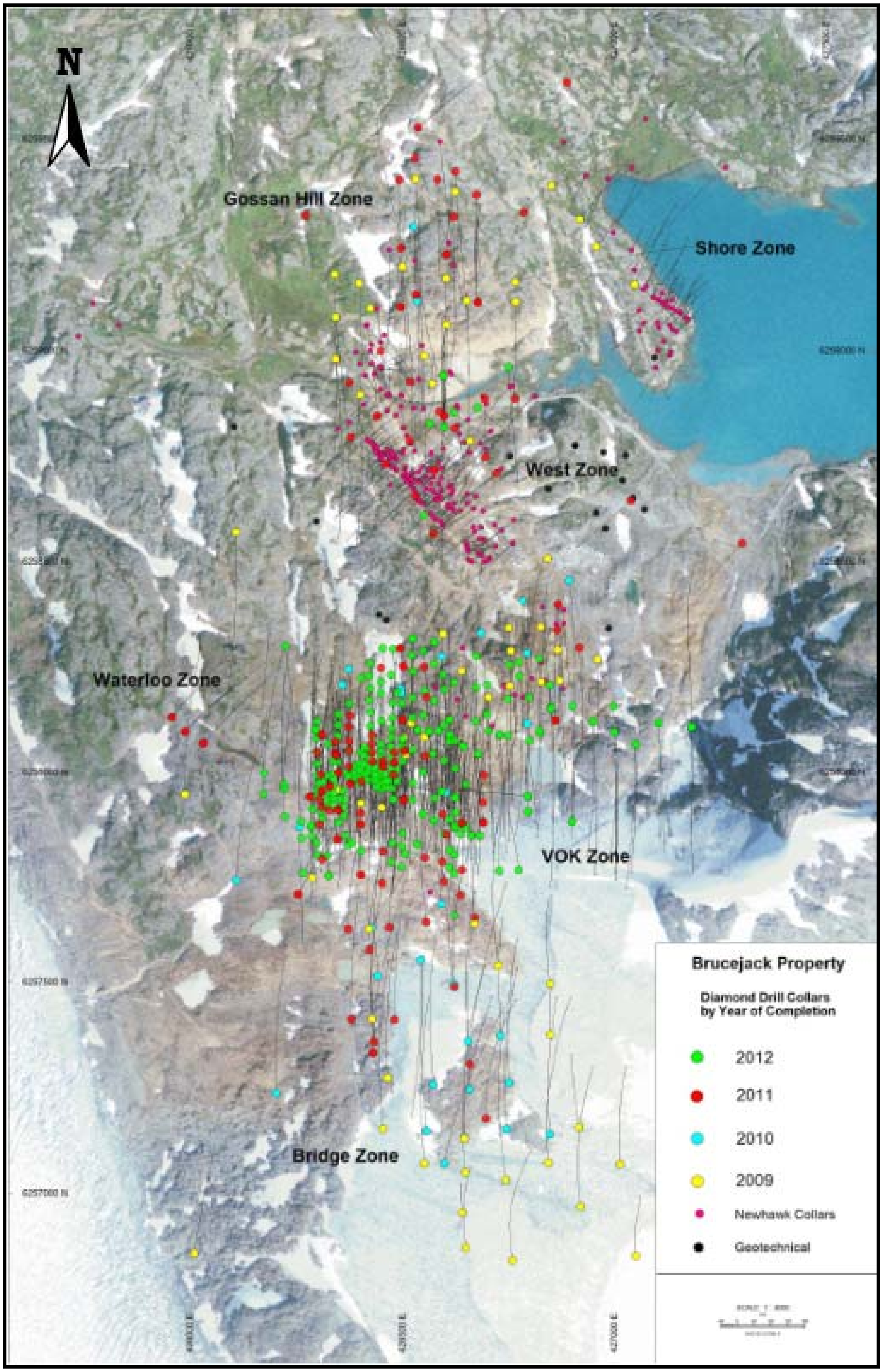
The geologist responsible for logging a drillhole assigned sample intervals with the criteria that the intervals did not cross geologic contacts and the maximum sample length was 2 m. Within any geologic unit, sample intervals of 1.5 m long could be extended or

reduced to coincide with any geologic contact. Sample lengths were rarely greater than 2 m or less than 0.5 m, and generally average 1.5 m. Every drillhole was sampled in its entirety from top to bottom.

The logging data was directly introduced in a database, using the software DHLogger, at the moment of the logging by the trained geotechnicians and geologists.

It is Snowden's opinion that the core logging procedures employed are thorough and provide sufficient geotechnical and geological information. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results. Snowden believes the drilling has been conducted using industry standard guidelines.

Figure 10.3 Brucejack Property Diamond Drillhole Plan



11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Information in this section has been excerpted from Olssen and Jones (2012c), which was sourced from sections within Ghaffari et al. (2012), edited in part and modified.

11.1 SAMPLE PREPARATION BEFORE DISPATCH OF SAMPLES

Upon completion of the geological logging, the core was moved to the splitting area where the core was either split or sawn in half lengthwise using a wet diamond saw. All PQ core was sawn as the core was too big to fit into the splitters (Figure 11.1). Likewise, any sample intervals that contained visible gold or interesting mineralization were also sawn. All other core was cracked in half using a standard hammer/blade core splitter. One-half of the drill core was placed in a plastic sample bag with the appropriate sample tag and the other half was returned to its original position in the core box. The sample bags were placed in four or five rice sacks and flown by helicopter to the staging area. Individual work orders generally included between 80 and 120 samples, including standards, blanks, and field duplicates. At the staging area, a local expediter brought the samples to Stewart where the samples were loaded onto a 5 t truck and locked in the company's warehouse for the night. The next morning, the samples were driven to the ALS sample preparation facility in Terrace, BC.

Figure 11.1 Cutting PQ Core at the Brucejack Property

The cut PQ samples weighed approximately 10 kg. HQ samples weighed approximately 6 kg, and NQ samples weighed between 3 and 4 kg. These weights assume a nominal 1.5 m sample length. In general, the average sample size submitted to the analytical laboratory was 6.5 kg.

Pretivm's QP for field activities is Mr. Ken McNaughton, P.Eng.

11.2 Analytical Laboratory

The 2011 and 2012 programs on the Property used ALS as the principal laboratory. The samples that were originally sent to ALS in Terrace, BC, for sample preparation were then forwarded to the ALS facility in Vancouver, BC, for analysis.

ALS is an internationally recognized mineral testing laboratory operating in 16 countries and has an International Organization for Standardization (ISO) 9001:2000 certification. The laboratory in Vancouver has also been accredited to ISO 17025 standards for specific laboratory procedures by the Standards Council of Canada (SCC).

A second laboratory, SGS Canada, was used as a check and comparison with ALS.

11.2.1 Method

Samples at ALS were crushed to 70% passing 2 mm, (-10 mesh). Samples were riffle split and 500 g were pulverized to 85% passing 75 µm (-200 mesh). The remaining coarse reject material was returned to Pretivm for storage in their Stewart, BC, warehouse for possible future use.

Gold was determined using fire assay on a 30 g aliquot with an AA finish. In addition, a 33 element package was completed using a four-acid digest and ICP-atomic emission spectroscopy (ICP-AES) analysis, which included the silver.

11.2.2 Density Determinations

Density determinations were completed by ALS using the pycnometer method on pulps from the drilling program. Pretivm's QP, Mr. Ken McNaughton, selected the samples as the programs were progressing to maintain good coverage over a wide range of locations and rock types. A total of 2,408 pulp specific gravity determinations were completed, including 705 determinations since the September 2012 estimate.

As part of the 2012 drilling program, Pretivm selected a portion of the samples (207 samples) for core density measurements, as well as pulp specific gravity measurements, in order to determine whether there was any impact on the density as a result of porosity.

The core density measurements were carried out by ALS using a standard Marcey method calculation on wax-coated core samples.

The results of the comparison indicated that the core density was on average 6% lower than the pulp specific gravity within the siliceous zone and 9% lower on average in all other rock types. As a result the pulp specific gravity measurements–which were used to estimate density in the model–were factored by 6% (siliceous zone) or 9% (other rock types) prior to statistical analysis and estimation.

11.3 Quality Assurance and Quality Control

11.3.1 Data – 2011

During 2011, data were entered into a data capture tool suited for import to a database. Data were then sent digitally to Caroline Vallat at GeoSpark Consulting Inc. (GeoSpark).

Updated sampling data were provided regularly, including down hole primary sample details, as well as reference to duplicate samples, and standards and blanks inserted throughout the sampling. The drillhole header, survey, and down hole attribute data were also provided on a regular basis.

To the best of GeoSpark's knowledge, the sample data were provided in their original digital state, and there is no reason to suspect any data tampering.

Data provided were loaded into a relational database suited for quality assurance (QA)/quality control (QC) on the analytical results reported by ALS, Vancouver. The database was reviewed regularly to ensure that the data remained functional for use. This included, for example, eliminating down hole interval overlaps and records beyond end of hole depths, as well as addressing any data entry issues related to the down hole attribute codes. Updates were also made to the sampling table wherever QA/QC measures revealed data entry issues related to the sample identities or details. For example, there were some issues where the wrong standard identity was entered and the analytical results returned clearly revealed the correct standard identity. All updates to sample identities were tracked within the database.

All analytical results were entered directly from ALS and SGS Canada assay certificates to a database managed by GeoSpark. The analytical results were then provided to Pretivm personnel for internal use.

Ongoing review of the analytical results took place in order to remedy any suspect analytical results. Re-analyses were requested from the laboratory whenever there was a question of the accuracy of results reported. In addition, internal laboratory repeat and field duplicates were reviewed in order to monitor the repeatability or precision of results reported.

Any re-analyses were further reviewed and the results were assigned to the primary samples, which were denoted with a suffix of "R" meaning re-run.

Duplicates, standards, and blanks were inserted approximately every twentieth sample and amount to 5.86%, 5.76%, and 5.98% relative to the total number of primary samples

submitted to ALS. In addition, a representative set of check samples was submitted to SGS Canada for analysis using similar analytical methods and techniques. The check samples served to define any bias in the primary results.

This amount of QA/QC data is sufficient to represent the quality of the sample analytical results reported by ALS.

Ongoing documentation of the analytical QA/QC results was provided to Pretivm (Vallat 2011).

Vallat (2011) concluded:

> *With consideration of inhomogeneity within the Brucejack project mineralization as a function of nugget mineralization and the nugget effect, and thorough review of the analytical results reported on field duplicates, a satisfactory level of precision has been inferred for the primary sample results. Additionally, initial review of the internal lab repeat results at the time of analysis reporting has increased the confidence in results reported by ALS Chemex.*
>
> *Mineral concentrations reported on standard and blank materials have been consistently monitored in order to remove any concern of local contamination or instrumentation issues.*
>
> *The detailed review of the standard and blank results has inferred that there is strong accuracy in the primary sample results reported by ALS Chemex.*
>
> *Check sampling has shown that there is no need for concern of bias in the primary sample results reported.*
>
> *The quality assurance and quality control measures taken and addressed herein have allowed for overall confidence in the analytical results reported for the 2011 Brucejack project.*

11.3.2 DATA – 2012

Throughout the Brucejack 2012 exploration program, drillhole data was entered on site using a local database platform. Weekly updates of the drillhole sample details were provided to GeoSpark for update of the local master QA/QC database.

Periodically, additional drillhole details including drillhole collars, surveys, and down hole lithology records were exported from the camp based data system and provided to GeoSpark for update of the Brucejack 2012 master database.

The Brucejack 2012 data was maintained within a relational database housing drillhole collar, survey, sampling, assay, and down hole lithology details. Additionally, a separate database was used for the QA/QC data of all received assay certificates.

As assay certificates were received, the data was imported directly to the QA/QC database and the analytical results for the duplicate pairs, standards, and blanks were reviewed prior to sending database exports to Pretivm personnel. QA/QC data was documented regularly in order to maintain communications in this regard.

A master database for the Project was compiled to include all data from the Project in order to improve data transferability. This master database was compiled from multiple source databases including:

- The pre-2011 Brucejack database compiled by Silver Standard, which included data from 2009 and 2010. This database source was a relational database system suited for exploration and mining data management.
- The historic Brucejack database including all data for drillholes drilled prior to 2009. This database was compiled and verified by Silver Standard following the initial database compilation and analytical QA/QC results and verification by GeoSpark.
- The 2011 and 2012 Pretivm relational database, compiled by GeoSpark for Pretivm as previously described.

The master Brucejack database was provided to Snowden and the database was then further reviewed for any potentially problematic data.

The master database was, at the time of this report, regularly updated and maintained by GeoSpark. All new data was imported from camp database exports and original assay certificates provided. Original assay certificates were reviewed for QA/QC as they were received, and the original data, provided by Pretivm, was checked for potentially problematic data. The database was reviewed to ensure that the data remained functional for use by eliminating down hole interval overlaps and records beyond end of hole depths, and addressing any data entry issues related to the down hole attribute codes, etc. The sampling table was also updated wherever QA/QC measures revealed data entry issues related to the sample identities or details. For example, there were some issues where the wrong standard identity was entered and the analytical results returned clearly revealed the correct standard identity. Updates to sample identities were tracked within the database.

11.3.3 2012 QA/QC

The QA/QC analytical results for the Project are quantitatively sufficient to represent precision, accuracy, and bias.

ALS of Vancouver, BC, reported the primary sample analytical results for the drillholes identified as SU-289 to SU-585.

Secondary check samples analyses were analyzed by SGS Canada of Vancouver, BC.

Primary sample gold concentrations were analyzed using fire assay with AA finish. Silver, copper, and an additional 31 elements were analyzed using four-acid digestion with ICP analysis.

Secondary check samples were analyzed using similar analytical methods.

The analytical results of the field duplicates, blanks, and standards were reviewed as they were reported, in order to maintain high-quality results throughout the exploration program.

Standards and blanks that were determined to have been reported in excess of the defined acceptable limits for the materials were re-analyzed along with a sample in their vicinity. This allowed for improvement of any samples with questionable accuracy.

In addition, the field duplicates were monitored for sample pairs with poor correlation due to the nature of the mineralization having localized high-grade narrow vein mineralization. Sample pairs where the duplicate was returned at a much greater grade than the primary sample, and within the greater than 5 g/t Au category were flagged as being in the high-grade region.

Pretivm was provided with ongoing documentation of the analytical QA/QC results along with updated database exports containing the sample details associated with the analytical results.

Vallat (2012) generated a summary QA/QC report reviewing the analytical results, which concluded:

> *The quality assurance and quality control measures taken by Pretivm Resources have been sufficient to maintain high quality results.*
>
> *The review of field duplicates, blanks and standards, and check samples has allowed for inference of a reasonable level of precision, good accuracy, and insignificant levels of bias within the primary sample results reported by ALS Global related to the Brucejack 2012 project including drill holes SU-289 to SU-585.*
>
> *The analytical results can be inferred to be of sufficient quality to represent the project.*

11.4 AUTHOR'S OPINION ON DATE SAMPLE PREPARATION, SECURITY AND ANALYTICAL PROCEDURES

Procedures undertaken to date by Pretivm have been under the supervision and security of the issuer's staff, as far as drill core sampling prior to dispatch. Laboratory sample reduction and analytical procedures have been conducted by independent accredited companies with acceptable practices.

Pretivm ensures quality control is monitored through the insertion of blanks, certified reference materials, and duplicates.

It is Snowden's (the author of this section) opinion that the sample preparation, security, and analytical procedures are satisfactory and appropriate for resource evaluation.

12.0 DATA VERIFICATION

Independent sampling and site verification visits were undertaken by P&E in 2011 and by Snowden in 2012.

The information in this section pertaining to the P&E work has been excerpted from Olssen and Jones (2012c), which was sourced and updated from sections within Ghaffari et al. (2012).

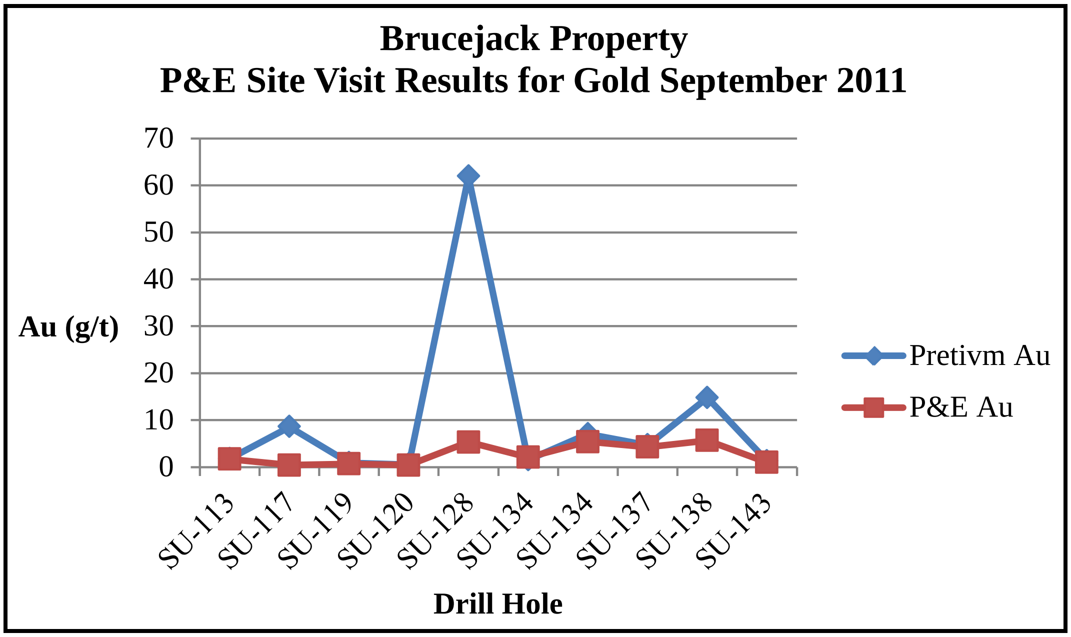
12.1 SITE VERIFICATION AND INDEPENDENT SAMPLING BY P&E

The Property was visited by Mr. Fred Brown, CPG, Pr.Sci.Nat., from September 13 to 15, 2011. Independent verification sampling was completed on diamond drill core, with ten samples distributed in nine holes collected for assay. An attempt was made to sample intervals from a variety of low- and high-grade material. The chosen sample intervals were then sampled by taking the remaining half-split core. The samples were then documented, bagged, sealed with packing tape, and brought by Mr. Brown to ALS in Terrace, BC, for analysis.

At no time, prior to the time of sampling, were any employees or other associates of Pretivm advised as to the location or identification of any of the samples to be collected.

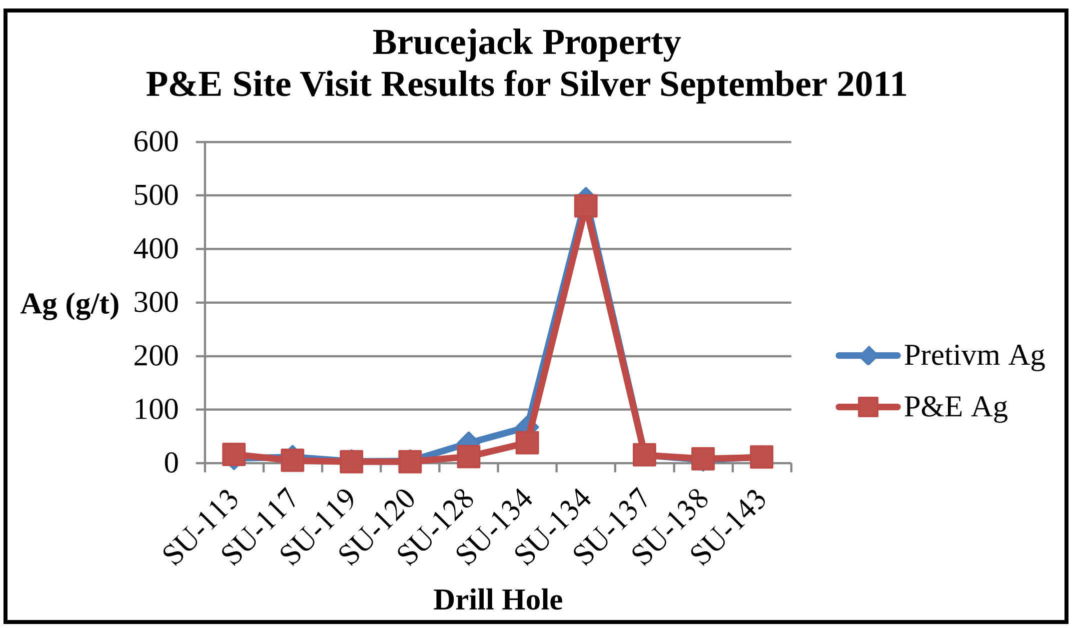
A comparison of the P&E independent sample verification results versus the original assay results are shown in Figure 12.1 and Figure 12.2.

Figure 12.1 P&E Independent Site Visit Sample Results for Gold



Source: Ghaffari et al. (2012)

Figure 12.2 P&E Independent Site Visit Sample Results for Silver



Source: Ghaffari et al. (2012)

12.2 DATA VERIFICATION BY SNOWDEN

Snowden's QP, Mr. Ivor Jones, visited the Brucejack site on February 15 and 16, 2012 and June 3 to 6, 2013, and takes overall responsibility for data verification in this report. The following items were verified and completed:

- cross-check of Pretivm drill logs with drill core; example core was reviewed with Mr. Ken McNaughton
- core handling, storage, and security at Pretivm's core storage facility in Stewart
- core logging process, alignment, recovery, mark-up and core sawing, sampling
- insertion of blanks, certified reference material
- core shack at the Brucejack camp
- review of drill logs, assay records and interpretations at Pretivm's office in Vancouver
- review of NI 43-101 technical reports incorporating the GeoSpark data validation work completed in 2011.

Snowden carried out a basic statistical and visual validation of the data prior to estimation and did not find any significant issues.

From June 8 to 10, 2012, Mr. Adrian Martínez Vargas completed sample validation under the supervision of Ms. Lynn Olssen and Mr. Ivor Jones. The following items were verified:

- cross-check of collar coordinates
- core logging process, alignment, recovery, mark-up and core sawing, sampling
- insertion of blanks, certified reference material
- core shack at the Brucejack camp
- sample transportation and delivery to ALS facilities in Terrace, BC.

On June 9, 2012, Mr. Martínez Vargas collected 12 samples from 6 drillholes for assay. The samples were selected randomly as two contiguous intervals per drillhole. Each sample was taken as the halved interval of HQ cores, then documented, bagged, and sealed. All samples were selected from cores with good recovery, with a weight averaging 6.74 kg. Mr. Martínez Vargas transported the samples by car to Terrace and then by airplane to Vancouver, BC. The samples were under the direct supervision of Mr. Martínez Vargas for the duration of the transportation.

The samples were sent to the ALS laboratories in Vancouver, BC, for assaying. The sample preparation and assaying protocol requested of ALS was the same used by Pretivm at the ALS facilities in Terrace. Additionally two different standards, provided by Pretivm were assayed in order to test the accuracy of the laboratories. The results for the standards assays are shown in Table 12.1.

Table 12.1 Standard Verification

ID	Snowden Au	Snowden Ag	Pretivm Au	Pretivm Ag	Nominal Au	Nominal Ag
A	10.65	283	9.97	275	9.97±0.58	276.0 ± 17.1
B	0.86	40.3	0.87	39.3	0.87±0.09	39.3 ± 4.6

The comparison between the grade in the samples collected by Snowden and Pretivm are shown in Figure 12.3 and Figure 12.4.

Figure 12.3 Sample Verification Results for Gold Grades

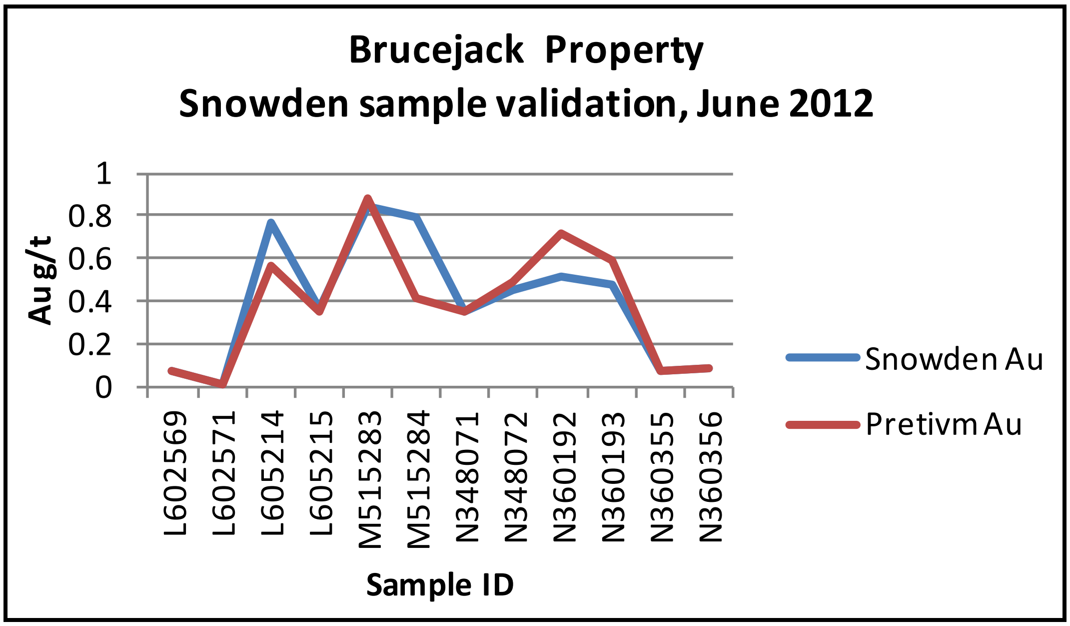
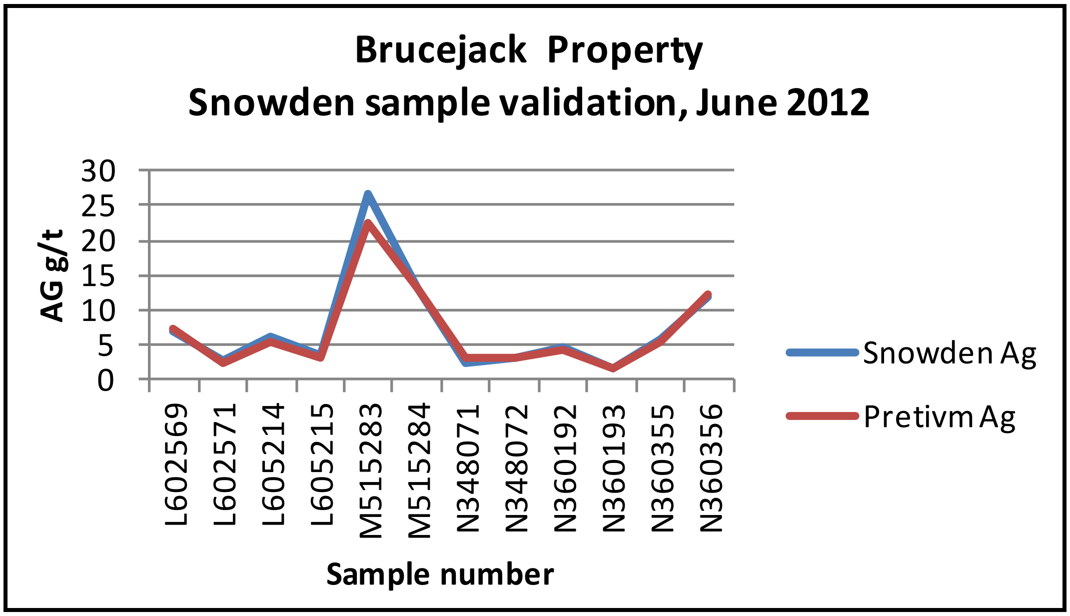


Figure 12.4 Sample Verification Results for Silver Grades



Snowden has not undertaken a complete data verification study; however, sufficient checks have been completed to satisfy Snowden that the Brucejack drilling and sampling data is suitable to use in estimating a mineral resource.

13.0 MINERAL PROCESSING AND METALLURICAL TESTING

13.1 INTRODUCTION

Several testing programs were completed to investigate the metallurgical performance of the Brucejack mineralization, including recent test work conducted between 2009 and early 2013, and historical test work conducted between 1988 and 1990 for the feasibility study by CESL Limited (CESL) (1990).

After 2009, metallurgical test work was conducted on mineral samples from the VOK Zone, the West Zone, and adjacent mineralization deposits such as the Galena Hill Zone and the Gossan Hill (R8) Zone. This study focuses on the mineralization of the VOK Zone and the West Zone.

Gold and silver are the key economic metals in the mineralization of the Project. The metallurgical test programs conducted after 2009 include head sample characteristics, gravity concentration, gold/silver bulk flotation, cyanidation, and the determination of various process related parameters. Early test work, after 2009, focused on developing the flowsheet of gravity concentration, bulk flotation, and flotation concentrate cyanidation. This test work also examined the metallurgical responses to the flowsheet of gravity concentration followed by whole ore leaching. Later test work concentrated on the gravity-flotation concentration flowsheet.

Comprehensive metallurgical test work programs were conducted in 2012 and 2013 to support the feasibility study; this test work focused on assessing the metallurgical performance of the West Zone and the VOK Zone mineralization. The main testing programs are summarized in Table 13.1. Although cyanidation is not proposed for this study, the results from the test programs are also summarized in this section.

Table 13.1 Major Metallurgical Testing Programs

Year	Program ID	Laboratory**	Gravity	Flotation	Grindability	Cyanidation	Others
2013	1208011	Inspectorate	√	√	√	√	√
2012	11489	Hazen	-	-	√	-	-
2012	KRTS20734-A	Knelson	√	-	-	-	-
2012	MS1399	Met-Solve	√	-	-	-	-
2012	-	Pocock	-	-	-	-	√
2012	12012	JZM					√
2012	1106811	Inspectorate	√	√	√	√	√

table continues…

Year	Program ID	Laboratory**	Gravity	Flotation	Grindability	Cyanidation	Others
2010-2011	1004608	Inspectorate	√	√	√	√	-
2009-2010	0906609	Inspectorate	√	√	√	√	-
Before 1990*	-	Various	√	√	√	√	√

Notes: *From Feasibility Study Sulphurets Property by CESL
**Hazen = Hazen Research Inc.; Inspectorate = Exploration & Metallurgical Testing Inspectorate America Corporation; Met-Solve = Met-Solve Laboratories Inc.; Knelson = FLSmidth Knelson; Pocock = Pocock Industrial Inc.; JZM = Joe Zhou Mineralogy Ltd.

13.2 Historical Test Work

Historical test work used composite samples collected from the West Zone and the R8 Zone. The feasibility study prepared in 1990 indicated that the Brucejack mineralization consists of apparently erratic veins and lenses containing metallic gold (native gold) and metallic silver (native silver), together with a variety of sulphide minerals in a quartz-rich environment, within a zone of altered volcanic rocks . Gold occurs as a range of relatively coarse grains (40 to 100 µm) to fine grains (less than 40 µm) locked in either pyrite or quartz gangue. Silver occurs in small amounts in metallic form, while most silver is intimately associated with, or a component of, various sulphide minerals. The major minerals in the samples are listed in Table 13.2.

Table 13.2 Mineralogical Assessment

Mineral	Content (%)
Pyrite	9.7
Sphalerite	0.5
Tetrahedrite	0.1
Jalpaite	0.1
Ruby Silver	0.05
Galena	0.05
Chalcopyrite	Trace
Native Gold	Trace
Native Silver	Trace
Gangues	89.5

Source: CESL (1990)

Metallurgical testing included gravity separation, flotation, cyanidation, and roasting pre-treatment.

The test work indicated that gravity separation would recover a significant portion of the contained gold. Cyanide leaching on the gravity tailings produced good overall gold recoveries, but poor silver recoveries (less than 40%). As reported in the 1990 feasibility study, it was indicated that the poor silver recoveries were attributed to the silver occurrence in the form of relatively insoluble silver sulphides such as tetrahedrite and proustite.

The gold and silver minerals responded well to the flotation concentration. The reagent scheme screening tests showed that the addition of collector 3418A would improve the recovery of the precious metals and reduce concentrate mass pull. The test work also indicated that the addition of lime, to increase slurry pH from 8.1 to 10.5, could substantially reduce the concentrate weight from 6.8 to 1.5%.

Similar metallurgical performances were produced from the West Zone samples and the R8 Zone samples. However, the test results appeared to indicate that the R8 mineralization might require finer primary grinding.

Using a combined process of gravity separation and flotation, CESL projected the overall gold and silver recoveries to be approximately 89% and 83% for the West Zone mineralization, and 88% and 85% for the R8 Zone. The projections for the blend of the two zones are detailed in Table 13.3.

Table 13.3 Metallurgical Performance Projection

Products	Mass Recovery (%)	Grade (g/t)		Recovery (%)	
		Au	Ag	Au	Ag
Gravity Concentrate	0.2	1,139.0	3,966.0	22.5	1.1
Flotation Concentrate	4.5	143.4	12,665.0	66.4	82.4
Tailings	95.3	1.2	119.7	11.1	16.5
Head	100.0	9.3	777.6	100.0	100.0

Source: CESL – Blend (1990)

13.3 2009 TO 2013 TEST WORK

Inspectorate–previously Process Research Associates Ltd. (PRA)–carried out preliminary metallurgical test work investigating the metallurgical performance of the Brucejack mineralization from 2009 to early 2011. During 2012 and 2013, Inspectorate conducted further comprehensive metallurgical test work on the VOK Zone and West Zone mineralization; this test work was conducted to optimize process conditions, improve metallurgical performance, and support the feasibility study. These test programs were conducted under the supervision of Frank Wright, P.Eng., a metallurgist contracted by Pretivm.

13.3.1 SAMPLE DESCRIPTION

The samples used for the test work beginning in 2009 were generated from numerous diameter drill core intervals that were produced from various geological exploration programs. Drillhole distribution is presented in Figure 10.3.

13.3.2 2012 TO 2013 TEST SAMPLES

Inspectorate used two sets of drill core interval samples for the 2012 to 2013 test programs, which focused on optimizing the process flowsheet and investigating variations in the metallurgical performances of the samples.

The 2012 test work was conducted on 102 drill core samples collected in 2011. Six composite samples were generated from the individual drill core intervals; four composites were from the VOK Zone (VOK-1, VOK-2, VOK-3, VOK-4) and two master composites were from the West Zone (WZ-1, WZ-2). A master composite, labelled as Composite BJ-A, was re-blended from portions of five of the master composites. The composition of the master composites is summarized in Table 13.4 and Table 13.5.

Table 13.4 Master Composites (2012 Test Program)

	Composite					
	VOK-1	VOK-2	VOK-3	VOK-4	WZ-1	WZ-2
Sample Labels	210	208	119A	225	233	279
	213	122B	119C	226	285	288
	223	127A	122C	114A	121C	121B
	246	128B	122D	122A	131B	131A
	135A	150C	128A	157D	131C	143C
	135B	157E	135C	163A	143A	154A
	157C	170C	150A	163B	143B	154B
	195A	176A	150B	170A	154C	162A
	200B	176B	157A	170B	162D	162B
	202B	190A	157B	190B	222C	162C
	219A	193A	193C	202A	240C	212A
	224A	193D	200A	237B	282B	222A
	224B	193E	219C	241B	282C	222B
	230A	219B	224D	252A	284A	240A
	230B	232B	230B	252B	284B	240B
	232A	232C	237A	252C	-	282A
	238A	241A	-	-	-	-
	238B	250C	-	-	-	-
	250B	-	-	-	-	-
	253A	-	-	-	-	-
	253B	-	-	-	-	-

Table 13.5 Composite BJ-A Composition (2012 Test Program)

Sub-composite	Weight (kg)
VOK-2	180
VOK-3	75
VOK-4	110
WZ-1	75
WZ-2	60

In 2013, Inspectorate conducted further confirmation test work using 28 drill core samples collected from the VOK Zone in 2012. The drill core intervals were blended into eight composites according to spatial locations. The composite samples were further blended to generate two master composites, MU and ML composite, representing the upper zone and lower zone of the VOK deposit, respectively.

Table 13.6 summarizes information on the drill core intervals used to generate the composite samples.

Table 13.6 Composite Samples (2013 Test Program)

	Composite							
	SWU	SEU	SWL	SEL	NWU	NEU	NWL	NEL
Hole Number	SU-357	SU-315	SU-338	SU-302	SU-454	SU-334	SU-304	SU-316
	SU-390	SU-394	SU-340	SU-312	SU-490	-	SU-350	SU-327
	SU-447	SU-398	SU-342	-	-	-	SU-364	SU-334
	SU-451	SU-468	SU-419	-	-	-	-	-
	SU-476	SU-484	-	-	-	-	-	-
	-	SU-507	-	-	-	-	-	-

13.3.3 2010 to 2011 Tests Samples

The composite samples prepared for the 2010 to 2012 test work were originally from the West Zone, the Galena Hill Zone, and the Bridge Zone. The drillhole interval samples from the Galena Hill Zone (identified as GH2) and the Bridge Zone (identified as BZ2) were grouped into high- and low-grade composites; however, the test work was focused on the high-grade composite samples only. The WZ1 composite sample was also tested, which was comprised of separate drillhole intervals from the Gossan Hill Zone, R8 Zone, and West Zone. The composite samples and individual sample identifications are presented in Table 13.7.

Table 13.7 Conceptual Master Compositing List (2010/2011)

Sample ID	Zone	Hole ID
Composite GH2 (High Grade)		
SU-005	Galena Hill	SU-05
SU-006- A	Galena Hill	SU-06
SU-033	Galena Hill	SU-33
SU-54 A	Galena Hill	SU-54
SU-76 B	Galena Hill	SU-76
Composite BZ2 (High Grade)		
SU-021-B	Bridge Zone	SU-21
SU-025	Bridge Zone	SU-25
SU-058 A	Bridge Zone	SU-58
SU-58 B	Bridge Zone	SU-58
SU-64 B	Bridge Zone	SU-64
SU-69 A	Bridge Zone	SU-69
SU-69 B	Bridge Zone	SU-69
SU-69 C	Bridge Zone	SU-69
SU-75 C	Bridge Zone	SU-75
SU-78 C	Bridge Zone	SU-78
SU-10 C	Bridge Zone	SU-10
Composite WZ1		
SU-032-A	Gossan Hill	SU-32
SU-036-A	Gossan Hill	SU-36
SU-036-B	Gossan Hill	SU-36
SU-42 A	Gossan Hill	SU-42
SU-63 A	Gossan Hill	SU-63
SU-66A	Gossan Hill	SU-66
SU-032-B	R8 Zone	SU-32
SU-032-C	R8 Zone	SU-32
SU-42 B	R8 Zone	SU-42
SU-63 B	West Zone Footwall	SU-63
SU-66 B	West Zone Footwall	SU-66
SU-67 A	Gossan Hill	SU-67
SU-67 B	Gossan Hill	SU-67
SU-74 A	Gossan Hill	SU-74
SU-88 A	Gossan Hill	SU-88
SU-88 B	Gossan Hill	SU-88
SU-98	Main West Zone	SU-98
SU-103	Main West Zone	SU-103

The variability tests also used composite samples SU-98, SU-76B, SU-32A, SU-32C, and SU-33.

13.3.4 2009 to 2010 Test Samples

In October and November 2009, PRA received two batches of assay reject samples. The first batch had 378 samples with a total weight of 1,695 kg (including packing bag weight), while the second batch had 198 samples and with a total weight of 950 kg (including packing bag weight).

The samples were grouped into 16 composite samples, which were labelled as: SU-4, SU-5, SU-6A, SU-6B, SU-10, SU-19, SU-21A, SU-21B, SU-25, SU-27, SU-032A, SU-032B, SU-032C, SU-033, SU-036A, and SU-036B.

The composite samples were further grouped into zone composite samples representing:

- the West Zone and the Gossan Hill Zone (Composite R8)
- the Bridge Zone (Composite BZ)
- the Galena Hill Zone (Composite GH).

13.3.5 Sample Head Analyses

In 2012, ALS and Inspectorate performed head analyses on six master composites and some of the individual drill core intervals. The results from both laboratories are shown in Table 13.8.

There is a significant deviation in the assay results between the two laboratories and also between the assay methods. This deviation indicates a substantial nugget effect on the gold and silver assay. The two West Zone composites produced higher silver grades, particularly for master composite WZ-2. Inspectorate also assayed the drill core drill interval samples that were used in the variability tests.

Table 13.8 Head Assay Comparison (2012)

	ALS				Inspectorate			
	Fire Assay (g/t)		Metallic Assay (g/t)		Fire Assay (g/t)		Metallic Assay (g/t)	
Sample ID	Au	Ag	Au	Ag	Au	Ag	Au	Ag
VOK-1	5.8	12	11.7	20	11.2	19.7	6.9	20.1
VOK-2	19.9	22	10.0	13	16.9	29.6	9.9	16.6
VOK-3	26.6	27	53.2	43	45.5	34.8	102.4	65.6
VOK-4	1.6	36	2.7	45	10.1	19.9	3.0	42.5
WZ-1	4.8	25	6.4	32	6.1	36.1	5.8	32.5
WZ-2	6.5	405	6.5	421	4.6	407	4.9	478.7
BJ-A	-	-	-	-	14.1	55.5	12.9	52.5
122C	0.33	1.08	-	-	<1.0	1.2	-	-
135C	2.09	2.92	-	-	1.47	2	-	-
219C	76.5	46	-	-	71.59	67.8	-	-

table continues…

Sample ID	ALS				Inspectorate			
	Fire Assay (g/t)		Metallic Assay (g/t)		Fire Assay (g/t)		Metallic Assay (g/t)	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag
127A	47	31	-	-	28.66	31.1	-	-
176B	1.28	8	-	-	<1.0	18.2	-	-
193A	0.25	1.05	-	-	<1.0	1.1	-	-
208	2.68	5.09	-	-	1.33	2.8	-	-
232B	19.95	15	-	-	10.48	19.1	-	-
135B	8.17	111	-	-	6.76	123.1	-	-
195A	12.8	12	-	-	7.39	5.98	-	-
219A	66	54	-	-	62.63	45.7	-	-
223	3.07	12	-	-	2.91	15.6	-	-
230A	5.31	9	-	-	2.37	11.9	-	-
238B	7.14	16	-	-	1.89	14.5	-	-
253B	4.56	5	-	-	<1.0	5.4	-	-

The assay data for the 2013 test composites are shown in Table 13.9.

Table 13.9 Head Assay Comparison (2013)

Sample ID	Au	Ag	Ag			
	Metallics (g/t)		ICP (g/t)	S (tot) (%)	C Graph (%)	As (ppm)
SEL	1.44	27.9	32.9	4.51	0.08	655
NWL	0.91	4.0	4.9	2.39	0.07	249
SEU	9.47	9.8	10.0	4.35	0.08	822
NEL	4.89	15.1	8.0	2.16	0.08	399
NEU	2.59*	3.1	4.0	3.19	0.08	374
SWL	14.83	12.5	8.9	3.09	0.08	481
SWU	9.24	14.0	12.0	4.06	0.13	690
NWU	27.40	20.7	28.7	2.86	0.07	238
BMS**	1.87	10.1	10.9	4.11	0.06	1,154
MU	10.28	12.3	9.9	3.54	0.07	525
ML	5.62	8.3	9.9	2.69	0.07	368

Notes: *Initial sample showed 500 g/t gold in coarse fraction, but re-running produced a significantly lower head as shown in Table 13.9.
**Represents a stope planned for producing a bulk sample during summer 2013. Heavy media separation test was conducted using the sample.

Table 13.10 shows the head grade assay for the 2010 and 2011 test composite samples. The gold content of the samples ranges from 2.5 to 52.9 g/t.

Table 13.10 Metal Contents of Composite Samples (2010 to 2011)

Composite	Head Grade (g/t)	
	Au	Ag
Composite GH2	4.93	52.9
Composite BZ2	0.91	7.7
Composite WZ1	1.79	25.4
Composite SU-98	73.30	2.5
Composite SU-76B	12.60	13.0
Composite SU-32C	11.00	10.4
Composite SU-32A	3.80	25.8
Composite SU-33	3.68	22.1

Table 13.11 shows the head assay of the 2009 and 2010 composites. The assay data reveals that there is a significant variation between the grades obtained from standard fire assay and metallic analyses procedures. This indicates that the gold in some of the samples occurs in the form of nugget gold.

Table 13.11 Metal and Sulphur Contents of Composite Samples (2009 to 2010)

Sample ID	Au(1) (g/t)	Au (CN)(2) (g/t)	Au(3) (g/t)	Ag(4) (g/t)	Ag (CN) (g/t)	S(-2) (%)	C(org)(5) (%)	Cu(4) (ppm)	As (%)
SU-4	1.86	-	1.75	3.9	-	2.67	0.22	57	0.113
SU-5	0.99	-	1.10	34.8	-	1.58	0.10	235	0.026
SU-6A	1.36	-	1.98	67.3	-	3.63	0.06	101	0.020
SU-6B	1.05	-	5.23	12.9	-	3.79	0.19	90	0.029
SU-10	0.71	-	0.76	8.3	-	1.89	0.13	77	0.011
SU-19	1.35	-	1.57	6.6	-	2.03	0.25	133	0.010
SU-21A	0.62	-	0.64	10.3	-	2.39	0.14	70	0.026
SU-21B	5.23	-	5.05	12.3	-	2.07	0.18	96	0.031
SU-25	1.64	-	2.12	11.4	-	1.86	0.22	34	0.025
SU-27	0.64	-	0.91	4.0	-	1.21	0.15	23	0.033
SU-032A	2.46	1.70	2.24	13.3	11.7	3.50	0.11	66	0.016
SU-032B	0.84	0.78	1.42	71.1	73.8	3.11	0.35	57	0.007
SU-032C	1.90	1.62	3.06	1.9	4.0	2.93	0.29	27	0.024
SU-033	2.17	2.10	3.42	24.5	29.8	3.08	0.21	63	0.018
SU-036A	1.40	0.68	1.30	10.2	8.8	3.23	0.22	104	0.046
SU-036B	0.64	0.41	0.55	3.8	3.0	3.56	0.33	26	0.028
Comp R8	1.14	-	1.44	-	-	-	-	60	0.022
Comp GH	1.65	-	1.73	-	-	-	-	131	0.022
Comp BZ	1.53	-	1.67	-	-	-	-	77	0.020

Notes: (1) whole sample assay; (2) CN = cyanide soluble; (3) metallic analyses; (4) by ICP; (5) org = organic carbon.

13.3.6 Ore Hardness Test Work

Table 13.12 and Table 13.13 show the Inspectorate and Hazen grindability test results from various test programs on the Brucejack mineralization. On average, the mineralization appears to be moderately hard. In general, the ball grindability data of the ore is relatively consistent although the Bond ball mill work index (BWi) for the mineral samples ranges from 13.8 to 17.2 as shown in Table 13.12.

Table 13.12 Conventional Grindability and Crushability Test Results

Sample ID	BWi (kWh/t)	Cut Particle Size (Screen Aperture) (µm)	RWi (kWh/t)	CWi (kWh/t)	UCS (psi)	Ai (g)
Inspectorate (2013)						
MU (Upper Zone Master Composite)	15.6	106	-	-	-	-
ML (Lower Zone Master Composite)	15.0	106	-	-	-	-
Hazen (2012)						
VOK HW 1	14.2	149	14.4	12.3	20,910	0.2254
VOK Ore 1	14.4	149	15.6	11.4	15,680	0.2125
VOK Ore 2	14.4	149	14.6	11.1	8,510	0.1384
VOK Ore 3	15.4	149	17.9	10.4	9,000	0.0903
VOK Ore 4	14.2	149	15.2	9.3	11,800	0.3820
VOK Ore 5	13.8	149	14.3	7.9	5,770	0.2474
VOK Ore 6	14.4	149	13.5	8.9	11,500	0.2385
WZ HW 1	12.2	149	13.2	6.9	2,520	0.0388
WZ Ore 1	16.7	149	16.7	11.8	22,390	0.3069
WZ Ore 2	15.3	149	15.1	10.7	15,530	0.3535
WZ Ore 3	15.8	149	15.5	10.3	20,310	0.6599
WZ Ore 4	15.5	149	17.0	9.5	26,460	0.2479
Inspectorate (2012)						
VOK-1 Master Composite	15.8	74	-	-	-	-
VOK-2 Master Composite	15.3	74	-	-	-	-
VOK-3 Master Composite	15.8	74	-	-	-	-
VOK-4 Master Composite	15.7	74	-	-	-	-
WZ-1 Master Composite	17.2	74	-	-	-	-
WZ-2 Master Composite	15.7	74	-	-	-	-
Inspectorate (2009 to 2010)						
BZ Composite	16.4	105	-	-	-	-
GH Composite	15.6	105	-	-	-	-
R8 Composite	16.2	105	-	-	-	-

Note: RWi = Bond rod mill work index; CWi = Bond crushing mill work index; UCS = unconfined compressive strength ; Ai = abrasion index

Table 13.13 SMC Test Results (2012)

Sample ID	DWi (kWh/m³)	A	b	Axb	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	ta	Specific Gravity
VOK HW 1	5.76	52.8	0.92	48.6	16.7	12.0	6.2	0.45	2.79
VOK Ore 1	6.37	56.6	0.77	43.6	18.1	13.2	6.8	0.41	2.79
VOK Ore 3	7.12	62.9	0.62	39.0	20.2	15.0	7.8	0.40	2.75
VOK Ore 5	4.61	52.3	1.16	60.7	13.9	9.5	4.9	0.56	2.81
WZ HW 1	4.89	55.2	1.08	59.6	14.1	9.8	5.1	0.53	2.90
WZ Ore 2	7.08	66.7	0.59	39.4	19.9	14.8	7.7	0.37	2.76
WZ Ore 4	6.32	69.9	0.62	43.3	18.3	13.3	6.9	0.41	2.75
Average	6.02	59.5	0.82	47.7	17.3	12.5	6.5	0.44	2.79
Average – VOK	5.97	56.2	0.87	48.0	17.2	12.4	6.4	0.45	2.79
Average – WZ	6.10	63.9	0.76	47.4	17.4	12.6	6.6	0.44	2.80

Note: DWi = drop weight index; Mia = coarse ore work index provided directly by SMC Test®; Mih = high pressure grinding roll (HPGR) ore work index provided directly by SMC Test®; Mic = crushing work index provided directly by SMC Test® ta =low-energy abrasion component of breakage

Contract Support Services, Inc. conducted primary grinding circuit simulations based on the test results from Hazen and Inspectorate. The simulation results for a mill feed rate of 2,700 t/d are summarized as follows:

- SAG mill/ball mill/pebble crusher (SABC) arrangement:
  - a 19 ft diameter by 8 ft long (effective grinding length (EGL)) SAG with 1,290 kW of installed power
  - a 13 ft diameter by 22 ft long ball mill with 1,470 kW of installed power (drawing 1,417 kW)
  - a pebble crusher with 45 to 50 kW of installed power.
- SAG mill/ball mill (SAB) arrangement:
  - a 19 ft diameter by 8 ft long (EGL) SAG with 1,290 kW of installed power
  - a 13 ft diameter by 22 ft long ball mill with 1,470 kW of installed power (drawing 1,429 kW).

13.3.7 SAMPLE SPECIFIC GRAVITY

The specific gravity of the Brucejack mineral samples are shown in Table 13.14 and Table 13.15. The specific gravity data varied narrowly from 2.71 to 2.87.

Table 13.14 Sample Specific Gravity (2012)

Sample ID	Specific Gravity
VOK-1 Master Composite	2.87
VOK-2 Master Composite	2.83
VOK-3 Master Composite	2.77
VOK-4 Master Composite	2.80
WZ-1 Master Composite	2.74
WZ-2 Master Composite	2.76

Table 13.15 Sample Specific Gravity (2009 to 2010)

Sample ID	Specific Gravity	Sample ID	Specific Gravity
SU-4	2.79	SU-25	2.71
SU-5	2.74	SU-27	2.74
SU-6A	2.82	SU-032A	2.73
SU-6B	2.84	SU-032B	2.73
SU-10	2.76	SU-032C	2.72
SU-19	2.76	SU-033	2.78
SU-21A	2.75	SU-036A	2.82
SU-21B	2.77	SU-036B	2.78

13.3.8 FLOTATION TEST WORK

PRIMARY GRIND SIZE

Since 2009, different primary grind sizes have been tested by Inspectorate on the various composite samples . PAX and A208 were used as collectors, methyl isobutyl carbinol (MIBC) was used as a frother, and copper sulphate was used as an activator. The head samples were pre-concentrated by centrifugal gravity separation followed by bulk flotation.

In 2012, the optimum primary grind size was further evaluated using the process conditions developed by Inspectorate during the previous test programs. The primary grind test sizes ranged from 80% passing 38 µm to 80% passing 114 µm. Figure 13.1 and Figure 13.2 show therelationships between primary grind size and overall gold and silver recoveries (gravity and flotation) .

Figure 13.1 Gold Recovery versus Primary Grind Size (2012)

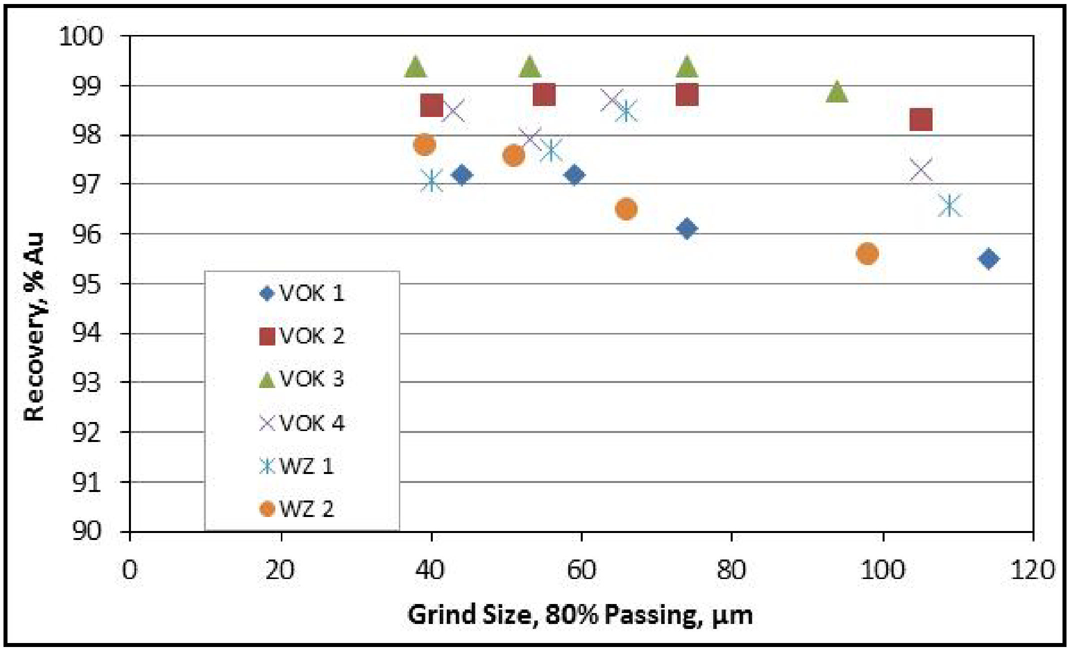
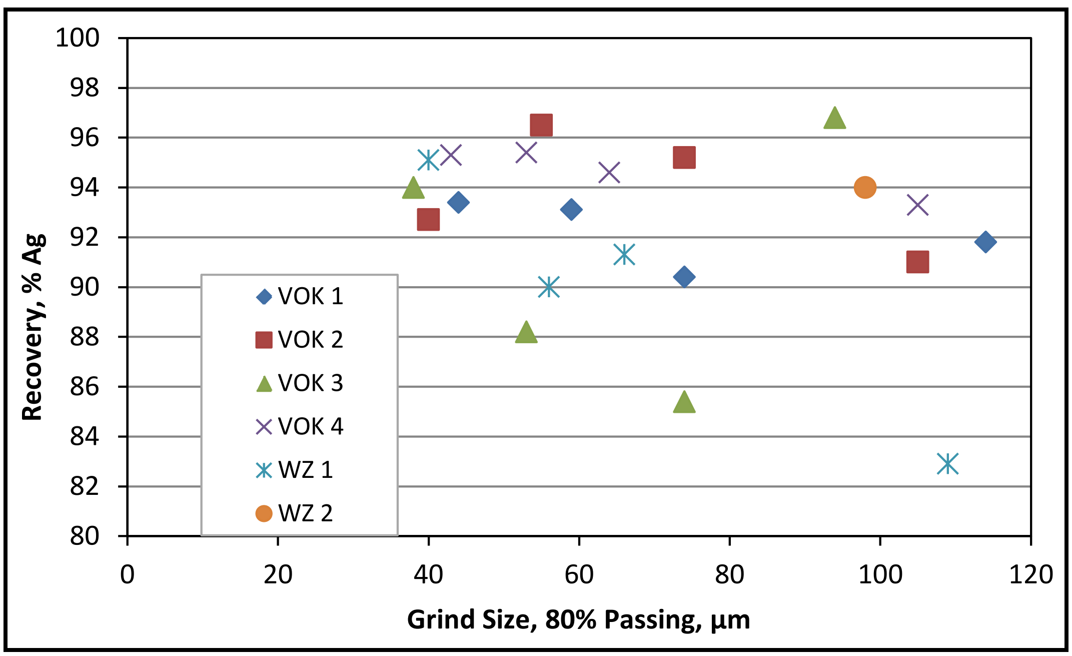


Figure 13.2 Silver Recovery versus Primary Grind Size (2012)



The results indicate that gold and silver responded very well to the tested process conditions. At the grind size of 80% passing 70 µm, approximately 96 to 97% of the gold, and 91 to 92% of the silver, were recovered to the gravity concentrate and bulk flotation concentrate. Gold in the mineralization shows a better metallurgical response to the simple and conventional flowsheet when compared to silver.

In general, there was a reduction in gold and silver recoveries when there was an increase in the grind size. However, the effect of the primary grind size on the overall metal recoveries was insignificant, especially when the grind size was finer than 80% passing 60 to 80 µm. A grind size of 80% passing 74 µm was selected for the variability and locked cycle tests.

In the 2013 test program, the master composite samples generated from the upper zone and lower zone of the VOK deposit (labeled as MU and ML composites) were tested for their metallurgical response to three different grind sizes ranging from 80% passing between 79 µm to 114 µm. No significant difference was noted in gold and silver recoveries when the primary grind size changed.

In the 2010 to 2011 testing program, similar tests were conducted to assess the effect of primary grind size on gold and silver recoveries. The test results, shown in Figure 13.3 and Figure 13.4 appear to indicate that a finer grind size produces better metal recovery.

Figure 13.3 Effect of Primary Grind Size on Gold Recovery (2010 to 2011)

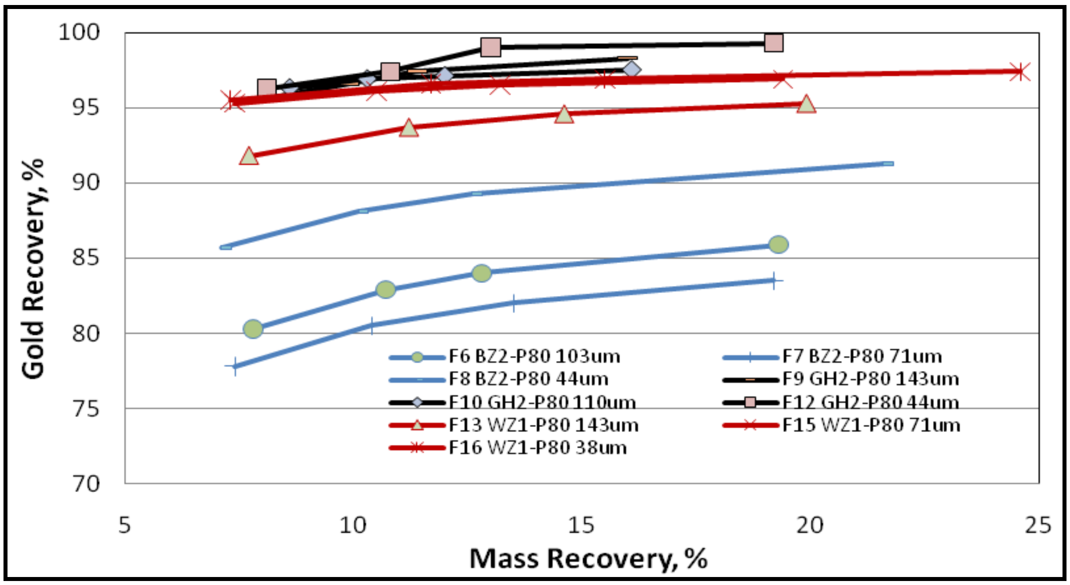
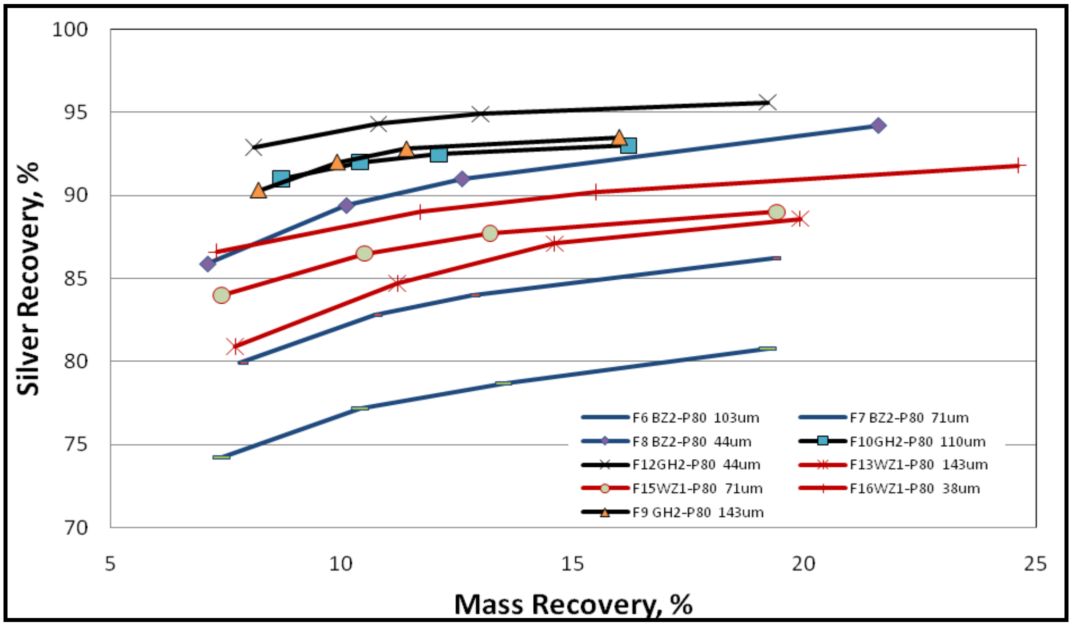


Figure 13.4 Effect of Primary Grind Size on Silver Recovery (2010 to 2011)

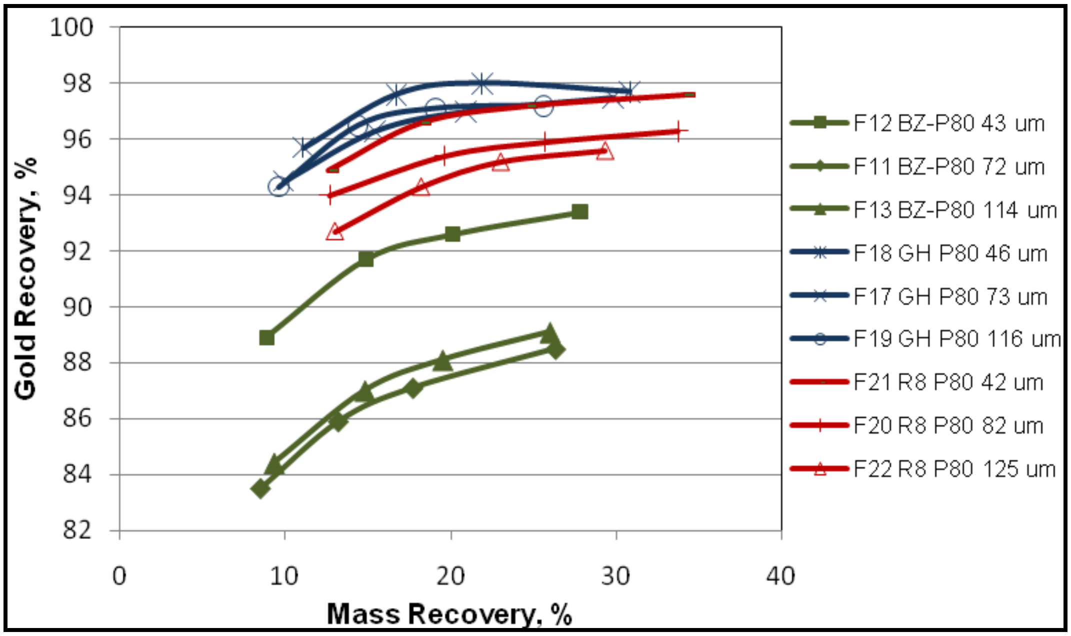


Gold and silver recoveries from the Galena Hill Zone sample and the West Zone sample were higher than the Bridge Zone sample. At a grind size of 80% passing 143 µm, the gold recoveries of the Galena Hill Zone sample and the West Zone sample were approximately 97% and 95%, respectively.

Silver recoveries are lower than gold recoveries for the Galena Hill Zone and the West Zone samples. However, for the Bridge Zone sample, the difference in gold and silver recoveries is much smaller.

The 2009 to 2010 test program investigated the relationship between metal recovery and primary grind size as well. Figure 13.5 shows that gold recovery improves when the primary grind size is finer than 70 µm. The improvement becomes much less significant at a grind size between 80% passing 70 µm and 80% passing 125 µm. The test results also indicate that gold recovery increases with concentrate mass pull, in particular when the mass pull is less than 15 to 20%.

Figure 13.5 Effect of Primary Grind Size on Gold Recovery (2009 to 2010)



There is a substantial difference in metallurgical response between the Bridge Zone mineralization and mineralization at the Galena Hill, West, and Gossan Hill zones. The gold recovery of the Bridge Zone sample is approximately 87% at a primary grind size of 80% passing 114 µm and a mass recovery of 15%; however, the Galena Hill sample produces a higher than 96% gold recovery at similar test conditions.

Reagents and Slurry pH

The 2012 test work studied the effect of reagent regimes on gold and silver flotation performances. The reagents tested include:

- collectors: PAX, PAX+A208, and 3418A+A208
- frother: MIBC and D250
- regulator: copper sulphate ($CuSO_4$).

The samples showed similar metallurgical responses to these reagent regimes, although there was some variation in metallurgical responses among the tests. The effect of the flotation collectors on the gold and silver recoveries are compared in Figure 13.6 and Figure 13.7, respectively.

Figure 13.6 Collector Screening Tests – Gold Recovery (2012)

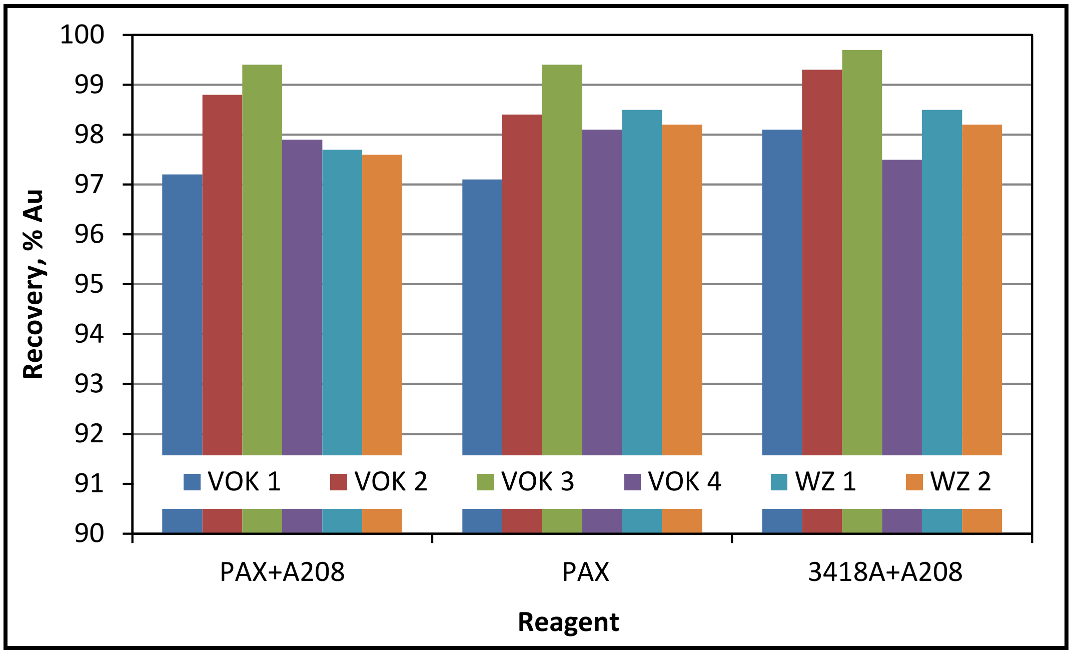
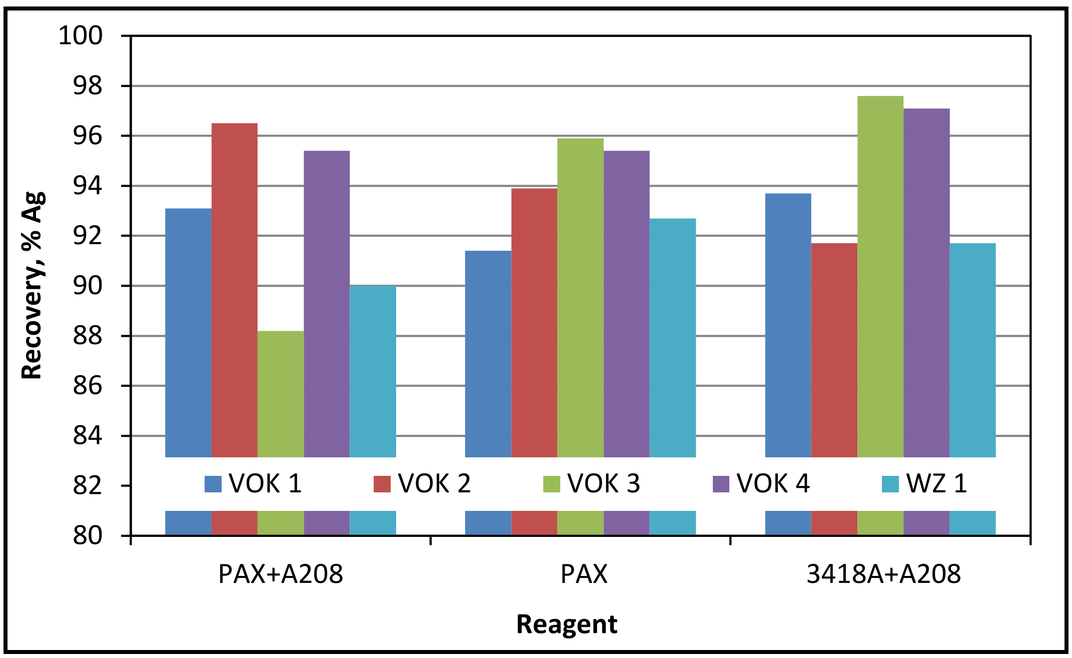


Figure 13.7 Collector Screening Tests – Silver Recovery (2012)

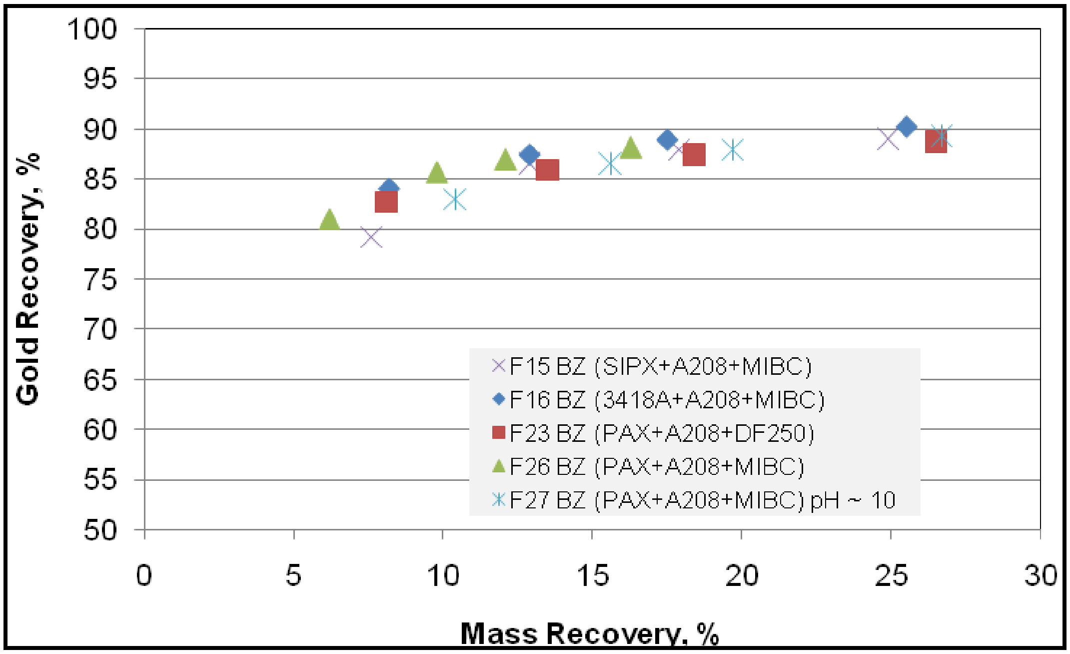


Copper sulphate has been used in the test work since 2009; however, the 2012 and 2013 test results indicate that the addition of copper sulphate does not improve metal recovery.

In 2012 and 2013, most of the flotation tests were conducted at a natural slurry pH that was developed from the previous test programs. The tests that were conducted at an elevated pH (10.5) adjusted by lime with 3418A and A208 as collectors did not show a significant influence of slurry pH on metal recoveries.

The 2009 to 2010 test program investigated the effect of flotation reagents and slurry pH on metallurgical performance. Figure 13.8 shows the test results of the Bridge Zone composite sample. It appears that the reagents and slurry pH have an insignificant effect on gold recovery.

Figure 13.8 Effect of Reagent and Slurry pH on Gold Recovery (2009 to 2010)



Note: Test F27 was conducted at a higher pH; the other tests were conducted at a natural pH.

Cleaner Flotation Test Work

In the early stages of the 2012 test program, a master composite labelled as Composite BJ-A, was used to study the metallurgical responses to the cleaner flotation. Two different cleaner procedures were tested: one procedure upgraded the rougher and scavenger concentrates separately after regrinding, and the other procedure upgraded the combined rougher and scavenger concentrate. On average, both cleaner procedures produced similar results. Test results from the combined concentrate are shown in Figure 13.9 and Figure 13.10. The results indicate that upgrading efficiencies are good for gold and silver. Gold appears to have better cleaner efficiency than silver.

Figure 13.9 Effect of Cleaner Flotation on Gold Recovery (Composite, 2012)

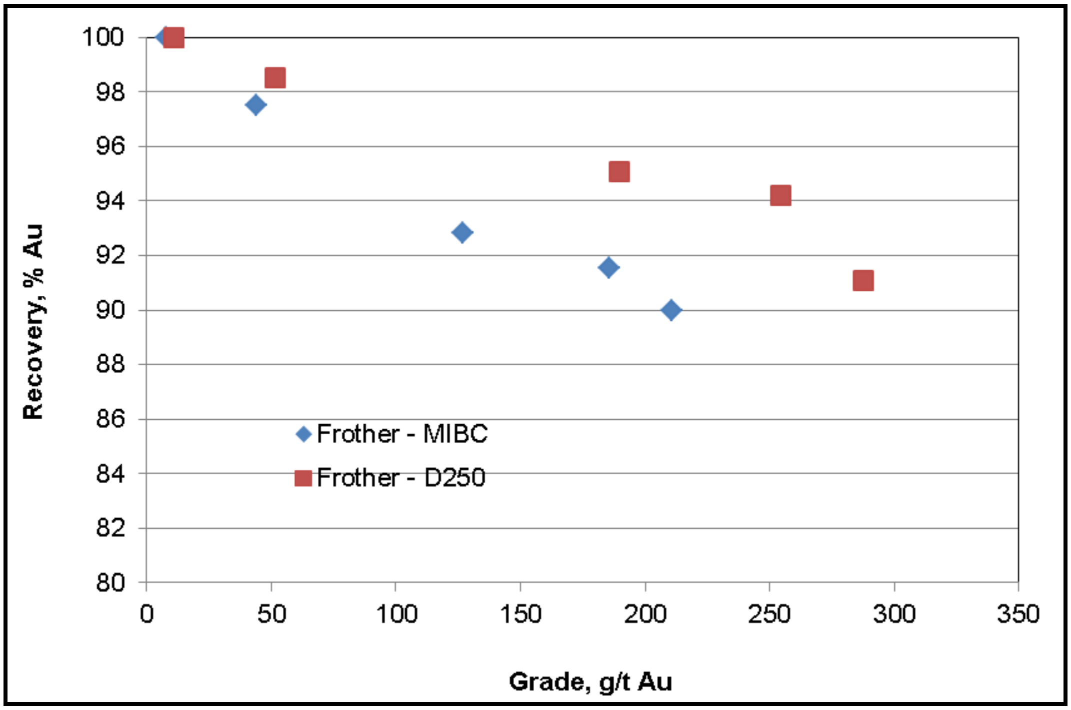
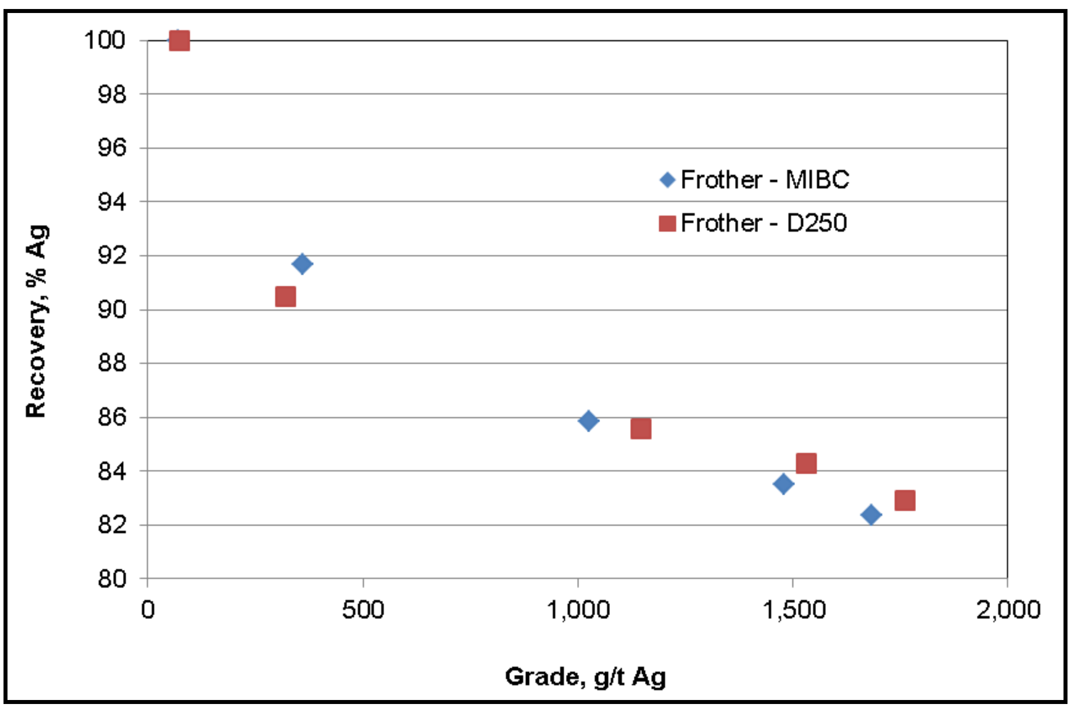


Figure 13.10 Effect of Cleaner Flotation on Silver Recovery (Composite, 2012)



Further cleaner tests were conducted on some of the 2012 and 2013 variability test samples. The cleaner flotation efficiency curves are shown in Figure 13.11 to Figure 13.14. The results show that there are significant variations in the cleaner flotation performance of the rougher/scavenger concentrates. The variations may be caused by fluctuations in flotation feed grades and differences in mineralogy. On average, gold showed better upgrading efficiencies than silver in these tests.

Figure 13.11 Effect of Cleaner Flotation on Gold Recovery (Interval Samples, 2012)

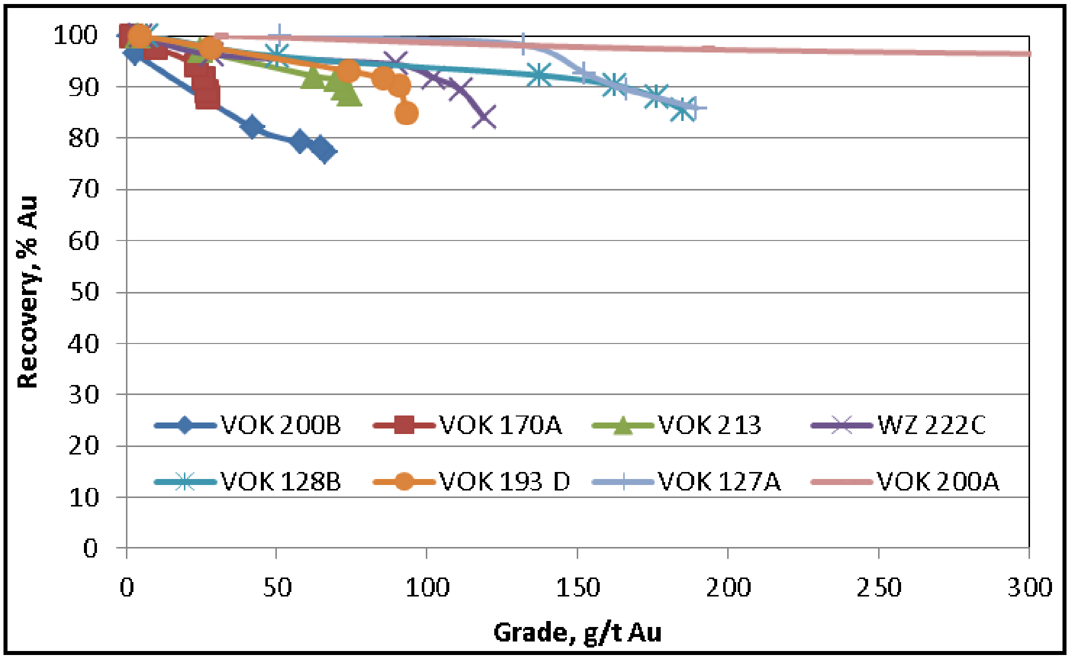


Figure 13.12 Effect of Cleaner Flotation on Gold Recovery (Composites, 2013)

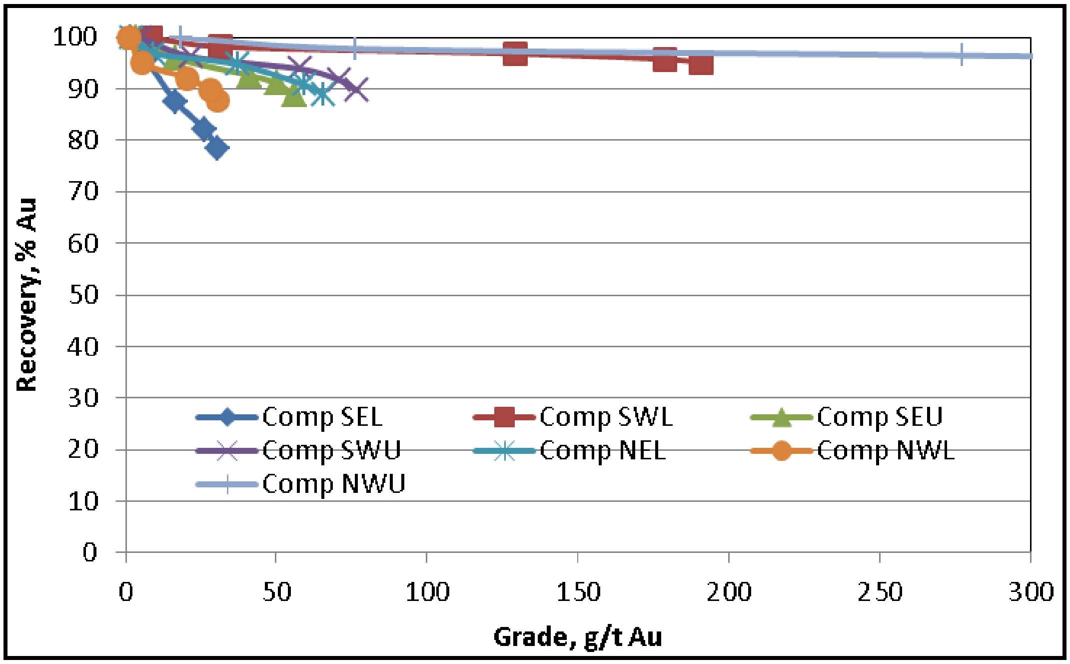


Figure 13.13 Effect of Cleaner Flotation on Silver Recovery (Interval Samples, 2012)

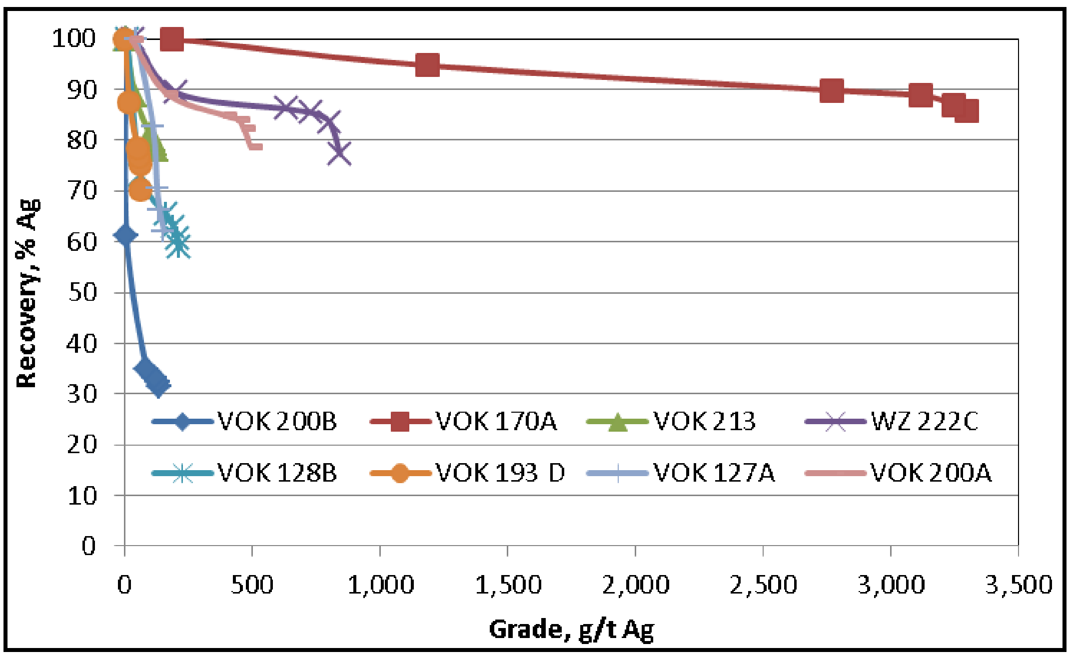
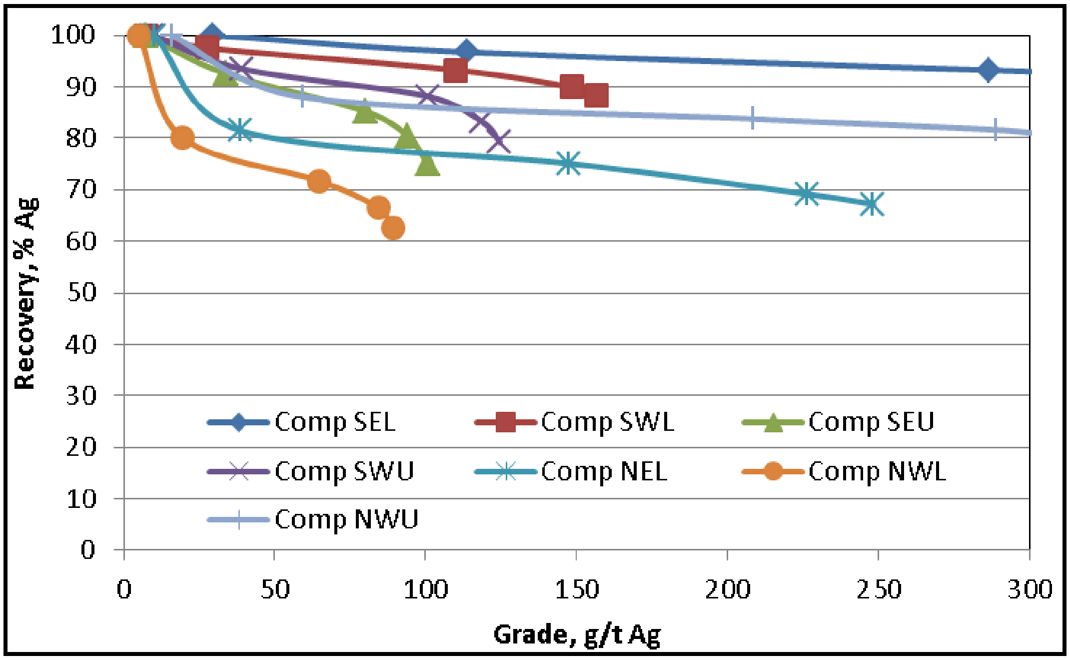
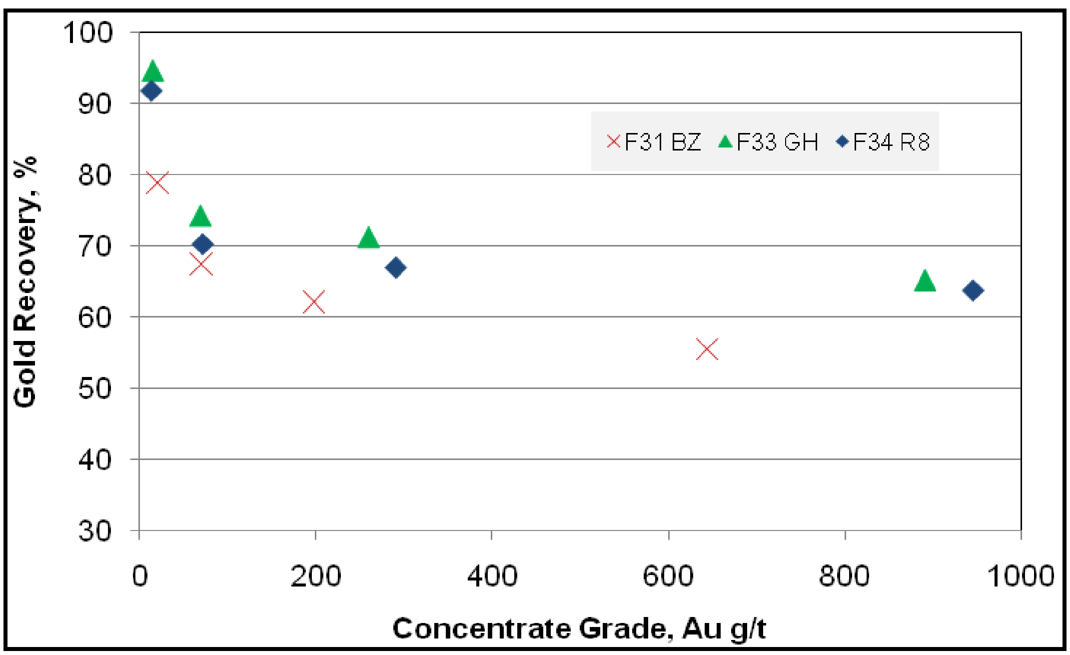


Figure 13.14 Effect of Cleaner Flotation on Silver Recovery (Composites, 2013)



The 2009 to 2010 testing program also studied the effect of upgrading the rougher flotation concentrates on metal recovery. The test results indicate that the cleaner flotation was able to substantially upgrade the concentrates from the Brucejack mineral samples. However, as shown in Figure 13.15, gold recovery was significantly reduced at the first cleaner flotation stage.

Figure 13.15 Effect of Cleaner Flotation on Gold Recovery (2009-2010)



Other Flotation Tests

Potential preg-robbing effects were noted during cyanide leaching tests; therefore, the 2012 test program explored whether it was possible to remove the carbonaceous material by pre-flotation. Although the tests showed that the pre-flotation should be capable of removing a portion of the carbonaceous material, the gold reporting to the carbonaceous concentrate was high (approximately 35% of the gold reporting to the carbonaceous concentrate from the VOK-1 composite sample) and significant carbonaceous material still remained in the tailings of the pre-flotation step.

Exploratory tests were conducted in 2013 to study the upgrading potential of head samples by heavy media separation. No encouraging results were produced.

Gravity Concentration Test Work

Metallic gold determination tests and gravity concentration tests conducted during the 2009 test programs showed that the Brucejack mineralization contains a significant amount of fine nugget gold grains. The metallic gold determination test results are shown in Table 13.16 to Table 13.18.

The results indicate that free gold occurrence is substantially different from sample to sample. The VOK-3 and VOK-4 composite samples may contain significant amounts of native gold; however, the VOK-1 and VOK-2 composite samples may contain significantly less nugget gold. Compared to the VOK Zone samples, the West Zone samples, on average, appear to contain less native gold grains. The 2009 to 2011 test results show similar gold occurrence patters as those identified in the 2012 test work.

Table 13.16 Metallic Gold Test Results – Composite Samples (2012)

Sample ID	Screen Tyler Mesh	Grade (g/t)		Distribution (%)		
		Au	Ag	Au	Ag	Mass
VOK-1 Composite	150	27.31	23.02	13.8	4.0	3.5
	-150	6.11	19.98	86.2	96.0	96.5
	Total	**6.85**	**20.09**	**100**	**100**	**100**
VOK-2 Composite	150	13.12	35.76	10.6	17.1	8.0
	-150	9.59	14.96	89.4	82.9	92.0
	Total	**9.87**	**16.62**	**100**	**100**	**100**
VOK-3 Composite	150	1,377.00	362.59	66.9	27.5	5.0
	-150	35.65	50.05	33.1	72.5	95.0
	Total	**102.38**	**65.59**	**100**	**100**	**100**
VOK-4 Composite	150	23.86	24.77	41.4	3.0	5.2
	-150	1.83	43.5	58.6	97.0	94.8
	Total	**2.97**	**42.54**	**100**	**100**	**100**
WZ-1 Composite	150	20.6	25.11	28.0	6.0	7.8
	-150	4.49	33.1	72.0	94.0	92.2
	Total	**5.75**	**32.47**	**100**	**100**	**100**

table continues…

Sample ID	Screen Tyler Mesh	Grade (g/t)		Distribution (%)		
		Au	Ag	Au	Ag	Mass
WZ-2 Composite	150	10.27	164.63	14.5	2.4	6.9
	-150	4.49	501.95	85.5	97.6	93.1
	Total	**4.89**	**478.66**	**100**	**100**	**100**
Composite BJ-A	150	126.66	123.8	42.0	13.1	5.37
	-150	9.91	46.8	58.0	86.9	94.63
	Total	**16.18**	**50.94**	**100**	**100**	**100**
Composite BJ-A	150	98.2	91.7	36	8.3	4.73
	-150	8.68	50.6	64	91.7	95.27
	Total	**12.92**	**52.54**	**100**	**100**	**100**

Table 13.17 Metallic Gold Test Results – Composite Samples (2009 to 2011)

Sample ID	Screen Tyler Mesh	Grade (Au g/t)	Distribution (%)	
			Mass	Au
Composite R8	+150	6.95	4.8	23.1
	-150	1.16	95.2	76.9
	Total	**1.44**	**100.0**	**100.0**
Composite GH	+150	6.66	7.9	30.3
	-150	1.31	92.1	69.7
	Total	**1.73**	**100.0**	**100.0**
Composite BZ	+150	3.89	5.4	12.6
	-150	1.54	94.6	87.4
	Total	**1.67**	**100.0**	**100.0**

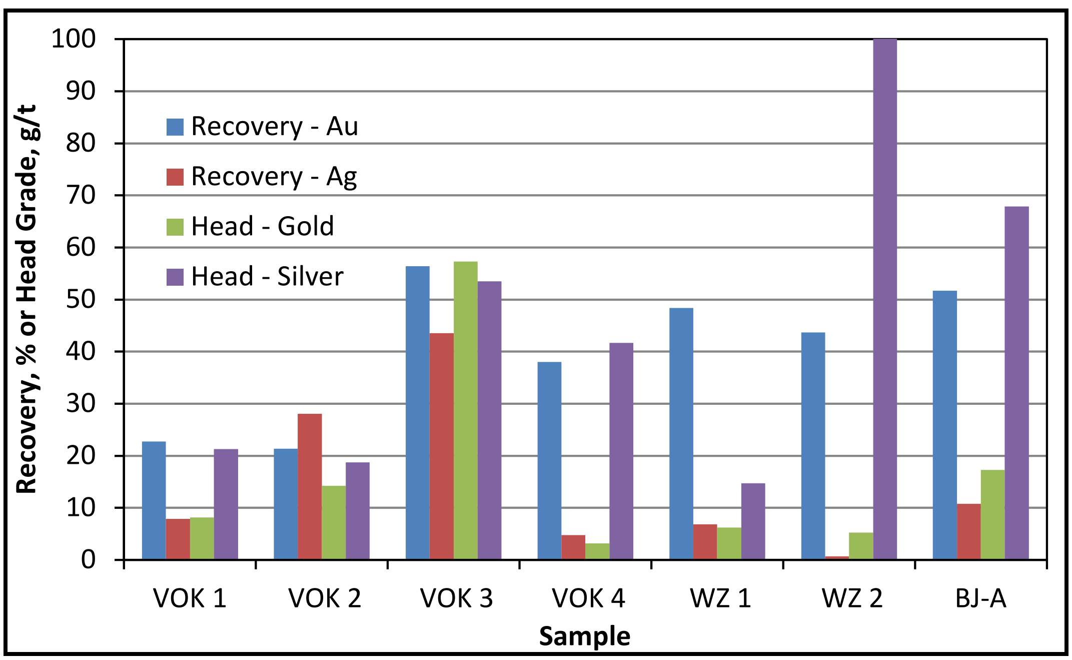
Table 13.18 Metallic Gold Test Results – Individual Samples (2009-2010)

Sample ID	Screen Tyler Mesh	Grade (g/t)		Distribution (%)		
		Au	Ag	Au	Ag	Mass
SU-4	+150	1.91	1.0	9.4	4.1	8.6
	-150	1.74	2.2	90.6	95.9	91.4
	Total	**1.75**	**2.1**	**100.0**	**100.0**	**100.0**
SU-5	+150	2.99	29.3	11.5	3.8	4.2
	-150	1.02	32.7	88.5	96.2	95.8
	Total	**1.10**	**32.6**	**100.0**	**100.0**	**100.0**
SU-6A	+150	9.25	50.6	21.8	4.2	4.7
	-150	1.62	56.9	78.2	95.8	95.3
	Total	**1.98**	**56.6**	**100.0**	**100.0**	**100.0**
SU-6B	+150	100.1	94.0	73.7	27.1	3.8
	-150	1.43	10.1	26.3	72.9	96.2
	Total	**5.23**	**13.3**	**100.0**	**100.0**	**100.0**
SU-10	+150	2.11	2.1	11.4	2.0	4.1
	-150	0.70	4.3	88.6	98.0	95.9
	Total	**0.76**	**4.2**	**100.0**	**100.0**	**100.0**
SU-19	+150	1.65	3.0	4.6	3.2	4.4
	-150	1.57	4.2	95.4	96.8	95.6
	Total	**1.57**	**4.1**	**100.0**	**100.0**	**100.0**
SU-21A	+150	0.64	4.3	3.7	2.0	3.7
	-150	0.64	8.2	96.3	98.0	96.3
	Total	**0.64**	**8.1**	**100.0**	**100.0**	**100.0**
SU-21B	+150	22.0	2.5	34.8	3.0	8.0
	-150	3.58	6.9	65.2	97.0	92.0
	Total	**5.05**	**6.5**	**100.0**	**100.0**	**100.0**
SU-25	+150	2.63	15.0	9.0	10.7	7.3
	-150	2.08	9.8	91.0	89.3	92.7
	Total	**2.12**	**10.2**	**100.0**	**100.0**	**100.0**
SU-27	+150	2.70	0.5	7.5	2.5	2.5
	-150	0.86	0.5	92.5	97.5	97.5
	Total	**0.91**	**0.5**	**100.0**	**100.0**	**100.0**
SU-32A	+150	6.49	15.1	14.2	4.7	4.9
	-150	2.02	15.7	85.8	95.3	95.1
	Total	**2.24**	**15.7**	**100.0**	**100.0**	**100.0**
SU-32B	+150	8.28	51.0	38.1	4.7	6.5
	-150	0.94	73.1	61.9	95.3	93.5
	Total	**1.42**	**71.7**	**100.0**	**100.0**	**100.0**
SU-32C	+150	10.9	9.0	37.1	22.0	10.4
	-150	2.15	3.7	62.9	78.0	89.6
	Total	**3.06**	**4.2**	**100.0**	**100.0**	**100.0**
SU-33	+150	22.6	29.6	59.6	7.8	9.0
	-150	1.52	34.9	40.4	92.2	91.0
	Total	**3.42**	**34.4**	**100.0**	**100.0**	**100.0**
SU-36A	+150	2.12	9.5	15.4	7.9	9.4
	-150	1.21	11.4	84.6	92.1	90.6
	Total	**1.30**	**11.2**	**100.0**	**100.0**	**100.0**
SU-36B	+150	0.69	7.9	12.3	20.4	9.9
	-150	0.54	3.4	87.7	79.6	90.1
	Total	**0.55**	**3.8**	**100.0**	**100.0**	**100.0**

The 2012 and 2013 tests routinely incorporated gravity concentration because there is a significant portion of gold present as nugget grains in the mineralization. Gravity concentration tests were conducted on the head composite samples and the flotation concentrate samples. Two stages of gravity concentration were conducted–the first stage by centrifugal concentration, and the second stage by panning. The test results indicated that most of the samples responded well to gravity concentration. Figure 13.16 shows the gravity concentration results achieved on the 2012 composite samples at a primary grind size of 80% passing between 50 and 60 µm (the BJ-A composite sample was ground to 80% passing approximately 74 µm). Gold recovery reporting to the panning concentrates varied from 21 to 56%, while silver recovery varied from 1 to 44%. The gravity concentration recovery from the BJ-A composite was approximately 52% for gold and 11% for silver.

The variability tests showed that the average gravity concentration recovered approximately 45.8% of gold and 21.4% of silver (total unweighed recoveries of the panning concentrates) from the 71 samples tested, with average head grade values of 21.5 g/t gold and 105 g/t silver. The average panning concentrate grades obtained were 21.7 kg/t gold and 15.4 kg/t silver. The tests also explored the recovery of nugget gold from 11 of the reground rougher flotation concentrates that were generated from the variability tests. On average, gravity concentration recovered 24.5% of the gold and 11.6% of the silver from the rougher flotation concentrates. The panning concentrate grades averaged 1.6 kg/t gold and 3.5 kg/t silver.

Figure 13.16 Gold Recovery by Gravity Concentration – Composite Samples (2012)



Note: The WZ-2 sample contains approximately 446 g/t silver.

The 2009 to 2010 test results indicated that most of the samples responded well to gravity concentration, especially the reground flotation concentrates (Table 13.19). For the flotation concentrates produced from the zone composite sample, approximately 29 to 45% of the gold was recovered into the gravity concentrates containing over 1,000 g/t gold; however, metallurgical responses for silver were not the same as for gold.

The test results also indicated that some of the samples (such as the SU-36B sample) were less amenable to the gravity concentration process.

Table 13.19 Gravity Concentration Test Results (2009 to 2010)

Test ID	Sample ID	Primary Grind/ Regrind Size	Grade (g/t)		Recovery (%)	
			Au	Ag	Au	Ag
GF35	BZ	P_{80} 131 µm	685	428	17.0	4.6
GF37	R8	P_{80} 116 µm	70.5	677	2.7	1.8
GF36	GH	P_{80} 116 µm	158	495	11.0	1.8
GF41	GH	P_{80} 116 µm	331	339	25.7	1.4
FG38	R8	P_{80} <25 µm	1,081	1,222	35.6	2.6
FG39	GH	P_{80} <25 µm	1,918	3,103	44.8	4.5
FG40	BZ	P_{80} <25 µm	1,079	984	29.3	5.9
FG42	SU-32B	P_{80} <25 µm	801	4,193	22.6	1.4
FG43	SU-33	P_{80} <25 µm	5,810	8,341	43.9	4.9
FG44	SU-36A	P_{80} <25 µm	3,337	1,653	42.3	4.0
FG45	SU-36B	P_{80} <25 µm	217	337	10.6	2.4

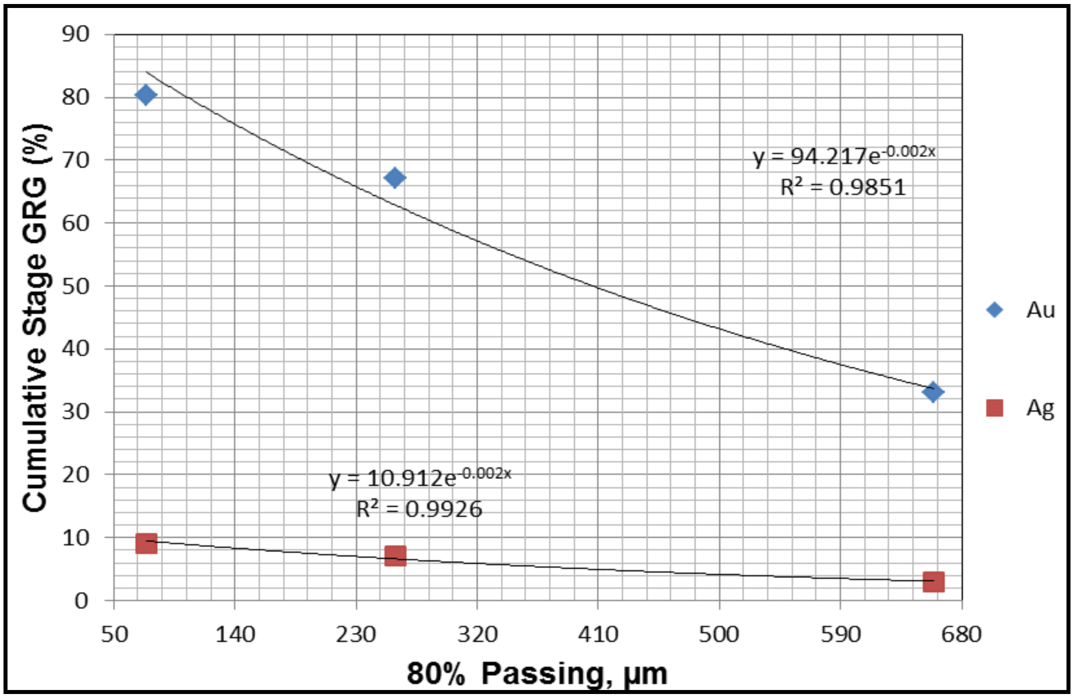
In 2012, Met-Solve and Knelson each performed gravity recoverable gold (GRG) and gravity recoverable silver (GRS) tests to investigate the gold and silver recoveries by gravity concentration and cyanide leaching.

Knelson used the extended gravity recoverable gold and silver (E-GRG) procedure with intensive cyanidation for the tests. The E-GRG test results(Figure 13.17) show the following liberation and gravity recovery characteristics for gold and silver:

- With three stages of grinding and gravity concentration, the GRG value is estimated to be 80.3% and the GRS value is 9.1% at a grind size of 80% passing 74 µm.
- The gold head grade of the samples was 17 g/t, with a final gravity gold tailings grade of 3.2 g/t. The gravity concentrate gold grain sizes corresponding to the P_{20}, P_{50}, and P_{80} values for the sample are 59, 125, and 304 µm, respectively. Accordingly, the GRG gold grains are classified as coarse to very coarse.
- The silver head grade of the sample was 58.6 g/t.

Intensive cyanidation test work on the gravity concentrate produced very encouraging gold and silver recoveries. Gold and silver extractions were 99.5% and 86.9% after the concentrates were leached for 24 hours.

Figure 13.17 Cumulative Stage GRG versus Grind Size for Gold and Silver



According to the GRG/GRS test results, Knelson conducted simulations to determine the gold recovery from a centrifugal gravity concentration/intensive leaching circuit and from a gravity concentration/tabling circuit. Total gold recovery by a centrifugal gravity concentration/intensive leaching circuit was estimated in a range from 39 to 71% depending on the capacity of centrifugal concentrators and gold occurrence (native gold or electrum). When using a gravity concentration/tabling circuit, the expected gold recovery would be between 32 and 65%. The simulation results are summarized in Table 13.20.

Table 13.20 Gravity Concentration Modelling Results (2012)

Equipment	Feed to Gravity (t/h)	Circulation Load Treated (%)	Concentrating Cycle Time (min)	Gravity Recovery (% total Au)	Gravity Concentrate	
					(kg/d)	(g/t)
Centrifugal gravity concentrate upgrading by acacia reactor, assuming gold is present as native gold						
XD20	40	11	15	46	1,920	12,445
XD30	80	21	20	56	2,520	11,500
QS40	140	37	30	63	2,976	10,857
2 x QS40	280	74	30	69	5,952	5,965
2XQS48	365	97	30	71	6,240	5,863
Centrifugal gravity concentrate upgrading by tabling, assuming gold is present as native gold						
XD20	40	11	15	38	1,920	10,168

table continues...

Equipment	Feed to Gravity (t/h)	Circulation Load Treated (%)	Concentrating Cycle Time (min)	Gravity Recovery (% total Au)	Gravity Concentrate (kg/d)	Gravity Concentrate (g/t)
XD30	80	21	20	48	2,520	9,796
QS40	140	37	30	55	2,976	9,528
2 x QS40	280	74	30	63	5,952	5,408
2 x QS48	365	97	30	65	6,240	5,363
Centrifugal gravity concentrate upgrading by acacia reactor, assuming gold is present as electrum						
XD20	40	11	15	39	1,920	10,490
XD30	80	21	20	50	2,520	10,116
QS40	140	37	30	57	2,976	9,834
2 x QS40	280	74	30	65	5,952	5,567
2 x QS48	365	97	30	67	6,240	5,511
Centrifugal gravity concentrate upgrading by tabling, assuming gold is present as electrum						
XD20	40	11	15	32	1,920	8,451
XD30	80	21	20	42	2,520	8,528
QS40	140	37	30	49	2,976	8,503
2 x QS40	280	74	30	57	5,952	4,954
2 x QS48	365	97	30	60	6,240	4,956

Met-Solve also conducted GRG and intensive leach tests, as well as gravity concentration circuit simulations to estimate the recoveries of gold and silver from the mineralization.

Table 13.21 summarizes the test results, which show:

- The sample had a high GRG value of 80.7%, which is in agreement with the test results produced by Knelson.
- The extractions from the centrifugal concentrates by intensive leach were 99.2% for gold and 92.2% for silver.
- The intensive cyanide leach option on the gravity concentrate is predicted to have better overall gold and silver recovery than the concentrate tabling process.

Table 13.21 Gravity Separation Test Results

Elements	Head Grade Calculated (g/t)	Recovery (%)		
		Gravity Concentration Three- stage Falcon	Intensive CN Leach	Gravity Upgrading Table
Gold	18.7	80.7	99.2	61.1
Silver	63	33.7	92.2	42.5

Met-Solve performed mathematical modelling based on the data obtained from the batch test work. Table 13.22 shows the mathematical modelling results for gold recovery from the primary grind circuit. As predicted, approximately 49.3% of the gold would report to the centrifugal concentrator when 35% of the hydrocyclone underflow is sent to the gravity concentrator.

Table 13.22 Mathematical Model Results – Gold Recovery

Feed Grade (g/t Au)	GRG Content (%)	Mass Split to Gravity (%)	Total Hydrocyclone Underflow Tonnage (t/h)	Total Mass to Ball Mill (t/h)	Total Mass to Centrifugal Concentrator (t/h)	Recovery to Primary Gravity Concentrator (% Au)
23.6	83.2	13.3	564	489	75	28.8
		26.6		414	150	43.3
		35.0		367	197	49.3
		44.3		314	250	54.3
		70.9		164	400	63.3

CYANIDE LEACH TEST WORK

Inspectorate conducted cyanide leach tests to investigate gold extraction from various samples, including head samples, flotation concentrate samples, and flotation tailings samples.

During the 2012 test program, substantial cyanide leach test work was conducted on:

- the composite samples
- the individual drill core interval samples
- the flotation concentrates produced from the samples.

The initial tests used a direct cyanide leach procedure to investigate gold and silver extraction from the head samples. A high cyanide concentration (3 g/L sodium cyanide) was used to ensure that lixiviate concentration was not the leach rate limiting factor for gold and silver extraction. The primary grind size was targeted at 80% passing 50 to 70 µm. The slurry pH was maintained at 10.5 and the leach retention time was 48 hours. In an effort to improve metal extraction from the samples with poor extraction rates, additional procedures were applied, such as finer grinding, carbon-in-leach (CIL), and extended leach retention time. The gold and silver extraction rates for the 31 test samples are shown in Figure 13.18 and Figure 13.19, respectively.

Figure 13.18 Gold Cyanide Extraction – Whole Ore Leach (2012)

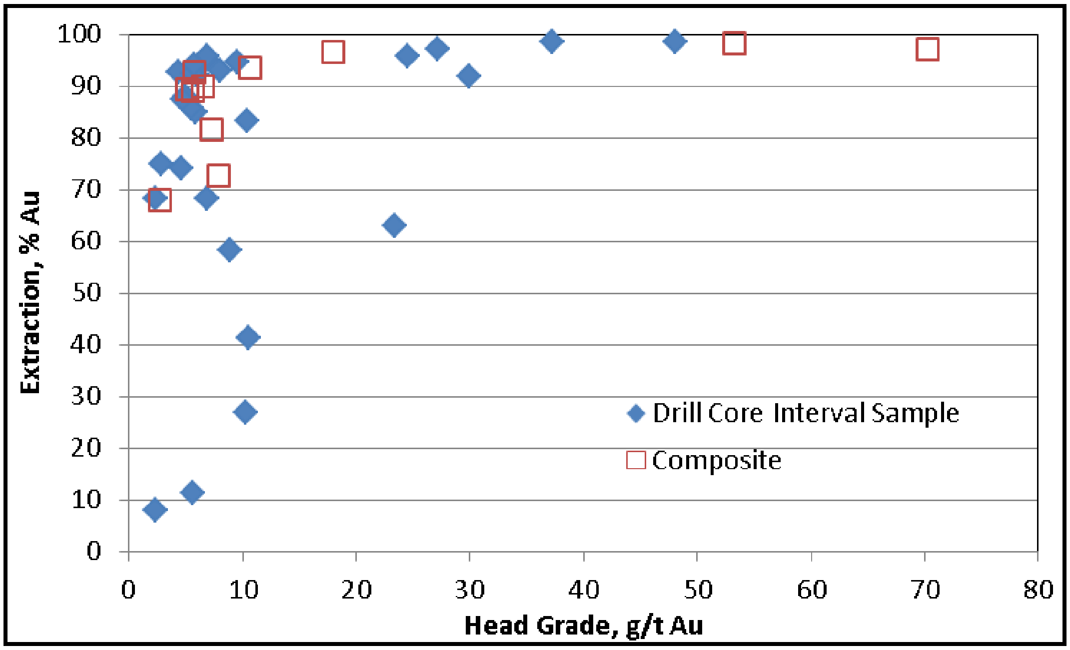
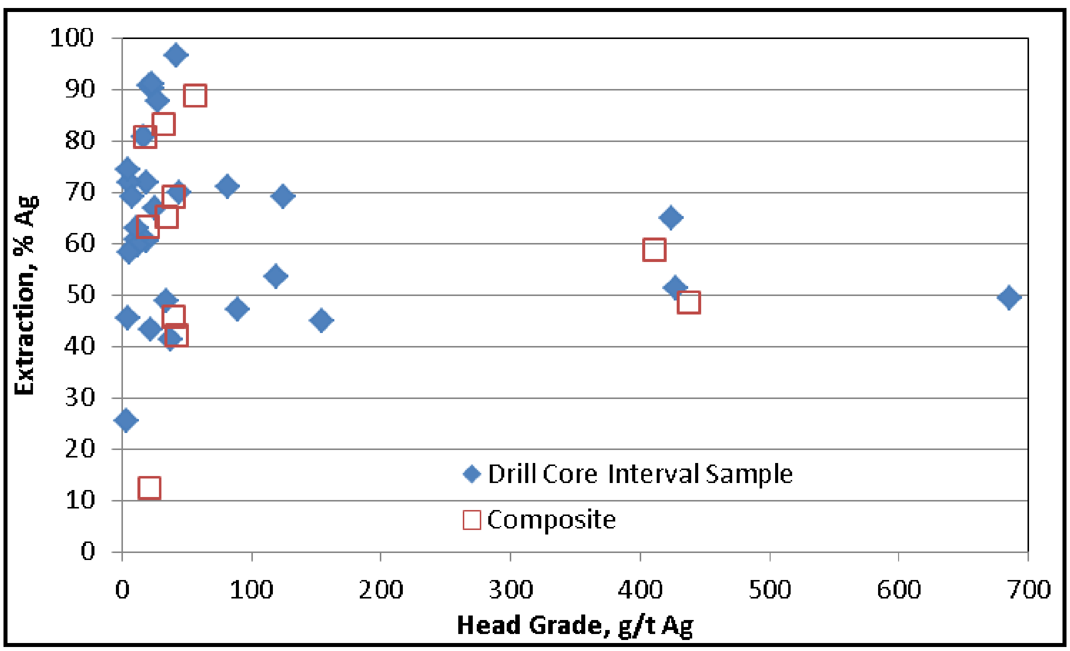


Figure 13.19 Silver Cyanide Extraction – Whole Ore Leach (2012)



The results show substantial variations in metal extraction. For the drill core samples, extractions range from 8 to 99% for gold with an average of 77.6%, and from 25 to 96% for silver with an average of 62.8%. For the composite samples, extractions range from 61 to 99% for gold with an average of 87.8%, and from 12 to 89% for silver with an average of 62.6%.

Some of the 2012 test samples showed a poor metallurgical response to the cyanidation procedures. As reported by Frank Wright, the poor responses may be the result of the adverse effect of increased arsenic, graphite, and electrum content–which are related to the mineralogical nature.

Cyanide leach tests were also conducted on the rougher flotation concentrates and cleaned scavenger flotation concentrates produced from the composite samples. The test results are shown in Figure 13.20 for gold and Figure 13.21 for silver. The average gold extraction rate of the blended rougher concentrate was 96% for the VOK Zone sample, and 89.5% for the West Zone sample. Most of the scavenger concentrates produced similar gold extraction rates as the rougher concentrate, excluding the VOK-1 and VOK-4 samples, which were only 60% and 38%, respectively. The average silver extraction rate of the blended rougher concentrate was 77% for the VOK Zone sample, and 54% for the West Zone sample. The average silver extraction from the scavenger concentrates was 78%, although there was some variation in the extractions.

Figure 13.20 Gold Cyanide Extraction – Concentrate Leach (2012)

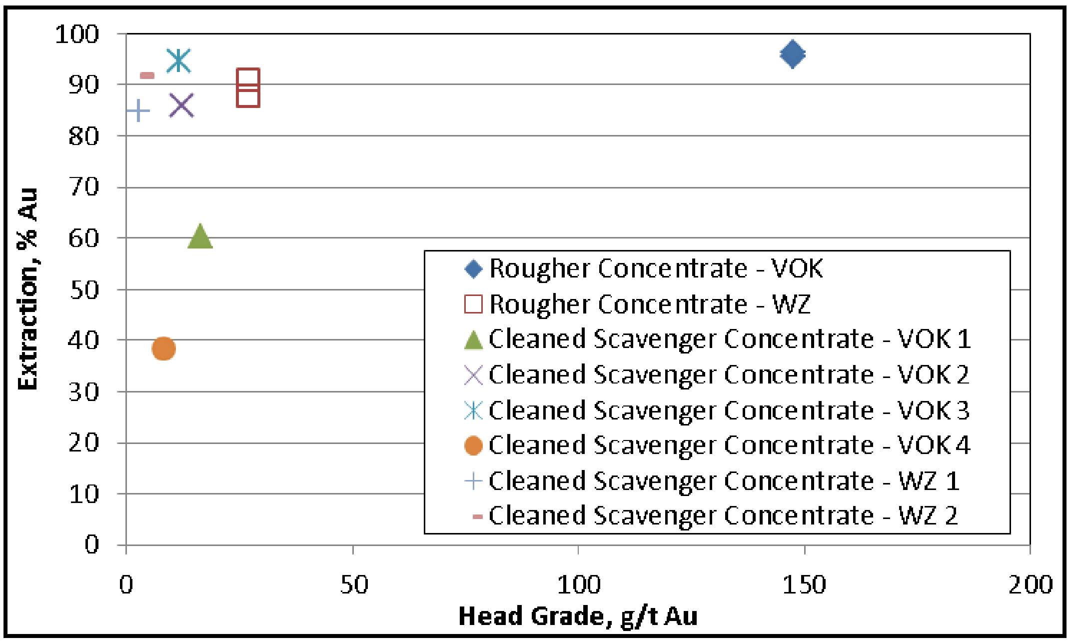
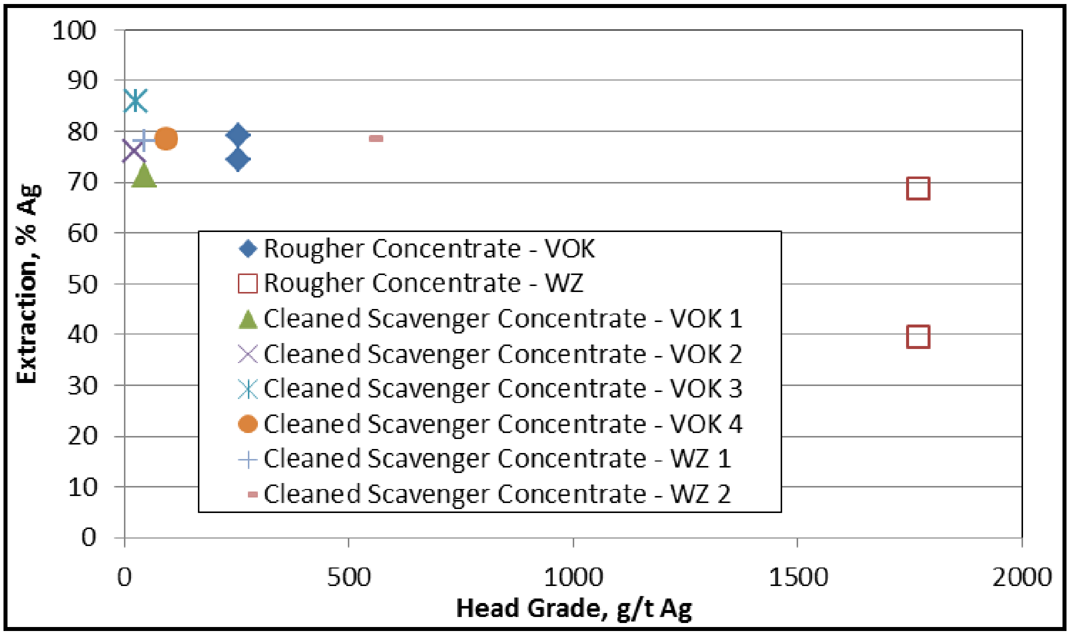


Figure 13.21 Silver Cyanide Extraction – Concentrate Leach (2012)



In 2013, further cyanide leach tests were conducted on the concentrates produced from the locked cycle flotation tests. The concentrates were leached for 48 hours while the cyanide concentration in the solution was maintained at 3 g/L sodium cyanide. The maximum gold extraction was between 90 and 95% after approximately 24 hours. Silver extraction was between 80 and 85% after the concentrates were leached for 48 hours.

In 2012, Joe Zhou Mineralogy Ltd. (JZM 2012) conducted a detailed study on two leach residue samples to determine mineralogy and gold/silver deportment. These samples included Composite VOK-1 (Test C8) and Composite WZF-1 (Test C11) from the West Zone footwall.

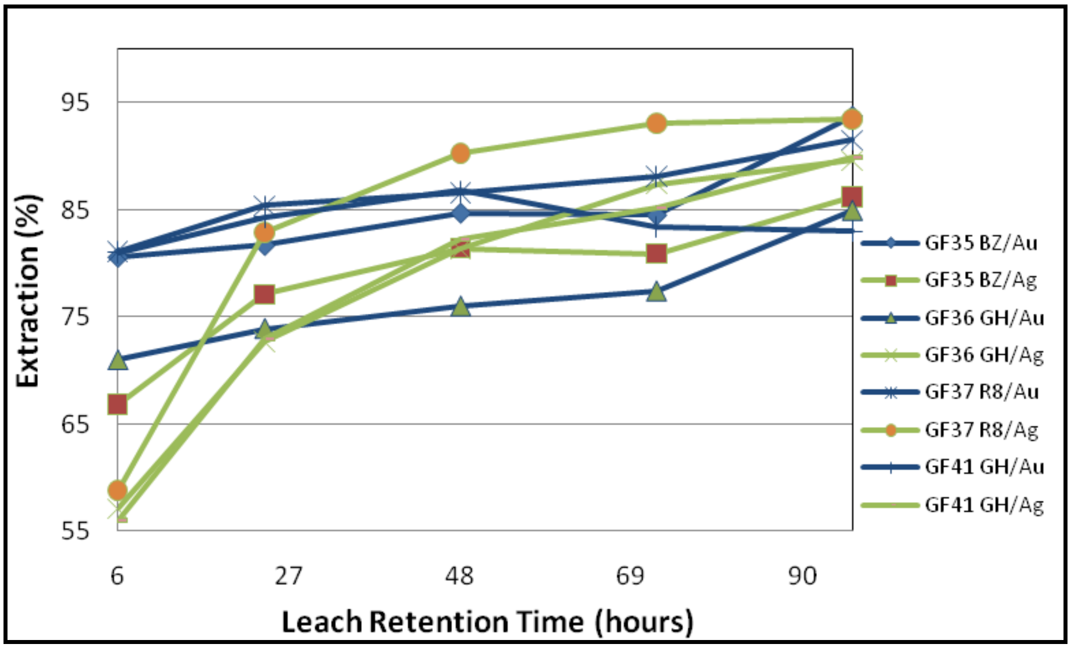
The study indicated that the gold in both leach residue samples occurred mainly as submicroscopic gold, with a modest amount of microscopic gold. The submicroscopic/microscopic gold occurrences are the main cause of gold losses, since conventional leaching cannot recover the gold without finer grinding and pre-oxidation. The silver in both leach residue samples occurred mainly as liberated particles, with some attached and locked particles. Submicroscopic silver also occurred in pyrite. The liberated and attached silver minerals should be recoverable using an extended leach retention time at a higher cyanide concentration. However, locked silver minerals cannot be recovered without further regrinding. Table 13.23 summarizes the gold occurrences in various forms in the residue.

Table 13.23 Occurrences of Gold in Leach Residues

Sample ID	Au (g/t)	Gold Distribution (%)			Gold in Other Minerals	Total
		Microscopic	Submicroscopic			
		Electrum	Pyrite	Arsenopyrite		
Leach Residue C8 (VOK)	10.4	12.5	72.9	2.1	12.5	100.0
Leach Residue C11 (West Zone Footwall)	8.3	21.8	52.1	7.5	18.6	100.0

During the 2010 to 2011 test program, cyanidation tests were conducted on the flotation concentrates using a combination of gravity concentration and flotation concentration. The sodium cyanide concentration used was 3 g/L and the slurry pH was adjusted by lime to 10.5. Figure 13.22 shows the effect of leach retention time on gold extraction. At a leach retention time of 27 hours, the leaching process extracted between 73 and 86% of the gold and between 73 and 82% of the silver. An increase in the leach retention time beyond 72 hours improved the gold and silver recoveries by up to approximately 10%.

Figure 13.22 Bulk Concentrate Leach Retention Time Test Results (2010 to 2011)



During the 2009 to 2010 test work, leach tests were also conducted on head samples and concentrate samples. The leaching test results on the head samples are summarized in Table 13.24. The tests were conducted at a pH of 10.5 and a sodium cyanide concentration of 3 g/L, with three different primary grind sizes.

Table 13.24 Head Sample Cyanidation Test Results (2009-2010)

Test No	Sample ID	Grind Size (P_{80} µm)	Calculated Head (g/t)		Extraction (%)		Residue Grade (g/t)		Consumption (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
C1	BZ	71	1.79	9.68	81.0	59.7	0.34	3.9	2.09	0.28
C2	BZ	40	2.01	10.3	85.1	63.0	0.30	3.8	2.17	0.37
C3	BZ	127	2.35	10.6	84.7	57.5	0.36	4.5	1.97	0.23
C4	GH	72	1.41	40.2	77.9	67.6	0.31	13.0	1.91	0.24
C5	GH	42	1.35	38.3	76.3	72.0	0.32	10.8	1.94	0.23
C6	GH	119	1.49	36.6	72.4	68.6	0.41	11.5	1.77	0.23
C7	R8	78	1.37	26.4	75.9	65.2	0.33	9.2	1.71	0.32
C8	R8	44	1.24	24.5	75.0	68.2	0.31	7.8	2.02	0.32
C9	R8	131	1.34	25.2	73.8	63.2	0.35	9.3	1.85	0.33

At a leach retention time of 48 hours, gold extraction from the head samples ranged from 72 to 85%; silver extraction from the head samples was lower, ranging from 58 to 72%. The influence of primary grind size on the gold and silver recoveries was relatively insignificant. The test results indicated that gold extraction from Composite BZ was better than Composites GH and R8, possibly because Composite BZ had a higher gold head grade than the other two samples. It appears that all the samples tested would need a longer leach retention time because leaching was not complete when the tests were terminated. Sodium cyanide consumption varied from 1.7 to 2.2 kg/t.

Further tests were conducted on the flotation concentrates that were reground to 90% passing 25 µm. The tests used a high sodium cyanide concentration of 5 g/L, and an increased leach retention time of 96 hours. The test results are summarized in Table 13.25.

The test results indicate that between approximately 79 and 86% of the gold can be extracted from the reground concentrates. The tests also produced similar silver extraction results. The addition of potassium permanganate ($KMnO_4$), lead nitrate ($Pb(NO_3)_2$), and oxygen did not improve gold extraction. The leach retention time required for gold ranged from approximately 48 to 72 hours; however, silver required a longer leach retention time. Cyanide consumption was high, ranging from 13.7 to 16.0 kg/t sodium cyanide, which was possibly due to the high cyanide dosage (5 g/L sodium cyanide) and fine grind size.

Table 13.25 Concentrate Cyanidation Test Results (2009 to 2010)

Test No.	Sample ID*	Pre-treatment	Calculated Head (g/t)		Extraction (%)		Residue Grade (g/t)		Consumption (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
C10/F24	R8	Regrind	8.0	125	86.0	86.7	1.13	16.6	13.8	0.55
C11/F25	GH	Regrind	8.6	203	79.4	87.3	1.77	25.8	15.4	1.08
C12/F26	BZ	Regrind	11.6	56	82.6	79.7	2.02	11.3	15.6	0.61
C13/F24	R8	$KMnO_4$ to regrind	8.1	123	82.7	85.5	1.40	17.9	13.7	0.41
C14/F25	GH	Regrind + oxygen in leach	9.2	129	72.5	81.2	2.54	24.3	16.0	1.77
C15/F26	BZ	$Pb(NO_3)_2$ to regrind	10.7	55	69.9	73.2	3.22	14.8	14.5	1.53

Note: * Rougher + scavenger concentrate.

Based on the findings of the preliminary test work, Inspectorate conducted further testing using a combination of flotation, gravity concentration, and cyanidation to recover gold and silver from the Brucejack mineralization. There were three different process combinations:

- primary grind, gravity concentration, rougher/scavenger flotation, and regrind on the flotation concentrate, followed by cyanidation on the reground concentrate (Flowsheet A)
- primary grind, rougher/scavenger flotation, regrind on the flotation concentrate, and gravity concentration on the reground concentrate, followed by cyanide leaching on gravity tailings (Flowsheet B)
- primary grind, gravity concentration, rougher/scavenger flotation, regrind on the flotation concentrate, gravity concentration on the reground concentrate, followed by cyanide leaching on the gravity tailings, and intensive leaching on the panning tailings (Flowsheet C).

Test results for the three different process combinations are presented in Table 13.26 to Table 13.28.

As shown in Table 13.26, the flotation and gravity concentration recovered approximately 94% of the gold from the BZ sample, and 97% of the gold from the R8 and GH samples. The gold leach extraction rates from the flotation concentrates were higher than 91% for the BZ and R8 samples. Compared to the other two samples, the GH sample showed a lower gold cyanide extraction at approximately 84% on average.

Table 13.26 Test Results - Gravity Concentration, Flotation and Cyanide Leach Combined Flowsheet (Flowsheet A) (2009 to 2010)

Test ID/Sample ID	Primary Grind/ Regrind Sizes	Grade (g/t)		Recovery/Extraction* (%)	
		Au	Ag	Au	Ag
GF35/Composite BZ					
Gravity Concentrate	P_{80} 131 µm	685	428	17.0	4.4
Flotation Concentrate	-	18.6	45.2	77.1	77.3
Leach on Flotation Concentrate	P_{90} <25 µm	-	-	93.7	86.2
Head	-	4.5	10.9	-	-
GF37/Composite R8					
Gravity Concentrate	P_{80} 116 µm	70.5	677	2.7	1.8
Flotation Concentrate	-	11.5	158	94.4	91.0
Leach on Flotation Concentrate	P_{90} <25 µm	-	-	91.5	93.7
Head	-	2.8	39.5	-	-
GF36/Composite GH					
Gravity Concentrate	P_{80} 116 µm	158	495	11.0	1.8
Flotation Concentrate	-	9.6	200	85.5	92.5
Leach on Flotation Concentrate	P_{90} <25 µm			84.9	89.6
Head	-	1.9	36.3	-	-

table continues…

Test ID/Sample ID	Primary Grind/ Regrind Sizes	Grade (g/t)		Recovery/Extraction* (%)	
		Au	Ag	Au	Ag
GF41/Composite GH					
Gravity Concentrate	P_{80} 116 µm	331	339	25.7	1.4
Flotation Concentrate	-	7.7	186	71.8	92.8
Leach on Flotation Concentrate	P_{90} <25 µm	-	-	83.0	89.9
Head	-	1.8	34.5	-	-

Notes: *Extraction refers to flotation concentrate. Leach retention time: 96 hours. Cyanide concentration: 5 g/L.

As shown in Table 13.27, Flowsheet B produced a much higher gold gravity concentration recovery from the BZ, GH, and R8 samples when compared to Flowsheet A. Also, the tests indicated that the leach retention time for the gravity concentration tailings was significantly reduced. It appears that most of the leachable gold in the gravity concentration tailings was extracted within 25 hours (approximately 90% or more of the leachable gold was extracted within 6 hours).

Flowsheet B was also used to test the SU-32B, SU-33, SU-36A, and SU-36B samples. Gold and silver flotation recoveries obtained from these samples were similar to those achieved from three zone composite samples; however, the gold and silver leaching extraction rates were lower.

The SU-32B and SU-36B samples also produced lower gold recoveries at the gravity concentration stage.

Table 13.27 Test Results – Flotation, Gravity Concentration and Cyanide Leach Combined Flowsheet (Flowsheet B) (2009 to 2010)

	Primary Grind/ Regrind Sizes	Concentrate Grade (g/t)		Recovery/Extraction (%)	
		Au	Ag	Au	Ag
GF38/Composite R8*					
Flotation Concentrate	P_{80} 128 µm	7.51	106	94.1	88.6
Gravity Concentrate	P_{94} 33 µm	1,081	1,222	35.6	2.6
Gravity Tailing	-	4.68	103	58.5	86.0
Leach on Gravity Tailing	-	-	-	91.8	83.6
Head	-	2.03	26.5	-	-
GF39/Composite GH*					
Flotation Concentrate	P_{80} 141 µm	12.9	212.1	97.1	98.7
Gravity Concentrate	P_{90} <25 µm	1,918	3,103	44.8	4.5
Gravity Tailing	-	4.68	103.2	52.3	94.2
Leach on Gravity Tailing	-	-	-	86.2	68.7
Head	-	1.99	32.1	-	-
GF40/Composite BZ*					
Flotation Concentrate	P_{80} 133 µm	8.60	44.4	85.1	97.3
Gravity Concentrate	P_{90} <25 µm	1,079	984	29.3	5.9

table continues...

	Primary Grind/ Regrind Sizes	Concentrate Grade (g/t)		Recovery/Extraction (%)	
		Au	Ag	Au	Ag
Gravity Tailing	-	4.68	103	55.7	91.4
Leach on Gravity Tailing	-	-	-	80.9	68.7
Head	-	1.70	7.68	-	-
GF42/Composite SU-32B**					
Flotation Concentrate	P_{80} 109 µm	4.71	382	93.1	90.8
Gravity Concentrate	P_{80} <25 µm	801	4193	22.6	1.4
Gravity Tailing	-	3.57	376	70.5	89.3
Leach on Gravity Tailing	-	-	-	78.7	78.6
Head	-	0.99	82.3	-	-
GF43/Composite SU-33**					
Flotation Concentrate	P_{80} 92 µm	13.5	164	98.5	93.3
Gravity Concentrate	P_{80} <25 µm	5,810	8,341	43.9	4.9
Gravity Tailing	-	7.50	156	54.6	88.4
Leach on Gravity Tailing	-	-	-	87.6	78.2
Head	-	2.32	29.7	-	-
GF44/Composite SU-36A**					
Flotation Concentrate	P_{80} 138 µm	8.95	45.7	97.0	94.5
Gravity Concentrate	P_{80} <25 µm	3,337	1,653	42.3	4.0
Gravity Tailing	-	5.05	43.8	54.7	90.5
Leach on Gravity Tailing	-	-	-	61.5	66.2
Head	-	2.12	11.1	-	-
GF45/Composite SU-36B**					
Flotation Concentrate	P_{80} 96 µm	2.71	19.3	91.5	95.0
Gravity Concentrate	P_{80} <25 µm	217	337	10.6	2.4
Gravity Tailing	-	5.05	43.8	80.9	92.6
Leach on Gravity Tailing	-	-	-	56.9	63.3
Head	-	0.58	4.0	-	-

Notes: *Extraction is referred to gravity concentration tailings; leach retention time = 25 hours; direct cyanide leach; cyanide concentration = 5 g/L.
**Extraction is referred to gravity concentration tailings; leach retention time = 24 hours; CIL; cyanide concentration = 3 g/L.

As shown in Table 13.28, Flowsheet C produced a higher than 60% gold recovery from the WZ1, GH2, and SU98 samples through two stages of gravity concentration. This indicated that a significant amount of gold in the mineralization occurred in the form of nugget gold with a wide range of grain sizes. However, a much smaller amount of silver occurred as native silver. The test results also indicated that intensive cyanide leaching produced higher than 93% gold and silver extractions from the high-grade gravity cleaner concentration tailings (panning tailings). The gold leaching recoveries from the centrifugal gravity concentration tailings were less than 65% for the WZ1 and GH2 samples, which may imply that a portion of the gold is intimately associated with the host minerals of the samples tested.

Further gravity concentration test work was conducted on the blended rougher flotation concentrate produced from the various flotation tests. The centrifugal gravity tailings were subjected to cyanide leaching. As shown in Table 13.29, gravity concentration on the reground concentrates recovered 37% of the gold from the GH2 concentrate and 29% of the gold from the WZ1 concentrate. The gold leaching extractions from the gravity tailings were 84% for the GH2 sample and 75% for the WZ1 sample.

Table 13.28 Test Results - Gravity Concentration, Flotation, Secondary Gravity Concentration and Cyanide Leach Combined Flowsheet (Flowsheet C) (2010 to 2011)

	Primary Grind/ Regrind Sizes	Concentrate Grade (g/t)		Recovery/Extraction (%)	
		Au	Ag	Au	Ag
GF26/Composite GH2					
Primary Gravity Concentrate	P_{80} 125 µm	1,808	183	36.4	0.32
Flotation Concentrate	P_{80} 125 µm	16	302	62.0	98.6
Secondary Gravity Concentrate	P_{80} 7.1 µm	1,116	2,650	51.5	8.1
Gravity Rougher Tailing	-	5	189	31.3	76.9
Gravity Panning Tailing	-	70	927	17.2	15
Intensive Leach on Gravity Panning Tailing	-	-	-	93.6	95.6
Leach on Gravity Rougher Tailing	-	-	-	61.3	64
Head	-	5	53	-	-
Overall Recovery	-	-	-	90.4	71.2
GF27/Composite SU98*					
Primary Gravity Concentrate	P_{80} 123 µm	11,959	186	33.2	0.3
Flotation Concentrate	P_{80} 123 µm	214	556	66.2	98.8
Secondary Gravity Concentrate	P_{80} 6.9 µm	13,281	11,323	79.1	27.7
Gravity Rougher Tailing	-	35	264	19.6	61.0
Gravity Pan Tailing	-	69	1,412	1.4	11.3
Intensive Leach on Gravity Panning Tailing	-	-	-	95.3	95.8
Leach on Gravity Rougher Tailing	-	-	-	97.2	66.9
Head	-	73	205	-	-
Overall Recovery	-	-	-	99.1	78.9
GF25/Composite WZ1					
Primary Gravity Concentrate	P_{80} 120 µm	1,151	194	26.4	0.4
Flotation Concentrate	P_{80} 120 µm	12	163	70.6	97
Secondary Gravity Concentrate	P_{80} 7 µm	646	1,600	50.4	9.1
Gravity Rougher Tailing	-	3	107	31.0	77.3
Gravity Pan Tailing	-	71	716	18.6	13.6
Intensive Leach on Gravity Panning Tailing	-	-	-	94	96.5
Leach on Gravity Rougher Tailing	-	-	-	64.3	66.9
Head	-	2	25	-	-
Overall Recovery	-	-	-	88.7	72.5

Note: *Composite SU-98 is from the area between the West Zone and Galena Hill Zone.

Further cyanide leach tests were carried out on the leach residue, which was further re-ground to 80% passing 10 µm. The test results in Table 13.29 show that additional regrinding and re-leaching extracted 13% more gold and 51% more silver from the leaching residue.

Table 13.29 Gravity/Leaching Test Results on Re-ground Flotation Concentrate (2010 to 2011)

		Concentrate Grade (g/t)		Recovery/Extraction (%)	
	Regrind Size	Au	Ag	Au	Ag
GR3/C25/C29 GH2 Blended Rougher Concentrate					
Re-ground Flotation Concentrate	P_{80} 25 µm	23	458	-	-
Gravity Concentrate	-	2,366	2,849	37	3
Gravity Tailings	-	18.8	442.8	63	97
Leach on Gravity Tailings	-	-	-	84	75
Leach Residue Regrinding	P_{80} <10 µm	2.94	117.7	-	-
Leach on Reground Residue	-	-	-	11	52
Secondary Leach Residue	-	2.05	56.4	-	-
GR2/C24/C28 WZ1 Blended Rougher Concentrate					
Reground Flotation Concentrate	P_{80} 25 µm	8.6	177.4	-	-
Gravity Concentrate	-	941	1,543.5	29.3	2.6
Gravity Tailings	-	6.9	175.3	70.7	97.4
Leach on Gravity Tailings	-	-	-	74.8	82.6
Leach Residue Regrinding	P_{80} <10 µm	1.96	34.5	-	-
Leach on Reground Residue	-	-	-	14.3	50.4
Secondary Leach Residue	-	1.74	18	-	-

VARIABILITY TEST WORK

Since 2011, three variability test work programs were carried out on various samples generated from the VOK Zone, the West Zone, and the other adjacent deposits.

Variability Test Work (2012 and 2013)

The initial stage of the 2012 variability test program studied the metallurgical responses of various VOK Zone and West Zone drill core interval samples to a conceptual gravity-flotation flowsheet. The flowsheet produced a gravity concentrate, a rougher flotation concentrate, and the scavenger concentrate that was reground and cleaned. The rougher flotation concentrate and the cleaned scavenger flotation concentrate were combined, reground, and cyanide-leached to further determine the cyanidation variability responses of the concentrates.

The latter stage of the 2012 variability test program and the 2013 variability test program used similar procedures at the gravity concentration and rougher scavenger flotation stage, but the resulting rougher and scavenger concentrates were re-ground and cleaned to produce a gold-silver bearing concentrate.

The gravity and flotation test results are summarized in Figure 13.23 and Figure 13.24.

Figure 13.23 Metal Recovery - Gravity and Bulk Flotation Flowsheet (2012)

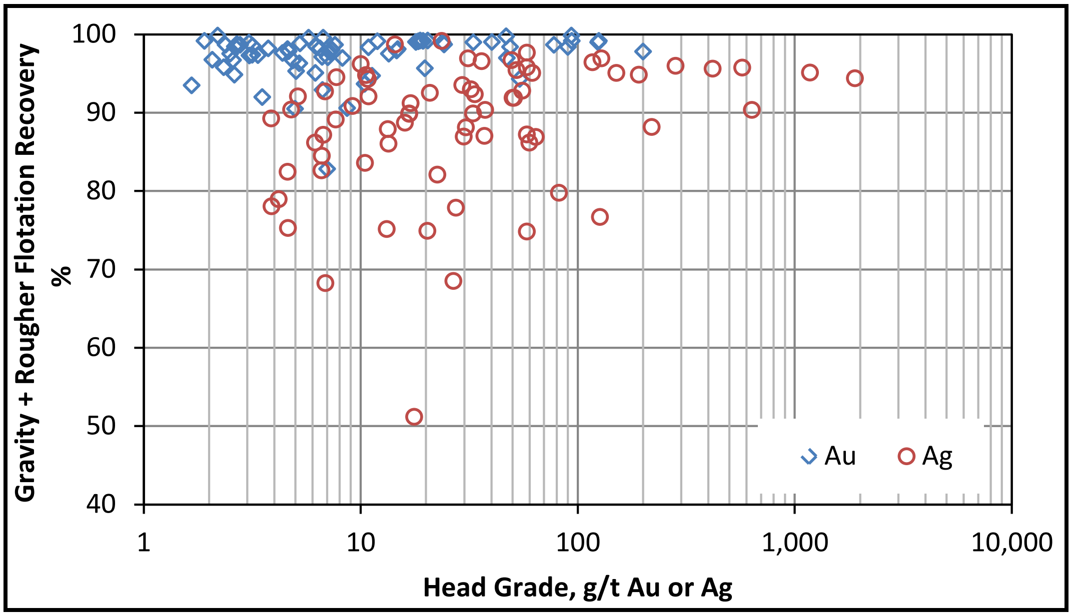
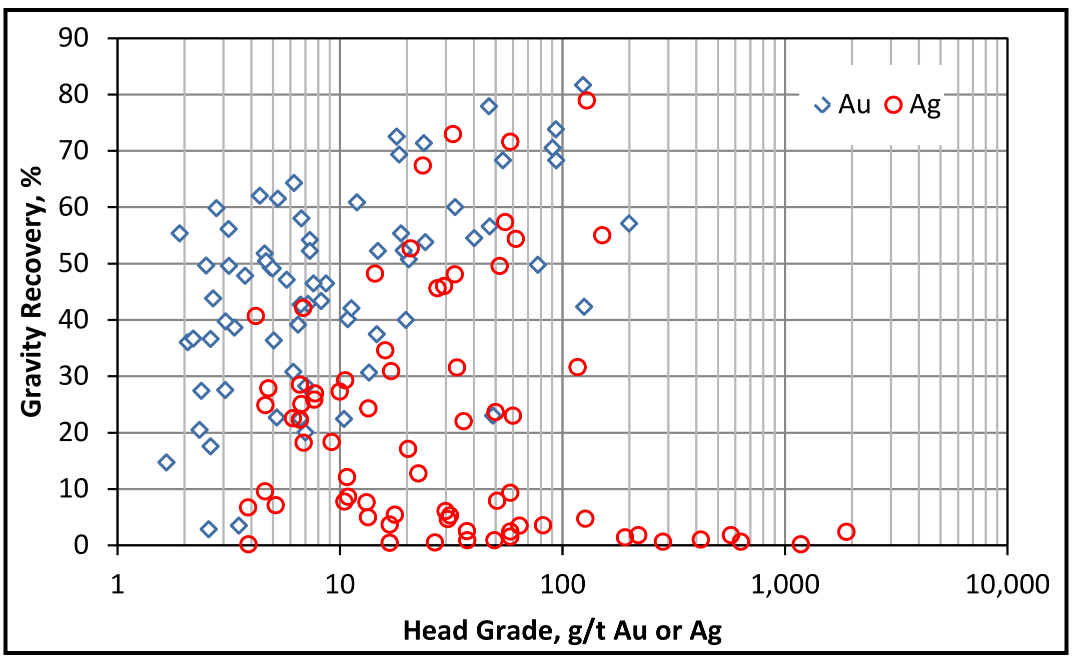


Figure 13.24 Metal Recovery - Gravity Concentration (2012)



In general, the test results were consistent with with the results produced from the composite samples. The overall gold and silver recoveries (gravity concentration and bulk flotation combined flowsheet) increased with increasing head grades. The recoveries for the samples responded well to the gravity/bulk flotation procedure,

including samples with low head grades. Gold recovery varied from 82.8 to 99.8%, averaging 97.2%, while the head gold grade fluctuated from 0.5 to 200 g/t, averaging 21.5 g/t. At the silver head grade range of 3.9 to 1,897 g/t , the silver recovery varied from 51.2 to 99.1%, averaging 88.5%. Gold and silver reporting to the tailings increased with head grade, which implies that more aggressive procedures may be used to improve gold and silver recoveries during periods of the higher-grade feeds (e.g. finer primary grinding and reagent modifications).

A substantial fluctuation was also noted in the gold and silver gravity concentration recoveries (Figure 13.24). Recoveries varied substantially, from 2.8 to 81.7% for gold and from 0.2 to 78.9% for silver. The variation is typically a result of the nugget gold effects. In general, gold recovery by gravity concentration increased with head grade; however, silver recovery reduced with head grade. On average, gold and silver recoveries reporting to the concentrate of the gravity separation procedure were 45.3% and 21.2%, respectively.

There is a good relationship between head gold grade and gravity concentrate gold grade (i.e. concentrate grade increased with feed grade). The average gold grade of the concentrate produced from gravity concentration tests was 21.7 kg/t. However these tests did not show a good correlation between head grade and gravity concentrate grade for silver. The average silver grade of the gravity concentrate was 15.4 kg/t. These relationships are shown in Figure 13.25 and Figure 13.26.

Figure 13.25 Gravity Concentrate Grade versus Head Grade – Gold (2012)

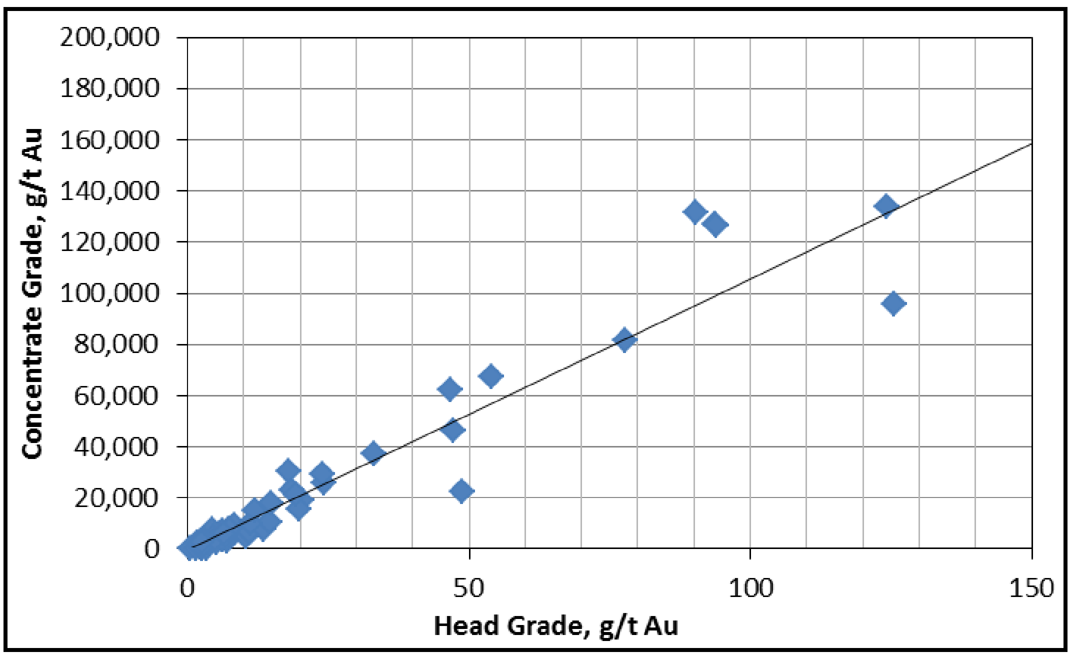
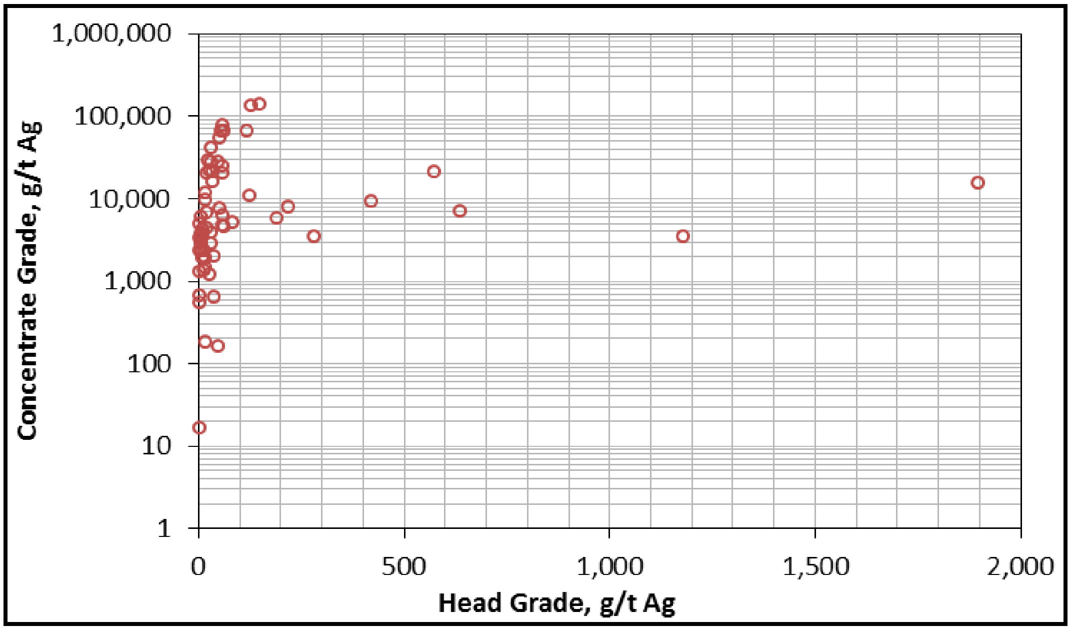


Figure 13.26 Gravity Concentrate Grade versus Head Grade – Silver (2012)



Variability Test Work (2010 to 2011)

In 2011, Inspectorate conducted seven variability tests on various samples, which included three samples from the West Zone and four samples from the Galena Hill Zone. The tests studied the metallurgical responses of these samples to Flowsheet C, developed from the composite samples. The test results are summarized in Table 13.30.

Table 13.30 Variability Test Results (2010 to 2011)

	Grade (g/t)		Recovery/Extraction (%)		
	Au	Ag	Au	Ag	Grind Size
GF26/Composite GH2 – Head	4.9	52.9	100.0	100.0	Primary Grind Size: P_{80} 125 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	1,808	183	36.4	0.32	
Flotation Concentrate	16.5	302	62.0	98.6	
Secondary Gravity Concentrate	1,116	2,650	51.5	8.1	
Intensive Cyanide Leaching	-	-	93.6	95.6	
Cyanide Leaching	-	-	61.3	64	
Overall Recovery	-	-	91.0	71	
GF27/Composite SU98– Head	73.3	205	100.0	100.0	Primary Grind Size: P_{80} 123 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	11-959	186	33.2	0.3	
Flotation Concentrate	214	556	66.0	99.1	
Secondary Gravity Concentrate	13-281	11,323	79.1	27.7	
Intensive Cyanide Leaching	-	-	95.3	95.8	
Cyanide Leaching	-	-	97.2	66.9	
Overall Recovery	-	-	99.0	79.0	

table continues...

	Grade (g/t)		Recovery/Extraction (%)		Grind Size
	Au	Ag	Au	Ag	
GF25/Composite WZ 1 – Head	1.8	25.4	100.0	100.0	Primary Grind Size: P_{80} 120 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	1,151	194	26.4	0.44	
Flotation Concentrate	9.1	128	70.9	97.5	
Secondary Gravity Concentrate	646	1,600	50.4	9.1	
Intensive Cyanide Leaching	-	-	94.0	96.5	
Cyanide Leaching	-	-	64.3	66.9	
Overall Recovery	-	-	89.0	73.0	
GF32/Composite SU 33/GH – Head	3.68	22.1	100.0	100.0	Primary Grind Size: P_{80} 125 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	751	201	27.0	1.7	
Flotation Concentrate	9.5	50.4	71.1	91	
Secondary Gravity Concentrate	690	818	53.0	10.0	
Intensive Cyanide Leaching	-	-	87.8	83.0	
Cyanide Leaching	-	-	74.6	78.4	
Overall Recovery	-	-	92.0	77.0	
GF30/Composite SU-32C/WZ – Head	11	10.4	100.0	100.0	Primary Grind Size: P_{80} 165 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	6,006	201	58.0	4.2	
Flotation Concentrate	38.0	37.6	41.2	89.3	
Secondary Gravity Concentrate	678	1,133	22.2	21.8	
Intensive Cyanide Leaching	-	-	96.8	90.9	
Cyanide Leaching	-	-	91.2	68.8	
Overall Recovery	-	-	97.7	79.1	
GF31/Composite SU-32A/WZ – Head	3.8	25.8	100.0	100.0	Primary Grind Size: P_{80} 161 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	592.6	203	35.9	2.3	
Flotation Concentrate	8.6	64	61.1	84.4	
Secondary Gravity Concentrate	4,142	2,958	83.4	23.9	
Intensive Cyanide Leaching	-	-	89.0	86.0	
Cyanide Leaching	-	-	85.0	71.0	
Overall Recovery	-	-	96.0	71.0	
GF28/Composite SU-76B/GH – Head	12.6	130	100.0	100.0	Primary Grind Size: P_{80} 116 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	3617	196	49.9	0.3	
Flotation Concentrate	22.8	374	48.8	94.6	
Secondary Gravity Concentrate	1,893	5,301	66.5	11.3	
Intensive Cyanide Leaching	-	-	95.0	96.0	
Cyanide Leaching	-	-	81.0	67.0	
Overall Recovery	-	-	97.0	80.0	

The results from these variability tests indicated:

- There was no significant variation in metallurgical performance between the West Zone and Galena Hill Zone mineralization.
- In general, the samples tested were amenable to the combined procedure consisting of gravity separation, flotation, and cyanide leaching. The overall average gold recovery was 94.5%, which was approximately 19% higher than the average silver recovery.
- The overall gold recovery increased, with an increase in gold head grade. It appears that the overall silver recovery variation with head grade was less significant, although silver head grade ranged widely from 10 to 205 g/t.
- The regrind size was finer than 80% passing 10 µm.

The results from the samples tested using Flowsheet B and Flowsheet C are plotted in Figure 13.27 for gold and Figure 13.28 for silver.

Figure 13.27 Variability Test Results – Gold Metallurgical Performance (2010-2011)

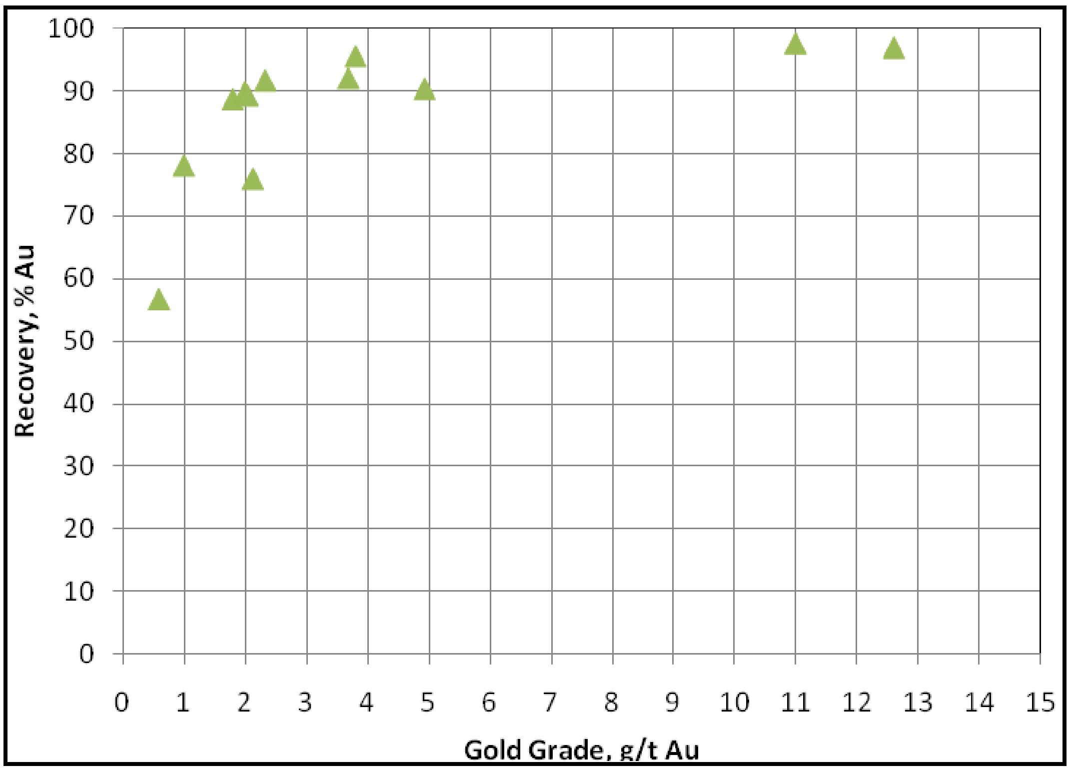
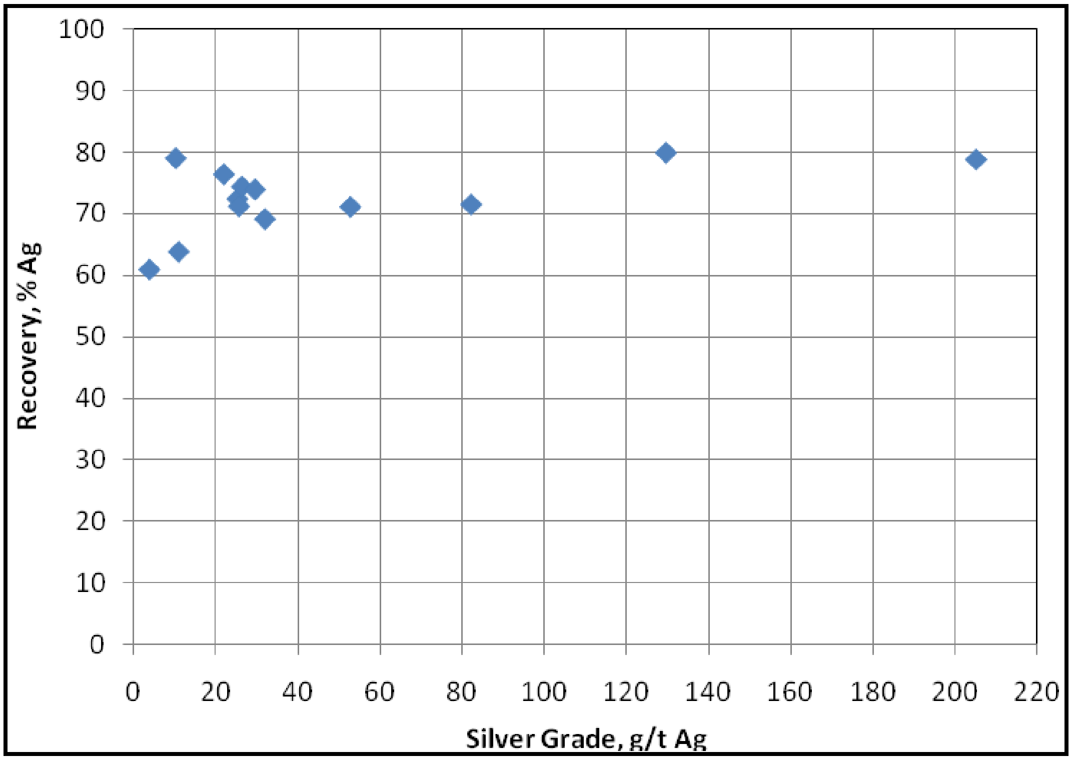


Figure 13.28 Variability Test Results – Silver Metallurgical Performance (2010-2011)



13.3.9 Locked Cycle Test (2012 and 2013)

In order to better examine recovery and concentrate grade, four locked cycle tests were completed in 2012 on two master composites, which consisted of one blend from the VOK Zone (VOK-1, -2, -3, -4) and one from the West Zone (WZ-1, -2). The procedure included:

- primary grinding targeting a moderate size of 80% passing 80 to 85 µm
- gravity concentration
- rougher and scavenger flotation with the scavenger concentrate recycled
- rougher concentrate cleaner flotation.

For tests FLC1 and FLC3, the rougher concentrates were reground prior to cleaner flotation. In an effort to activate gold- and silver-bearing minerals, copper sulphate was added during the rougher and cleaner flotation stages.

In 2013, two separate locked cycle tests were conducted on two composites generated from the upper and lower zones of the VOK deposit. The test procedure used was similar to that used for the locked cycle tests in 2012.

The results of all the six locked cycle tests are shown in Table 13.31 and summarized as follows:

- The average metal recoveries from the VOK Zone composites were approximately 97.8% for gold and 94.3% for silver. Approximately 53.9% of the gold and 28.6% of the silver reported to the gravity separation concentrate. The flotation concentrate contained approximately 130 g/t gold, 252 g/t silver, and 0.68% arsenic.
- Average recoveries from the master composite of the West Zone were approximately 94.0% for gold and 90.8% for silver. Approximately one-third of the gold reported to the gravity separation concentrate. The flotation concentrate contained 48.6 g/t gold, 2,800 g/t silver, and 0.24% arsenic.
- The addition of copper sulphate, together with regrinding of the rougher flotation concentrates, did not appear to improve the recoveries of the target metals.

A review of the locked cycle test procedures and results indicated that the flowsheet tested was robust for the mineralization.

Table 13.31 Locked Cycle Tests Results

Composite	Test No.	Head Grade Calculated			Gravity Concentration				Flotation						
					Recovery		Concentrate Grade		Concentrate Grade				Recovery	
		Au (g/t)	Ag (g/t)	S (%)	Au (%)	Ag (%)	Au (kg/t)	Ag (kg/t)	Au (g/t)	Ag (g/t)	S (%)	As (ppm)	Au (%)	Ag (%)
VOK-1 to -4	FLC1	24.2	33.6	2.92	54.2	30.5	11.7	9.1	181.3	354	48.1	8,249	43.9	61.7
VOK-1 to -4	FLC2	24.2	31.8	2.96	48.6	27.1	9.9	7.9	175.6	341	46.9	6,930	49.3	67.0
WZ-1 and -2	FLC3	6.0	225	3.03	32.0	1.3	1.7	2.7	52.6	3,096	43.5	2,622	59.2	88.5
WZ-1 and -2	FLC4	6.3	240	3.10	36.5	1.1	2.5	2.8	44.6	2,490	34.7	2,228	60.2	90.7
VOK ML	FLC2	10.3	12.5	3.41	48.0	21.6	4.3	2.4	83.8	152	52.2	5,801	48.5	71.7
VOK MU	FLC1	12.1	13.4	2.70	64.9	35.1	6.0	3.6	78.1	160	49.5	6,059	33.9	62.4

13.3.10 Solids Liquid Separation Tests Work

In 2012, Pocock Industrial conducted solids liquid separation (SLS) tests on the flotation concentrate and flotation tailings samples. The test program included sample particle size analysis, flocculants screening and evaluation, static and dynamic thickening tests, pulp rheology, and filtration studies.

The results of the thickening tests are summarized in Table 13.32, Table 13.33, and Figure 13.29. To enable the best performance for both flotation concentrate and flotation tailings, Hychem AF 304 was selected as the flocculant, which is a medium to high molecular weight anionic polyacrylamide, with a 15% charge density.

A concentration of approximately 20% is recommended for the flotation concentrate thickener feed solids. Using a conventional thickener under the recommended conditions, the unit area for the concentrate is in the range of 0.19 to 0.24 $m^2/t/d$, while the underflow solids concentration is from 68 to 72%.

Pocock recommended a concentration of approximately 10% for the tailings thickener feed solids. Using a conventional thickener under the recommended conditions, the unit area for tailings is within the range of 0.43 to 0.48 $m^2/t/d$, while the underflow solids concentration is in the range of 61 to 65%.

The recommended hydraulic rate is 5.2 to 6 $m^3/m^2/h$ for a high rate concentrate thickener, and 4.2 to 5.0 $m^3/m^2/h$ for a high rate tailings thickener.

The maximum design thickener underflow density is suggested to 72% for the concentrate and 65% for the tailings.

Table 13.32 Conventional Thickening Test Results for Flotation Concentrate

Feed Solids (%)	CCD Stage Simulated	Flocculant		Rise Rate ($m^3/m^2/h$)	Maximum Test		Solids Density (%)			Unit Area ($m^2/t/d$)		
		Dose (g/t)	Concentration (g/L)		Density (%)	Time (min)						
20	Stage 1	30.0	0.1	6.31	75.7	120	68	70	72	0.21	0.22	0.24
20	Stage 1	35.0	0.1	6.83	76.3	120	68	70	72	0.19	0.20	0.21
20	Stage 2	30.0	0.1	6.49	75.1	120	68	70	72	0.19	0.21	0.22
20	Stage 3	25.0	0.1	6.35	74.4	120	68	70	72	0.19	0.20	0.21
20	Stage 1	40.0	0.1	5.26	75.7	150	68	70	72	0.18	0.20	0.21
25	Stage 1	35.0	0.1	2.28	76.9	120	68	70	72	0.20	0.21	0.23
25	Stage 2	30.0	0.1	2.10	76.4	120	68	70	72	0.21	0.23	0.24
25	Stage 3	25.0	0.1	2.27	74.4	120	68	70	72	0.20	0.22	0.23
30	Stage 1	35.0	0.1	1.37	77.3	180	68	70	72	0.32	0.35	0.38
30	Stage 2	30.0	0.1	1.26	75.7	180	68	70	72	0.31	0.34	0.37
30	Stage 3	25.0	0.1	1.20	76.5	180	68	70	72	0.32	0.35	0.38

Notes: Flocculant (Hychem AF 304) is a medium to high molecular weight anionic polyacrylamide, with 15% charge density.
CCD = counter current decantation.

Table 13.33 Conventional Thickening Test Results for Flotation Tailings

Feed Solids (%)	Flocculant Dose (g/t)	Flocculant Concentration (g/L)	Rise Rate ($m^3/m^2/h$)	Maximum Test Density (%)	Maximum Test Time (min)	Solids Density (%)			Unit Area ($m^2/t/d$)		
7.5	30.0	0.1	9.71	63.3	150	61	63	65	0.50	0.52	0.53
7.5	35.0	0.1	10.07	66.0	150	61	63	65	0.40	0.42	0.43
7.5	40.0	0.1	9.69	66.0	150	61	63	65	0.41	0.42	0.44
10.0	35.0	0.1	8.14	64.9	120	61	63	65	0.41	0.43	0.45
15.0	30.0	0.1	4.96	65.2	150	61	63	65	0.43	0.45	0.47
15.0	35.0	0.1	4.44	65.2	120	61	63	65	0.42	0.43	0.45
15.0	40.0	0.1	3.39	62.8	150	61	63	65	0.43	0.45	0.47
15.0	45.0	0.1	3.10	62.8	150	61	63	65	0.42	0.44	0.46
20.0	35.0	0.1	0.84	65.2	180	61	63	65	0.72	0.76	0.80

Figure 13.29 Flotation Concentrate CCD Wash Test Results

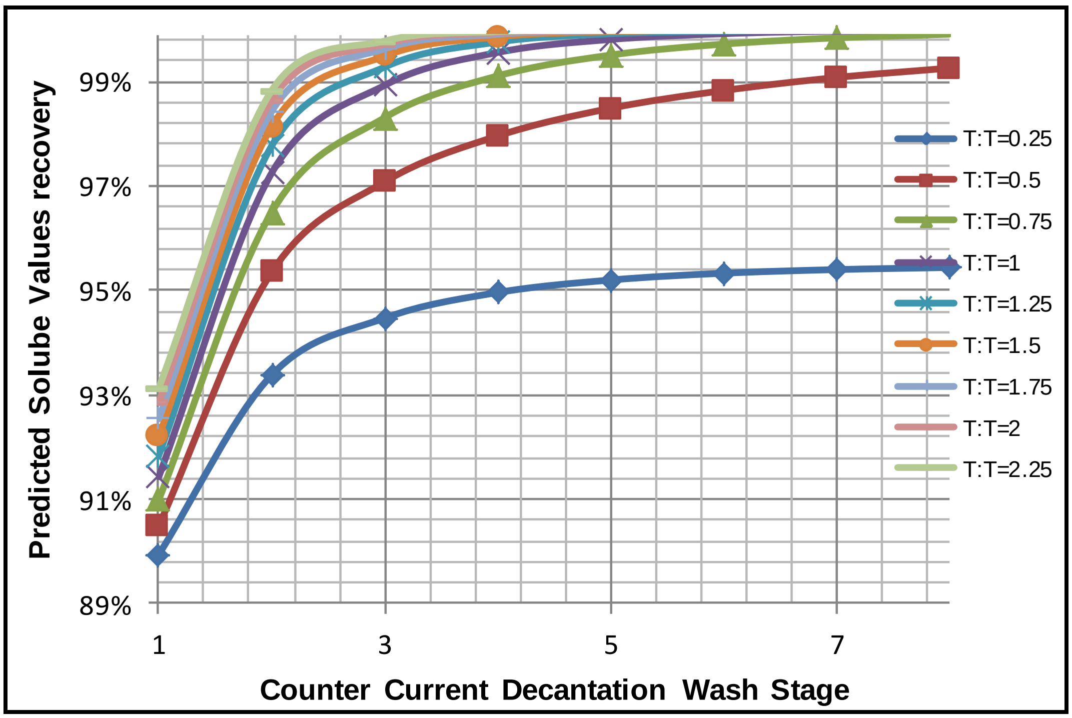


Table 13.34 Recommended Thickening Design Parameters

Material	pH	Flocculant Dose (g/t)	Thickener Feed Solids (%)	Underflow Density (%)	Unit Area for Conventional Thickener Sizing (m^2/t/d)	Hydraulic Rate for High Rate Thickener Sizing (m^3/m^2/h)	Thickener Type Recommended
Flotation Concentrate	10.78	35 to 45	20	68 to 72	0.190 to 0.225	5.2 to 6.0	High-Rate or High Density
Flotation Tailings	10.38	35 to 45	10	61 to 65	0.430 to 0.475	5.0 to 5.9	High-Rate or High Density
Flotation Tailings	8.34	90 to 100	10	61 to 65	0.430 to 0.475	4.2 to 5.0	High-Rate or High Density

The production rates indicates that vacuum filtration would likely be an uneconomical option for concentrate and tailings dewatering. A horizontal recess plate filter is recommended; the plate filter cake moisture content (Table 13.35) may be decreased by choosing a membrane-type filter press to get a high-pressure cake.

Table 13.35 Filtration Test Results and Sizing Summary

Filter Type	Material	Filter Feed Solids (%)	Bulk Cake Density (kg/m^3)	Filter Sizing Basis (m^3/t)	Filter Cake Moisture (%)	Filter Cycle Time (min)
Horizontal Recess Plate	Flotation Concentrate*	68.3	2155.9	0.580	12.6	18.1
	Flotation Tailings	58.0	824.7	1.516	17.6	19.4
Filter Type	**Material**	**Filter Feed Solids (%)**	**Bulk Cake Density (kg/m^3)**	**Filter Production Rate (kg/m^2/h)**	**Filter Cake Moisture (%)**	**Filter Aid (g/t)**
Horizontal Belt Vacuum	Flotation Concentrate	59.5	1723.3	116.8	29.3	200 (Hychem AF304)
	Flotation Concentrate		2047.8	49.7	24.0	None
	Flotation Tailings	59.0	1460.0	546.2	29.8	320 (Hychem AF340)
	Flotation Tailings		1757.9	124.0	24.4	None

Note: *With blowing and membrane squeezing.

13.3.11 CONCLUSIONS

A review of preliminary test work on samples from the Brucejack deposit has led to the following conclusions:

- The Brucejack mineralization is moderately hard.
- The test results suggest that the mineralization is amenable to the gravity/flotation combined recovery process. The recovery flowsheet should include:
  - gravity concentration to recover coarse free gold and silver
  - flotation to produce rougher and scavenger concentrates
  - regrinding on the rougher and scavenger concentrates
  - gravity concentration to recover fine free gold and silver
  - cleaner flotation to produce a final concentrate for sale
  - smelting on gravity concentrate to produce gold/silver doré.
- The test results indicate that there is a significant variation in the metallurgical performance of mineral samples taken from different parts of the deposit.
- The test results show that flotation concentrate responds reasonably well to direct leaching using cyanide, excluding a few of samples containing higher graphite (carbon), arsenic, or electrum content. The cyanide leaching conditions have not yet been optimized.

  Further improvements should be evaluated for gold and silver recoveries, including oxidation pre-treatment.
- The gravity concentrates responded very well to intensive leach by cyanide.

13.3.12 RECOMMENDATIONS

Further test work is recommended to:

- Conduct large scale pilot plant tests to confirm the findings of the test work completed to date.
- Further investigate the response of the sample material to the cyanide leach process.

13.4 METALLURGICAL PERFORMANCE PROJECTION

The proposed flowsheet used for the current study will include a gravity/smelting and flotation process, to produce gold-silver doré and a flotation concentrate containing gold and silver. The metallurgical performance of the Brucejack mineralization has been projected according to the results generated from the locked cycle and variability tests.

There is significant variation in metallurgical performance, which is possibly due to nugget effects; therefore, the projections were based on average data, summarized as follows:

- The gold and silver recoveries reporting to gold-silver doré were projected based on the gravity concentration test results, GRG gravity circuit simulations, and experience. It was assumed that the smelting recovery is 99.5%. The slag from smelting will be ground and tabled to recover gold-silver alloy grains, and the table tailings will be blended with the flotation concentrate.
- The gold and silver recoveries reporting to the gold-silver flotation concentrate were estimated using the average locked cycle test results achieved at a primary grind size of 80% passing approximately 75 µm, and the regression equations that were derived from the plots of the variability test results.
- The flotation concentrate grade was estimated by the sulphur grade of the concentrates produced from the locked cycle tests.

For materials with head grades beyond the range of the tested heads, the metal recovery and concentrate grade estimates were assumed based on test results and experience. The projections are detailed in Table 13.36 for the VOK Zone mineralization and Table 13.37 for the West Zone mineralization.

Table 13.36 Metallurgical Performance Projection – VOK Zone

Head Grade (g/t)	Gold and Silver Recovery (%)
Doré - Gold	
< 0.5	= 0
0.5 to 9.99	= - 0.147*((Head Grade, g/t) ^{2}) + 5.68*(Head Grade, g/t) - 1.214
9.99 to 40	= 7.32*ln(Head Grade, g/t) + 24.012
> 40	= 52
Doré - Silver	
<3.0	= 23
3.0 to 130	= (34.643*(Head Grade, g/t) $^{-0.48}$)
130 to 400	= 1.5
> 400	= 1.0
Flotation Concentration - Gold	
< 0.5	= 30
0.5 to 5.64	= 85.44*(Head Grade, g/t) $^{0.056}$ + 0.147*(Head Grade, g/t) ^{2} - 5.68*(Head Grade, g/t) + 2.6
5.64 to 9.99	= 1.18*ln(Head Grade, g/t) + 0.147*((Head Grade, g/t) ^{2}) - 5.68*(Head Grade, g/t) + 94.694
9.99 to 40	= - 6.14*ln(Head Grade, g/t) + 69.558
> 40	= 46.5
Flotation Concentration - Silver	
< 3.0	= 35
3.0 to 130	= 2.793*ln(Head Grade, g/t) - (34.643*((Head Grade, g/t) $^{-0.48}$)) + 78.07
130 to 400	= 90.5
> 400	= 91.5

Table 13.37 Metallurgical Performance Projection – West Zone

Head Grade (g/t)	Gold and Silver Recovery (%)
Doré - Gold	
< 0.5	= 0
0.5 to 9.99	= - 0.147*((Head Grade, g/t) ^{2}) + 5.68*(Head Grade, g/t) - 1.214
9.99 to 40	= 7.32*ln(Head Grade, g/t) + 24.012
> 40	= 52
Doré - Silver	
<3.0	= 23
130	= (34.643*(Head Grade, g/t)$^{-0.48}$)
130 to 500	= 1.5
> 500	= 1.0
Flotation Concentration - Gold	
< 0.5	= 30
0.5 to 5.64	= 85.44*(Head Grade, g/t)$^{0.056}$ + 0.147*(Head Grade, g/t)^{2} - 5.68*(Head Grade, g/t) + 2.
5.64 to 9.99	= 1.18*ln(Head Grade, g/t) + 0.147*((Head Grade, g/t)^{2}) - 5.68*(Head Grade, g/t) +94.094
9.99 to 40	= - 6.14*ln(Head Grade, g/t) + 68.958
> 40	= 45.9
Flotation Concentration - Silver	
< 3.0	= 35
3.0 to 130	= 2.9741*ln(Head Grade, g/t) - (34.643*((Head Grade, g/t)$^{-0.48}$)) + 73.956
130 to 500	= 2.9741*ln(Head Grade, g/t) + 72.456
> 500	= 91.5

14.0 MINERAL RESOURCE ESTIMATES

Snowden previously completed mineral resource estimates for different areas of the Project. Initially (September 2012), these estimates were aimed at evaluating the broader lower-grade mineralized areas including Bridge Zone, Gossan Hill, and Shore Zone (Olssen and Jones 2012b). No estimates have been prepared for these zones to consider the potential for underground extraction.

More recent estimates, also completed by Snowden, were focused on evaluating the higher-grade areas. These estimates were reported at a high-grade cut-off for potential underground extraction. They include:

- a mineral resource estimate for West Zone (Olssen and Jones 2012a)
- a mineral resource estimate for the VOK mineralized area (Olssen and Jones 2012c).

14.1 DISCLOSURE

The mineral resources were prepared by Snowden under the supervision of QP Ivor Jones, Executive Consultant and an employee of Snowden Mining Industry Consultants Inc. (Olssen and Jones, 2012c). The effective date of the resource estimate was November 20, 2012.

The author, by way of experience and qualifications, is a QP as defined by NI 43-101 and is independent of Pretivm.

14.2 KNOWN ISSUES THAT MATERIALLY AFFECT MINERAL RESOURCES

The author of this section is not aware of any permitting, legal, title, taxation, socio-economic, and marketing or political issues that could materially affect the mineral resource estimates.

14.3 ASSUMPTIONS, METHODS AND PARAMETERS

The estimates were prepared in the following steps:

- data validation
- data preparation (this and subsequent steps are summarized below)
- exploratory data analysis

- geological interpretation and modelling
- establishment of block models
- compositing of assay intervals
- consideration of grade outliers
- variogram analysis
- derivation of kriging plan
- grade value estimation
- grade estimate validation
- deduction for prior mined volume
- classification of estimates with respect to the CIM definition standards
- resource tabulation and resource reporting.

14.4 DATA PROVIDED

The drillhole database used by Snowden for the resource estimate was provided by Caroline Vallat from GeoSpark. This database is in Microsoft Access® format and contains collar, survey, assay, and specific gravity data.

Digital terrain models (DTMs) for the topographic elevation and the base of the ice cap were provided by Pretivm, together with solids for the lithological domains and mineralized corridors in VOK.

For the resource estimate, Snowden used all drillholes with collars lying inside of the area covered by the topography; which comprises a rectangular area with coordinates 425450 mE to 427550 mE and 6256450 mN to 6260550 mN (Figure 14.1). The data used in the estimates exclude intervals with no gold and silver values.

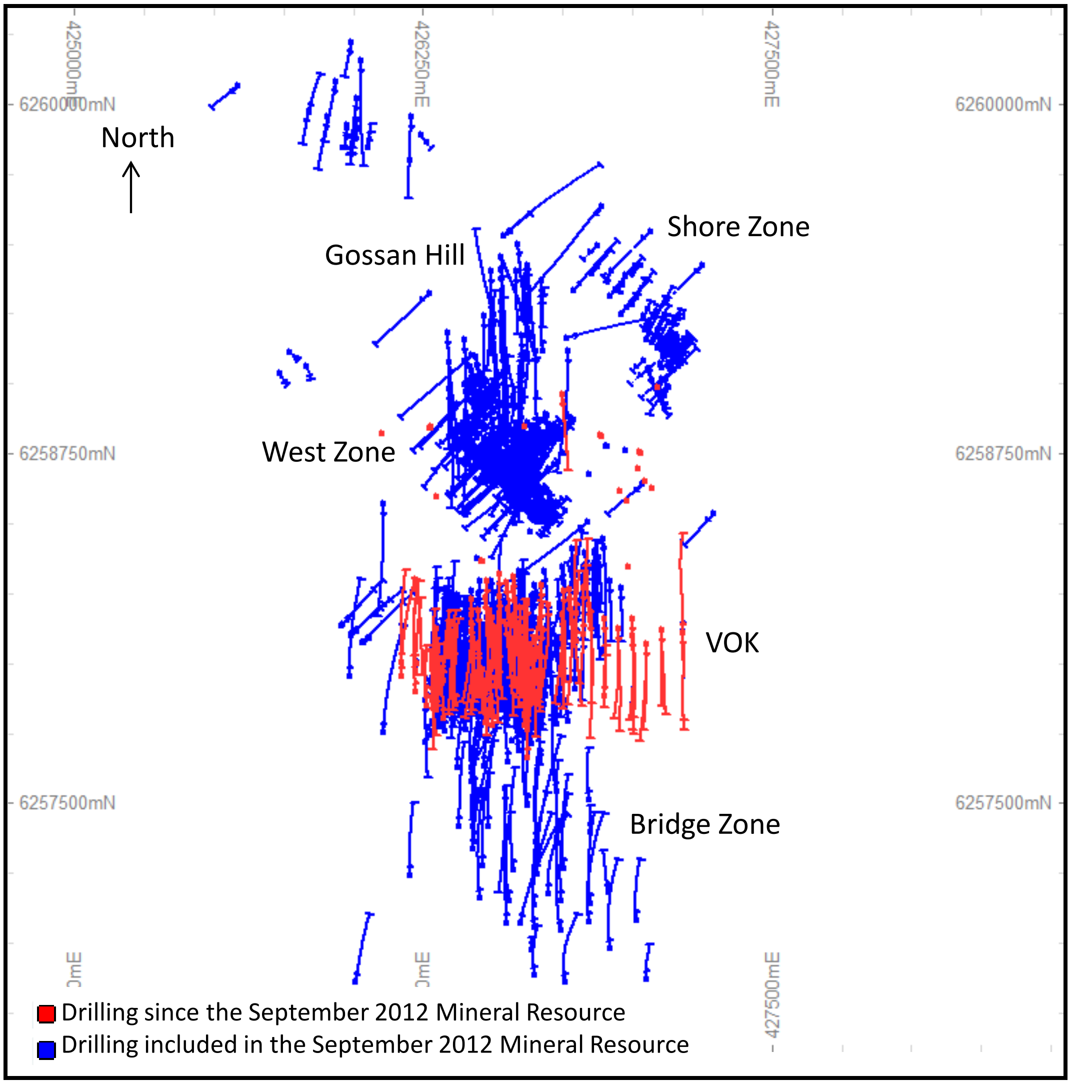
The total input data comprises 1,417 drillholes including 439 underground drillholes in West Zone (24,704 m), 400 historical surface drillholes (35,745 m), and 578 surface drillholes completed since 2009 (224,402 m).

The input data for the VOK estimate (between 6257520 mN and 6258440 mN) comprises 490 drillholes for 174,182 m.

The input data for the West Zone estimate comprises 756 drillholes (62,208 m) including 439 underground drillholes (24,688 m), 269 historical surface drillholes (21,321 m) and 48 surface drillholes (17,199 m) completed since 2009.

The sample database and the topographic surface were reviewed and validated by Pretivm prior to being supplied to Snowden. Snowden carried out basic validation checks and found no material issues with the database supplied.

Figure 14.1 Plan Showing the Distribution of Exploration Drillholes



14.5 GEOLOGICAL INTERPRETATION AND MODELLING

14.5.1 VOK

At VOK, lithological interpretations were used together with a nominal 1 g/t to 3 g/t Au cut-off grade to define high-grade corridors based on analysis of the statistical grade populations.

High-grade gold mineralization at VOK occurs in a series of west-northwest trending sub-vertical corridors of structurally re-oriented vein stockworks and subordinate vein breccias, hosted in a folded sequence of fine grained volcanic siltstone and sandstone, variably silicified polylithic volcanic conglomerate, fragmental latite volcanic rocks, bounded by relatively massive porphyritic latite flows. Gold mineralization also appears

to form a series of seemingly stratigraphy-parallel pods within the steeply-dipping polylithic conglomerate, and immediately overlying fragmental volcanic rocks in the core of the syncline. These pods appear to be spatially associated with intensely silicified zones that appear to have acted as pressure seals, which generated local overpressure conditions and subsequent depressurization by hydraulic and/or tectonic brecciation.

While there is some indication of a hard boundary between the polylithic conglomerate, and underlying fine-grained volcano-sedimentary and porphyritic latite flow volcanic rocks (previously termed "Jurassic" and "Triassic", respectively) in places, the mineralization crosses this contact. Gold and silver mineralization statistics for the volcanic and volcano-sedimentary rock units present in VOK are very similar, suggesting (along with visual assessment) that they can be treated as part of the same population for estimation. The boundary between the mineralized domains in the polylithic conglomerate and underlying volcano-sedimentary and porphyritic flow volcanic rocks is a hard contact in places due to the unconformity on the boundary with a generally weakly-mineralized siliceous unit along most of the contact. There are places where this siliceous unit does not exist and the boundary is gradational, with the sub-vertical mineralized corridors crossing the contact (Figure 14.2). In these places a soft boundary approach was used for grade estimation.

Recent geological interpretation has shown that the area previously covered by Galena Hill is actually an extension of VOK, and this area has now been incorporated into VOK.

14.5.2 West Zone

West Zone was interpreted for the April 2012 estimate using a nominal 0.3 g/t Au cut-off grade (Olssen and Jones, 2012a). There are no high-grade corridors defined at West Zone.

14.5.3 Domains Used for Modelling

After review of the grade distributions for the mineralized corridors and different rock types at VOK, Snowden chose to evaluate/model each mineralized corridor using hard boundaries, but combining the various lithologies within the mineralized corridor. There were some exceptions where the corridors were interpreted to cross the lithological boundaries, and these contacts were treated as soft.

The surrounding low-grade (background) domain was estimated as a single domain using soft boundaries, but excluding the high-grade population from within the high-grade domains.

The mineralized corridors within VOK change orientation locally. This was addressed by using a series of "search domains" with locally adjusted orientations applied for estimation. The boundaries between these search domains were treated as soft for estimation.

West Zone was estimated using a single mineralized domain.

Figure 14.2 Cross-section Showing Lithological Interpretation and Mineralized Domain Interpretation at VOK

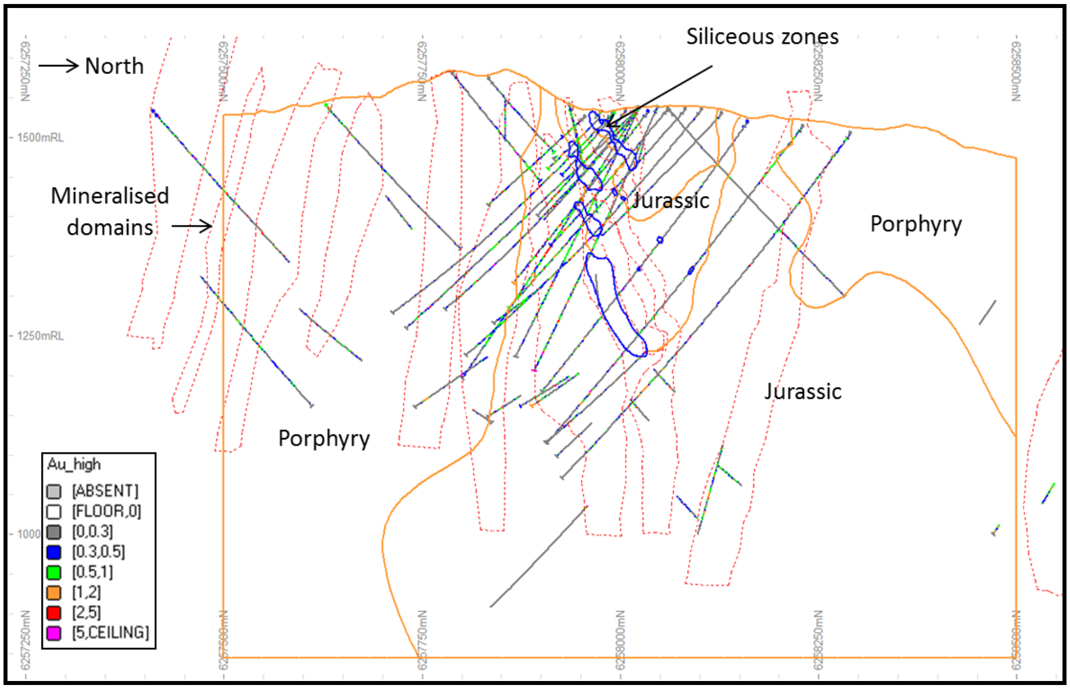
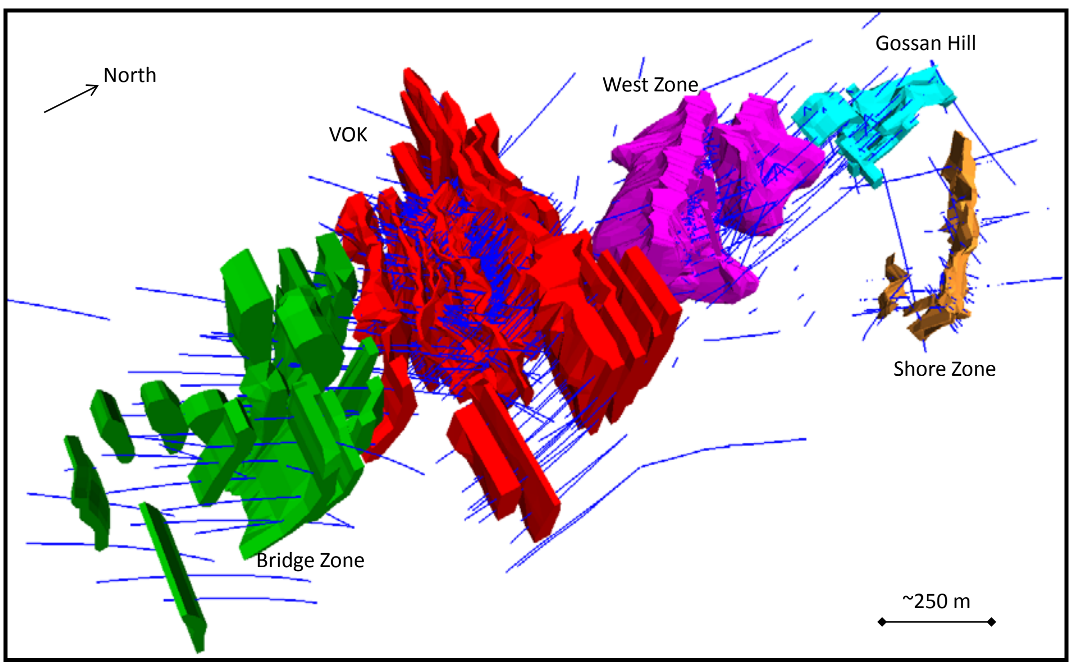


Figure 14.3 Orthogonal View of Mineralized High-grade Domain Interpretation



14.6 COMPOSITING OF ASSAY INTERVALS

All data was composited to the dominant sample length of 1.5 m prior to analysis and estimation.

14.6.1 SUMMARY STATISTICS

Statistical analysis of the gold and silver data was carried out by lithological domain (at VOK) and mineralized domain (high grade and low grade).

Review of the statistics for VOK indicated that the grade distributions for the mineralization within the different lithologies are very similar, and as a result these were combined for analysis.

Initial review of the individual high-grade corridors within each area showed that they have similar statistical distributions, therefore the individual corridors within each area were combined for statistical analysis and variography.

All high-grade domains, including the West Zone domain, exhibit a strong positive skewness with high coefficient of variation and extreme grades. The low-grade domains also show positively skewed distributions but have lower coefficients of variation and few extreme grades.

Table 14.1 summarizes the statistics for gold and silver for the mineralized domains. Due to clustering of the drilling, the data for West Zone has been declustered for statistical analysis.

Table 14.1 Summary Statistics of Composited Data for Mineralized Domains – VOK

Statistic	Gold g/t	Silver g/t
Samples	57,895	57,895
Minimum	0.00	0.02
Maximum	14,043.71	9,383.44
Mean	3.39	8.85
Standard Deviation	93.63	56.32
Coefficient of Variation	27.61	6.36
Variance	8,767.00	3,172.00
Skewness	86.95	93.40

Table 14.2 Summary Statistics of Composited Data for Mineralized Domains – West Zone

Statistic	Gold g/t	Silver g/t
Samples	33,089	33,089
Minimum	0.00	0.25
Maximum	1,657	37,636
Mean	1.40	30.66
Standard Deviation	17	219
Coefficient of Variation	11.84	7.15
Variance	275	48,020
Skewness	66.98	62.65

14.6.2 Extreme Values – Gold and Silver

Between 2 and 5% of the data within the mineralized high-grade domains appear to lie in a separate higher-grade population, which contains significant extreme grades. The treatment of these extreme grades is discussed in Section 14.7.

14.7 Consideration of Grade Outliers and Estimation Method

Assay populations from gold deposits are generally skewed and contain high-grade outliers that can introduce bias to mineral resource estimates.

Both West Zone and VOK exhibit extremely skewed grade populations where the high grades and the majority of the metal are located in less than 5% of the data, with individual raw gold grades of up to around 41,500 g/t Au. As a result of this population distribution, standard estimation techniques have been found to significantly over smooth the grades.

Discussions with Pretivm and analysis of the data indicated that the mineralization could be split into a pervasive background mineralization and a separate high-grade but discreet mineralization style. In order to model this style of mineralization without smearing grade, Snowden separated the lower-grade "background" population from the higher-grade population and estimated them independently. For gold in the West Zone and VOK, a threshold of 5 g/t Au was selected to separate the two populations, based on review of the population statistics and graphs. The silver data was treated using the same method with a threshold of 50 g/t Ag for VOK and 300 g/t Ag for the West Zone (Figure 14.4).

The relatively low skewness and the presence of only few samples with extreme grades in the low-grade domains allowed for the estimation of grades using ordinary kriging with a top cut. A top cut of 4 g/t Au and 100 g/t Ag was selected for VOK, based on the point of disintegration seen on the histogram and log probability plot. A top cut of 3 g/t Au and 100 g/t Ag was selected for West Zone.

In the high-grade domains (including West Zone)–with the separation of the lower- and higher-grade populations–the lower-grade population is amenable to the ordinary kriging method of grade interpolation.

Multiple indicator kriging was selected for estimation of the higher-grade population within the high-grade domains, to control the skewness of the data. Multiple indicator kriging involves modelling variograms at a series of grade thresholds, which allows the range of continuity to be reduced at the higher grades. A mathematical model was then used to define the top end of the grade distribution. The result of this estimation method is that, while no top cut is used to limit the higher grades, the higher grades are limited in their influence using a mathematical model based on the higher-grade data to estimate grades in the top class. This is based on probability rather than using the individual extreme grades in the dataset for grade estimation.

Figure 14.4 Log Probability Plot Showing the Threshold Between Lower- and Higher-grade VOK Population

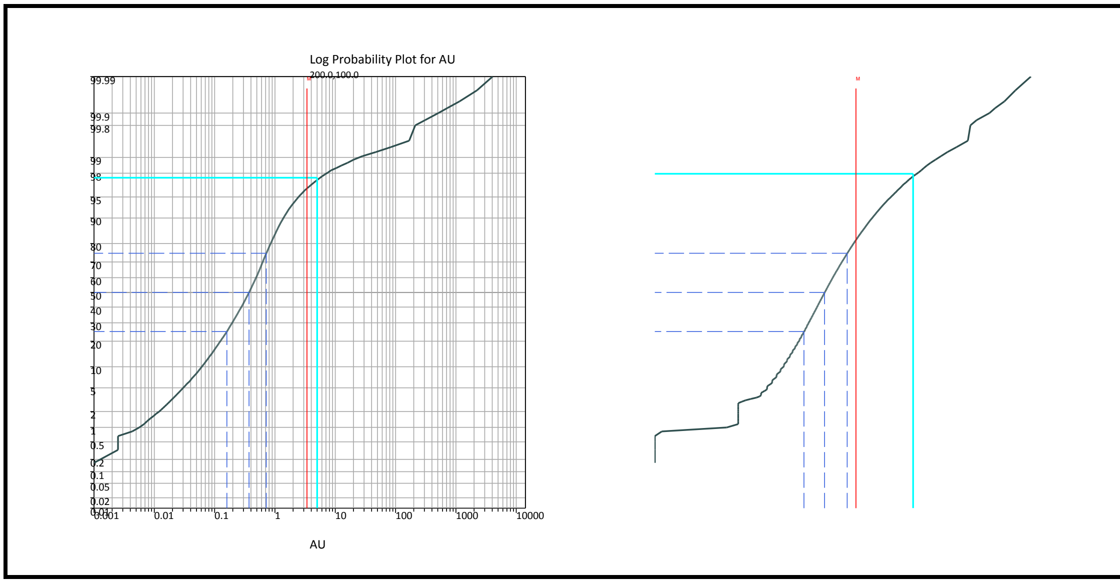
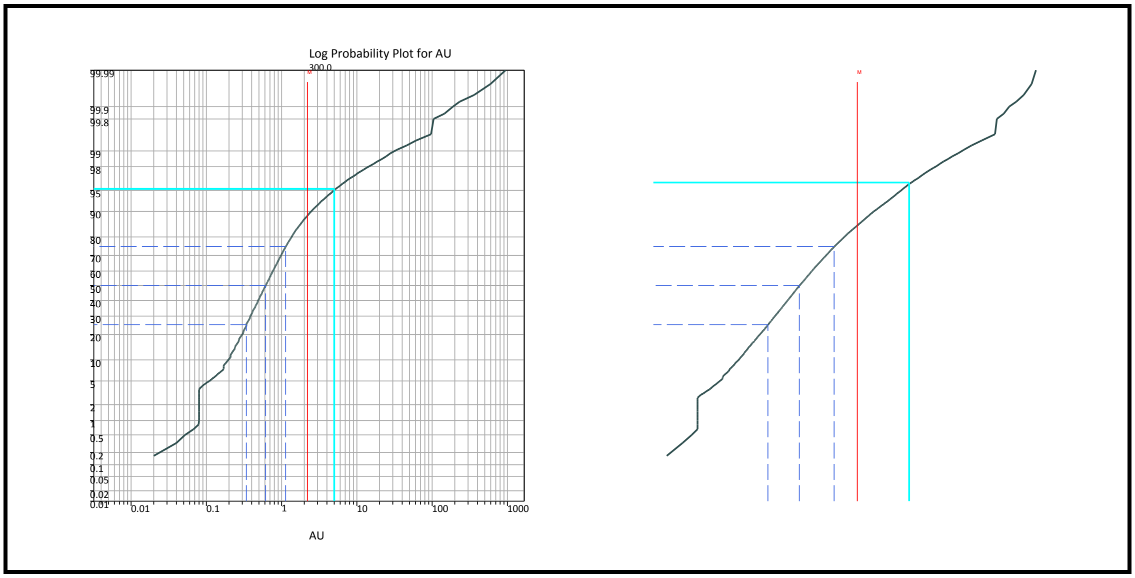


Figure 14.5 Log Probability Plot Showing the Threshold Between Lower- and Higher-grade Populations (West Zone)



14.8 Variogram Analysis

14.8.1 High-grade Domains - Low-grade Population

Due to the positively skewed nature of the grade distributions, normal scores experimental variograms were modelled for gold and silver for the estimates of the lower-grade population within the high-grade domains.

For the low-grade population in VOK and West Zone, variograms were calculated and modelled using only data below the thresholds of 5 g/t Au and 50 g/t Ag. Due to the locally changing orientations in the VOK area, the orientations of the variogram models were locally adjusted to reflect the interpretation.

The normal scores models were back-transformed prior to estimation.

Table 14.3 Parameters to Describe Gold Grade Continuity for the Low-grade Population Estimates Within the High-grade Domains

Area	Grade	Orientation	Nugget	Structure 1		Structure 2		Structure 3	
				Sill	Range	Sill	Range	Sill	Range
VOK	Gold	00→090	-	-	10	-	60	-	-
		-90→000	0.17	0.52	10	0.31	100	-	-
		00→000	-	-	10	-	50	-	-
	Silver	00→090	-	-	13	-	30	-	250
		-90→000	0.14	0.41	18	0.19	60	0.26	250
		00→000	-	-	13	-	40	-	140
West Zone	Gold	00→120	-	-	20	-	194	-	-
		-80→030	0.21	0.63	25	0.16	341	-	-
		10→030	-	-	7	-	35	-	-
	Silver	00→120	-	-	17	-	248	-	-
		-80→030	0.15	0.46	21	0.39	294	-	-
		10→030	-	-	6	-	59	-	-

14.8.2 HIGH GRADE DOMAINS - HIGH-GRADE POPULATION

Indicator variograms for gold were calculated and modelled for the high-grade populations within the high-grade domains for VOK and West zones.

Given the small proportion of data above the high-grade population threshold grade, experimental variograms were poorly structured for VOK and West Zone. As a result, experimental variograms were modelled for the 50th percentile of the distribution and then adapted for the 10, 20, 30, 40, 60, 70, 80, 90, and 95th percentiles of the distribution (Table 14.4).

As with the low-grade population, due to the locally changing orientations in the VOK area, the orientations of the variogram models were locally adjusted to reflect the interpretation.

It was not possible to model variograms for silver in West Zone. Given the high correlation between gold and silver (approximately 0.9 correlation coefficient), the gold variograms were also used for the estimation of the silver high-grade population for West Zone.

The upper tail of the high-grade population distributions, above the 95 percentile, was modelled using a hyperbolic or power mathematical model for each area for gold and silver.

Table 14.4 Parameters to Describe Gold Grade Continuity for a Range of Indicators for the High-grade Population Estimates Within the High-grade Domains in VOK

Grade	Cut-off (percentile)	Orientation	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
Gold	10,20,30,40, 50,60,70	00→090	-	-	15	-	60
		-90→000	0.57	0.18	6	0.25	40
		00→000	-	-	4	-	15
	80,90	00→090	-	-	12	-	50
		-90→000	0.57	0.18	4	0.25	30
		00→000	-	-	2	-	12
	95	00→090	-	-	9	-	30
		-90→000	0.57	0.18	3	0.25	20
		00→000	-	-	2	-	9
Silver	10,20,30,40, 50,60,70	00→090	-	-	10	-	20
		-90→000	0.40	0.45	5	0.15	15
		00→000	-	-	5	-	10
	80,90	00→090	-	-	8	-	16
		-90→000	0.40	0.45	4	0.15	12
		00→000	-	-	4	-	8
	95	00→090	-	-	6	-	12
		-90→000	0.40	0.45	3	0.15	9
		00→000	-	-	3	-	6

Table 14.5 Parameters to Describe Gold Grade Continuity for a Range of Indicators for the High-grade Population Estimates Within the High-grade Domains in the West Zone

Grade	Cut-off (percentile)	Orientation	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
Gold	10,20,30,40, 50,60,70	-90→000	-	-	7	-	17
		00→240	0.56	0.06	4	0.38	13
		00→330	-	-	4	-	13
	80,90	-90→000	-	-	6	-	14
		00→240	0.56	0.06	3	0.38	10
		00→330	-	-	3	-	10
	95	-90→000	-	-	4	-	10
		00→240	0.56	0.06	2	0.38	8
		00→330	-	-	2	-	8

14.8.3 HIGH-GRADE DOMAINS - PROBABILITY

In order to estimate the proportion of the high-grade population within each block, an indicator variogram was calculated and modelled for the mineralized domains at the population threshold (5 g/t Au and 50 g/t Ag for VOK, 5 g/t Au and 300 g/t Ag for West Zone).

A lower cut-off was used for the West Zone to remove the background "waste" and improve the quality of the indicator variograms. The lower cut applied was 1 g/t Au and 30 g/t Ag.

As with the low-grade and high-grade populations, the orientations of the variogram models were locally adjusted to reflect the interpretation due to the locally changing orientations in the VOK area. Table 14.6 summarizes the variogram models used for the probability estimate.

Table 14.6 Parameters to Describe Gold Grade Continuity at the Low-grade/High-grade Population Threshold

Area	Cut-off Threshold	Orientation	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
VOK	Gold 5 g/t	00→090	-	-	3	-	20
		-90→000	0.50	0.28	3	0.22	20
		00→000	-	-	3	-	15
	Silver 50 g/t	00→090	-	-	15	-	40
		-90→000	0.44	0.35	8	0.21	40
		00→000	-	-	3	-	15
West Zone	Gold 5 g/t	-90→000	-	-	4	-	10
		00→240	0.37	0.45	4	0.18	10
		00→330	-	-	2	-	4
	Silver 300 g/t	-90→000	-	-	5	-	9
		00→240	0.41	0.21	2.5	0.38	4
		00→330	-	-	2	-	3

14.8.4 LOW-GRADE DOMAINS

The low-grade domains were estimated using the variograms defined for the low-grade populations within the high-grade domains.

14.8.5 DENSITY

Given the amount of density data, a single omni-directional variogram was calculated and modelled for the total Brucejack deposit. Table 14.7summarizes the variogram model.

Table 14.7 Parameters to Describe Density Continuity

Orientation	Nugget	Structure 1		Structure 2	
		Sill	Range	Sill	Range
Omni-directional	0.05	0.47	20	0.48	100

14.9 ESTABLISHMENT OF BLOCK MODELS

A Datamine block model with cell dimensions of 10 mE by10 mN by10 mRL was coded to reflect the surface topography, base of overburden, lithological contacts, and the mineralization domains. This block model was used for estimation of the density, low-grade domains and the low-grade mineralized population within the high-grade domains at VOK and the majority of West Zone.

Within the well informed portion of West Zone, with close spaced drilling of around 5 m by 5 m to 10 m by 10 m, the parent cell size was reduced to 5 mE by 5 mN by 5 mRL for estimation of the background grades and low-grade mineralized population.

Two small scale discretized block models were created for the multiple indicator kriging estimates so that these point estimates could be subsequently re-blocked to take into account the correct degree of smoothing at the final block size. The discretized block models have parent cells sizes of 2.5 mE by 2.5 mN by 2.5 mRL for VOK and the majority of West Zone and 1.25 mE and 1.25 mN by 1.25 mRL for the well informed portions of West Zone.

14.10 GRADE INTERPOLATION PARAMETERS

14.10.1 HIGH-GRADE DOMAINS - LOW-GRADE POPULATION

The lower-grade population, within the high-grade domains, was estimated using ordinary kriging into 10 m by 10 m by 10 m parent blocks for VOK and most of West Zone. In the well informed portion of West Zone a 5 m by 5 m by 5 m parent block was used.

At VOK, the mineralized corridors were estimated with hard boundaries for estimation, except where the corridors merge together. A series of "search domains" with locally adjusted orientations applied for estimation to account for the local variability in the orientation of the corridors. The boundaries between these search domains were treated as soft for estimation.

For the West Zone, the high-grade domains were treated as a single domain for estimation for each area.

Gold and silver grades were interpolated using 1.5 m composites with all data above the population threshold removed (5 g/t Au and 50 g/t Ag for VOK, and 5 g/t Au and 300 g/t Ag for West Zone). Estimation parameters were established from the variography

analysis. Any target blocks that remained uninformed after the first pass search were subsequently estimated in a second search pass using a broader search ellipse and different restrictions.

In VOK and West Zone, the interpolation was controlled by:

- minimum/maximum numbers of composites: 20/26 per block (8/26 for pass 2)
- discretisation: 4 by 4 by 4
- maximum number of composites per hole: 8
- search ellipse:
  - West Zone: 200 m by 300 m by 30 m (400 m by 600 m by 60 m for pass 2)
  - VOK: 60 m by 100 m by 20 m (120 m by 200 m by 40 m for pass 2).

14.10.2 High-grade Domains - High-grade Population

The higher-grade populations were estimated using multiple indicator kriging to control the skewness of the data. Indicator variograms were modelled up to the 95 percentile of the data with a mathematical model used to define the top end of the grade distribution. The threshold for the 95 percentile of the higher-grade population is:

- VOK: 429 g/t Au and 503 g/t Ag
- West Zone: 93 g/t Au and 3,479 g/t Ag.

The higher-grade populations were estimated into small-scale discretized blocks of 2.5 mE by 2.5 mN by 2.5 mRL for VOK and the majority of West Zone and 1.25 mE and 1.25 mN by 1.25 mRL for the well informed portions of West Zone. The estimates were re-blocked into parent blocks twice the size of those used for the lower grade population estimates to further limit the influence of the highest grades in the highest-grade areas.

For VOK, the mineralized corridors were estimated with hard boundaries for estimation, except where the corridors merge together. As with the low-grade populations, a series of "search domains" with locally adjusted orientations were applied for estimation to account for the local variability in the orientation of the corridors. The boundaries between these search domains were treated as soft for estimation.

For West Zone, the high-grade domains were treated as a single domain for estimation for each area.

Gold and silver grades were interpolated using 1.5 m composites with all data below the population threshold removed (5 g/t Au and 50 g/t Ag for VOK, and 5 g/t Au and 300 g/t Ag for West Zone).

Estimation parameters were established from the variography analysis. Any target blocks that remained uninformed after the first pass search were subsequently estimated in a second search pass using a broader search ellipse and different restrictions. The

maximum number of samples was kept small in the second search pass to prevent single extreme grades influencing the block estimates at a great distance.

The interpolation was controlled by:

- minimum/maximum numbers of composites:
  - West Zone: 8/20 per block (2/8 for pass 2)
  - VOK: 6/16 per block (2/6 for pass 2)
- discretisation: 1 by 1 by 1 (indicator kriging)
- maximum number of composites per hole: 10
- search ellipse:
  - West Zone: 50 m by 50 m by 50 m (150 m by 150 m by 150 m for pass 2)
  - VOK: 50 m by 50 m by 20 m (150 m by 150 m by 60 m for pass 2).

14.10.3 High-grade Domains - Probability

The proportion of the higher-grade mineralization was estimated into each block and used to combine the two estimates in the determination of the overall block grade. For example, if a block had a probability of 5% high grade then the final block grade would combine 95% of the low-grade estimate with 5% of the high-grade estimate. The influence of the high-grade population above the 95 percentile is therefore greatly restricted.

An indicator estimate was run at the population threshold for gold and silver (5 g/t Au and 50 g/t Ag for VOK, and 5 g/t Au and 300 g/t Ag for West Zone), using all of the data within the mineralized high-grade domains.

Estimation for VOK and West Zone was into the small-scale discretized blocks used for the high-grade population estimates. The resultant probabilities were re-blocked into parent blocks the same size of those used for the lower-grade population estimates.

The same domains and boundary conditions applied to the high-grade and low-grade populations were used for the probability estimate. Probability for gold and silver were interpolated using 1.5 m composites. Estimation parameters were established from the variography analysis and used a single search pass only to prevent spreading the probability estimate too far in all areas except VOK.

In VOK there has been a significant increase in drilling and the use of a single search for the probability resulted in over smoothing of the estimate. As a result a two pass search was used with the first search kept very tight to prevent over smoothing in well informed areas. The second search was set similar to the single search used for the other areas. The maximum number of samples was also reduced in the first search pass and the maximum number of composites per drillhole was reduced.

For VOK and West Zone the interpolation was controlled by:

- Minimum/maximum numbers of composites:
  - West Zone: 5/50 per block
  - VOK: 5/20 per block (5/50 for pass 2)
- discretisation: 1 by 1 by 1 into small scale discretized blocks
- maximum number of composites per hole: 10 (West Zone), 6 (VOK)
- Search ellipse:
  - West Zone: 75 m by 75 m by 30 m
  - VOK: 35 m by 35 m by 10 m (70 m by 70 m by 20 m for pass 2).

For VOK, approximately 3% of the data is above 5 g/t Au and the average proportion of high-grades within the blocks was estimated at 3% within the Indicated portions of the estimate. For West Zone, around 5% of the data is above 5 g/t Au and the average proportion of high-grades within the blocks was estimated at 5% within the Measured and Indicated portions of the estimate.

14.10.4 Low-grade Domains

The background low-grade domain for VOK was estimated with ordinary kriging using dynamic anisotropies to locally adjust the search and variogram orientations to reflect the main trends of the folding in this area. The search and variogram parameters used were the same as for the low-grade population within the high-grade domains.

The background low-grade domain for West Zone was estimated with ordinary kriging into a domain using the same search and variogram parameters as for the low-grade population within the high-grade domains.

The background low-grade domains were estimated using 1.5 m top cut composites with soft boundaries between the low-grade and high-grade domains, but excluding the high-grade population from within the high-grade domains.

14.11 Density Estimation and Assignment

Density was estimated using simple kriging, with a global mean of 2.54 t/m^3, into 10 mE by 10 mN by 10 mRL parent blocks. Density values were interpolated using 1.5 m composites with parameters established from the variography analysis.

The interpolation was controlled by:

- minimum/maximum numbers of composites: 8/30 per block(2/8 for pass 2)
- discretisation: 4 by 4 by 4
- maximum number of composites per hole: 9

- search ellipse: 300 m by 300 m by 300 m (90 m by 900 m by 900 m for pass 2).

Outside of these areas, the average density of 2.54 t/m³ was applied. There is little variation in density between the different rock types.

A density of 0.9167 t/m³ was assigned to the ice existing in the glacial cap.

14.12 Prior Mining

A 3D wireframe model of the underground development and stopes for West Zone was represented by wireframes, and coded in the block model to ensure that the reported mineral resource estimates were depleted for prior mining.

No mining has occurred at VOK.

14.13 Model Validation

Grade estimates and models were validated by undertaking global grade comparisons with the input drillhole composites, visual validation of block model cross sections, and by grade trend plots.

14.13.1 Global Comparisons

The final grade estimates were validated statistically against the input drillhole composites. Table 14.8 and Table 14.9 provide a comparison of the estimated grades compared to the input grades (declustered where required) for the global estimates within the mineralized domains. This statistical comparison shows that the domains validate reasonably well globally. Note that in VOK the comparison is only for the Indicated resources to remove areas of extrapolation at depth.

Table 14.8 Comparison of the Mean Composite Grade with the Mean Block Model Grade for the Mineralized Domains in the West Zone

	Mineralized Domain	
	Gold (g/t)	Silver (g/t)
Number of Samples	33,089	33,089
Composite Mean	1.40	30.66
Estimated Mean	1.32	28.54

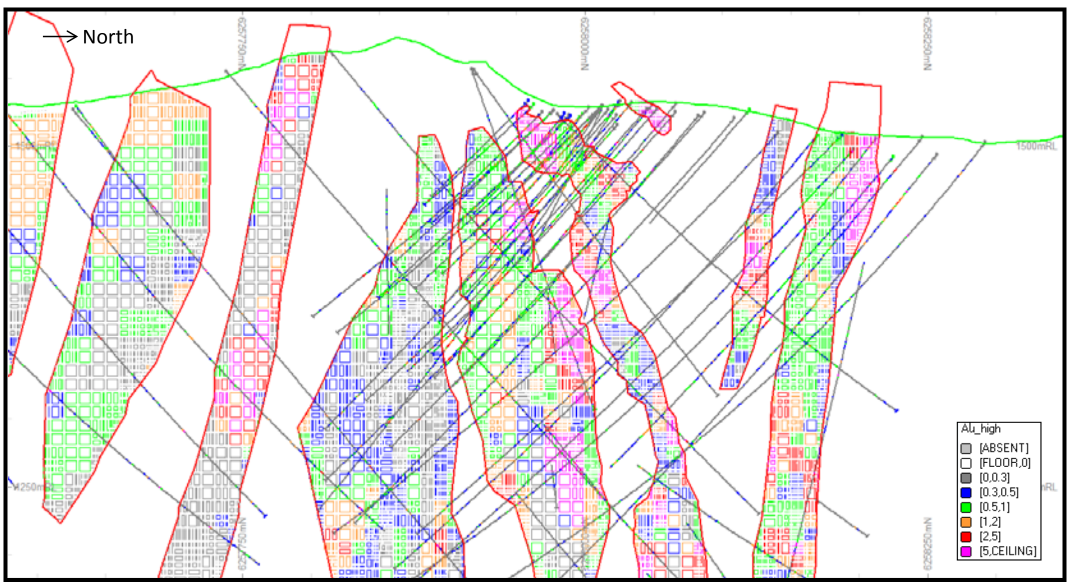
Table 14.9 Comparison of the Mean Composite Grade with the Mean Block Model Grade for the Mineralized Domains in VOK

	Mineralized Domain	
	Gold (g/t)	Silver (g/t)
Number of Samples	57,895	57,895
Composite Mean	3.39	8.85
Estimated Mean	3.03	7.95

14.13.2 Visual Validation

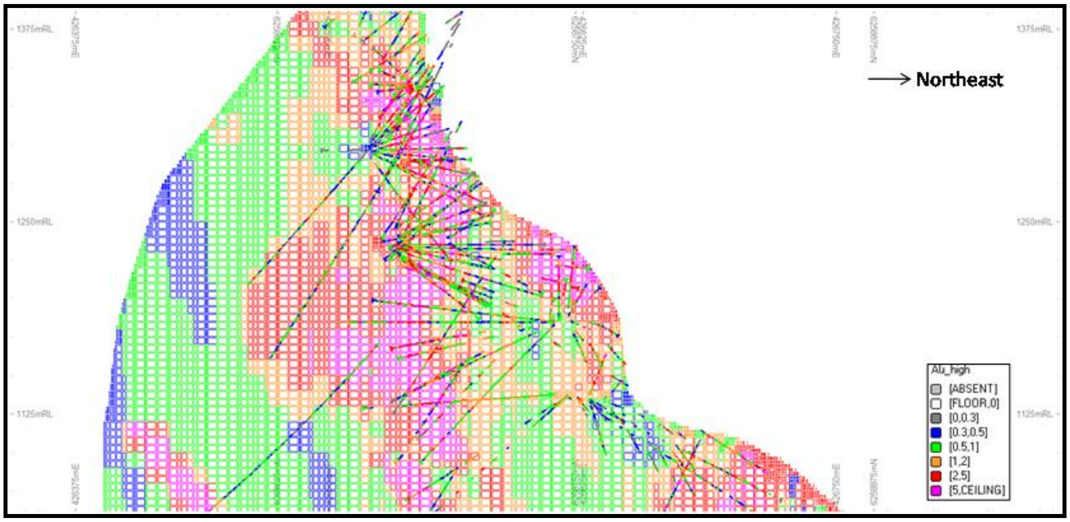
The gold and silver estimates show a good visual correspondence with the input composite grades. Example sections through the higher-grade portions of the main mineralized areas are illustrated in Figure 14.6 and Figure 14.7 for VOK and West Zone, respectively.

Figure 14.6 Example Cross-section Showing Estimated Gold Grades Compared to Input Composites Within the Mineralized Domains for VOK



Note: Drillholes are shown with –20 m clipping.

Figure 14.7 Example Oblique Section Showing Estimated Gold Grades Compared to Input Composites Within the Mineralized Domains for West Zone



Note: Drillholes are shown with +/- 10 m clipping.

14.13.3 GRADE TREND PLOTS

Sectional validation graphs were created to assess the reproduction of local means and to validate the grade trends in the model. These graphs compare the mean of the estimated grades to the mean of the input grades (declustered for West Zone) within model slices (bins). The graphs also show the number of input samples on the right axis, to give an indication of the support for each bin.

Validation graphs were created for the low-grade domains and high-grade domains including the low-grade population estimates, the high-grade population estimates, the probability estimates and the final combined estimates for each area (Olssen and Jones 2012c). Within VOK validation graphs were created for each mineralized corridor.

These graphs indicate that there is good local reproduction of the input grades in both the horizontal and vertical directions. The high-grade population estimate is quite smooth compared to the input data as expected. This smoothing was incorporated into the high-grade population estimate to prevent the over influence of the individual high-grade samples when the high-grade and low-grade estimates are recombined.

14.14 RESOURCE CLASSIFICATION

The resource classification definitions used for this estimate are those published by the CIM in their document “CIM Definition Standards”.

> *Measured Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of*

the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

Indicated Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Inferred Mineral Resource: that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

In order to identify those blocks in the block model that could reasonably be considered as a mineral resource, the block model was filtered by a cut-off grade of 5 g/t AuEq.

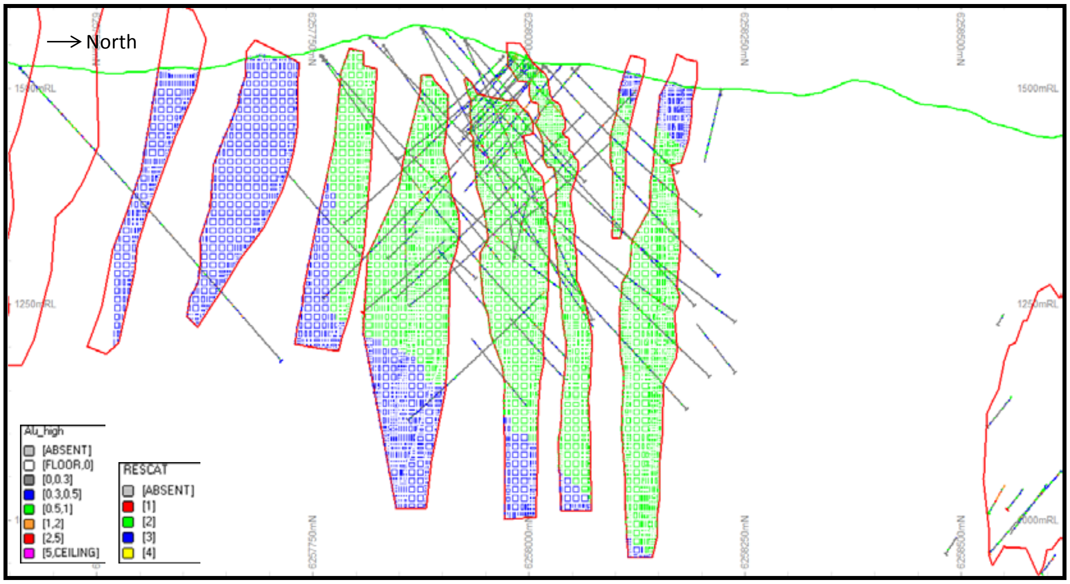
The blocks occurring above 5 g/t AuEq were classified as Measured, Indicated or Inferred.

Classification was applied based on geological confidence, data quality and grade variability. Areas classified as Measured Resources (“Rescat 1”) are within the well informed portion of the West Zone where the resource is informed by 5 m by 5 m or 5 m by 10 m spaced drilling. Areas classified as Indicated Resources (“Rescat 2”) are informed by 20 m by 20 m to 20 m by 40 m drilling within the West Zone and VOK. The remainder of the mineral resource is classified as Inferred Resources (“Rescat 3”) where there is some drilling information and the blocks lie within the mineralized interpretation.

With respect to some of the areas in the high-grade zones at VOK that have drill spacing nearing that of the Measured Resource of West Zone, Snowden has elected to retain the Indicated Resource classification until underground sampling has demonstrated a high level of confidence in these estimates. This is because of the high-grade nature of the mineralization.

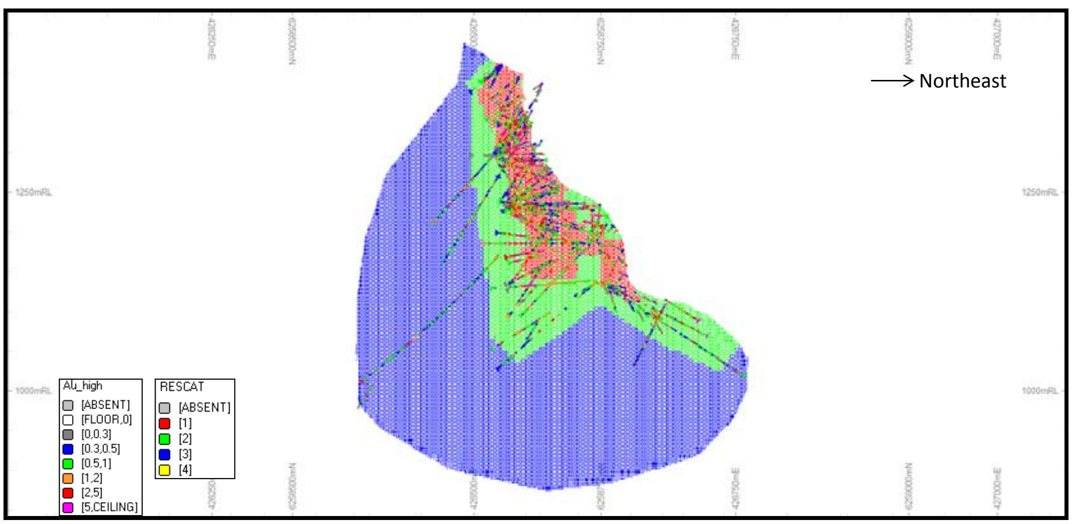
Figure 14.8 and Figure 14.9 illustrate example sections through the main areas of mineralization, coloured by resource classification (or “Rescat”) for VOK and West Zone, respectively.

Figure 14.8 Example Cross-section Showing Classification of Resource Estimate for VOK with Drilling Coloured by Gold Grade



Note: Drillholes are shown with ±20 m clipping. Codes are green for Indicated and blue for Inferred Resource categories.

Figure 14.9 Example Oblique Section Showing Classification of Resource Estimate for the West Zone with Drilling Coloured by Gold Grade



Note: Drillholes are shown with ±20 m clipping. Codes are red for Measured, green for Indicated and blue for Inferred Resource categories

14.15 RESOURCE REPORTING

The mineral resources are reported above a cut-off grade of 5 g/t AuEq, which reflects the potential economics of a high-grade underground mining scenario. The AuEq value for each block is calculated according to the formula (AuEq = Au + Ag/53) based upon prices of $US1,590/oz and $US30/oz for gold and silver, respectively. Recoveries for gold and silver are assumed to be similar.

High-grade mineral resources for VOK and West Zone are summarized in Table 14.10 and Table 14.11, respectively.

Table 14.10 VOK Mineral Resource Estimate Based on a Cut-off Grade of 5 g/t AuEq – November 2012(1)(4)

Category	Tonnes (millions)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Indicated	16.1	16.4	14.1	8.5	7.3
Inferred(2)	5.4	17.0	15.7	2.9	2.7

Note: (1) Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues. The Mineral Resources in this report were classified using the CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
(2) The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.
(3) Contained metal and tonnes figures in totals may differ due to rounding.
(4) The gold equivalent value is defined as AuEq = Au + Ag/53.

Table 14.11 West Zone Mineral Resource Estimate Based on a Cut-off Grade of 5 g/t AuEq – April 2012(1)(4)(5)

Category	Tonnes (millions)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Measured	2.4	5.85	347	0.5	26.8
Indicated	2.5	5.86	190	0.5	15.1
Measured + Indicated	4.9	5.85	267	0.9	41.9
Inferred(2)	4.0	6.44	82	0.8	10.6

Notes: (1), (2), (3), and (4) - See footnotes to Table 14.10

14.15.1 CONTRIBUTION OF LOW-GRADE, HIGH-GRADE AND EXTREME GRADE POPULATIONS

Within the final combined estimates, the proportion of metal attributable to the low-grade (less than 5 g/t Au), high-grade (5 g/t Au to 95 percentile) and extreme-grade (greater than 95 percentile) populations was calculated. This information is reported in Table

14.12 and Table 14.13 for the combined Measured plus Indicated and the Inferred portions of VOK and West Zone resources.

Table 14.12 Contribution of Grade Populations to the Estimate – VOK

Classification	Population		Tonnes (Mt)	Gold Metal (koz)	Gold Grade (g/t)	Contribution Gold Metal (%)
Indicated	Low grade	<5 g/t Au	14.8	318	0.67	4
	High grade	5-429 g/t Au	1.2	6,041	155	71
	Extreme grade	>429 g/t Au	0.12	2,127	558	25
	Total		16.1	8,484	16.4	100
Inferred	Low grade	<5 g/t Au	4.9	89	0.57	3
	High grade	5-429 g/t Au	0.49	2,566	164	87
	Extreme grade	>429 g/t Au	0.01	276	579	9
	Total		5.4	2,935	17.0	100

Table 14.13 Contribution of Grade Populations to the Estimate – West Zone

Classification	Population		Tonnes (Mt)	Gold Metal (koz)	Gold Grade (g/t)	Contribution Gold Metal %)
Measured + Indicated	Low grade	<5 g/t Au	4.2	144	1.06	16
	High grade	5-93 g/t Au	0.67	777	36.2	84
	Extreme grade	>93 g/t Au	0.001	2.4	97.7	0.3
	Total		4,918	924	5.84	100
Inferred	Low grade	<5 g/t Au	3,570	93	0.81	11
	High grade	5-93 g/t Au	378	526	43.3	64
	Extreme grade	>93 g/t Au	49	208	133	25
	Total		3,997	827	6.44	100

15.0 MINERAL RESERVE ESTIMATES

15.1 GENERAL

The mine design and mineral reserve estimation have been completed to a level of appropriate for feasibility studies. The mineral reserve estimate stated herein is consistent with the CIM Standards on mineral resources and mineral reserves and is suitable for public reporting. As such, the mineral reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The mineral reserves were developed from the resource model, "bjbm_1211_v2_cut", which was provided to AMC by Snowden–on behalf of Pretivm–in November 2012.

15.2 CUT-OFF GRADE

A NSR cut-off grade of $180/t of ore was used to define the mineral reserves, which is unchanged from the value used for both the June 2011 PEA (Ghaffari et al. 2011) and the February 2012 Updated PEA (Ghaffari et al. 2012). In the 2012 Updated PEA, site operating costs were estimated at approximately $171/t based on a 1,500 t/d operation, of which $104/t was attributable to mining. The cut-off grade thus provided a minimum margin of $9/t of ore mined.

The feasibility study provides the platform for increasing the accuracy of cost estimations relative to the 2011 and 2012 PEAs. Furthermore, the design production rate has been increased from 1,500 to 2,700 t/d. Average site operating costs over the LOM are now evaluated as follows:

- Mining: $94.4/t
- Processing: $18.2/t
- Surface Services: $19.6/t
- G&A: $25.5/t
- Total: $157.7/t

The $180/t NSR cut-off grade provides a minimum $22.3/t operating margin on ore mined.

15.3 NET SMELTER RETURN MODEL

AMC generated the NSR values for mineral reserves using the parameters provided by Tetra Tech and summarized in Table 15.1. The NSR for each block in the resource model was calculated as the payable revenue for gold and silver, less the costs of refining, concentrate treatment, transportation, and insurance.

The metal price assumptions are US$1,350/oz gold and US$22/oz silver. Costs assume a Cdn$ to US$ exchange rate of 1.00.

The NSR contributions for both flotation concentrate and doré were calculated individually, combined, and assigned to each block in the resource model.

Note that the base case economic parameters for this study may vary from the NSR model inputs, due to additional metallurgical knowledge acquired through the duration of the study. AMC believes that the magnitude of any parameter variation has no material impact on the study findings.

Table 15.1 Net Smelter Return Parameters

Items		Units		Parameters
Currency	Conversion Rate	Cdn$/US$	-	1.00
Metal Prices	Gold	US$/oz	-	1,350.00
	Silver	US$/oz	-	22.00
Doré				
Process Recoveries	Gold	%	For Au <0.22 g/t	0.00
		%	For Au ≥0.22 g/t and Au <9.85 g/t	$-0.147 \times Au^2 + 5.68 \times Au - 1.214$
		%	For Au ≥9.85 g/t and Au <40.00 g/t	$6.517 \times \ln(Au) + 25.61$
		%	For Au ≥40.00 g/t	50.00
	Silver	%	For Ag <3.00 g/t	23.00
		%	For Ag ≥3.00 g/t and Ag <260.00 g/t	$39.508 \times Ag^{-0.454} - 1$
		%	For Ag ≥260.00 g/t	1.50
Selling Costs	Metal Payable – Gold	%	-	99.80
	Metal Payable – Silver	%	-	99.80
	Refining Charge – Gold	Cdn$/oz	-	2.00
	Refining Charge – Silver	Cdn$/oz	-	0.60
	Transport/Port Handling Costs	Cdn$/oz-gold	-	1.00
	Insurance (Net Invoice Value)	% NIV	-	0.15
Flotation Concentrate				
Process Recoveries	Gold	%	For Au < 0.22 g/t	20.00
		%	For Au ≥0.22 g/t and Au <6.30 g/t	$85.44 \times Au^{0.056} - (-0.147 \times Au^2 + 5.68 \times Au - 0.714)$
		%	For Au ≥6.30 g/t and Au <9.85 g/t	$1.972 \times \ln(Au) + 91.774 - (-0.147 \times Au^2 + 5.68 \times Au)$
		%	For Au ≥9.85 g/t and Au <40 g/t	$1.972 \times \ln(Au) - 6.517 \times \ln(Au) + 64.95$
		%	For Au ≥40.00 g/t	48.00
	Silver	%	For Ag <3.00 g/t	35.00
		%	For Ag ≥3.00 g/t and Ag <260.00 g/t	$6.5399 \times \ln(Ag) - (39.508 \times Ag^{-0.454}) + 58.593$
		%	For Ag ≥260.00 g/t	92.50

table continues...

Items		Units		Parameters
Selling Costs	Metal Payable – Gold	%	For Au <0.25 g/t	0.00
		%	For Au ≥0.25 g/t and Au <3.00 g/t	$88.935 \times Au^{0.03} + 2$
		%	For Au ≥3.00 g/t and Au <11.50 g/t	$0.6677 \times \ln(Au) + 93.27$
		%	For Au ≥11.50 g/t	95
	Metal Payable – Silver	%	For Ag ≤10.00 g/t	0
		%	For Ag >10.00 g/t	95
	Penalty Charge – Arsenic	Cdn$/t-concentrate	-	0.3481 x (Ratio of As (ppm) / S (%) in Mill Feed)
	Treatment Charge	Cdn$/t-concentrate	-	200.00
	Refining Charge – Gold	Cdn$/oz	-	N/A
	Refining Charge – Silver	Cdn$/oz	-	N/A
	Transport/Port Handling Costs	Cdn$/t-concentrate	-	227.97
	Concentrate Production	% of mill feed	-	8.00
	Concentrate Moisture	%	-	9.000
	Insurance (Net Invoice Value)	% NIV	-	0.15

Note: NIV = net invoice value

15.4 MINING SHAPES

AMC used the Mineable Shape Optimizer (MSO) module from the Datamine Studio 3 mine planning software package to produce design excavations (shapes) that meet both the cut-off grade and operational design criteria.

The design criteria constrain the geometry of all planned excavations to what is achievable through the planned mining methods. Section 16.0 provides further detail on mining shapes and design parameters.

The preliminary shapes were individually refined to minimize the amount of sub-economic material within the shape volume that is inseparable from profitable material due to the practical constraints of mining.

15.5 DILUTION AND RECOVERY ESTIMATES

In the evaluation of mineral reserves, modifying factors were applied to the tonnages and grade of all mining shapes (both stoping and development) to account for dilution and ore losses that are experienced at all mining operations.

Ore dilution includes overbreak into the design hanging wall and design footwall, and also into adjacent backfilled stopes. Diluting materials are assumed to carry no metal values in the estimation of mineral reserve grades.

The largest component of dilution at Brucejack will be paste backfill due to its inherently weaker strength, compared to the hanging wall and footwall rock masses for any given dimensions of exposure.

Ore losses (recovery factors) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, problematic rock conditions, losses in fill, and blasting issues.

The factors used in generating the diluted and recovered ore tonnage (mineral reserves) are listed in Table 15.2.

Table 15.2 Dilution Factors and Recovery Factors by Type of Excavation

Type of Excavation	Dilution Factor* (%)	Recovery Factor* (%)
Primary Stopes	6.8	97.5
Secondary Stopes	15.2	92.5
Sill Pillar Stopes**	15.2	75.0
Ore Cross-cuts	4.0	100.0
Production Slashing	7.5	100.0

Notes: *Expressed on a weight basis.
**Includes stope ore to 30 m beneath the surface crown pillar.

The dilution factors were calculated from standard overbreak assumptions that are based on AMC's experience and benchmarking of similar long-hole open stope operations.

- All stopes (primary and secondary) carry 0.8 m of dilution from rock overbreak into the design hanging wall and design footwall, and 0.3 m of backfill dilution from the floor.
- Secondary stopes carry an additional 1.0 m of backfill dilution on each wall that exposes a primary stope.
- Sill pillar stopes are treated as secondary stopes, given the additional backfill dilution that can be expected from the roof.
- Ore cross-cuts carry 0.5 m of dilution from rock overbreak into the design hanging wall and design footwall.
- Production slashing of secondary stopes carries 0.5 m of backfill dilution on each wall that exposes a primary stope.

Further to the recovery factors applied to the various types of excavations in Table 15.2, the anticipated recovery of ore in proximity to the Brucejack Fault zone has been capped at 75%. This percentage was applied to all excavations within the fault zone, which is currently estimated to host approximately 1.35 Mt (7%) of in situ mineral reserves.

The application of the above parameters yields an overall LOM ore recovery of 93% and an overall ore dilution of 10%. The employment of parallel production drillholes in stoping operations at Brucejack will provide improved dilution control in comparison to fan drilling; this is discussed further in Section 16.0.

15.6 Orebody Description

Mineral reserves delineated at the $180/t NSR cut-off define an orebody consisting of numerous independent lenses in the VOK Zone and two distinct lenses in the West Zone, extending over a 550 m vertical distance, from the 1,000 m elevation to surface (approximately 1,550 m elevation).

15.6.1 VOK Zone

The VOK Zone hosts 21 lenses that each contain 60,000 t or more of mineral reserves. The mineral reserves are distributed between these lenses as follows:

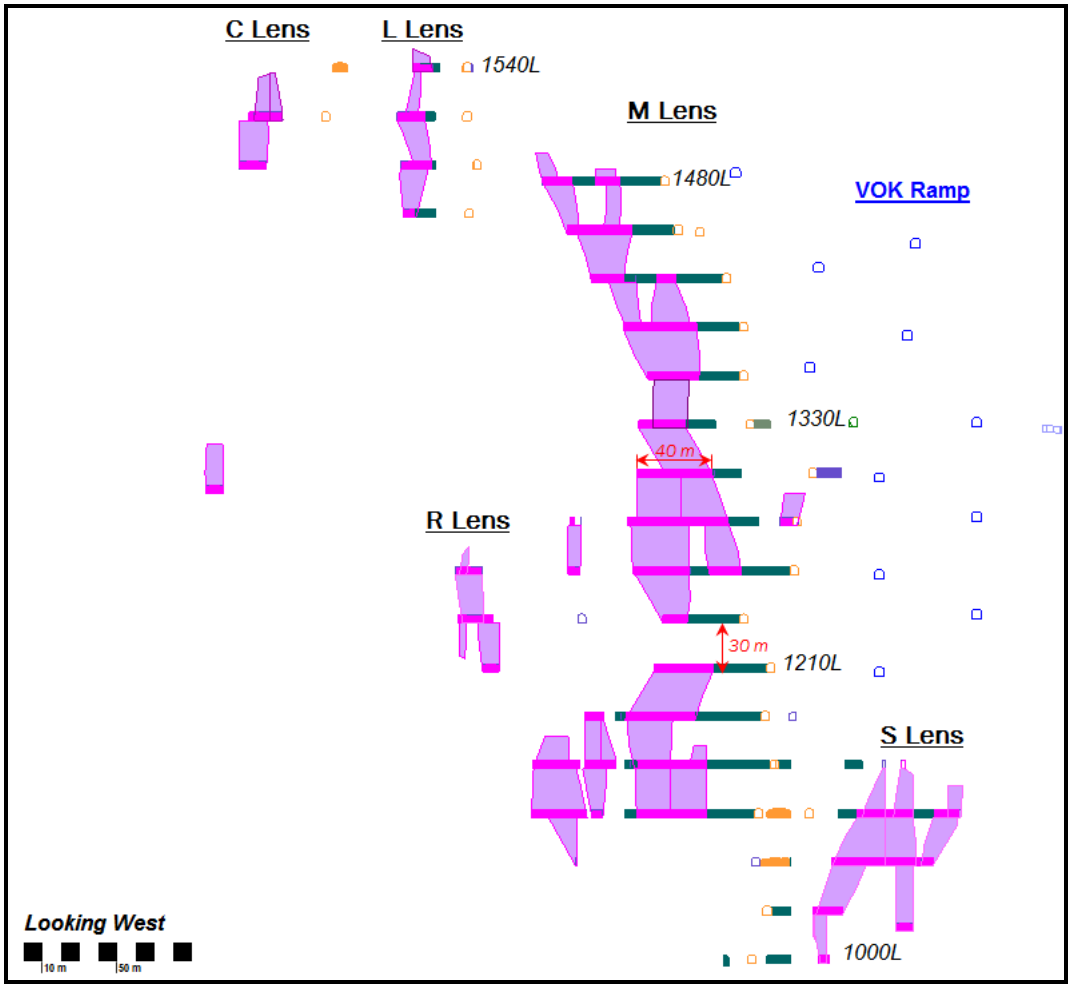
- 40% are within the Main Lens (M Lens)
- 30% are within the next three largest lenses
- The remaining 30% are distributed across 18 smaller lenses.

Mineral reserves in the Galena Hill area are proximal to the VOK Zone and have been considered as part of the VOK mineral reserves.

Strike length varies considerably with elevation, but generally the M Lens has a strike length of approximately 300 m. All other lenses in the VOK Zone strike 100 m or less in length.

Orebody thickness varies considerably by lens and by elevation but, on average, the M Lens is 25 m thick in the lower elevations (below 1,210 Level), 60 m thick in the mid-elevations (1,210 to 1,360 Level), and 25 m thick in the higher elevations (above 1,360 Level). The other lenses average approximately 20 to 25 m thick, although narrow reserves have been delineated down to a minimum 3 m mining thickness. The remaining lenses also vary but are, on average, approximately 20 m thick. The VOK Zone has a slight plunge towards the east. Figure 15.1 illustrates typical widths found in the VOK Zone.

Figure 15.1 Cross-section through the VOK Zone LOM Mining Shapes

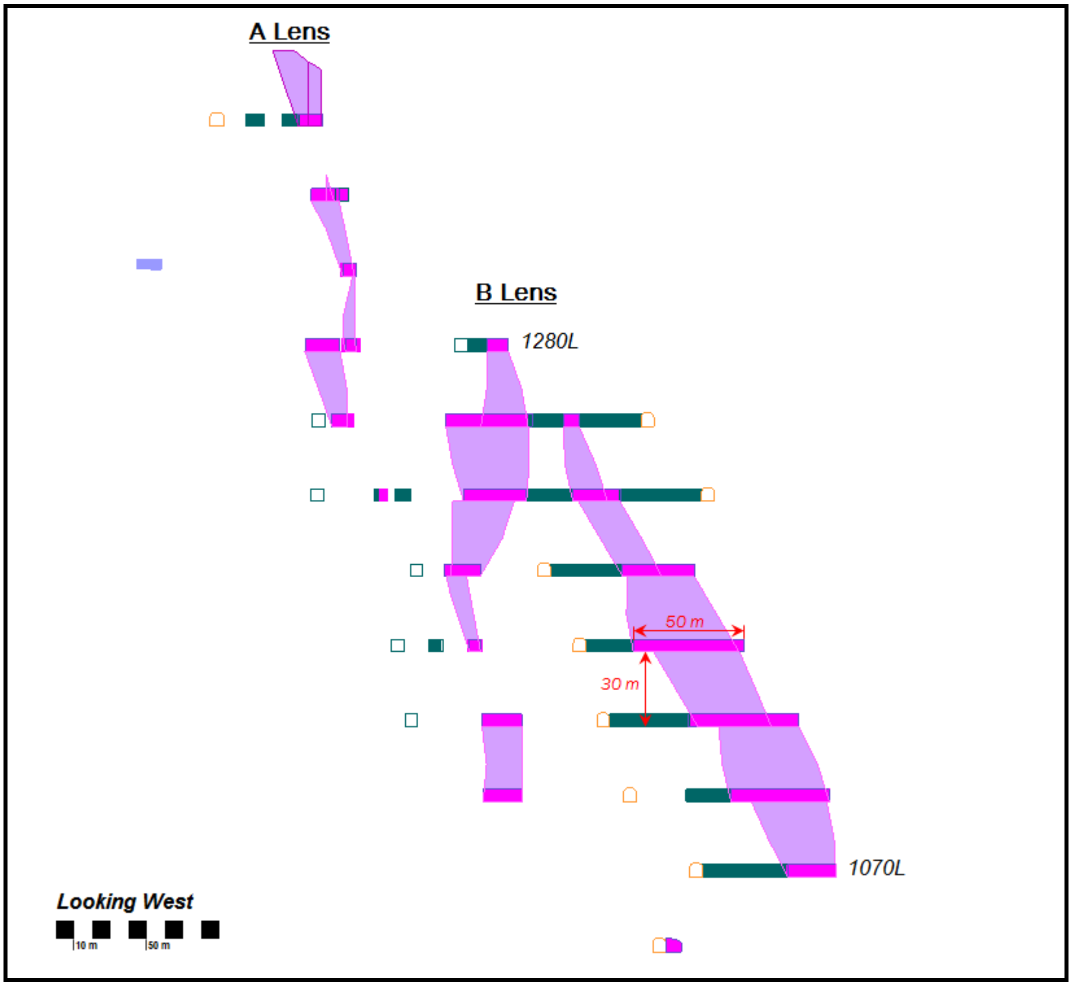


15.6.2 WEST ZONE

Mineral reserves within the West Zone are contained within five lenses, two of which (A and B lenses) host 90% of West Zone reserves. Strike lengths vary considerably with elevation, averaging approximately 100 m in the larger A and B lenses, while the smaller lenses are no more than 40 m along strike.

The A and B lenses have an average thickness of approximately 30 m, while the smaller lenses average only 15 m. Figure 15.2 illustrates typical widths found in the West Zone.

Figure 15.2 Cross-section through the West Zone LOM Mining Shapes



15.7 MINERAL RESERVES

Mineral reserves tabulated by zone and by reserve category are presented in Table 15.3. All mineral reserves are scheduled in the LOM plan, which is presented in Section 16.0.

The mining blocks divide the mineral reserves into logical parcels consistent with the mining sequence, and form the basis of the LOM development and production schedule discussed in Section 16.0.

Figure 15.3 illustrates the division of mineral reserves into lower, middle, and upper blocks in the VOK Zone. The West Zone is divided into lower and upper blocks as shown in Figure 15.4.

Table 15.4 presents the mineral reserves by mining areas (blocks).

Table 15.3 Brucejack Mineral Reserves* by Zone and by Reserve Category

Zone		Ore Tonnes (Mt)	Grade		Metal	
			Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
VOK Zone	Proven	-	-	-	-	-
	Probable	15.1	13.6	11	6.6	5.3
	Total	**15.1**	**13.6**	**11**	**6.6**	**5.3**
West Zone	Proven	2.0	5.7	309	0.4	19.9
	Probable	1.8	5.8	172	0.3	10.1
	Total	**3.8**	**5.8**	**243**	**0.7**	**30.0**
Total Mine	Proven	2.0	5.7	309	0.4	19.9
	Probable	17.0	12.8	28	7.0	15.4
	Total	**19.0**	**12.0**	**58**	**7.3**	**35.3**

Note: *Rounding of some figures may lead to minor discrepancies in totals. Based on Cdn$180/t cut-off grade, US$1,350/oz gold price, US$22/oz silver price, Cdn$/US$ exchange rate = 1.0.

Figure 15.3 Reserve Shapes and Mining Blocks in the VOK Zone

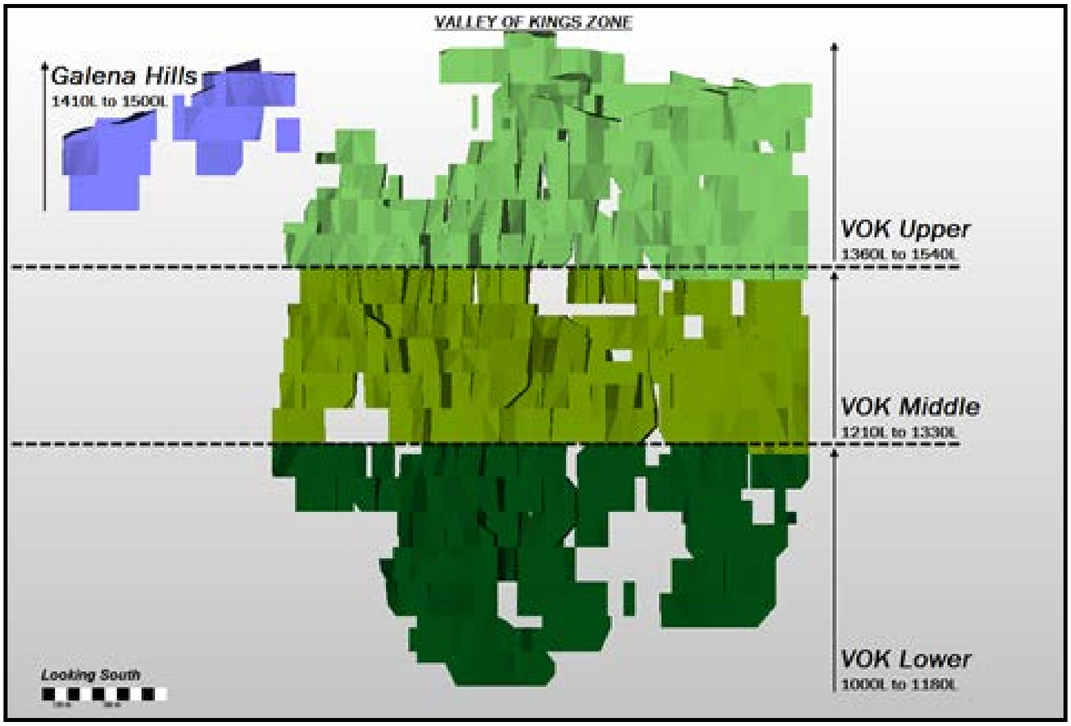


Figure 15.4 Reserve Shapes and Mining Blocks in the West Zone

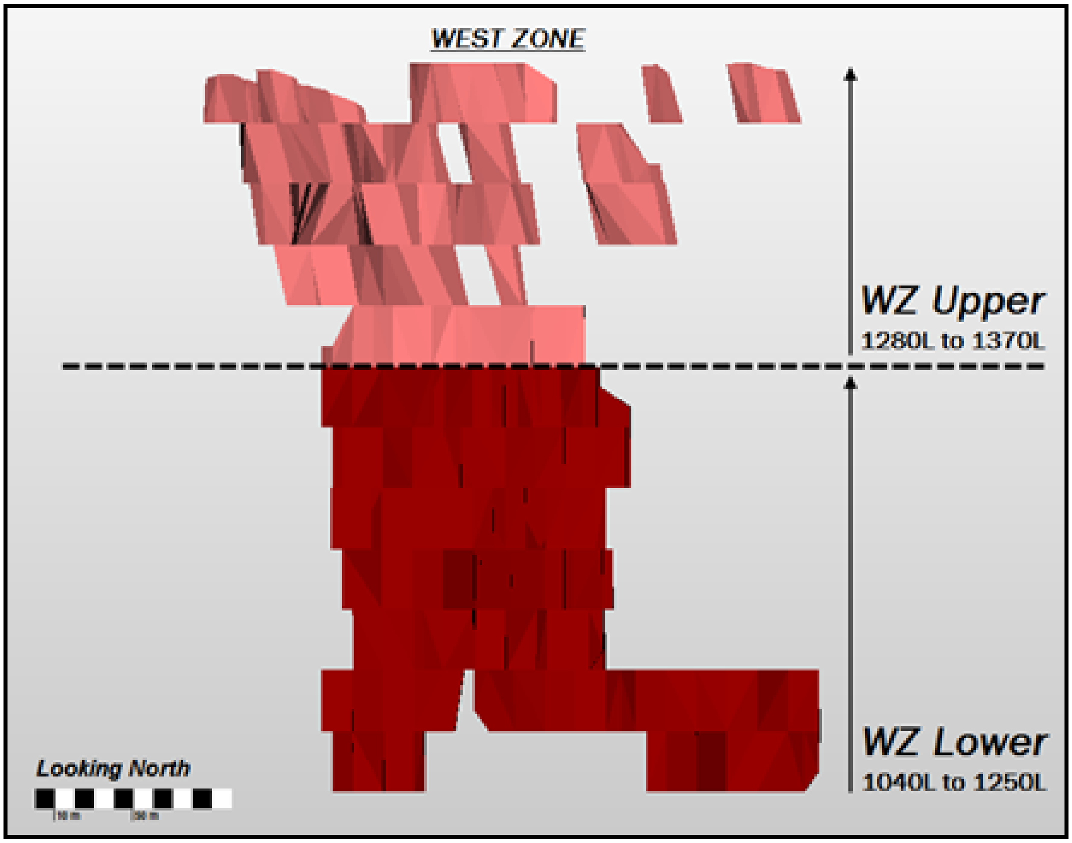


Table 15.4 Brucejack Mineral Reserves* by Mining Block

Mining Block	Ore Tonnes (Mt)	NSR ($/t)	Grade		Contained Metal	
			Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
VOK Upper	4.9	537	13.9	10	2.2	1.6
VOK Middle	6.7	487	12.7	10	2.8	2.1
VOK Lower	2.9	575	14.8	12	1.4	1.1
Galena Hills	0.7	571	14.5	26	0.3	0.6
VOK	**15.1**	**524**	**13.6**	**11**	**6.6**	**5.3**
WZ Upper	1.0	295	3.4	313	0.1	10.2
WZ Lower	2.8	364	6.6	218	0.6	19.9
WZ	**3.8**	**346**	**5.8**	**243**	**0.7**	**30.0**
Mining Block Total	**19.0**	**488**	**12.0**	**58**	**7.3**	**35.3**

Note: *Rounding of some figures may lead to minor discrepancies in totals. Based on Cdn$180/t cut-off grade, US$1,350/oz gold price, US$22/oz silver price, Cdn$/US$ exchange rate = 1.0.

16.0 MINING METHODS

16.1 GENERAL

The proposed underground mine design supports the extraction of 2,700 t/d of ore through a combination of transverse LHOS and longitudinal LHOS. Paste backfill is integral to the mine plan to maximize both orebody recovery and mining productivity. Modern trackless mobile equipment will be employed in the majority of mining activities.

A main decline from a surface portal in close proximity to the concentrator will be used to access the mine. A second decline, parallel to the main access decline, will be dedicated to conveying crushed ore directly to the concentrator via a 650 m long conveyor. The existing West Zone portal will also provide access (and egress) to the mine, although this route will not generally be required for day to day operations.

A fleet of LHDs and trucks will be used for material loading and transport from the various underground working areas through an internal ramp system that connects all levels to the centrally located crusher.

Permanent fans will provide ventilation by forcing air down the declines, through the internal ramps, and exhausting to surface via dedicated raises that connect the various working levels to surface in each zone. The primary fans will be located at each of the main surface portals, and complemented by booster fans located in the exhaust raises, in a common push-pull configuration. An electric mine air heating system will be used to take advantage of low electricity prices, with a propane system available as a back-up.

A pre-production development program that attains approximately 660 m/mo of advance will be required to establish the mine infrastructure and provide access to the initial stoping levels during the first two years of underground activity. Ongoing development, to sustain 2,700 t/d of ore production, will average approximately 475 m/mo during the first half of the mine life, and then decrease to approximately 275 m/mo in later years.

Major underground infrastructure will include crusher, conveyor belt, ventilation raises, fans, heating system, pumping stations, a maintenance facility, electrical substations, a fuelling facility, explosives magazines, refuges, mine communications and other ancillary installations.

16.2 Mine Design

16.2.1 Access and Ramp Infrastructure

The upper elevations of the West Zone and the VOK Zone bulk sample area on the 1,345 Level are currently accessible via the existing West Zone portal. The infrastructure development program will utilize this existing development, effectively developing the mine from the bulk sample access drive.

The new main access decline will join the main surface portal to the crusher tip and workshop area on the 1,330 Level. A main cross-over drive on the 1,330 Level will join the main decline to the two independent ramps servicing the VOK Zone and West Zone. Figure 16.1 illustrates the general development arrangement.

The internal ramps will connect all levels of the mine. The West Zone portal will be used for underground access until the completion of the twin declines and portal construction.

The southern ramp (VOK ramp) which will service the VOK and Galena Hill zones, and the northern ramp (West Zone ramp) which will service the West Zone, were both designed in a race-track configuration for safety, haulage efficiency, and to minimize wear on mobile equipment.

The VOK ramp will be developed up and down from the bulk-sample access development on the 1,345 Level. The West Zone ramp will be developed up and down from the main cross-over drive on the 1,330 Level.

The use of an independent ramp for each zone–as opposed to a single ramp servicing both zones–was selected in the interest of access and capital efficiency, given that the West Zone ramp will not be required until mid-way through the mine life.

For ease of entry and exit, ramps were designed with a 25 m turning radius and a 15% gradient, levelling out to a 5% gradient in proximity to a level access intersection. Passing bays were incorporated where required, typically at the level access. Figure 16.2 shows the ramp system for both zones in perspective view.

Figure 16.1 Mine Access and Development Infrastructure

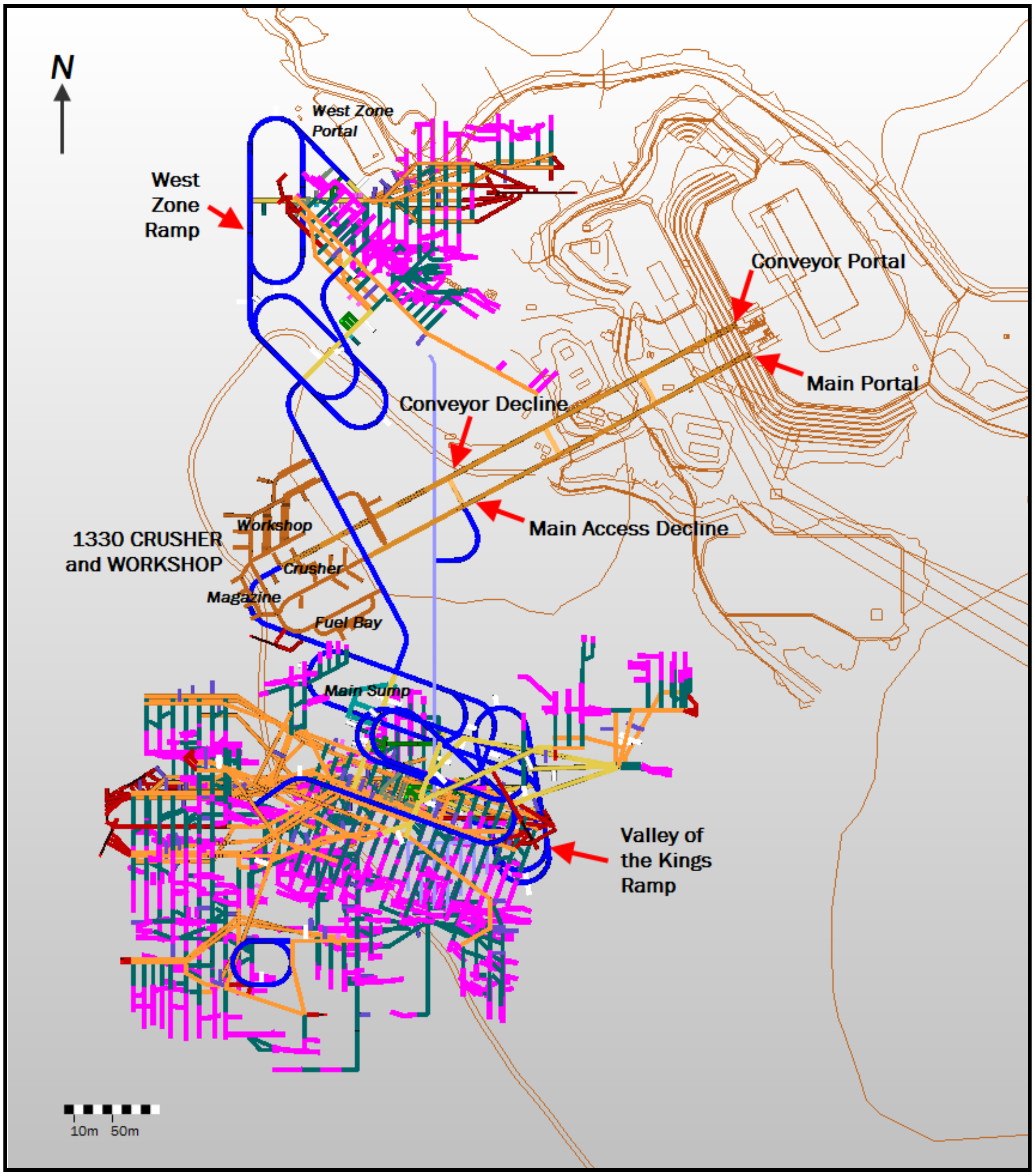
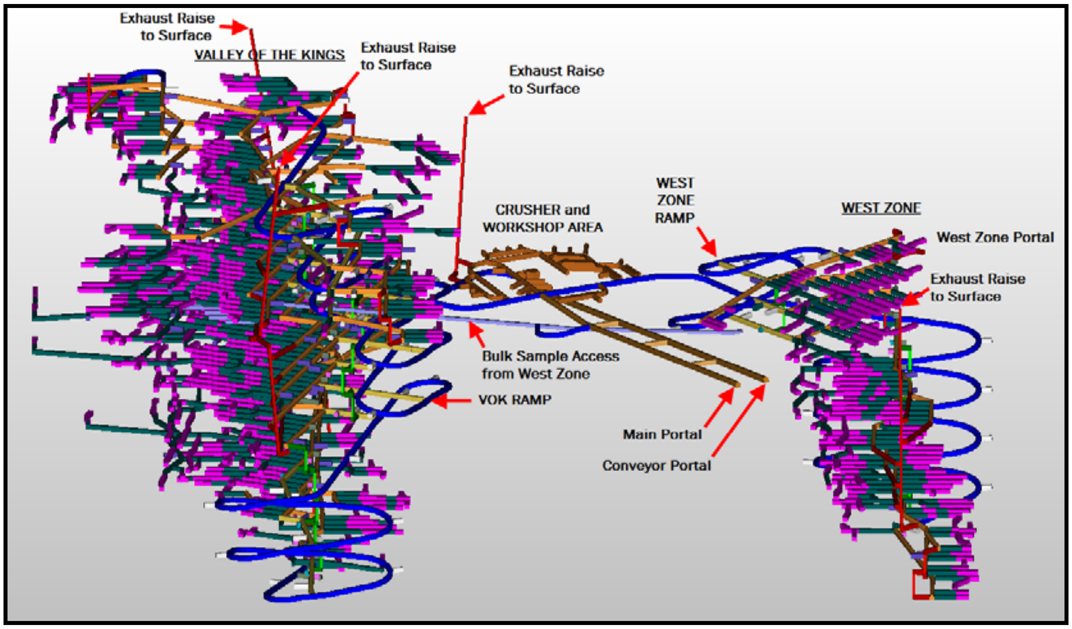


Figure 16.2 Brucejack Twin Declines and Ramp System

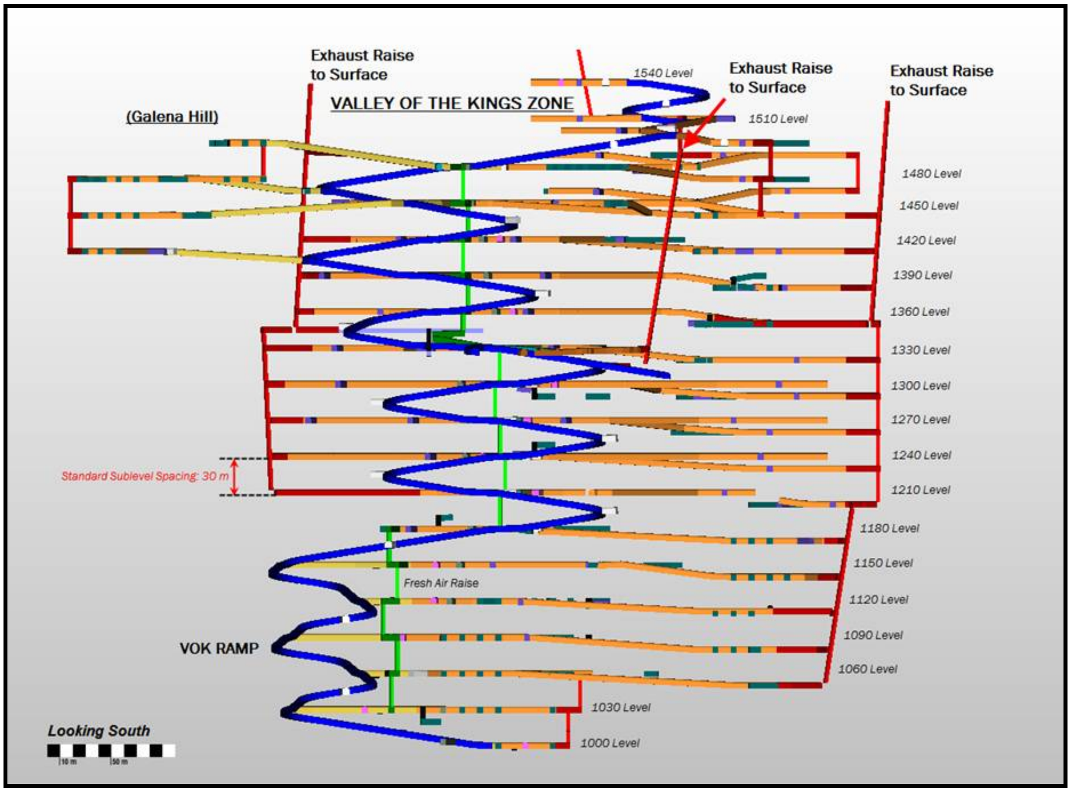


16.2.2 Level Development

Sublevels will be accessed from the ramps on a 30 m vertical interval that is defined by the planned stoping heights. Footwall and hanging wall drives will be set back a minimum of 22.5 m from the ore contact, whereas ramp development will be set back a minimum 50 m from the ore contact. This arrangement promotes long-term geotechnical stability and provides adequate space for the placement of a fresh air raise and other ancillary services between the ramp and level development.

Sublevels generally terminate at a ventilation raise at one or both ends, permitting the exhaust of contaminated air from activity on the level. Figure 16.3 illustrates the VOK Zone sublevel arrangement in long section.

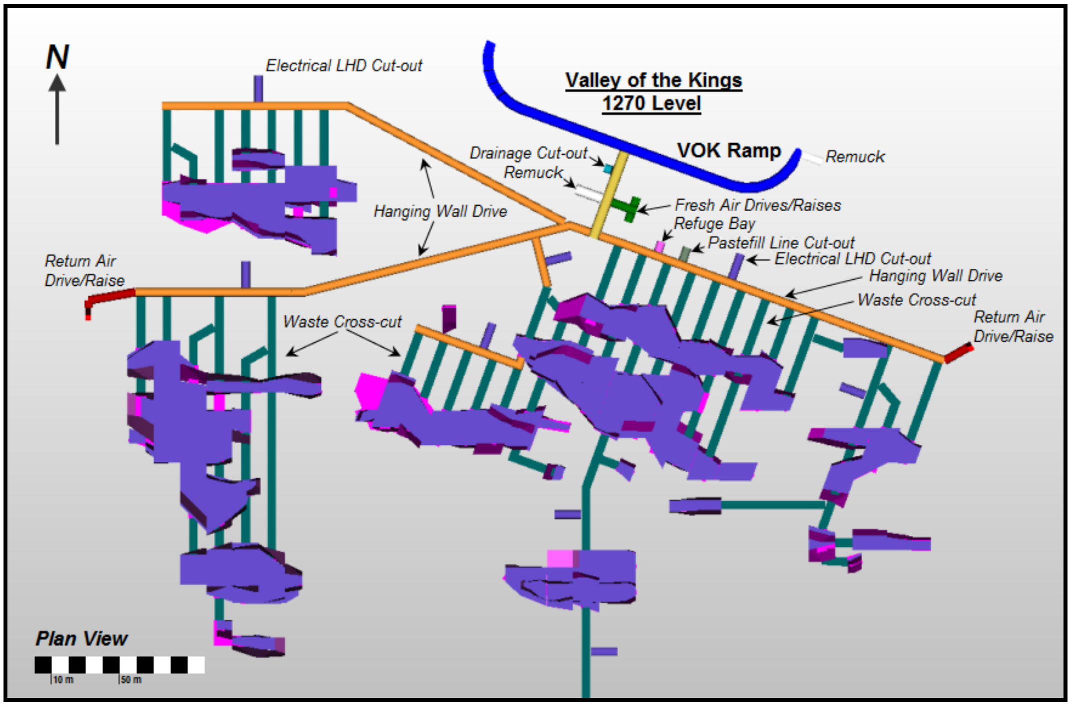
Figure 16.3 VOK Zone Sublevel Arrangement – Long Section



Level development will follow the general strike of the various lenses, providing access to the mineralized zones in a manner that promotes transverse mining wherever possible. Level development will generally be in the hanging wall, with hanging wall drives typically including excavations for sumps, refuges, transformers, remucks, paste fill line, and raise accesses.

Stope access crosscuts will be on 15 m spacings, with the exception of those levels where sill extraction or near-surface weathered ore will be recovered in smaller units that are designed on 10 m spacings. Figure 16.4 illustrates typical level development requirements.

Figure 16.4 Typical Level Plan – 1,270 Level in the VOK Zone



Development design considered equipment size, services, and required activity. Development design parameters are summarized in Table 16.1. Figure 16.5 and Figure 16.6 illustrate standard designs for hanging wall drives and the main decline, respectively.

Figure 16.5 Standard Design – Hanging Wall Drive

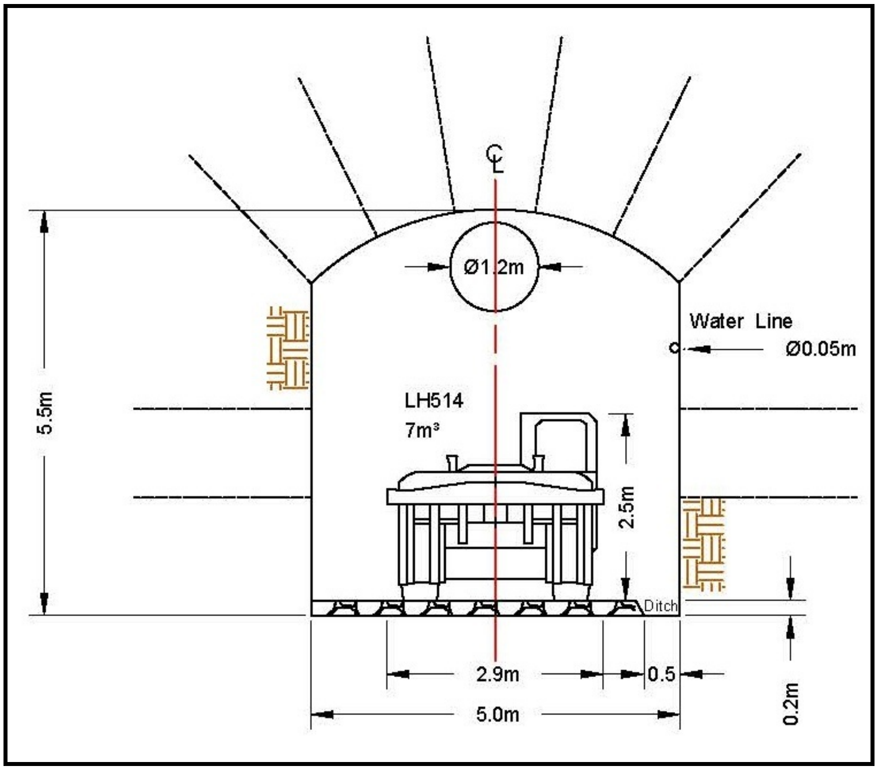


Figure 16.6 Standard Design – Main Decline

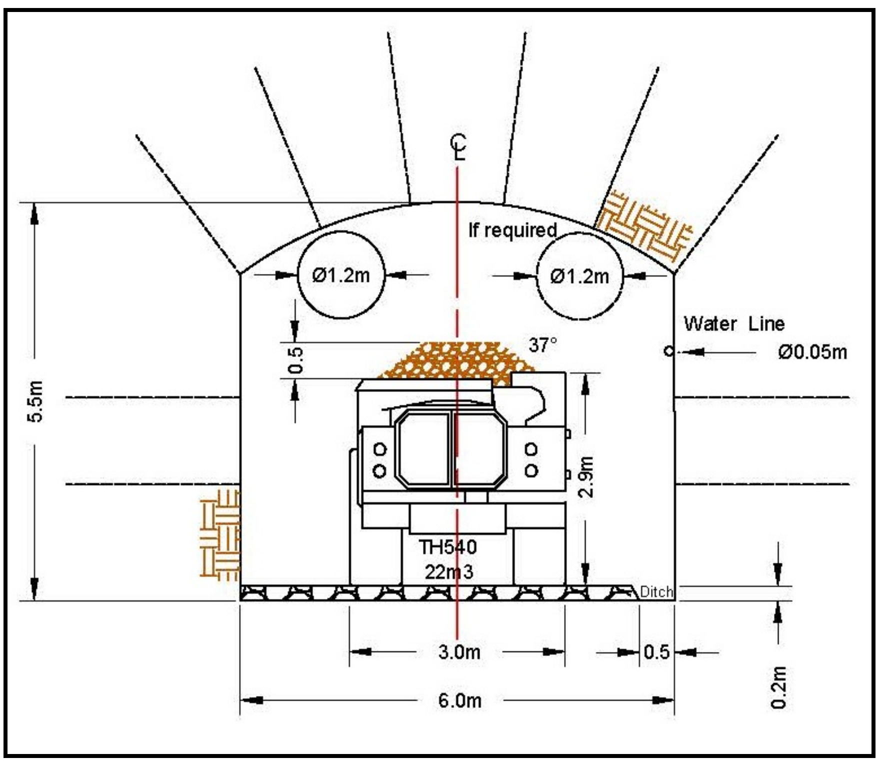


Table 16.1 Development Design Parameters

Parameter	Lateral																		Vertical		
	Remuck	Hanging Wall Drive	Access Drive	Electric LHD Cut-out	Ramp	Return Air Drive	Decline Cross-over Drive	Conveyor Decline	Main Access Decline	Infrastructure Drive	Drainage Cut-out	Waste Cross-cut	Main Cross-over Drive	Refuge Bay Cut-out	Ore Cross-cut	Fresh Air Drive	Return Air Drive	Pastefill Line Drive	Alimak Raise	Return Air Raise	Fresh Air Raise
Width (m)	6.0	5.0	6.0	5.0	6.0	5.0	6.0	6.5	6.0	5.0	5.0	5.0	6.0	5.0	6.0	5.0	5.0	5.0	3.0	3.0	3.0
Height/Length (m)	5.5	5.5	5.5	5.5	5.5	5.5	5.5	6.0	5.5	5.5	5.5	5.0	5.5	5.5	5.0	5.5	5.5	5.5	3.0	3.0	3.0
Arch (m)	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	0.0	2.0	2.0	0.0	2.0	2.0	2.0	0.0	0.0	0.0
Max Gradient (%)	2	15	2	2	15	2	5	15	15	2	2	2	15	2	2	2	2	2	-	-	-

16.2.3 STOPE DESIGN

AMC used the MSO module from the Datamine Studio 3 mine planning software package to produce conceptual stope shapes. Key design parameters used in MSO are summarized in Table 16.2. The conceptual stope shapes were refined as necessary in order to minimize the amount of planned dilution and to meet practical mining constraints.

Table 16.2 Stope Design Parameters

Parameter	Units	VOK Zone			West Zone		
		Standard	Weathered*	Sill Pillar	Standard	Weathered*	Sill Pillar
NSR Cut-off	$/t	180	180	180	180	180	180
Level Spacing	m	30	30	30	30	30	30
Stope Span	m	15	10	10	15	10	10
Minimum Mining Width	m	3	3	3	3	3	3
Minimum Waste Pillar Width	m	5	5	5	5	5	5
Minimum Footwall Dip	degrees	60	60	60	60	60	60
Minimum Hanging Wall Dip	degrees	60	60	60	60	60	60

Note: *Refers to stoping in weathered material immediately below the surface crown pillar. Weathered material extends 10 to 50 m below surface.

Individual areas meeting cut-off grade were evaluated against access development costs to determine economic viability, before including them in the mineral reserves. The LOM plan includes 782 stopes in the VOK Zone and 177 stopes in the West Zone. Figure 16.7 and Figure 16.8 are long-section views showing stope shapes generated by the MSO process.

Figure 16.7 Mineable Stope Shapes – VOK Zone

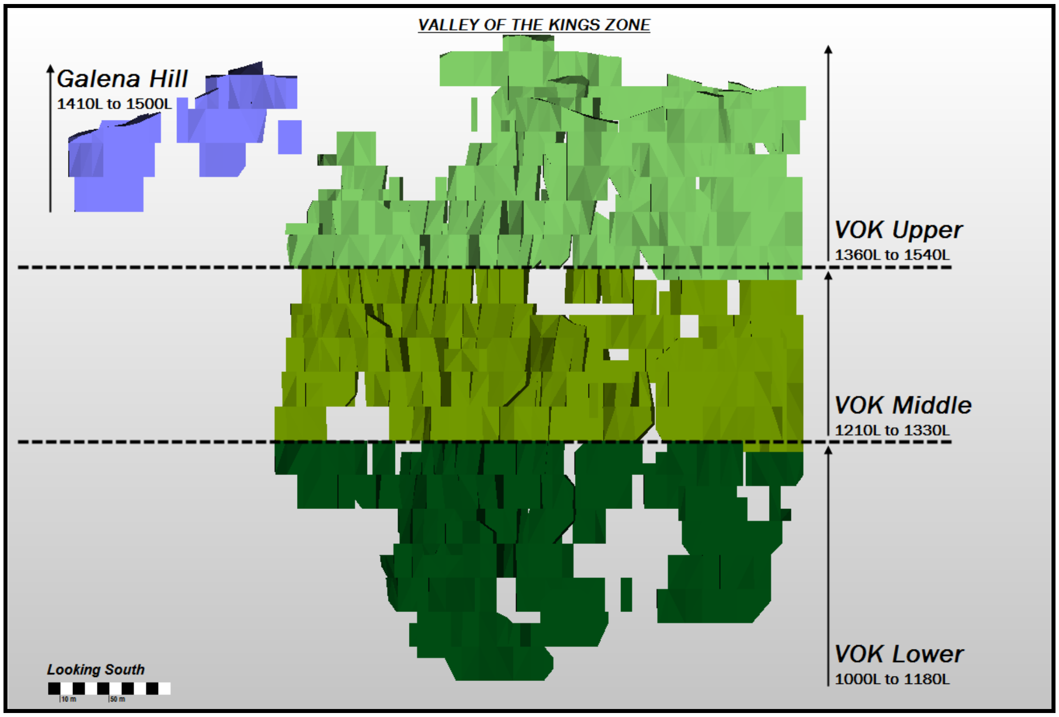
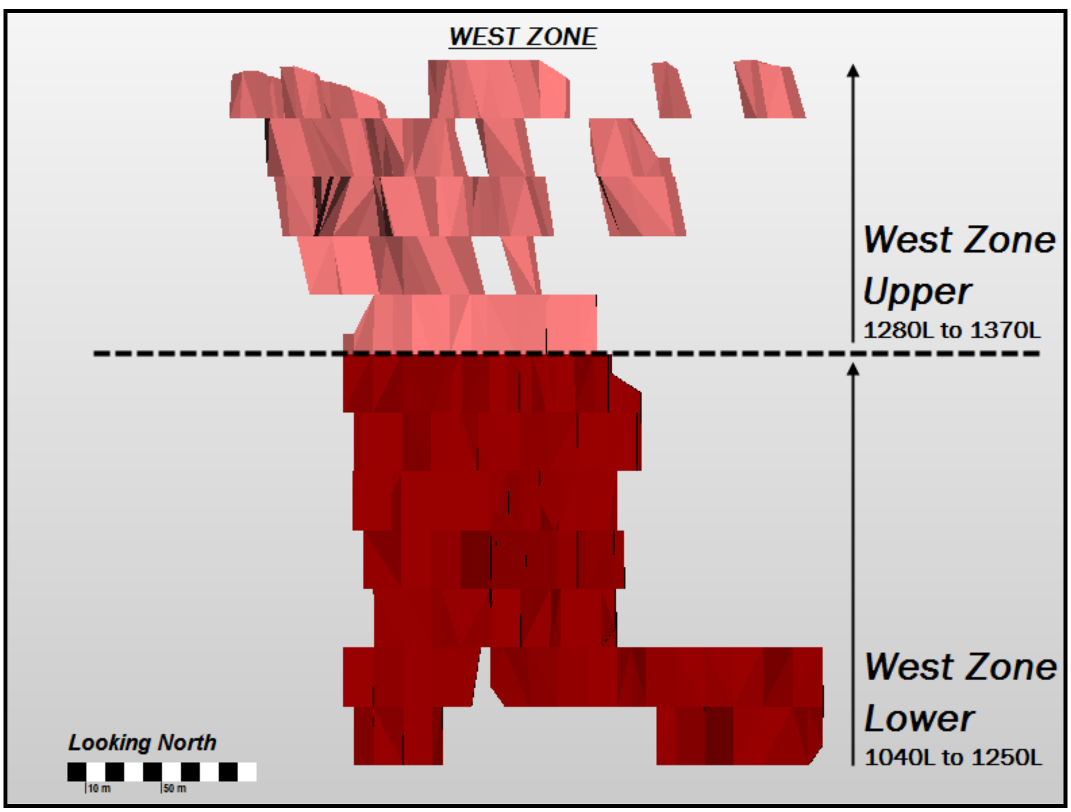


Figure 16.8 Mineable Stope Shapes – West Zone



16.3 MINING METHOD AND SEQUENCE

16.3.1 BLOCK DEFINITION

The orebody was divided into six logical blocks, defined by elevation, that facilitate 2,700 t/d of production through the creation of multiple working areas. Mining progresses upward from the lowest elevation in the block.

A number of factors that impact value and cash flow were taken into consideration when defining the block elevations:

- Three of the six blocks contain sills: Mining these sills will expose the cemented backfill of the stopes in the block immediately above. They will be relatively more problematic to mine due to the effects of increasing ground stress and the overhead fill, with lower recovery, higher dilution, and higher costs anticipated. Therefore, the block elevation selection tended toward minimizing the grade and contained metal within the sills.
- Pre-production development requirements: The first block to be scheduled for production is the VOK Zone is 1,360 Level to 1,540 Level. The block is close to the existing workings. This will allow the production ramp-up schedule to proceed in a reasonable time.
- Grade profile: Block definition impacts grade accessibility over time. The arrangement provided for the feasibility study assists with achieving higher grades earlier in the mine life.

16.3.2 STOPE CYCLE

The primary mining method will be transverse LHOS based on a standard primary/secondary sequence. No permanent pillars will be required and maximum ore extraction will be targeted.

The hanging wall drives will be completed, and a through ventilation circuit will be established before mining begins between any two levels.

A cross-cut will be driven from the hanging wall drive, through the centre of the stope, to the far ore contact on the undercut and overcut levels. The undercut level will have already been developed if stoping has progressed beyond the block starting level.

Cross-cuts on both levels will be cable-bolted from the central access to pre-support the roof prior to full-width slashing of the entire stope footprint. Slashing to the adjacent stope boundaries will expose paste fill walls in the case of secondary stope extraction.

Full width slashing will permit parallel production hole drilling across the entire width of the stope, and will reduce the potential for ore in stope corners to fail to break to design due to inadequate free face or poor explosives distribution. Ore recovery will be higher than a fan drilling alternative (in the absence of full-width slashing). Given the significant value of Brucejack ore, high recovery was an overriding criterion in the design.

Once the stope footprint is slashed out, a 750 mm pilot hole will be drilled in the slot raise location. Production drilling will follow in the raise and slot area, followed by the production rings as drilling progresses towards the near ore contact.

The raise and slot are generally opened in five firings or less. Production blasting and mucking will proceed cyclically until the stope is depleted and all ore has been mucked out. Transverse LHOS is a non-entry method, with remote mucking of blasted ore required once the drawpoint brow is open to the extent where the operator may be exposed to uncontrolled sloughing from the stope cavity.

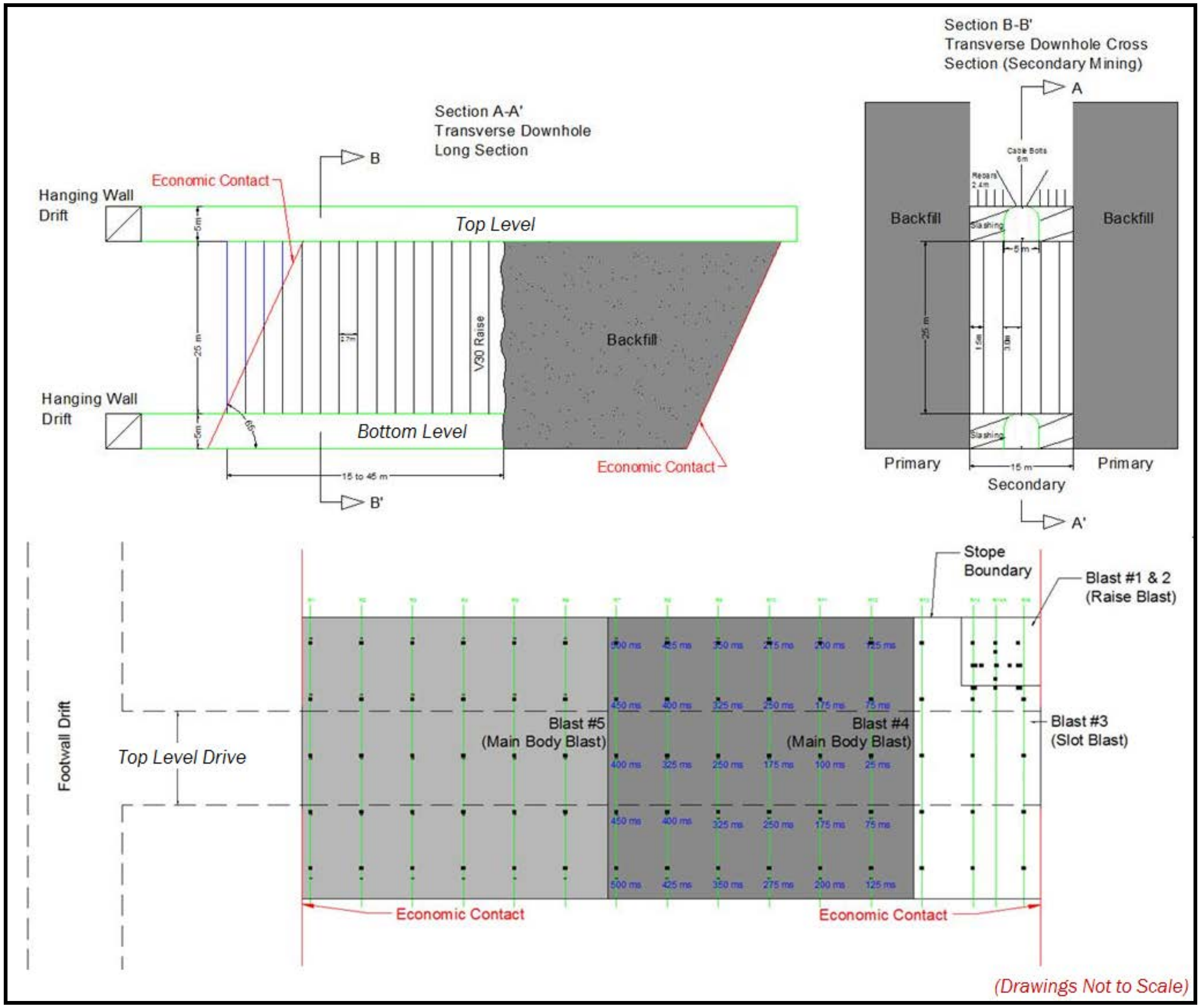
The empty stope will be remotely surveyed with cavity monitoring equipment. A barricade will be constructed in the drawpoint and the stope backfilled to just below the floor elevation of the top level. Crushed aggregate or ROM waste may be spread over the fill surface to reduce backfill dilution and increase trafficability of mucking equipment for the next lift of the stope.

The mining of sills and other areas, where top access is not available, will proceed in a similar manner; however, raise development and production drilling will be performed via uppers drilling from the bottom level. Figure 16.9 illustrates the typical LHOS design.

Longitudinal LHOS will also be employed at the Property, although significantly less ore tonnage will be recovered by this method in comparison to transverse LHOS. The longitudinal method will be used in thinner areas of the orebody, where the thickness of mineralization is less than 15 m, to avoid excessive access waste development. In contrast to transverse LHOS, mining will progress along the strike of the orebody to a common access point.

The overcut and undercut will be slashed to the footwall and hanging wall contacts. In all other respects, the stope cycle will be similar to transverse LHOS.

Figure 16.9 Typical LHOS Design

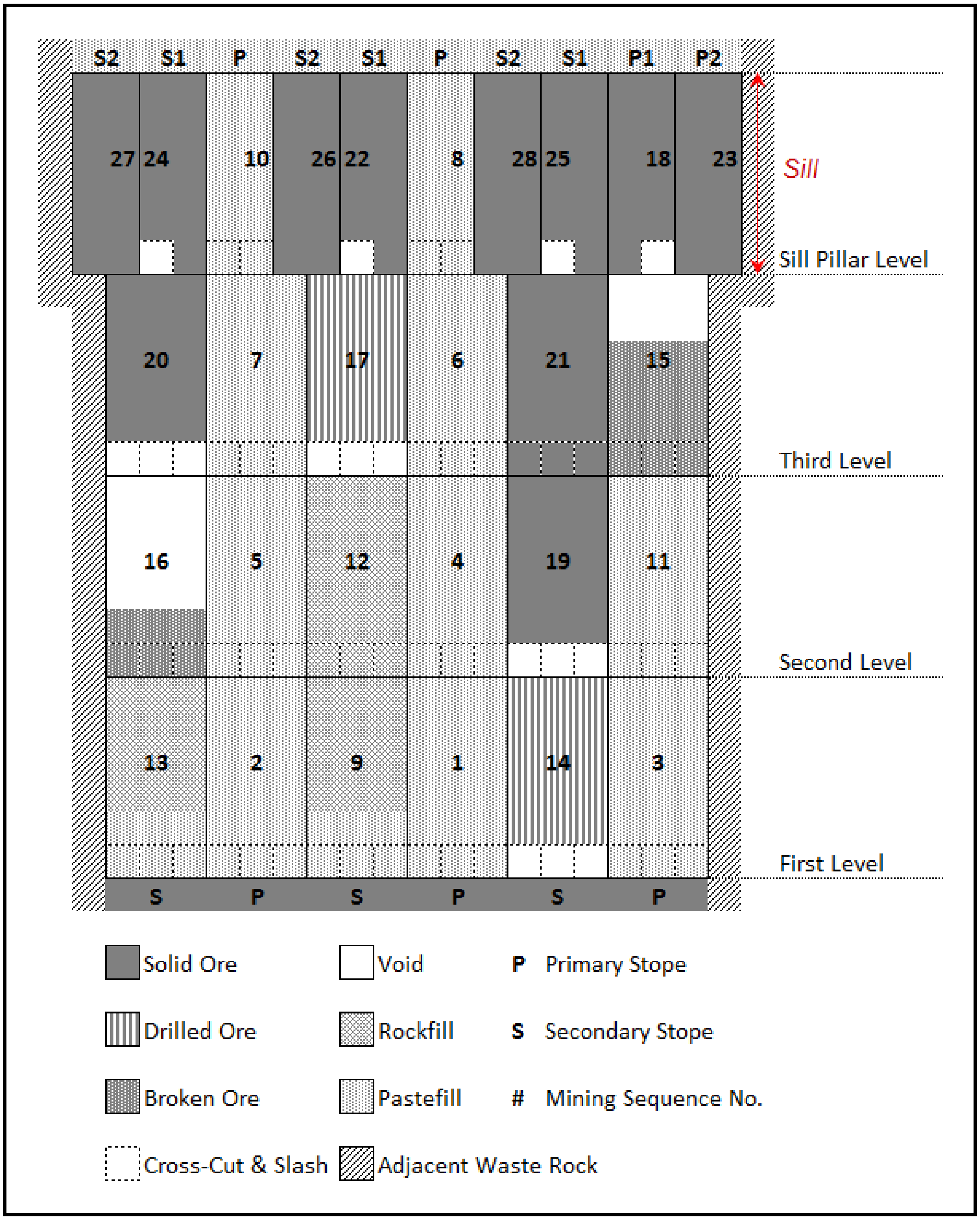


16.3.3 STOPE SEQUENCE

The mining sequence in any lens of a given block begins with the extraction of the primary stopes on the first (lowest) level. Wherever possible, the first primary stope will be located near the middle of the lens to develop a pattern of stope extraction that moves outwards to the extremities of the lens while progressing upwards towards the top. This generally promotes a favourable redistribution of ground stress, although many smaller lenses in the Brucejack orebody are either irregular in shape or of insufficient dimensions to properly develop this sequence.

When the adjacent primary stopes from the level above have been filled and cured, secondary stoping will commence. Figure 16.10 illustrates typical sequencing for the more massive lenses at the Property.

Figure 16.10 Example of Primary/Secondary LHOS at Brucejack Mine



16.3.4 Backfilling

The primary means of backfilling at the Property will be paste fill, generated from unclassified mill tailings mixed with adequate cementitious binder, to meet the strength requirements of re-exposure. Regular strength paste fill is commonly required where there will be re-exposure of vertical stope walls.

Stopes that will not be re-exposed by adjacent mining may be backfilled with unconsolidated waste and/or by paste fill with sufficient binder to remove any risk of future liquefaction (low-strength paste fill). High-strength paste fill will be required in the lower portion of all primary and secondary stopes that will be undercut by sill extraction from below. Table 16.3 tabulates the total projected paste fill volumes over the LOM by strength requirement and by binder dosage.

Table 16.3 LOM Paste Fill Requirements

Paste Type	LOM Quantity (m^3)	28-day Strength (kPa)	Binder Dosage (%)	Density Dry Paste (t/m^3)	Mass Dry Paste (t)	Binder Required (t)
High-strength Paste	139,000	800	5.5	1.46	203,000	11,000
Regular Paste	4,166,000	300	3.9	1.46	6,082,000	234,000
Low-strength Paste	1,714,000	100	2.8	1.46	2,503,000	69,000
Total	**6,019,000**	-	-	-	**8,788,000**	**314,000**

16.3.5 PASTE BACKFILL TEST WORK

Pretivm engaged AMC to undertake the first stage of a high-level study on the suitability of using mill flotation tailings for paste fill at the Brucejack mine. The results showed a higher-then-expected cement requirement for the range of required paste fill strengths. The density of the paste was low and resulting strengths required higher-than-expected cement content to achieve the target strengths.

Pretivm also engaged AMC to undertake second-stage laboratory testing. Stage 2 test work aimed to identify other classes of binders that would achieve target strengths at lower dosing rates. In particular, Stage 2 investigated the use of blended blast furnace slag and fly-ash with cement, as possibly better paste mix recipes.

The Stage 2 test work program included the following:

- material characterization tests in areas such as specific gravity and particle size distribution
- determination of paste fill density at a yield stress of 250 Pa as the benchmark for the paste fill mix
- unconfined compressive (UC) tests of mixes using General Purpose (GP) cement, slag, and fly-ash blend cements to look at the effect of adding fine ground iron blast furnace slag and fly-ash to the GP cement binder. Two slag blends were tested: MineCem (MC) containing 55% slag and Sunstate Slag Blend (SS) containing 35% slag. Medium-size fly-ash (FA) was also used.

As shown in Table 16.4, the Brucejack tailings paste fill mixes responded very favourably to the slag-based and fly-ash binders. The test program demonstrated a significant difference in the strength values for the paste fill mix with GP cement compared to the slag-based (MC and SS) and FA mixes. The following differences were noted:

- At 6% and 10% addition, consistently using MC binder (slag content 55%) produced a paste fill strength of more than double that of the GP mix.
- At 6% and 10% addition, the SS binder (slag content 35%) consistently increased paste fill strengths by over 50% compared to the GP mix.
- Using FA in the paste fill mixes reflects the expected lower strength gain in the early curing time (14 days) typical of FA mixes. However, the 28-day and final 56-day strengths steadily gained higher strength levels, showing the benefit of the FA in partly replacing the GP cement.

Table 16.4 Summary of UC Results

Batch	Tailings (%)	Cement/Binder	14 days	28 days	56 days
1	94	6% GP	405	448	565
2	90	10% GP	875	1,038	1,204
3	94	6% MC	909	1,145	1,428
4	90	10% MC	2,008	2,507	2,783
5	94	6% SS	577	738	903
6	96	10% SS	1,525	1,831	1,920
7	94	3% GP + 3% FA	340	537	681
8	90	5% GP + 5% FA	1,050	1,824	2,415

An outcome of the Stage 2 test work is the recommendation for further test work to determine appropriate binder dosages. For this study, AMC adopted industry standard dosages to achieve the required 28-day strengths, as outlined in Table 16.3.

16.3.6 Waste Management and Stope Filling

Considerable waste rock will need to be disposed of on an ongoing basis throughout the mine life.

Stopes will be filled with development waste wherever possible, but some waste will inevitably be hauled to surface for disposal in Brucejack Lake. All waste generated before the start of secondary mining must be hauled to surface given that it is unsuitable for backfilling primary voids without a cementitious binder.

It is normal that disused headings in mined-out areas are used for development waste disposal, and an allowance has been made in the waste disposal profile in this respect.

The disposal of waste rock in underground stopes has the effect of reducing the total void volume requiring paste backfill, and hence reduces the percentage of mill tailings that can be returned to underground. Table 16.5 tabulates the volumes of waste to be generated from milled ore and development headings, and the destination of these volumes over time. Over the LOM, 54% of development waste and 46% of tailings generated from milled ore will be placed back underground. The balance will be disposed of in Brucejack Lake.

Table 16.5 LOM Backfilling – Waste Rock and Mill Tailings

Year	Ore Tonnes ('000 t)	Total Tailings ('000 t)	Waste Tonnes ('000 t)	Waste Fill Volume (m^3)	Paste Fill Volume (m^3)	Tailings Underground ('000 t)	Waste to Surface ('000 t)
-2	5	-	575	-	-	-	575
-1	241	-	492	-	-	-	492
1	566	771	442	6,000	192,000	272	430
2	937	894	292	65,000	252,000	358	170
3	979	929	291	96,000	265,000	377	112
4	981	938	294	81,000	246,000	349	143
5	983	939	266	114,000	286,000	406	53
6	986	943	251	108,000	207,000	294	50
7	985	945	108	46,000	321,000	456	22
8	985	945	336	119,000	299,000	425	113
9	980	942	278	119,000	265,000	376	56
10	991	951	147	63,000	307,000	436	29
11	978	936	151	65,000	305,000	433	30
12	979	930	142	61,000	282,000	400	28
13	982	936	105	45,000	286,000	406	21
14	987	946	74	32,000	364,000	517	15
15	987	937	33	14,000	373,000	530	7
16	979	934	15	7,000	371,000	527	3
17	982	928	25	11,000	394,000	560	5
18	949	886	16	7,000	383,000	544	3
19	501	466	10	4,000	190,000	269	2
20	495	462	18	8,000	174,000	248	4
21	404	379	6	2,000	202,000	286	1
22	144	135	3	1,000	55,000	78	1
Total	18,986	18,072	4,369	1,074,000	6,019,000	8,548	2,366

16.4 Development and Production Schedule

16.4.1 Production Rate

During the early stages of the feasibility study, AMC performed a stope cycle time analysis on the average stope size for a select number of lenses. This analysis was completed to ascertain average production rates over the lens life, within the constraints of mining sequence. The findings were projected to all lenses in the orebody to form a preliminary aggregate schedule.

The preliminary schedule was further refined to maintain a constant production rate and optimize the grade profile over the LOM, although the level of detail remained at a high

level. The schedule indicated that a steady state production rate between 2,700 t/d and 3,000 t/d would be achievable.

In consultation with AMC, Pretivm approved the conservative end of the estimate, 2,700 t/d, as the target rate. Further scheduling performed on the detailed mine design confirmed the 2,700 t/d rate to be achievable with a high level of confidence.

16.4.2 Pre-production Development

Pre-production underground development will occur over an approximately 24-month timeframe before the first stope is extracted. Development drive ore produced during this period will be hauled directly to a surface stockpile pending commissioning of the conveyor hoisting system. Crushing of this early ore will be performed on surface.

The development strategy targets the VOK upper block as the first priority, followed by the more distant middle and lower blocks to sustain production. Development and construction of significant mine infrastructure including the declines, 1,330 Level workshop area, and crusher will be accomplished in parallel with the development of the VOK orebody.

Development of the West Zone will be deferred until the second half of the mine life, given the significantly lower-grade of mineralization.

The first stopes will be extracted between the 1,360 Level and 1,390 Level of the VOK Zone area. Critical path pre-production activities include:

- establishment of exhaust raise VR2 to support the development of the crusher area and workshops on the 1,330 Level
- access development to the top and bottom of the crusher chamber excavation and support of the crusher chamber, and installation of the crusher
- twin decline development from underground to the surface portal; when breakthrough is established, the ease of transport of materials and equipment for construction activities will be improved and conveyor installation can commence
- excavation and construction of the maintenance workshops, magazine, fuel bay, and other ancillary installations
- development of the VOK ramp to the 1,360 and 1,390 levels, development of the levels, and establishment of exhaust raise VR3.

Figure 16.11 illustrates the extent of development required for the main onset of stope production between the 1,360 and 1,390 levels. A total development requirement of 14,504 lateral metres and 780 vertical metres is planned in the first 24 months. Up to 660 m/mo of development advance will be required at the peak activity level. Relative to company strategy and goals beyond the feasibility study, AMC believes that opportunities may exist for further optimization of the development and production scheduling.

Figure 16.11 Extent of Mine Development at the Main Onset of VOK Stoping

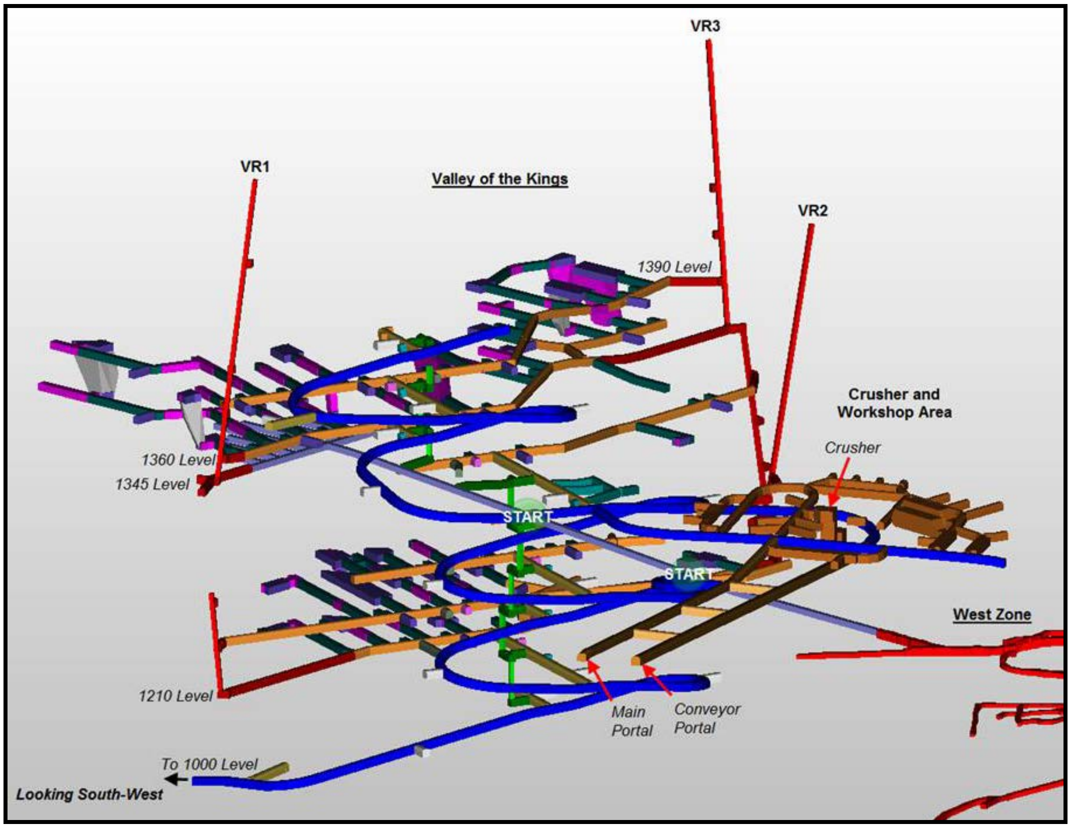
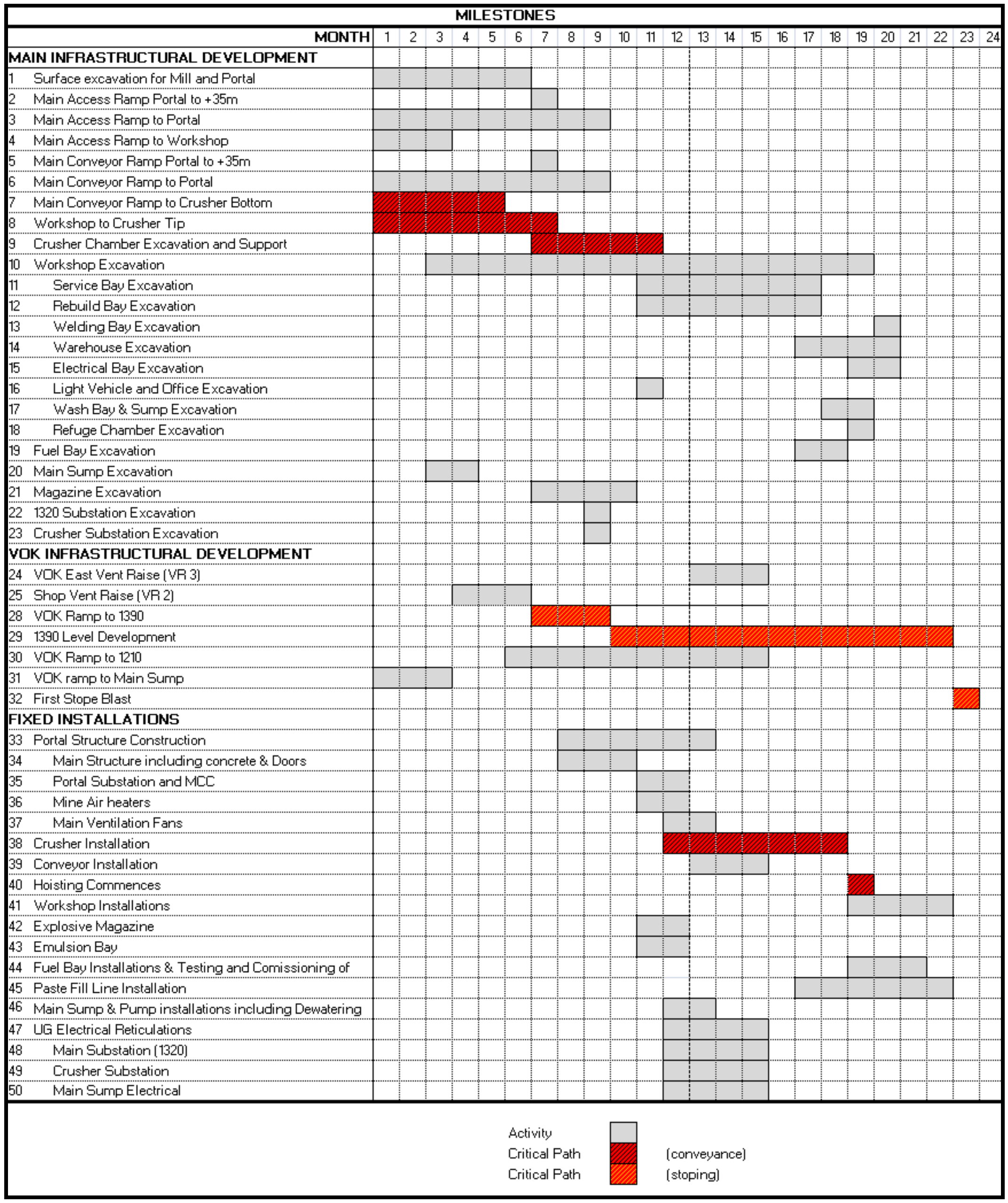


Figure 16.12 presents the critical path activities leading to the commissioning of the material handling system and initial stoping in the VOK Zone area.

Figure 16.12 Critical Path Construction and Development Activities



16.4.3 SUSTAINING DEVELOPMENT

Development of the VOK upper block alone is insufficient to ramp up and sustain 2,700 t/d of ore production. The VOK middle block must also be developed as a critical path activity. The following development will run in parallel with the VOK upper block development and mining, and will continue until the VOK middle block begins producing critical stope ore in the third year of activity:

- advancement of the VOK ramp downward to the 1,210 Level

- development of the 1,210 and 1,240 levels
- establishment of exhaust raises VR1 and VR3 from the 1,210 Level, connecting with existing portions of these raises in the VOK upper block.

The VOK ramp development will not be interrupted at the 1,210 Level, but will advance continuously to the bottom of the mine (1,000 Level) to access the high-grade ore hosted in the VOK lower block. The results of this strategy, which promote higher grades as early as possible in the mine life, are reflected in the generally declining trend of the LOM grade profile (Table 16.6).

Table 16.6 LOM Development Requirements

Year	Capital		Operational		Total	
	Lateral (meq)	Vertical (m)	Ore (meq)	Waste (meq)	Lateral (meq)	Vertical (m)
-2	6,188	285	57	286	6,531	285
-1	4,147	494	2,049	1,741	7,937	494
1	3,721	123	1,466	1,729	6,916	123
2	2,014	100	2,525	1,717	6,256	100
3	2,066	164	2,236	1,623	5,925	164
4	1,771	75	2,646	1,939	6,356	75
5	1,117	82	2,296	2,412	5,826	82
6	1,236	63	2,923	1,982	6,141	63
7	610	67	2,061	823	3,495	67
8	2,570	137	2,338	1,291	6,199	137
9	2,306	100	1,904	889	5,100	100
10	871	382	2,568	835	4,274	382
11	1,023	49	1,961	938	3,922	49
12	682	75	2,397	1,176	4,256	75
13	394	22	2,402	973	3,770	22
14	298	-	2,498	677	3,472	-
15	-	-	1,851	436	2,287	-
16	-	-	1,518	203	1,721	-
17	-	-	1,624	332	1,956	-
18	-	-	1,267	217	1,484	-
19	-	-	795	128	924	-
20	-	-	908	236	1,144	-
21	-	-	240	77	316	-
22	-	-	181	38	219	-
Total	31,014	2,217	42,712	22,697	96,423	2,217

Note: meq = metres equivalent

16.4.4 LOM Production Schedule

Full 2,700 t/d production is effectively achieved in Year 2, the fourth year of project activity.

Figure 16.14 illustrates the ramp-up to full production tonnage as the various blocks are brought into production.

Table 16.7 is a summary of projected LOM production tonnes and grade.

Figure 16.15 shows the LOM split of production by development, slashing, and stoping.

Figure 16.13 Life of Mine Production Schedule by Mining Block

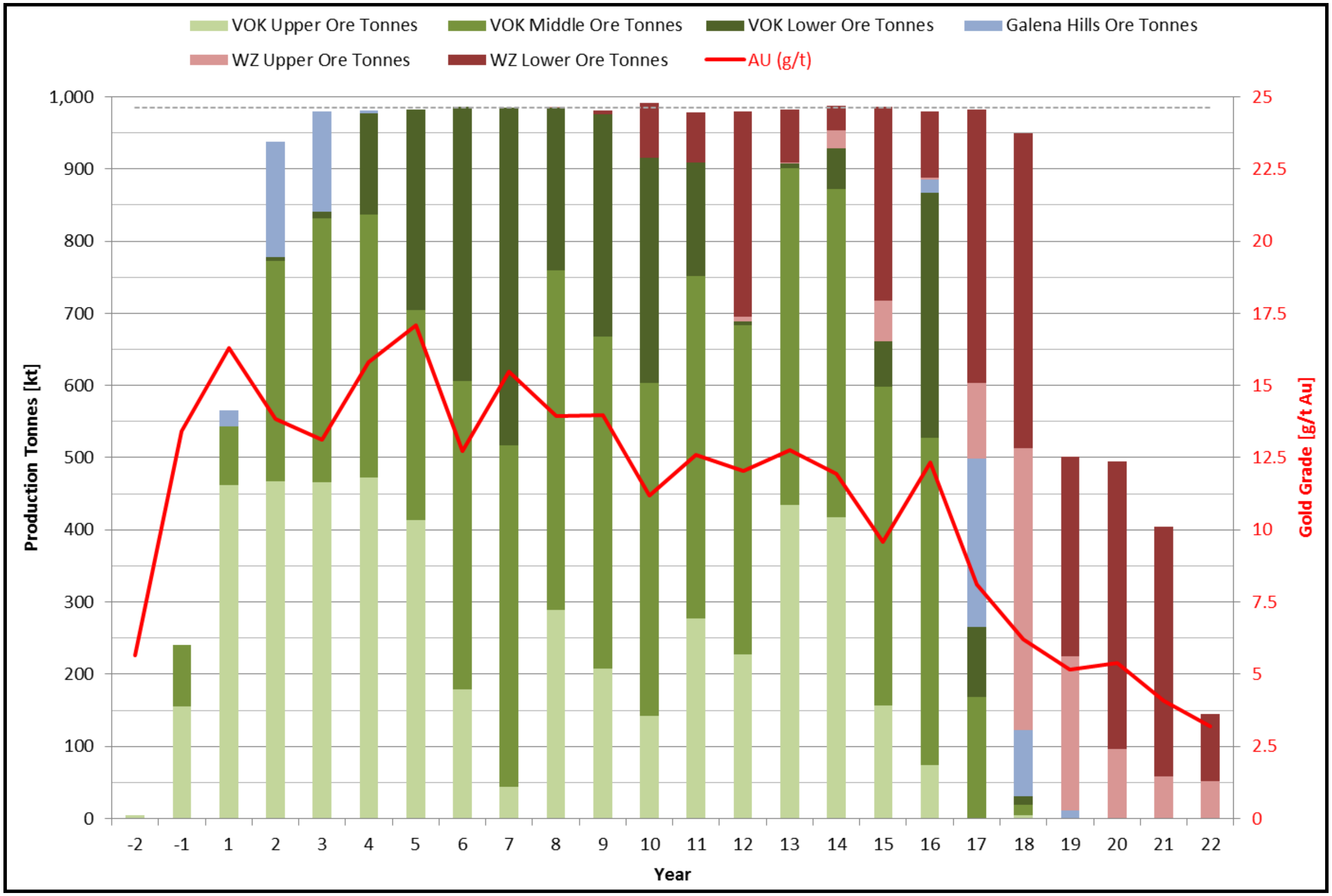


Figure 16.14 LOM Production Schedule by Activity

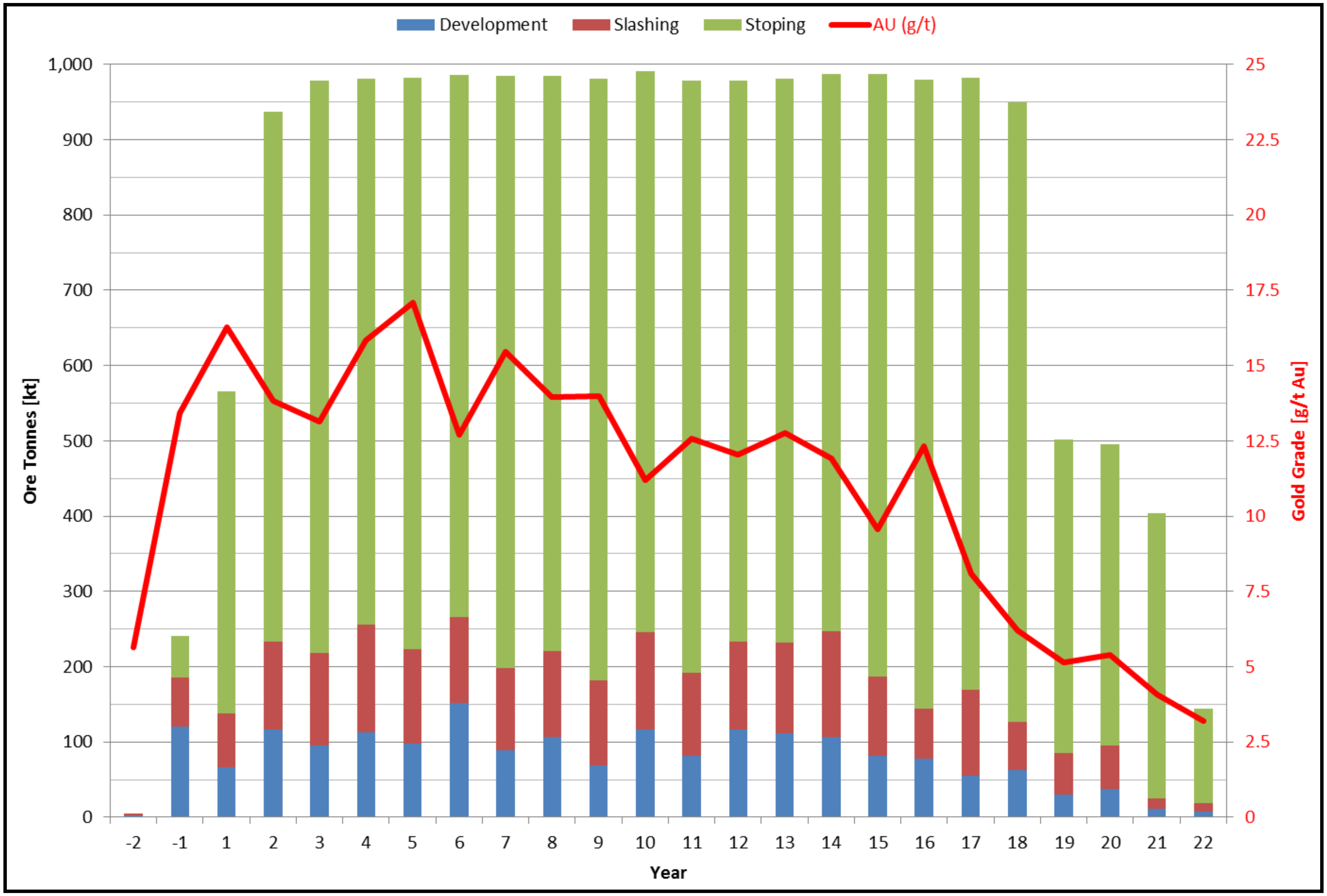


Table 16.7 LOM Tonnes and Grades

Year	Ore (Mt)	Au (g/t)	Ag (g/t)
-2	0.0	5.7	10
-1	0.2	13.4	11
1	0.6	16.3	13
2	0.9	13.8	11
3	1.0	13.1	11
4	1.0	15.8	12
5	1.0	17.1	14
6	1.0	12.7	9
7	1.0	15.5	11
8	1.0	14.0	10
9	1.0	14.0	11
10	1.0	11.2	18
11	1.0	12.6	14
12	1.0	12.1	55
13	1.0	12.8	19
14	1.0	11.9	20
15	1.0	9.6	60
16	1.0	12.3	36
17	1.0	8.1	98
18	0.9	6.2	237
19	0.5	5.2	289
20	0.5	5.4	333
21	0.4	4.1	282
22	0.1	3.2	269
Total	**19.0**	**12.0**	**58**

16.5 GEOTECHNICAL

16.5.1 OVERVIEW

Geotechnical designs and recommendations are based on the results of site investigations and geotechnical assessments, which included rock mass characterization, structural geology interpretations, excavation and pillar stability analyses, and ground support design.

Geotechnical site investigations completed to support the underground rock mechanics assessments included: geotechnical drilling and logging, oriented drill core measurements, borehole televiewer surveys, laboratory testing of rock core samples, and installation of borehole instrumentation to measure groundwater pressures. Geotechnical mapping of the dewatered historic underground workings was completed to provide structural geology information; the geotechnical performance of excavations in

the existing mine were also reviewed. The feasibility study site investigations were supplemented by a review of historical reports and inclusion of data collected during previous site investigation programs.

16.5.2 Rock Mass Properties

The rock mass of the Brucejack area was divided into eight geotechnical units based on characteristics of the rock mass.

The geotechnical units in the West Zone, in order of increasing competence, are as follows:

- The West Zone Fault Zone (WZ FZ) unit includes fault-disturbed rock. This unit is strong (according to the methods of ISRM (1978)) with fair rock quality designation (RQD) (Bieniawski 1976) and close to moderate discontinuity spacing.
- The West Zone Weathered Rock Zone (WZ WRZ) unit includes weathered near-surface rock. It is medium strong with good RQD and moderate discontinuity spacing.
- The West Zone Fresh Rock (WZ FR) unit comprises all remaining rock, which is very strong with excellent RQD and wide discontinuity spacing.

The geotechnical units in the VOK, in order of increasing competence, are as follows:

- The VOK Fault Zone (VOK FZ) unit includes fault-disturbed rock. The Fault Zone unit includes Brucejack Fault Zone rock and rock from all geologic units. It is strong with good RQD and close discontinuity spacing.
- The VOK Weathered Rock Zone (VOK WRZ) unit comprises near-surface weathered rock. This unit is strong with good RQD and close discontinuity spacing.
- Rock mass VOK Domain 1 (VOK D1) comprises the Argillite (ARG) geologic unit and is very strong with good RQD and moderate discontinuity spacing.
- Rock mass VOK Domain 2 (VOK D2) comprises the Porphyry (P1) and Silicified Rock (RHY) geologic units, which are strong with excellent RQD, and moderate discontinuity spacing.
- Rock mass VOK Domain 3 (VOK D3) comprises the Jurassic Conglomerate (JR), Triassic Sediment (TRS), and Andesite (ANDX) units, which are very strong with excellent RQD and wide discontinuity spacing.

Rock mass parameters used in design are summarized in Table 16.8.

Table 16.8 Rock Mass Properties

Unit	UCS (MPa)	GSI*	Unit Weight** (kN/m³)	m_i	m_b	S	E_{rm} (GPa)
VOK FZ	89	60	26.3	12	1.110	0.0023	5.13
VOK WRZ	50	63	28.6	17	1.879	0.0037	0.77
VOK D1	116	72	27.2	17	3.211	0.0144	9.76
VOK D2	95	70	27.1	19	3.186	0.0106	9.02
VOK D3	73	85	27.3	26	10.647	0.103	14.37
WZ FZ	77	57	26.3	12	0.928	0.0015	4.27
WZ WRZ	37	62	28.6	17	1.771	0.0032	0.73
WZ FR	116	85	27.3	21	8.599	0.103	16.77

Notes: *GSI are calculated from median rock mass parameters for each unit, where GSI = RMR '76.
**Unit weights are based on average results of specific gravity testing when possible.
The Hoek-Brown failure criteria were estimated assuming a disturbance factor ('D') of 0.8 for all units.
The Hoek-Brown curves were derived using a sigma3 maximum for a tunnel depth of 650 m.

16.5.3 BRUCEJACK FAULT ZONE

The Brucejack Fault lineament is currently the only known major structure that intersects the proposed mining footprint. It is a northerly striking anastomosing fault zone located along the western margin of the study area and extends north to the Iskut River Fault. In places the lineament appears to be several sub-vertical to moderately (greater than 60°) dipping fault strands braided together. The zone has normal faulting with variable displacement estimated at 500 to 800 m (ERSi 2010).

At the time this report was prepared, Pretivm had not yet developed a structural geology model for the Project. BGC reviewed drillhole data and core photographs in relation to the Brucejack Fault surface used in the PEA (Silver Standard 2010) to characterize the properties of the Fault Zone and review its proximity to the proposed mine workings. BGC's work focused on identifying the orientation, thickness, and rock mass characteristics of the Fault Zone. These interpretations were combined with the PEA fault surface to develop an updated 3D Brucejack Fault Zone surface, which was provided to Pretivm and the mine planning group to assist with ongoing project planning.

The Brucejack Fault Zone is comprised of a core of highly fractured rock with a zone of less fractured, fault-disturbed rock mass on either side. The width of the fault zone varies with depth and along strike from approximately 5 to 40 m. It is considered to be continuous along strike, and dips slightly to the east above the 1,325 m elevation, and dips slightly west below the 1,325 m elevation. For design purposes, the median RQD, Joint Condition, and point lead index Is50 value (ISRM 1985) are 62%, 16, and 3.5 MPa, respectively, compared to the ”excellent” median RQD value (91%) and median Is50 value of 6.5 MPa in the surrounding undisturbed VOK D2 rock mass.

16.5.4 Underground Rock Mechanics

Stope Design Criteria

Rock mechanics analyses were completed to estimate achievable spans for the proposed mine openings. Stope stability analyses for the observed lower quartile (“conservative”, Q’ = 10) and median (“base case”, Q’ = 40) rock masses were completed. The recommended maximum unsupported hydraulic radii vary from 1.9 to 3.1 for the backs and from 6.2 to 11.0 for the hanging walls, for the conservative and base case designs, respectively. The recommended maximum supported hydraulic radii vary from 4.1 to 5.6 for the backs and from 10.0 to 14.5 for the hanging walls, for the conservative and base case designs, respectively.

A preliminary MAP3D numerical model developed for the VOK Zone shows stress concentration and yielding proximal to the dense stope clusters in the middle of the VOK Zone, indicative of potential instability in the stope hanging walls and footwalls. This indicates some potential for increased dilution. Cable bolts could be installed into the hanging walls of dense stoping blocks to "tie" the hanging wall together until backfill is placed, to help reduce this dilution. Note that currently, the model is not considered sufficiently calibrated for quantitative design.

Stand-off Distances

Minimum stand-off distances between excavations of 10 m, 25 m, and 50 m are recommended for the raises, ramps, and underground crusher, respectively. The recommended stope stand-off distance from all hanging wall drives is 25 m. The proposed portal decline will be twinned, with a recommended minimum pillar thickness of approximately 10 m between the two excavations.

Rib Pillars

The rib pillars between cross-cuts were designed to be in waste and will not be recovered, but are considered temporary based on the short-term lifespan required for access to a given stope. The minimum recommended pillar width to height ratio for cross-cut rib pillars for the “base case” stope design is 1.1:1.0, and 3.3:1.0 for the “conservative case” stope design. If cross-cuts are developed within the weathered zone, the recommended rib pillar width to height ratio is 1.7:1.

The rib pillars between the open stoping blocks are intended to give temporary support to the mining block until the primary stopes are backfilled and the pillar can be recovered in the form of a secondary stope. Using the pillar stability graph method developed by Hudyma (1988) and tributary loading theory, the minimum recommended secondary stope span (rib pillar thickness) to primary stope span for the “base case” stope design is 1:1 for sublevel intervals of 30 m.

Analysis of the “conservative case” shows that high stresses may develop in the pillar core, and that some spalling and dynamic rockmass damage may be expected. This may result in spalling in 25% of rib pillars, and difficult drilling in approximately 25% of secondary stopes. If stopes are developed within the weathered zone, the minimum recommended secondary stope span (rib pillar thickness) to primary stope span is 1.5:1.

AMC estimated ore recovery in the weathered zone to be 75% in anticipation of stress-induced mining difficulties. Furthermore, the operational expenditure was escalated for ore in the weathered zone in consideration of the increased ground support requirements.

Analysis of the weathered zone stopes shows low confinement; ground support including resin-grouted rebar, mesh-reinforced shotcrete, and straps may be required to confine the pillar rock mass and prevent unravelling. Many of the near-surface stopes will actually extend below the weathered zone into the fresh rock, which will reduce the potential for rib pillar instability.

SILL PILLARS

The current design sill pillar thickness is 30 m. The numerical modelling analysis shows some relaxation in larger stope hanging walls, and stress concentrations in sill pillars within areas of the mine with denser stoping. The model shows that the bottom-up sequence concentrates stress in both VOK Zone sill pillars. Yielding is likely to occur prior to recovering the entire sill pillar, and therefore achievable sill pillar recovery may be less than 100%. The West Zone sill pillar is interpreted to be stable except for stress concentration in the sill pillar abutments. Stress concentration in pillar abutments is common in mines using centre-out sequencing, and does not necessarily indicate stability problems prior to full extraction of the sill pillar.

GROUND SUPPORT REQUIREMENTS

The structural stability of the proposed excavations was analyzed using an empirical design chart after Grimstad and Barton (1993) and Unwedge© (Rocscience 2003) to develop minimum ground support recommendations. Ground support analyses for primary (permanent "man-entry") and secondary (temporary "development") headings were conducted in each structural domain. Ground support recommendations are provided in Table 16.9.

Table 16.9 Ground Support Recommendations

Opening Type	Cross Section (w by h, m)		Ground Support Type	Length (m)	Spacing (m)	Shotcrete Estimate (%)	Additional Notes
Main access decline, ramps, and other haulage routes	6 by 5.5	Back	Fully-grouted #7 Dywidag	2.4	1.8 by 1.8	10	-
		Walls	Fully-grouted #7 Dywidag	1.8	1.8 by 1.8	10	
Level development	5 by 5.5	Back	Swellex Pm12	1.8	1.8 by 1.8	10	Fully-grouted #7 Dywidag can be used in direct substitution of Pm12 when desired for operational efficiency.
		Walls	Swellex Pm12	1.8	1.8 by 1.8	10	
Intersections	Includes 6 by 5, 5 by 5, three-way, four-way, and herringbone layouts	Back	Pre-support: Fully-grouted #7 Dywidag	2.4	1.8 by 1.8	10	Welded wire mesh should be installed on the back and upper portion of each wall for all intersections with an effective span greater than 6 m. Strap consumption estimate: 25% of pillars; 3 straps per pillar.
		Back	Long support: Coupled fully-grouted #7 Dywidag or cable bolts	5.0	2.4 by 2.4	10	
		Walls	Fully-grouted #7 Dywidag	1.8	1.8 by 1.8	10	
Full-width undercuts	5 m high by 6 m wide pilot	Back	Pre-support: Swellex Pm12	2.4	1.8 by 1.8	N/A	All support must be installed prior to slashing.
		Back	Long support: Bulbed cable bolts	6	2.4 by 2.4	N/A	
		Walls	-	-	-	N/A	
	15 m wide full undercut (post-slash)	Back	Swellex Pm12	2.4	1.8 by 1.8	25	All support except for shotcrete must be installed as each lateral slash is developed (prior to full width exposure)
		Walls	Swellex Pm12	2.4	1.8 by 1.8	25	
Portal	-	Back	Fully-grouted #7 Dywidag	1.8	0.8 by 0.8	100	1 m spaced steel sets in first 30 m and 100% coverage with minimum 50 mm thick steel-fibre reinforced shotcrete throughout the length of the portal
	-	Walls	Fully-grouted #7 Dywidag	1.8	1.8 by 1.8	-	
Raises	3 by 3	All	Fully-grouted #7 Dywidag	1.2	0.8 x 0.8	50	Staggered spacing. Reduced support may be feasible if man-access is not permitted.

Notes: Design factor of safety is 1.3.
Wall bolts must extend down to within 1.5 m of sill (floor).
Surface support should be installed when excavation intersects relatively poorer ground, faults, more persistent joints or narrower joint spacing, soft joint walls, groundwater seepage points, or "dead" sounding difficult to scale material.
Use shotcrete estimate percentage for mesh cost estimating if mesh is preferred surface support.
All estimates are provided for cost estimating purposes only.

FULL-WIDTH UNDERCUTS

The Project mine planners proposed full-width undercutting of select stopes. Ground support recommendations are provided in Table 16.10. Primary stopes should be tight-filled as best as possible.

MINING THROUGH THE BRUCEJACK FAULT

The feasibility study mine plan has development (stope access drifts) sub-parallel to the Brucejack Fault Zone. At the next level of study, it is recommended that developments near or within the fault zone be aligned perpendicular to the fault trend to minimize the exposure of fault-disturbed rock.

All developments through the Brucejack Fault Zone will require support with fully grouted #7 Dywidag bolts on a 1.5 m square pattern. Full coverage (sill to sill) of welded wire mesh and 75 mm of fibre-reinforced shotcrete is also recommended.

Stopes should be excavated in isolation and backfilled prior to any other production openings within the fault zone. The rock mass within the fault zone is not competent enough to form adequate rib or sill pillar strength between stopes. In each case, stopes should be constrained to either the host or fault disturbed rock. Excavations bridging the boundary will have unplanned dilution along the contact.

The preliminary recommendation for maximum supported back hydraulic radii is 2.5 (10 m by 10 m), and maximum unsupported hanging wall hydraulic radii is 3.75 (10 m by 30 m), for stopes within the fault zone. Cable bolt support consisting of 6 m single or double strand bulbed cable bolts on a 2.5 m square spacing in the back is recommended.

SURFACE RAISE LOCATIONS

The finalized raise locations should avoid fault-disturbed rock, and minimize intersection of weathered rock. The recommended pillar thickness between a raise and nearby development or production openings, including the decline access ramp, is 10 m.

UNDERGROUND CRUSHER AND OTHER MINE INFRASTRUCTURE

The proposed crusher excavation will require upper level access for trucks and lower level access for conveyor egress. The proposed excavation will consist of an upper truck dump and rock breaker level, which connects via a bin and hopper system to the lower level crusher station.

A localized set of data was reviewed to estimate the geotechnical properties of the rock mass at the proposed crusher location. The rock mass is entirely within the VOK D2 geotechnical domain. The 25th percentile values were used for design.

The excavation was designed for a factor of safety of 2.0, as the excavation must remain operable for the life of the mine, and opportunities for rehabilitation will be limited once the mine is in production. Two levels of support are recommended:

- Primary support consisting of galvanized, resin-grouted rebar (or an equivalent) and welded wire mesh (or fibre-reinforced shotcrete). The purpose of these elements is

to support and retain material between the cable bolt plates and to provide a shell of near-surface support. In addition, confining surface support (e.g. steel or heavy gauge mesh straps) is recommended for all noses and benches within the excavation to reduce the potential for unravelling. Fibre-reinforced shotcrete is recommended when infrastructure will make rehabilitation impractical.

- Secondary support consisting of cable bolts in the back and walls of the excavation. The purpose of these elements is to support larger wedges and increase long-term stability of the excavation.

The crusher chamber excavation should be completed in stages from the top heading to allow sequential support installation and minimize the dimensions of unsupported spans. A minimum radial standoff distance of 50 m is recommended to prevent stress interaction between the crusher and development or production openings. This recommendation also applies to offset from major structures (i.e. the Brucejack Fault Zone).

The ground support recommendations for the underground crusher and other mine infrastructure excavations are summarized in Table 16.10.

Table 16.10 Mine Infrastructure Excavations – Ground Support Recommendations

Area	Dimension (height x width (along trend) by length) (m)	Trend/ Plunge of Excavation	Design Factor of Safety	Lifespan	Support
Cap Magazine	3.05 by 6.1 by 3.05	152/21	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure).
Crusher Chamber - Lower level (conveyor)	18.1 by 14.2 by 9.4	057/01	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS. Cable bolts: Walls: 5.0 m length, bulbed strand, 2.5 m square spacing Back: 5.0 m length, bulbed strand, 2.5 m square spacing
Crusher Chamber - Upper level (truck)	17.3 by 7.6 by 9.3	057/01	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.5 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS. Cable bolts: Walls: 5.0 m length, bulbed strand, 2.5 m square spacing Back: 5.0 m length, bulbed strand, 2.5 m square spacing
Electricians and Millwrights Shop	5.5 by 16.2 by 5.5	270/01	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure).
Fuel and Lube Station	4.5 by 35 by 8.5	243/06	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure). Cable bolts: Back: 5.0 m length, bulbed strand, 2.5 m square spacing
Service Bay, Maintenance Bay, and Tire Bay	11.5 by 42 by 10.0	005/00	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure). Cable bolts: Walls: 5.0 m length, bulbed strand, 2.5 m square spacing Back: 5.0 m length, bulbed strand, 2.5 m square spacing

table continues...

Area	Dimension (height x width (along trend) by length) (m)	Trend/ Plunge of Excavation	Design Factor of Safety	Lifespan	Support
Powder Magazine	7.1 by 14.1 by 6.4	152/01	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure). Cable bolts: Walls: 5.0 m length, bulbed strand, 2.5 m square spacing Back: 5.0 m length, bulbed strand, 2.5 m square spacing
Refuge Station and Offices	4.6 by 15 by 5.2	062/01	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure).
Warehouse	5.5 by 27 by 5.5	270/01	2	LOM	Galvanized, resin-grouted rebar (or equivalent); 1.8 m length; 1.75 m square spacing. Welded wire mesh, 100% coverage on back and walls, coated with minimum 2" SFRS (shotcrete only required if rehab will not be practical due to installed infrastructure).

Notes: LOM = life of mine assumed 20 to 25 years; SFRS = steel fibre reinforced shotcrete

CROWN PILLAR

To maximize crown pillar recovery, the minimum recommended crown pillar thickness for the West Zone and VOK Zone is 15 m, with a maximum recommended stope span of 10 m for all stopes immediately below the crown pillar. As the recommended maximum span is narrower than the transverse width of the mineralized zones, transverse stopes immediately below the crown must be tight-filled as much as practicable to reduce the potential for crown pillar collapse. The crown pillar should be supported with 5.0 m long single strand bulbed cable bolts on 2.5 m spacing.

PORTAL

The portal should be excavated with a minimum cover of 12 m of rock above the crown (back) of the tunnel excavation. The rock face should be excavated in two or more benches of equal height with 75° bench face angles and a 5 to 6 m horizontal bench between them. Resin-grouted rebar bolts, screen, and fibre-reinforced shotcrete should be applied to the portal face to retain loose rock over the portal entrance.

16.6 HYDROGEOLOGICAL/GROUNDWATER

16.6.1 OVERVIEW

Conceptualization of the groundwater flow system at the Brucejack site was required to provide estimates of groundwater inflow to the existing and future underground mine workings. These inflow estimates were used to size dewatering equipment and as input to the process water balance. The results of site investigations and hydrogeologic testing completed between 2010 and 2012 were used to better understand the groundwater flow system. Site investigations were completed to evaluate the hydrogeologic conditions (e.g. hydraulic parameters of the bedrock, hydrostratigraphic units, and hydraulic gradients) in the vicinity of the existing and proposed underground workings, and included hydraulic response testing (e.g. packer testing, slug testing) and the installation of groundwater monitoring wells and vibrating wire piezometers. Data collected during site investigations are supplemented by ongoing monitoring of groundwater elevations and collection of water quality samples, and by data collected during adit dewatering activities at the site.

Details regarding the investigations and data used to develop the conceptual and numerical hydrogeologic models are summarized in BGC's numerical hydrogeologic model report entitled "Brucejack Project Feasibility Study Numerical Hydrogeologic Model" and dated June 18, 2013.

16.6.2 CONCEPTUAL HYDROGEOLOGIC MODEL

Surface topography has a pervasive influence on the groundwater flow system at the site. The elevation within the immediate project area ranges from approximately 1,350 m at the outlet of Brucejack Lake to over 2,000 m at the highest peaks. Measured groundwater elevations suggest that the water table is a subdued replica of topography, with depths to groundwater typically greater in the uplands relative to the valley bottoms. Groundwater enters the flow system from an infiltration of precipitation and snowmelt,

with lesser components supplied by surface water infiltration in lakes. Groundwater discharge zones are generally restricted to lakes, creeks, gullies, and breaks in slope.

The hydrostratigraphy of the site is composed of a thin, discontinuous layer of glacial till or colluvium underlain by bedrock. Thicker overburden deposits are confined to local sections of the valley bottom and are not present in the vicinity of the proposed underground mine.

Bedrock of the Project area can be broadly divided as follows:

- Triassic marine sedimentary and volcanic rocks of the Stuhini Group
- Jurassic sediments and volcanics of the Hazelton Group
- early Jurassic dikes, sills, and plugs of diorite, monzonite, syenite, and granite, the most common of which are grouped as the "Sulphurets Intrusions".

A general trend of decreasing bedrock permeability with depth is observed site wide, though permeability varies by two to three orders of magnitude at any given depth. There is no apparent relationship between hydraulic conductivity and the major structure in the immediate vicinity of the Project area (the Brucejack Fault).

16.6.3 NUMERICAL MODEL DEVELOPMENT AND CALIBRATION

The conceptual model described in Section 16.6.2 was used as the basis for the development of a numerical hydrogeologic model, built using the graphical user interface Groundwater Vistas (ESI 2011), and MODFLOW-Surfact code (Harbaugh et al. 2000; HydroGeoLogic 2011). The numerical model was calibrated in stages to available hydrogeologic data collected within the study area, comprising 67 packer and slug tests within bedrock, 40 hydraulic head targets, and approximately 3 months of volumetric discharge data available from mine dewatering activities that occurred in 2011 to 2012.

Prior to predictive simulation runs, an additional run was completed to account for ongoing dewatering at the site; dewatering activities are expected to continue until the start of mining operations, with the water level in the existing adit maintained at an approximate elevation of 1,300 masl.

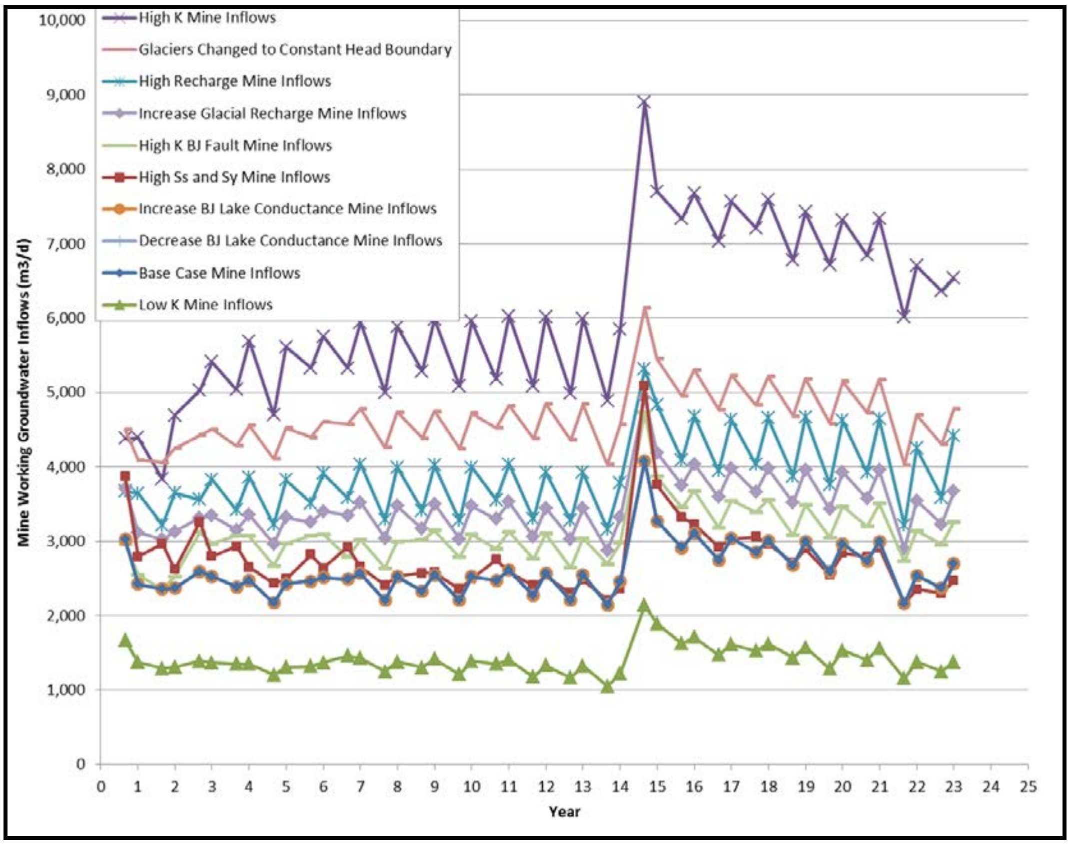
16.6.4 PREDICTIVE SIMULATIONS AND INFLOW ESTIMATES

Predictive simulations are based on a 21-year mine life, as stipulated in the underground mining plans provided by AMC on February 27, 2013. The underground mining stopes and associated developments were simulated using head-dependent boundaries constrained to outflow (i.e. drains). Drains representing the development (i.e. underground working access and egress ramps and declines) were activated according to the mining plan, and remained active throughout mining operations, while mining stopes were deactivated after a period of one year, at which point the stopes were assumed to be backfilled with paste. The conductance of Brucejack Lake was adjusted throughout the simulated operations to reflect tailings deposition.

The model simulated rate of groundwater inflow to the underground workings for the base case scenario is predicted to remain relatively stable throughout the development of the VOK Zone resource during the first 14 years of mine life, ranging between 2,250 m^3/d and 2,650 m^3/d. The rate of inflow to the underground workings is predicted to increase to a peak of approximately 3,750 m^3/d in Year 15 with the development of the West Zone resource. During Years 16 to 21 of mine life, predicted inflows range between 2,700 m^3/d and 3,000 m^3/d, before decreasing to approximately 2,400 m^3/d in the final two years of mine life.

A series of sensitivity scenarios were considered in addition to the base case scenario, as shown in Figure 16.15.

Figure 16.15 Estimated Inflow to Underground Workings for Base Case Predictive Simulation and Nine Sensitivity Scenarios



The inflow estimates are sensitive to the hydraulic properties of the bedrock in the Project area, as well as recharge applied to the model. Specifically, increasing the hydraulic conductivity by a factor of five results in predicted model inflow increased by a factor of 2.3, averaging approximately 5,900 m^3/d on an annual basis and peaking at approximately 8,900 m^3/d in Year 15 of mine operations. Increasing recharge by a factor of two results in 1.5 times more inflow, averaging approximately 3,800 m^3/d on an annual basis and yielding maximum annual flows of approximately 5,100 m^3/d. Given the uncertainty associated with hydraulic head conditions in glaciated areas, a sensitivity

run was completed representing glaciers as constant head boundaries with the boundary set to topography (i.e. the glacier surface). This run, which represents the maximum recharge scenario under glaciated areas, resulted in an increase in mine inflows by a factor of 1.8, with an estimated inflow averaging approximately 4,600 m^3/d on an annual basis.

The estimates of groundwater inflow to the underground workings provided should be revisited if significant deviations from the proposed mining plans are expected. The results of the sensitivity runs have been provided to show the range of inflow estimates predicted by the model. For planning purposes, the most conservative inflows are provided by the 'High K Mine Inflows" model run (the dark purple line, Figure 16.15) and the base case (the dark blue line, Figure 16.15) provides the best estimate of inflows using the existing dataset.

16.7 Mobile Equipment Requirements

16.7.1 Pre-production Phase

During the preproduction phase, over 14,000 m of lateral development must be completed to meet the ramp-up production schedule. Development during this period will peak at approximately 660 m/mo, but with the average over the pre-production timeframe at about 600 m/mo. A contractor will be utilized to supply manpower and expertise to excavate the initial capital waste development program of ramps, drifts, and raises that are detailed in the pre-production development program described in Section 16.4.2.

To reduce risk associated with contractor supplied fleets, a portion of the pre-production mining equipment will be supplied by Pretivm. Table 16.11 shows the breakdown of Pretivm and contractor supplied equipment for the pre-production development phase. The ancillary equipment during this period will be supplied by Pretivm.

Table 16.11 Contractor and Pretivm Equipment during Pre-production Development

Description	Contractor	Pretivm	Total
Two boom mining jumbo	1	3	4
LHD, 14 t (Development)	3	2	5
LHD, 14 t (Production)	0	0	0
Haulage truck, 40 t	0	3	3
Bolter	0	2	2
Cable bolter	0	1	1
Shotcrete sprayer	0	2	2
ITH long hole drill	0	1	1
Transmixer	0	2	2

Note: ITH = in-the-hole

Jumbos

Each jumbo can support 150 to 160 m/mo of development. Four jumbos will be required to meet the peak scheduled advance.

LHDs

Up to five diesel LHDs will be required to move the blasted material from the various development headings and from the mass excavations. The contractor will provide three diesel units for this period as the electric LHDs planned for the production phase will not have the flexibility of movement required during the pre-production phase.

Trucks

Three 40 t trucks will provide adequate haulage capacity for pre-production activities.

Bolters

Demand on bolting during pre-production development will be higher than during production as development peaks during this time. Both bolters required for production will be brought on line during this phase to provide the contractor with bolting capacity.

Cable Bolter

Cable bolting will not be required in the stope undercuts during pre-production development as few will be excavated. However, there will be a need for cable bolting in the shop excavations and in all of the intersections. A single cable bolter will be sufficient to handle this load, with any excess bolting being handled manually.

Shotcrete Sprayers

It is anticipated that 10% of all development over the LOM will require shotcrete. The entire workshop and crusher area will also require shotcrete. One shotcrete sprayer should be of sufficient capacity to meet the expected volume; however, a second unit is planned in the interest of ensuring capability for this critical task.

Transmixers

Two transmixers will be required to transfer the expected volume from the surface batch plant to the sprayers underground, to ensure no shotcreting delays.

Long Hole Drills

A long hole drill will be required for the crusher excavation, drop-raising of the fresh air raise to advance the VOK ramp, and early slot development in the VOK upper block. The drill may also be employed to drill cable bolt holes if the cable bolter is unavailable.

16.7.2 Production Phase

Pretivm will supply all equipment during production with the exception of Alimak raise climbers, which will be included in the raising contracts. Table 16.12 lists the required equipment for development, stoping, and support activities.

Table 16.12 Underground Development and Production Equipment List

Description	Availability (%)	Utilization (%)		Quantity
		Peak	Average	
Two boom mining jumbo	74	73	66	3
LHD, 14 t (diesel) (development)	74	69	62	2
LHD, 14 t (electric) (production)	74	68	68	5
Haulage truck, 40 t	74	87	84	6
Bolter	79	44	41	2
Cable bolter	79	45	45	2
Top hammer long hole drill	79	62	62	1
ITH long hole drill	79	62	62	3
100 kW generator set for electric LHD tramming	79	19	19	5
Explosives loader, diesel, emulsion (production)	79	-	-	2
Shotcrete sprayer	79	15	15	2
Transmixer	79	19	19	2

JUMBOS

Development advance will be approximately 475 m/mo during the first 10 years of production. A two boom unit, capable of drilling holes 4.3 m deep, was selected based on the average drift size. First principles estimates and benchmarking indicate that expected performance is 150 to 160 m/mo. Three units will be required.

LHDS

A study of LHD (loader) productivity determined that six, 14 t loaders will be required for the scheduled stope and development volumes. An additional unit will also be needed, dedicated to feeding the crusher. A total of seven units will be required.

AMC completed a trade-off study that compared electric units to diesel units, with the conclusion that electric loaders present a cost advantage with lower operating and ventilation costs for equivalent capital costs. Five electric loaders were selected for stope mucking and crusher service, with two diesel loaders selected for development headings, given the requirement for increased mobility.

The tramming distance that an electric LHD must travel when moving between the various working areas of the mine will generally exceed the radius of operation of the trailing cable that powers the LHD under normal loading circumstances. An allowance was made for trailer-mounted 100 kW generators to provide this autonomy.

It is anticipated that blasted ore will generally be dumped directly into the crusher feed bin and not into the blending and storage bays adjacent to the crusher. As such, the single electric LHD designated to the crusher area should have the capacity to handle any blending and remucking activities.

Haulage Trucks

AMC considered electric haulage trucks in the interest of minimizing ventilation airflow and heating requirements. The significant capital premium for equipping the mine with an electric fleet was found to be irrecoverable through reduced operating costs, such that standard diesel units were adopted for all haulage activity. Six units will be required to support the scheduled material movement of ore and waste.

Forty tonne trucks were determined to be the most cost effective truck capacity in consideration of utilization, capital cost and operating costs.

Bolters

Pattern bolting of development headings and stope backs will utilize both 2.4 m resin grouted rebar and 2.4 m swellex type friction bolts. Two units will be required for redundancy and should be equipped for the installation of either bolt type.

Cable Bolters

All stope undercuts and overcuts will be cable bolted with 6 m long cable bolts, and all intersections will be bolted with 5 m long cable bolts. A single bolter could be utilized; however, the utilization would be higher than 90% and it is likely that headings would stand waiting. Two cable bolters have been factored into the study.

Long Hole Drills

Production drilling will mostly be done with ITH hammer drills. Compared to top-hammer drills, ITH drills are more accurate for longer holes and are recommended for the 25 m average production hole length. The proposed Cubex Aries B drill is also able to ream 762 mm diameter slot raises, thus eliminating the need for a standalone slot raise machine or crew. This drill will also be utilized for service holes between sublevels.

In addition to the ITH drills, a single top hammer drill will be used. This will provide some flexibility to the production drilling fleet. The top hammer drill will be utilized for sill recovery, crown level stopes, stopes near the Brucejack fault where fan drilling is required (full width slashing is not anticipated for these stopes), and the recovery of resources at the top of lenses where uphole drilling is inevitable. One unit will be required.

Explosive Loaders

Two face chargers will be required for development loading, to load as many as six rounds per day. Each unit will require pumps for face charging with emulsion. One unit will be required at full production for loading uphole and downhole stopes with emulsion.

Shotcrete Sprayers

It is anticipated that 10% of development will require shotcrete. Shotcrete will also be required for paste fill exposures in stope development and barricade construction for backfilling and ventilation bulkheads. Two units will provide adequate capacity and redundancy for this critical activity. Wet fibre-reinforced shotcrete will be standard.

TRANSMIXERS

Shotcrete will be delivered from the surface batch plant. Only one truck is required; however, as demand is unlikely to be consistent–with average haul distances not always typical and varying shotcrete amounts required–two transmixers are recommended.

16.7.3 SUPPORT EQUIPMENT

Table 16.13 presents the complete list of support equipment.

Table 16.13 Support Equipment List

Item	Description	Availability (%)	Utilization (%)	Quantity
1	Personnel carrier, diesel, underground	79	16	2
2	Scissor lift truck, diesel	79	54	2
3	Lubrication service truck, diesel	79	41	1
4	Boom truck, diesel	79	75	2
5	Explosives truck, diesel, emulsion (transport)	79	29	1
6	Tractor	79	46	9
7	Utility vehicle	79	46	19
8	Telehandler, diesel	79	57	1
9	Wheel loader with tire handler	79	5	1
10	Motor grader (tracks and wheels)	79	75	1
11	500 cfm portable compressor	79	10	3

PERSONNEL CARRIERS

Though many employees will be transported in tractors, personnel carriers will be needed for most employee transport. The transporter selected is capable of carrying 22 people. With two transporters, 44 people can be brought to work each day. The remainder of the personnel will be brought underground in tractors or light vehicles.

SCISSOR LIFT TRUCKS

A scissor lift truck is required to install pipe and an average of 17 m/d of services. A two-man crew on a single scissor lift truck should be able to complete up to 6 m of service installation in 3 hours, enabling up to 4 headings to be installed in 12 hours with a single scissor lift truck. However, the scissor lift truck will also be required for hanging fans, installing power cables, and other general service activities. Two scissor lift trucks are therefore recommended. A tractor equipped with lifts can be used as a substitute if the scissor truck is down for maintenance.

LUBRICATION TRUCK

A lubrication truck will be required to fuel and lubricate all equipment that is not likely to return to the shop area at frequent intervals. Down time can be reduced by keeping equipment near the working headings. This will also help improve traffic flow on the ramp. This equipment will include LHDs, jumbos, long hole drills, bolters, and cable

bolters. The service truck will travel between these equipment pieces to perform daily servicing.

BOOM TRUCKS

A boom truck will be required to transport materials from surface to underground daily and to facilitate loading and unloading. Material stockpiles will be set up throughout the mine for supplies such as rock bolts, screen, resin, vent duct, etc.

EXPLOSIVES VEHICLES

At full production, explosives consumption is estimated to be 2.7 t/d of bulk emulsion. This will be delivered to the mine in nine custom-made ISO tanks, each with a capacity of 6,000 L or 7 t. A boom truck will transport the full tanks to the emulsion bays. Two emulsion pumps will be used to transfer from the full tank to the empty one in the bay. Consumption will average three tanks per week. All other explosives will be transported to the cap and powder magazine by the explosives handling tractor. Approximately 300 caps and 300 primers will be required daily.

TRACTORS

Tractors will be used to transport some personnel during shift change. They will also be used for nipping materials and general transport through the mine. All tractors will be equipped with a cargo/man carrying compartment in the back. Some tractors will also be equipped with man lifts to facilitate services installations, bulkhead construction, surveying, geological mapping, loading of development rounds, etc. The following crews will be issued tractors for use during their shifts:

- development blasters
- backfill crew
- mechanics
- electricians
- production blasters
- diamond drillers
- warehouse.

16.8 VENTILATION

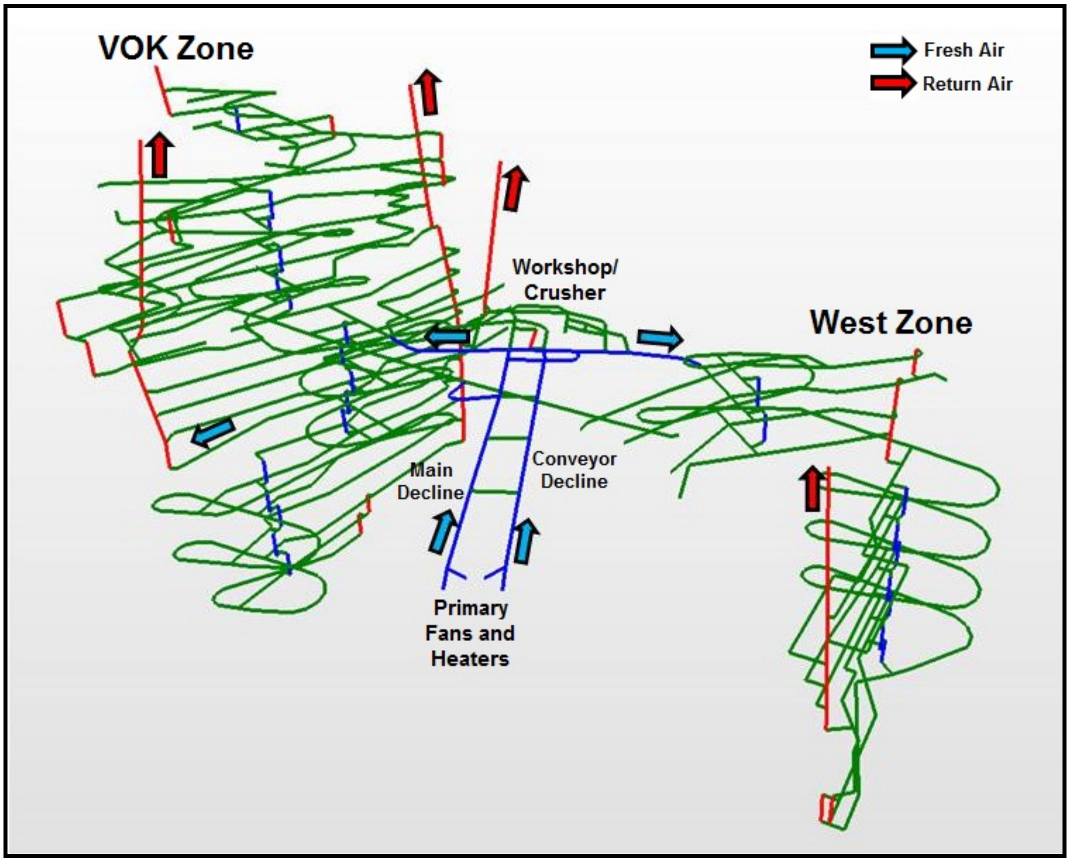
The ventilation system was designed to meet the requirement of the Health, Safety and Reclamation Code for Mines in British Columbia – 2008 (The Code), which requires a minimum of 0.06 m^3/s of ventilating air for each kilowatt of power of diesel powered equipment operating. The design is based on a “push-pull” configuration, with permanent surface fans located at the portal of the twin declines. The extremity of each active footwall will be fitted with an exhaust fan acting as the “pull” component of the system.

The VOK Zone and West Zone mining areas will be supplied with fresh air from a connection to the twin declines and each area will have at least one exhaust return air raise (RAR) to surface.

The underground crusher and workshop will have a dedicated RAR to prevent the introduction of dust and other contaminants into production areas. The volume of air flowing through the crusher and workshop areas will be controlled with a combination of fans and regulation.

Figure 16.16 shows an isometric view of the Brucejack ventilation system.

Figure 16.16 Brucejack Ventilation System – Looking West



16.8.1 TOTAL AIRFLOW REQUIREMENTS

Air flow requirements were determined based on the diesel engine exhaust (DEE) dilution provided at point of use for the number of required mining areas. An airflow allowance was also determined for underground infrastructure, leakage, and balancing inefficiencies.

Total airflow requirements were determined based on the anticipated concurrent activities during steady state production and development. The following activities are anticipated at the Property:

- three development headings advancing
- two production levels with stope mucking and truck loading
- two production levels with drilling/charging/servicing activities.

Airflow allocations, based on the steady state production and development scenario, are summarized in Table 16.14.

Table 16.14 Total Airflow Requirements

Areas		Flow (m^3/s)	Number	Total (m^3/s)
Development		38	3	114
Production	Stope Mucking and Loading	24	2	48
	Drilling/Charging/Services	13	2	26
Infrastructure	Crusher Chamber	24	1	24
	Workshop/Magazine	20	1	24
	Fuel Bay Loop	24	1	24
	Sacrificial Level	20	1	20
Leakage and Balancing	15%	-	-	45
Total (rounded)				**320**

16.8.2 Auxiliary Ventilation

All work areas in the mine not supplied with a split of fresh air must be ventilated using auxiliary systems. The most effective means for providing airflow to areas without primary airflow is typically with small diameter (up to 1,400 mm) axial fans combined with low leakage and flexible ducting.

Development Ventilation

During access and level development, distances up to 800 m will be ventilated using auxiliary systems. The peak auxiliary airflow for development activity will be required to dilute the emissions of one 40 t truck and one 14 t loader, amounting to 38 m^3/s of auxiliary airflow.

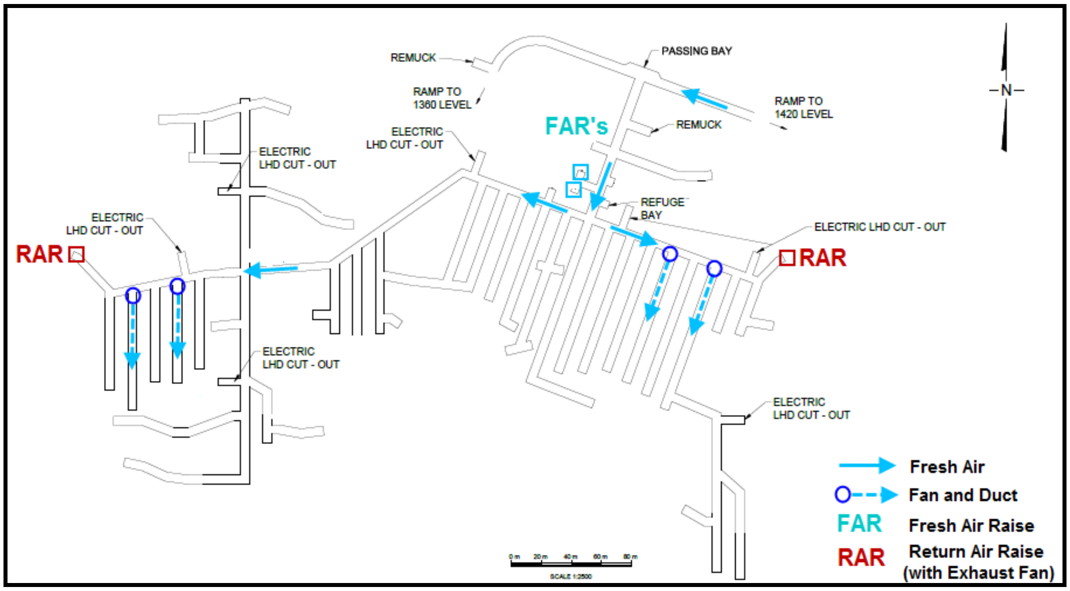
Modelling indicates that two ducts will be required. Each duct will have two 55 kW fans bolted together in series. The duct size is 1,200 mm in diameter. This will supply 38 m^3/s up to a distance of 850 m. This arrangement will allow for adequate overhead clearance for a fully-loaded 40 t truck.

Drawpoint Ventilation

An allowance of 12.5 m^3/s was made for each active drawpoint, for dust, blast fume, and diesel exhaust clearance. Modelling indicates that a single-stage 55 kW fan, with 900 mm diameter low-resistance, low-leakage ducting will supply the required airflow to a distance of at least 250 m. An increase of ducting size to 1,100 mm can be employed for stopes that require a longer forcing distance–up to 750 m.

Figure 16.17 shows a ventilation configuration for a typical production level.

Figure 16.17 Typical Production Level



Ventilation Modelling

AMC developed a ventilation model (using Ventsim) for the Project for three primary purposes:

- to validate the operability of the ventilation circuit ensuring airflow can be provided to all the required areas
- to ensure compliance with design criteria
- to determine fans duties and energy requirements.

Peak primary fan duties will occur at full production in conjunction with maximum development activities in the lowest levels of each ventilation district.

16.8.3 Permanent Primary Fans

Over the LOM, there will be a multitude of settings for the ventilation circuit, depending on the type of activities and their location throughout the mine. AMC modelled the circuit to reflect the peak primary fan duties that could be reasonably expected.

Primary fan requirements are summarized Table 16.15.

Table 16.15 Primary Fan Specifications

Description	Specification
Duty	Two each @ 160 m^3/s @ 1,121 Pa
Fan Diameter	2.84 m
Type	Horizontal mount axial mine fan
Configuration	There will be two forcing fans, each connected with ducting to the main access decline and conveyor decline
Voltage	4,160 V
Fan Motor	261 kW to 710 rpm, variable frequency drive capability

16.8.4 Mine Air Heating

This study assumes that all intake air entering the mine will be heated above the freezing point for the following reasons:

- protect the health and safety of personnel working or travelling in intake airways
- prevent the freezing of service water and discharge lines
- ensure reliable operation of conveying and other mechanical equipment in the decline
- maintain ice-free and safely trafficable roadways
- prevent rock surface (or shotcrete lining) expansion/contraction damage from freezing and thawing of rock joints in the upper parts of the intake airways.
- prevent ice build-up in airways that would potentially lead to unsafe conditions.

16.8.5 Conveyor Decline

The conveyor decline will be a main mine intake with planned dimensions of 6.5 m wide by 6.0 m high. Care will be required to ensure that the air speed in the conveyor decline is not too high otherwise dust from the conveyor will be picked up and carried into the working areas. It is the differential air velocity that needs to be considered in the design not the actual drift air velocity. When the drift velocity and the conveyed material are moving in opposition, as is the case with the Brucejack design, a reduced drift velocity is required. Given the planned conveyor speed of 1.0 m/s and the design maximum velocity of 5.5 m/s, the velocity in the conveyor decline should not exceed approximately 4.5 m/s.

Given that the conveyor will be located in a primary air intake, the risk of the conveyor catching fire must be managed. The design includes allowance for the following:

- fire retardant belt
- fire retardant grease and lubricants
- ventilation controls to isolate the air in the conveyor decline in the event of a fire

- regular inspection of the conveyor decline during operation in order to detect the development of faulty rollers or belt misalignment.

In the unlikely event of conveyor belt fire, fire doors placed in key areas would close and smoke would flow directly to the workshop/crusher exhaust raise.

16.8.6 Emergency Preparedness

In development of the ventilation strategy for the Project, consideration has been given to the potential for mine emergencies. As such, the following criteria have been established:

- In general, ramps will be in fresh air once developed.
- On almost all levels, escape can be either to a ramp or to the escape ladderway.
- In each ramp, escape may either be up the ramp or down the ramp to a safe area.
- One permanent 40 person refuge station will be established adjacent to the "T" junction servicing both West Zone and VOK Zone lodes.
- Other refuge chambers will be portable for flexibility of location at the most appropriate points in the mine.
- While the primary means of communication will be by radio, a stench system will be in place for introduction of ethyl mercaptan into both portals concurrently in the event of fire.
- Fire doors will be located in accordance with legislated requirements and to isolate areas of high fire potential to ensure noxious gases are not distributed through the mine workings.

There are a variety of incidents that will trigger the emergency response plan and/or evacuation plan. Such events may be fire, rock fall, injured personnel, or major ventilation equipment breakdown. Emergency coordination will occur from the control room where all information and communications can be monitored.

For the two surface portals, both of which will be supplied with fresh air, the vehicle portal will be considered the primary escape and the conveyor portal the secondary escape. Additionally, the existing West Zone portal will be available as an emergency egress point.

For the production stoping blocks, a ladderway will be installed in each of the raises located next to main ramps. The raises are sized to afford easy passageway.

The exhaust raises to surface will be for ventilation only and not used as a second means of egress. Therefore the exhaust raises will not have ladderways installed.

A static refuge station will be established adjacent to the "T" junction, servicing both West Zone and VOK Zone lodes. It is required to provide refuge during an emergency for 40 persons. This refuge station will provide refuge for personnel working predominantly

in the workshop/crusher/mine offices area. In review of crew numbers during the life of the mine, it is estimated that the maximum number of personnel underground working during any shift will be 86. It is estimated that 36 people will be working predominantly in the workshop area.

The remaining personnel working underground, namely the production development and service crews, will be provided refuge by means of five, 12-person mobile self-sufficient rescue chambers. These will be independent of a compressed air supply, with appropriate provisions for safe refuge. They will be located in areas where a secondary egress is not, or has not yet been established, and will be sited relative to the active working areas in order to be within the average walking pace duration of a personal self-rescuer device.

An automatic stench gas warning system will be installed on the supply side of the surface vehicle portal and conveyor portal. When fired, this system will release stench gas into the main fresh air system allowing the gas to permeate rapidly throughout the mine workings. Once stench gas is released, underground mine personnel would report immediately to the nearest mine refuge station or surface, whichever is closer.

The primary purposes of fire doors are to prevent noxious gases from reaching workers should they be trapped underground and to prevent fire from spreading as much as possible.

Fire doors will be required to isolate the following areas:

- workshop
- fuel bay
- conveyor decline.

Portal doors will also be designed to meet fire door criteria.

16.9 UNDERGROUND INFRASTRUCTURE

16.9.1 MINE DEWATERING AND SOLIDS HANDLING

Mine dewatering must accommodate groundwater inflows from the VOK Zone workings, the West Zone workings, and inflows from drill and other operating equipment. Total inflows are estimated to be approximately 100 L/s; however, to accommodate for uncertainty in the water inflow model, the design capacity for the pumping system is based on maximum inflows of 139 L/s.

Mine dewatering will be handled by a combination of submersible and horizontal centrifugal pumps located throughout the West Zone and VOK Zone working levels. The pumps will handle mine water via multiple lifts throughout the mine to minimize pump size and power. Lift stations will be designed with consistent vertical intervals of 60 m to maintain similar pump head characteristics. This reduces the amount of different pump sizes required, which will simplify maintenance.

On low-head applications, a submersible pump will be used. For a decline development heading, a submersible pump will be used at the drilling face to pump water to a sump. In sumps that require higher flows and heads, a submersible pump will feed a train of horizontal pumps. Each pump station has 100% redundancy such that the loss of any single pump will not disable the system.

Intermediary levels will drain via borehole to the main pumping levels. Each sump will stage water to the next higher sump in the mine. All of the underground dewatering sumps will report to the VOK Zone 1,330 Level sump through a network of boreholes and pressure piping, before leaving the mine and reporting at the surface 1,400 Level water treatment plant.

Every pump station will be equipped with dirty water and clean water sections, the water decanting into the clean water section before pumping. This will allow the majority of the slimes to settle prior to pumping. This is important as high levels of fines in the pump water can reduce impellor life on the pumps.

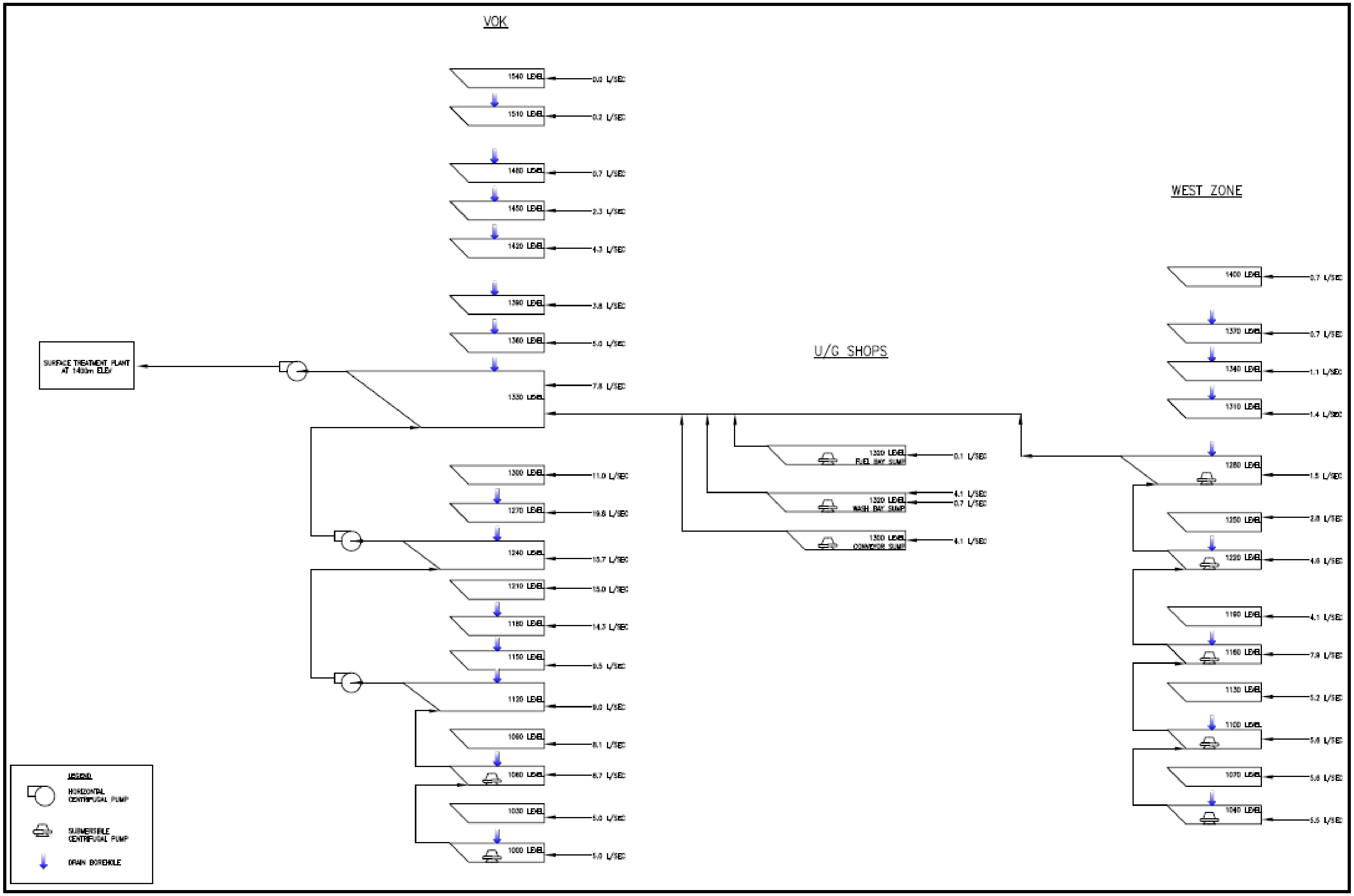
Discharge piping and pumps were sized to accommodate the water flows expected from each pumping station. Though lower stations will have higher inflows, they will be pumping water from fewer levels and will therefore have lower volumes of water to move than higher stations. It is assumed that the pumps will not be running full time, but will cycle as required. For this reason, pump capacity was designed to be larger than the inflow, such that the maximum required operation, relative to time, is limited to 80% or less for any pump. The pipe capacity was sized in a similar manner.

To minimize upfront capital, the pump procurement will be staged such that pumps only arrive as their assigned sumps are excavated. Table 16.16 shows the pump installation schedule. Figure 16.18 is a line diagram of the dewatering system.

Table 16.16 Pump Installation Schedule

Dewatering	Year																							
	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Service Water Pump	■		■																					
Development Sumps Pumps	■																							
VOK Level Sump 1,330	■																							
VOK Level Sump 1,240		■																						
VOK Level Sump 1,150			■																					
VOK Level Sump 1,120					■																			
VOK Level Sump 1,060							■																	
VOK Level Sump 1,000								■																
West Zone Level Sump 1,280											■													
West Zone Level Sump 1,220												■												
West Zone Level Sump 1,160													■											
West Zone Level Sump 1,100													■											
West Zone Level Sump 1,040													■											
Convey Sump 1,300		■																						
Shop Sump 1,320		■																						
Fuel Storage Pump		■																						

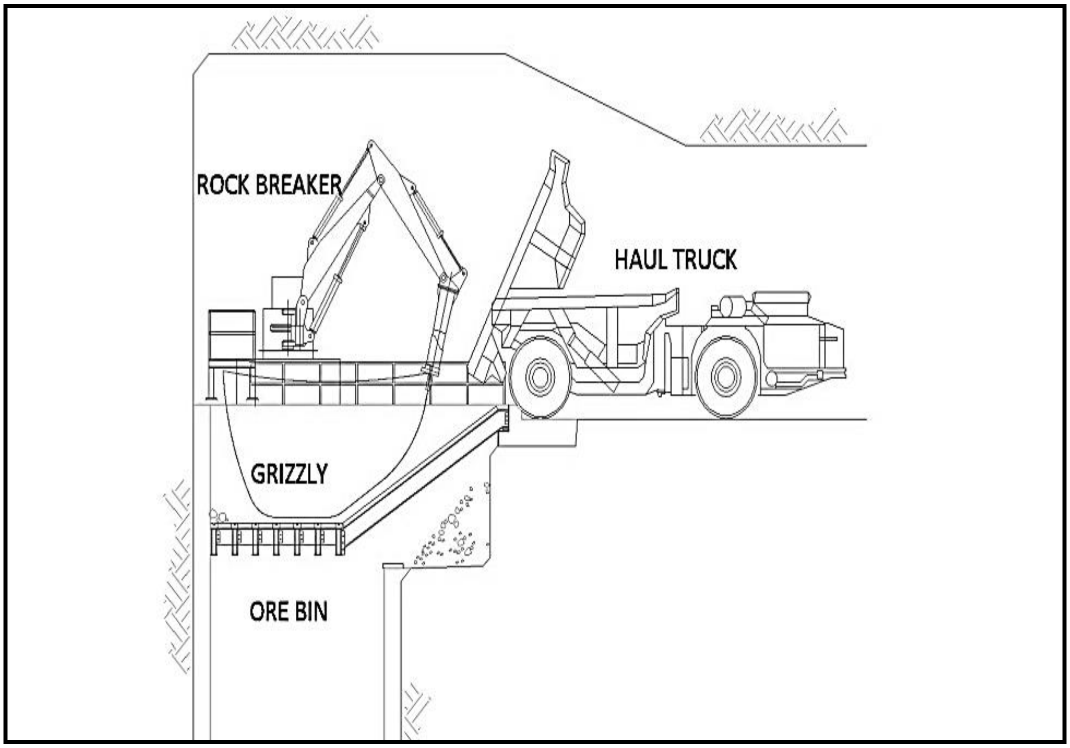
Figure 16.18 Dewatering Plan



16.9.2 MATERIAL HANDLING

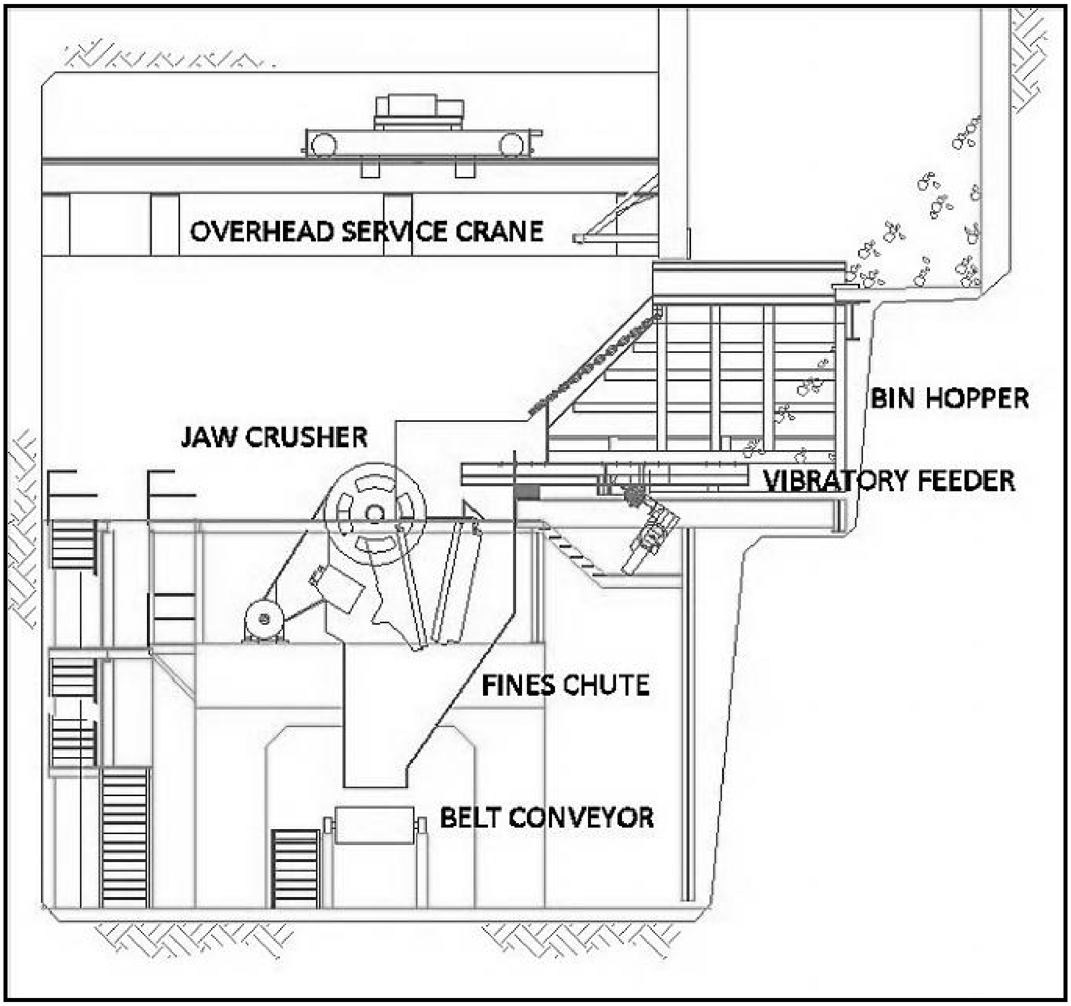
ROM material will be transported underground by truck from the West Zone and VOK Zone areas and deposited into the ore storage bays or directly onto the scalping grizzly (see Figure 16.19). Material stockpiled in the storage bays will be scooped and deposited onto the scalping grizzly by an electric LHD. At the scalping grizzly, material smaller than 400 mm will fall through to the ore bin and larger material will be broken down by a hydraulic rock breaker stationed above the grizzly screen.

Figure 16.19 Crusher Tipping



As shown in Figure 16.20, the 500 t capacity ore bin will feed material down through a hopper at the bottom of the bin to a vibratory feeder. This vibratory feeder will contain a grizzly screen that will transport large material to a jaw crusher and allow fines (less than 64 mm) to fall through and down the fines chute to the transfer conveyor. The jaw crusher will reduce the larger material down to 65 to 75 mm in size and drop this product down the fines chute to the transfer belt conveyor.

Figure 16.20 Crusher



The 42 in wide belting on the transfer conveyor will carry material from the crushing area to the main conveyor at a speed of 1 m/s and will be positioned at a 90° angle to the main conveyor. The ore on the transfer conveyor will move up, past a magnet that will remove any tramp iron, depositing this iron into a waiting bin (see Figure 16.21).

The main conveyor will move at a rate of 1 m/s transporting approximately 225 t/h of ore up the decline tunnel to the mill. The main conveyor will exit the decline tunnel through the portal structure where the 335 hp drive unit will be located. It will continue on from the portal to the mill through an enclosed, heated, rectangular gallery. The gallery structure will be elevated, continuing at a grade of 15% from the portal structure to the mill, allowing for traffic underneath (see Figure 16.22).

Figure 16.21 Transfer Conveyor from Crusher to Main Conveyor

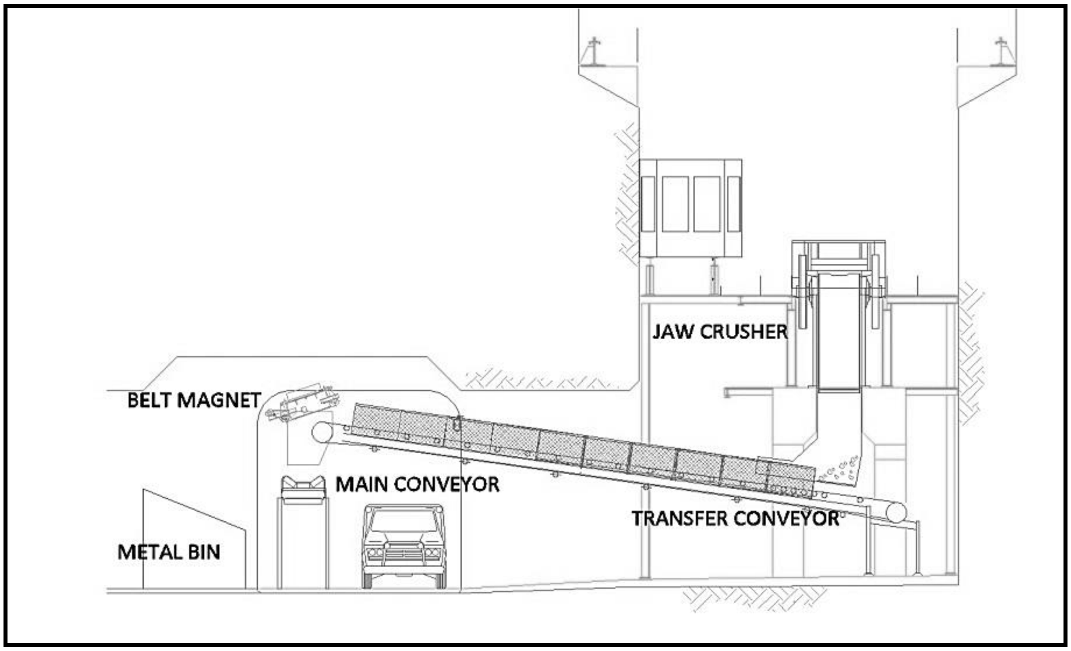
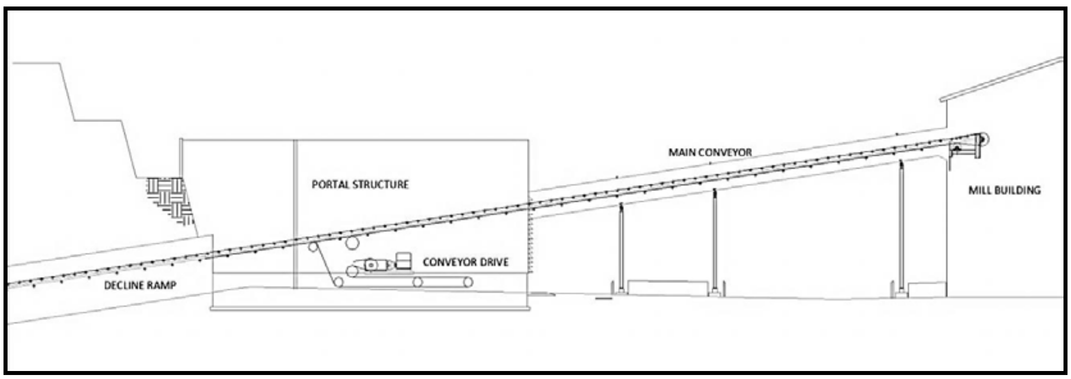


Figure 16.22 Conveyor from Decline to Mill



The main conveyor belt speed of 1 m/s, along with the expected air speed of 4 m/s down the decline will provide a combined speed of 5 m/s. This speed was assessed as reasonable in terms of keeping dust lifted from the belt to a minimum. Table 16.17 summarizes conveyor parameters.

Table 16.17 Conveyor Parameters

Conveyor	Width	Length	Speed	Incline Angle (°)
Transfer	42 in (1 m)	55 ft 7 in (16.940 m)	197 ft/min (1 m/s)	8.00
Main	42 in (1 m)	2,149 ft (654.981 m)	197 ft/min (1 m/s)	8.53

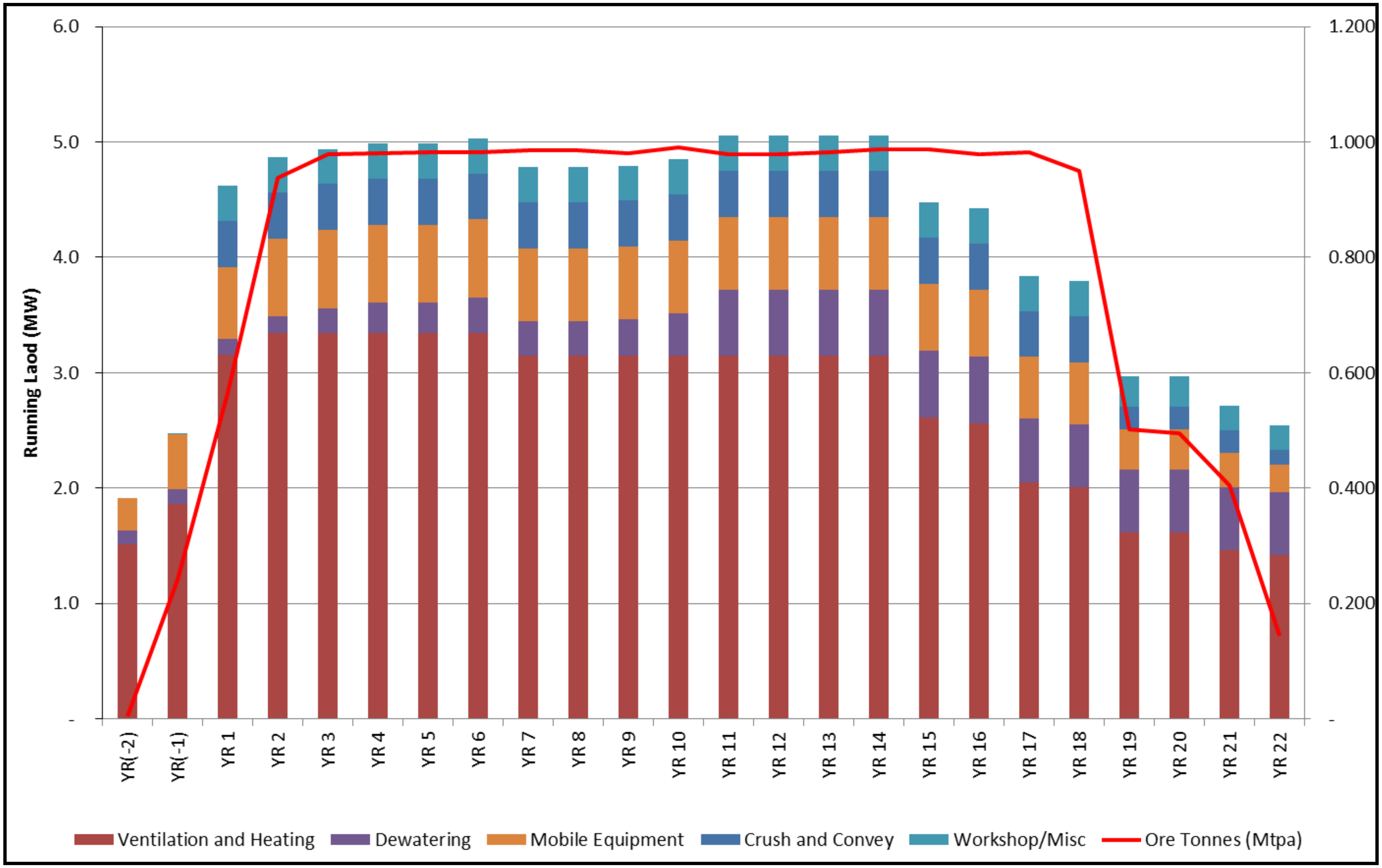
16.9.3 Power Requirements and Electrical Distribution

BC Hydro indicated that the total electric power supply available for the Brucejack site will be limited to a connected load of 20 MW. The maximum underground connected load to support full production and development activities will be approximately 9 MW, inclusive of ventilation and heating.

Considering the other key consumers of mine power such as the mill and paste plant, the power available for mine air heating will be limited to 4 MW. As the mine air heaters will at times require approximately 16 MW of power, a propane direct-fired system will make up the remaining heating requirement.

Figure 16.23 shows the growth of the power requirements over the LOM in relation to ore production. Ventilation and heating, mobile equipment, and dewatering are the main consumers of power. The maximum running load is estimated to be 5 MW and will occur when full production levels are achieved. As the mine is developed deeper, the dewatering power demand will increase due to a higher lifting head and increased inflows. As development activity and production decrease, the power requirements will also reduce.

Figure 16.23 Underground Power Requirement Profile



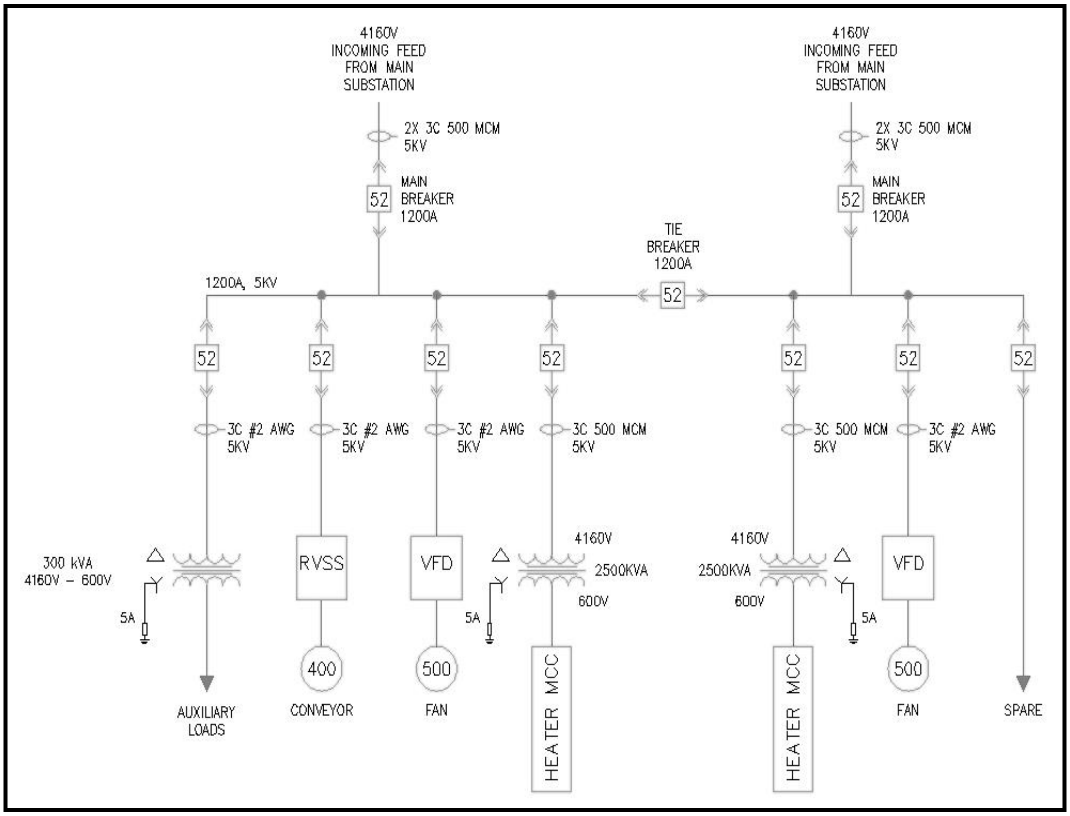
Electrical power will be supplied to the portal building by four separate 4,160 V feeder circuits from the site main substation. Two feeders (six conductors, 500 MCM each) will supply power to the 4,160 V distribution equipment located at the portal building. The other two feeders (six conductors, 350 MCM each) will continue on through the portals to the underground access declines, one feeder in the main access decline and one in the conveyor decline.

The portal building 4,160 V switchgear will incorporate two main incoming circuit breakers, a tie circuit breaker and suitable feeder circuit breakers. These will supply power to a 2,500 kVA electric heater unit, two supply transformers, two 500 hp fan drives, a 400 hp conveyor drive, and other auxiliary equipment via a 300 kVA delta-wye step down transformer with a 5 A continuous rated neutral grounding resistor. A 600 V, 600 A motor control center will be supplied by the step down transformer to distribute power to various small horsepower motors and a lighting panel located at the portal building.

Each 4,160 V decline feeder will terminate at respective substations located at the 1,300 crusher level and 1,320 shop level. A tie circuit will connect the two underground substations to allow for a redundant power feed system from either underground feeder; with 4,160 V isolating switches configured to allow all electrical equipment to be supplied by either the main access decline feeder or conveyor decline feeder from the surface. Step down transformers located at the 1,300 and 1,320 Level substations will provide a 600 V supply to various electrical loads at that level, including the crusher and main conveyor feed system, sump pumps, lighting, etc.

Additional 4,160 V feeders (three conductors, 500 MCM each), supplied from fused disconnects at the crusher and shops substations to both the VOK Zone and West Zone working levels, will provide power to portable substations to be used for development, pumping, ventilation, lighting, etc. Figure 16.24 is a schematic of the portal power distribution system.

Figure 16.24 Portal Power Schematic



16.9.4 Compressed Air

Due to the inefficient nature of mine wide compressed air systems, compressed air will be supplied by local area compressors. The underground maintenance and service bay area will have a dedicated compressor permanently installed, with air lines from the air receiver routed to convenient locations in the area.

In addition to the permanent compressors, several smaller, portable compressors will be available.

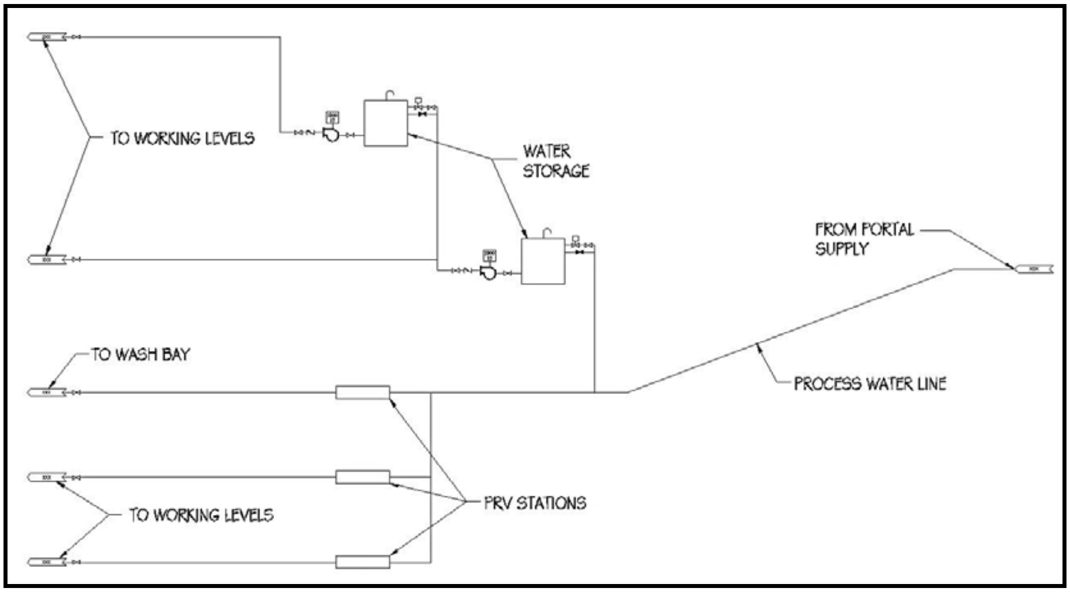
All mobile drilling equipment, including jumbos, long hole drills, bolters and cable bolters will be equipped with on board compressors. ITH drilling equipment will have portable adjacent compressors to meet their elevated pressure requirements.

16.9.5 Service Water Supply

Service water supply for drilling and dust control will be supplied via a 4 in steel line at the portal. The line will continue through the main decline ramp to the underground workings. Pressure reducing valves (PRV) will be supplied at the 1,320, 1,220, and 1,120 levels to reduce the supply pressure below 100 psig.

To supply service water to the higher-working levels, two lift stations will be required. Two water holding tanks will be positioned at the 1,390 and 1,480 levels. The 1,390 Level tank will be fed by the main header pressure in the decline ramp. An automated valve will control the tank level and a booster pump will feed working levels from the 1,390 Level up to the 1,480 Level. A tank positioned on the 1,480 Level will supply another booster pump to feed service water to the higher levels in the same fashion. Figure 16.25 is a schematic of the main water distribution system.

Figure 16.25 Mine Water Distribution Schematic



16.9.6 FUELING AND LUBRICATION

A fuel bay area will be located on the 1,320 infrastructure level between two automatic fire doors, as shown in Figure 16.26. The doors will be connected to a fire detection system that will close the doors if a fire is detected. A foam fire suppression system will be located inside the fuel bay area and will consist of a storage tank, piping, valving, detectors, and alarms.

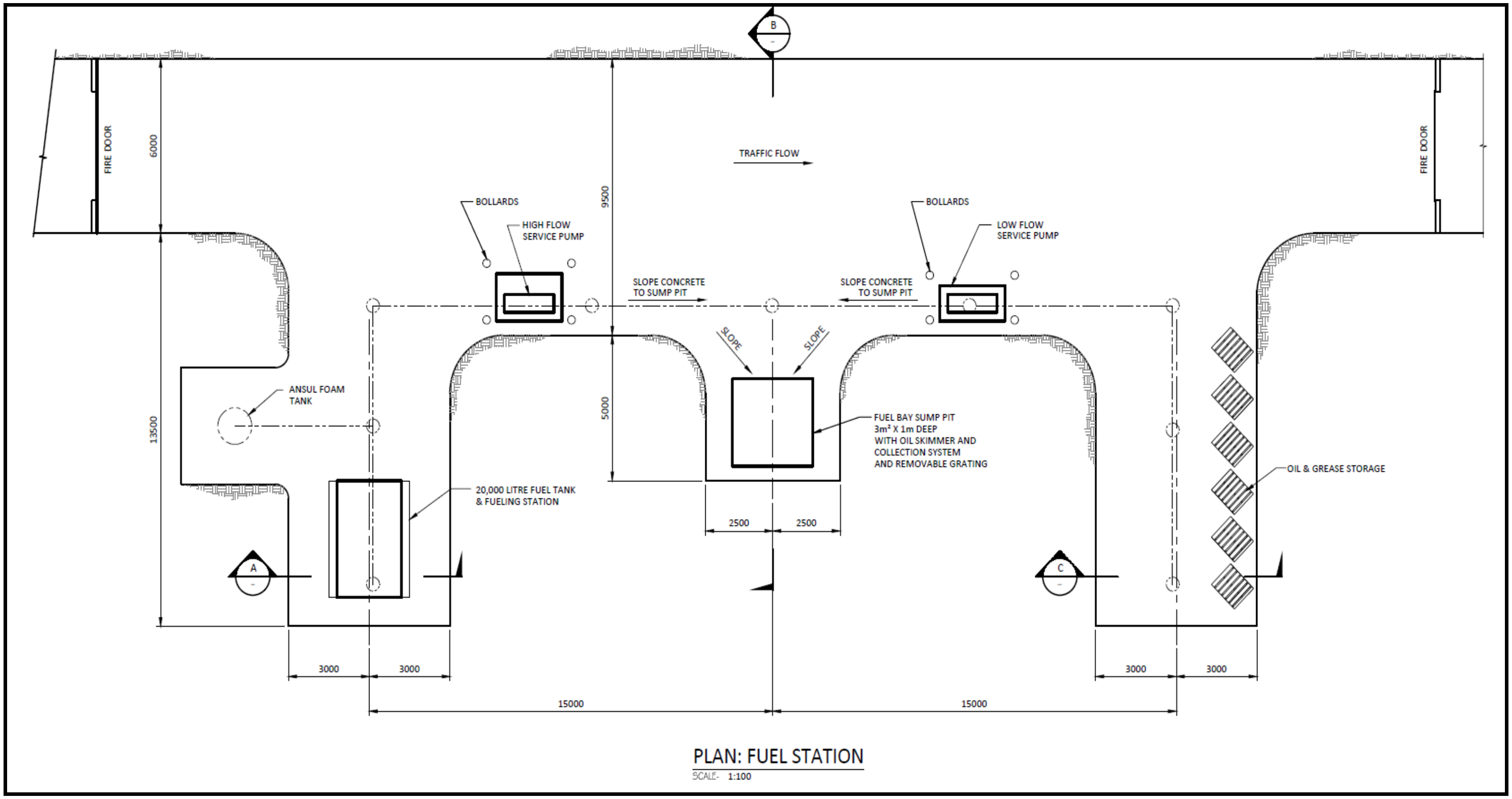
The fuel bay area will include a main drift and three bays. A 20,000 L fuel storage tank and a fuel delivery pumping system will be located in one of the bays located inside the fuel bay area.

The fuel storage tank will be filled from a surface storage tank via a dedicated fuel pipe line. A sump pit in another of the side bays will contain a sump pump and removable grating cover. The sump pump will report any collected water to the shop sump.

The third side bay will be used for oil and grease storage.

Vehicles requiring re-fueling or oil and greasing, will enter the fuel bay area from the west and exit to the east.

Figure 16.26 Fuel Bay Layout



16.9.7 WORKSHOP AND STORES

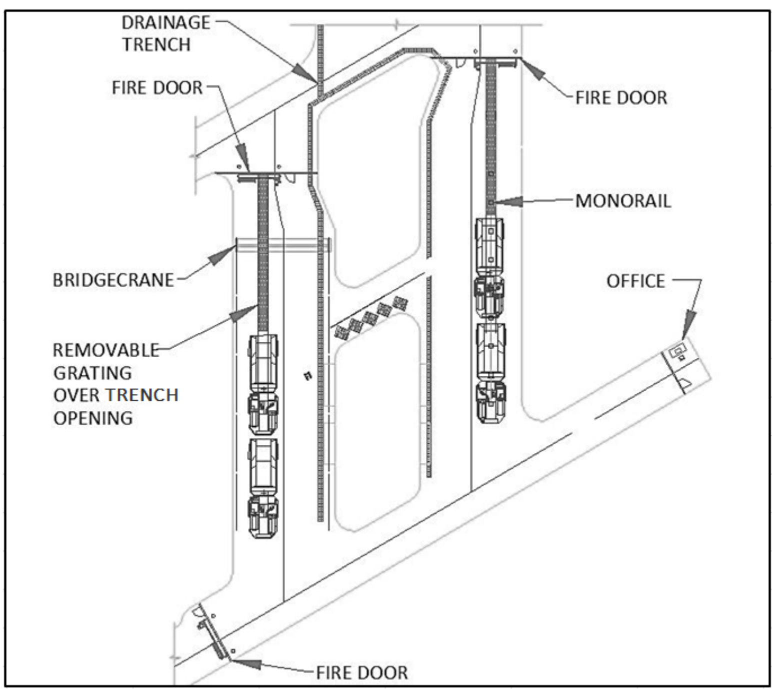
The maintenance area will consist of two large bays, each 10 m wide by 9.4 m high. A 2.1 m deep service trench will run the length of each bay to allow access to the undercarriage of the vehicles and give adequate headroom. The design accommodates up to two large vehicles per bay in service position over the trench. The two service bays will be joined by a cross-cut, allowing for forklift and foot traffic to move from one bay to the other without exiting the shop area. One sidewall along the length of each bay will accommodate tool cribs, with the cross-cut having common short-term storage of oil and greases. A drainage trench with covering grating will run the length of each bay to carry water to a nearby sump.

One service bay will be equipped with a 30 t bridge crane spanning the width and running the length of the bay. The other bay will have a monorail running its length.

The service area will be equipped with a stationary compressor and airlines to power air tools and provide compressed air as needed. A welding plug will also be sited in this area.

Three roll-up doors will separate the two maintenance bays from the rest of the mine. An office will be located at the end of a drift located in the maintenance area. Figure 16.27 illustrates the workshop layout.

Figure 16.27 Workshop Layout



16.9.8 EXPLOSIVES MAGAZINE

Two bays will be provided for the storage of bulk emulsions, each containing two 6,000 L storage tanks and a storage area. The entrance to the bays will be controlled with a roll-up door and a man-door. The length of each bay is approximately 12.8 m.

A third bay will be designated for the storage of caps and powder on wooden shelves. A concrete wall with a steel door will separate this bay from the rest of the mine works.

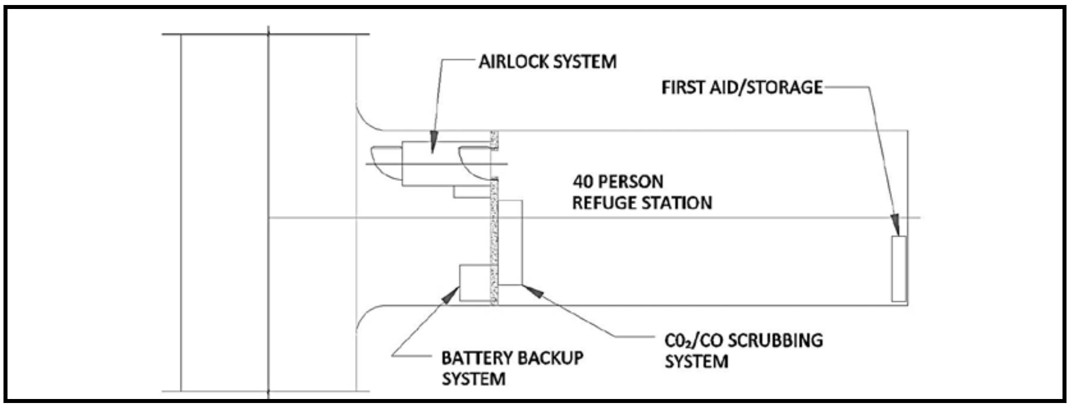
16.9.9 REFUGE STATIONS

A refuge station (see Figure 16.28) will be located between the two decline tunnels at the mine works. The station will accommodate 40 people and will be equipped with an airlock entrance, a battery back-up electrical system, an air conditioning unit, a carbon dioxide/carbon monoxide scrubbing unit, cache of oxygen-type cylinders, and emergency supply of first aid, food, water, and oxygen candles.

The refuge station will be located in a bay off a drift and will be separated from the drift by a concrete wall. Access to the station will be through an airlock system.

This refuge station can also be used as a lunchroom.

Figure 16.28 Permanent Refuge Station



16.9.10 COMMUNICATIONS

FIBER OPTICS AND PHONE AND RADIO COMMUNICATIONS

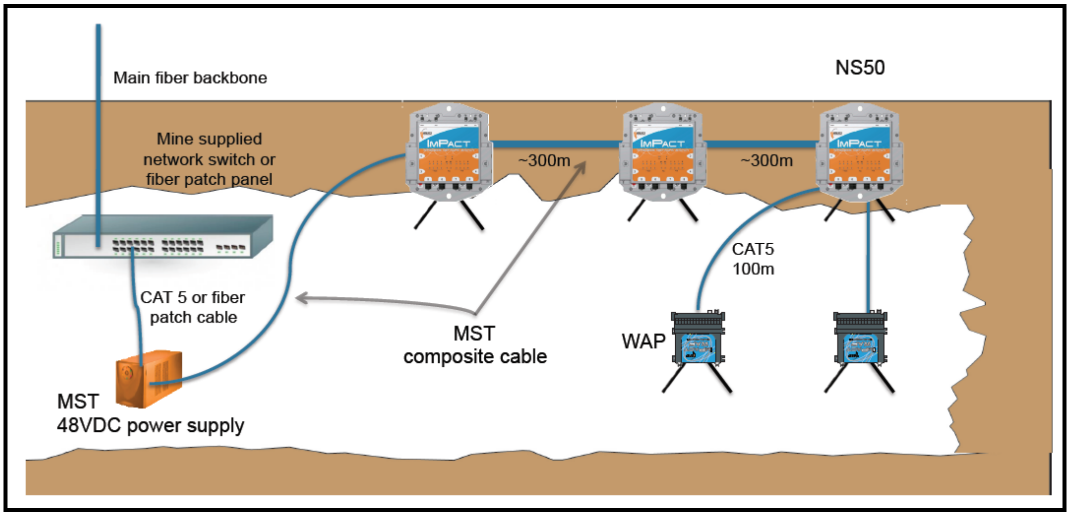
The underground wireless network infrastructure will consist of:

- Voice over Internet Protocol (VoIP) MinePhones
- cap lamps
- asset and personnel tracking
- vehicle intelligence
- proximity detection.

Radio communications will be established using a ruggedized, fiber-based, open standard, 802.11, high bandwidth, underground wireless network and a Strix Wireless Mesh surface network.

The backbone of the network will comprise of gigabit network switches connected by a composite cable that runs fiber and power to each device. Each switch will also house up to two wireless radios, giving pervasive wireless coverage along travel ways. This will also provide the ability to make continuous VoIP telephone calls from the portal to the face, and complete asset and personnel tracking. The system will also have redundancy to keep it running in the event that the fiber gets damaged. Figure 16.29 shows a schematic of the proposed typical underground communications system.

Figure 16.29 Underground Communications System Schematic



The network system "Head End Unit" will reside in the portal indoor substation. The two network backbone cables will branch out through the portals into the underground access declines–one in the main access decline and one in the conveyor decline. Amplifiers will be spaced out between ultra-high frequency (UHF) coax cable segments at no more than 350 m spacing. A communications cable will also branch out at drifts as necessary, with "end-of-line" termination antennas as required.

Personnel Tracking

Personnel tracking will be accomplished using a radio frequency identification (RFID) tag system. An integrated communications cap lamp will contain the RFID tag.

Vehicles will also contain RFID tags and UHF radios. The system will be integrated into MineDash, a browser-based tracking and reporting application, allowing operators and mine controllers to monitor, track and allocate personnel and resources.

Movement of personnel, vehicles, and other assets will be monitored throughout the mine. Having the ability to ensure that mine staff are accounted for in an emergency will increase safety and speed the provision of help to any injured personnel. Tracking

vehicles and assets can also lead to increased productivity and efficiency by eliminating time wasted looking for equipment underground.

Fixed Plant Monitoring and Control

A programmable logic controller (PLC) system will be used for fixed plant monitoring and control. The PLC system processor (main rack) will reside in the portal indoor substation. Remote PLC racks placed near equipment (as necessary) will monitor and control the underground systems, including, but not limited to:

- rock box levels
- crusher
- conveying equipment
- magnet
- substations
- sumps and pumps
- ventilation doors
- fuel delivery
- traffic control
- air quality and quantity.

A fiber optic backbone stemming from the portal substation will be used for remote input/output racks and Internet protocol network communications. A fiber to copper network switch at the portal substation will connect the fiber backbone. Two independent fiber cables will branch out through the portals into the underground access declines, one in the main access decline and one in the conveyor decline. Fiber to copper switches will be installed at substations and at remote input/output locations, bridging the network together.

The PLC system will be tied to the mill and control room on surface using a wireless antenna to bridge the underground network and the control room networks together.

Underground Stores Data Management

The underground stores data management system will use the fiber optic network to link its assets to a central web server, whereby any computer on the information technology network can access the database.

Collision Avoidance

A proximity detection system will allow drivers to be aware of other vehicles or personnel that are 60 to 120 m away from their vehicles. This facilitates overall safety in the mine. This system will also integrate with the integrated communications cap lamps.

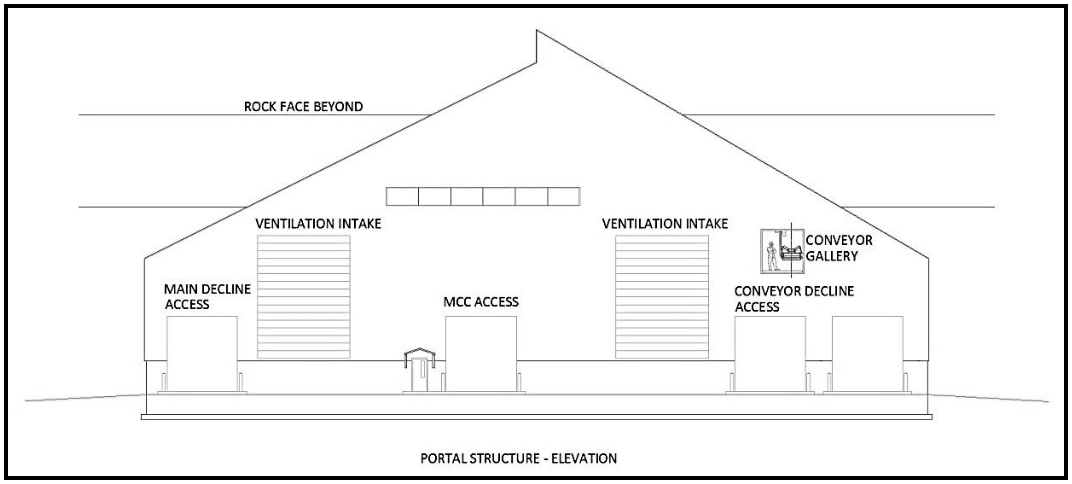
16.9.11 PORTAL STRUCTURE

A portal structure (see Figure 16.30) will be constructed at the access to the underground decline tunnels. The structure will span the area between both decline tunnels and will house the mine air heaters and ventilation fans, the conveyor drive motor and mechanism, and an electrical substation. The main decline conveyor will exit up from the tunnel and pass through the portal structure on its way to the mill. Access into the portal structure will be via one of four overhead doors.

The portal structure was designed to be built up against the mill site high wall and will be required to resist roof snow loads with pressures up to 400 kg/m^3. The roof was designed with 6:12 and 7:12 pitches to better shed snow. A ridgeline roof split will also help initiate snow movement from the roof.

A monorail located in the ceiling of the portal structure will allow for removal of the mine air fan motor and components.

Figure 16.30 Portal Structure Schematic



16.9.12 HEATING SYSTEM AND PROPANE STORAGE

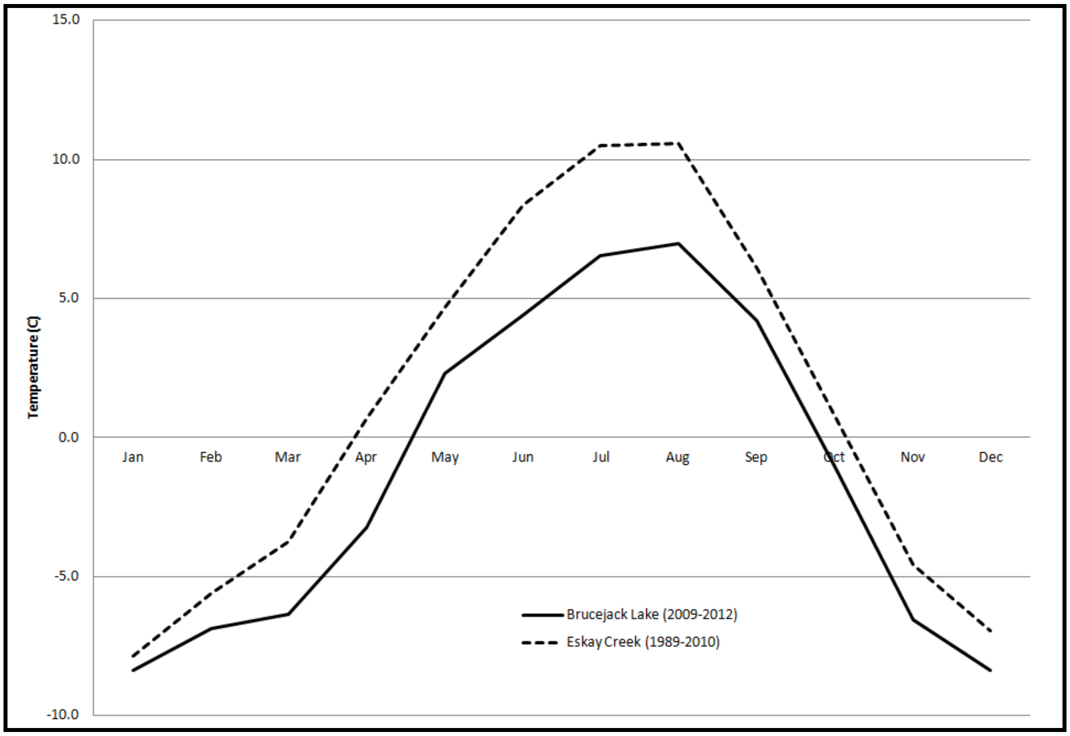
The energy to run mine air heaters is typically derived from natural gas, propane, electricity, and diesel. With exception of natural gas, the other energy types will potentially be available at the Project site. As part of the feasibility study, AMC examined the merits of alternative types of energy supply to run the heaters. Based on the results of the analysis and the site availability of electrical power, a hybrid electrical/propane powered mine heating system was determined as the best outcome for the Project.

CLIMATIC DATA

Climactic data from site was analyzed to quantify the amount of annual power and propane required for mine air heating. AMC was provided with a climatic data set collected from a weather station located adjacent to Brucejack Lake, at an elevation of 1,400 m.

The data was collected over a relatively short time period (2.5 years, October 2009 to March 2012) and may not be representative of longer-term averages. To address this concern, AMC analyzed climatic data collected from the weather station adjacent to the Eskay Creek Mine, located approximately 19 km northwest of the Brucejack site at an elevation of 900 m. Minimum, maximum, and mean daily temperatures were collected from 1989 to 2010, yielding 21 years of comparative data. Figure 16.31 shows the monthly mean temperatures collected from the two weather stations.

Figure 16.31 Monthly Mean Temperatures



The trend in temperature change over the course of a year at Eskay Creek parallels the Brucejack data, but is consistently higher in absolute value. It is probable that the 500 m elevation difference between the two stations accounts for this differential value. AMC concludes that the data collected from the Brucejack weather station is representative of long-term averages.

AMC has assumed the coldest temperature that can be anticipated at Brucejack Lake is approximately -35°C.

16.9.13 PROPANE SUPPLY AND STORAGE

Mine air heating will be the only consumer of propane for the underground operations. Surface infrastructure–for example the camp–will require propane, however, the storage of propane for this purpose will be independent of mine air heating. Based on the available climatic data from site, calculations were performed to estimate the annual

propane consumption. Table 16.18 shows the monthly and annual propane requirement.

Table 16.18 Propane Consumption

Month	Propane Consumption (L)
January	123,984
February	68,548
March	44,881
April	5,976
May	22
June	25
July	31
August	35
September	38
October	164
November	45,417
December	90,762
Total	**379,883**

Propane for mine air heating will be required to be delivered to site approximately six months of each year. The Project site will be accessible by a planned permanent route that begins at km 215 of Highway 37 which extends to the Bowser Transfer Station. At the Bowser Transfer Station, all equipment and supplies will be transferred onto a Husky-tracked vehicle for transport to the mine site.

The access route is subject to avalanche risk and extreme winter weather. To avoid mine production interruptions, a one-week supply of propane capable of supporting a constant air temperature of -25°C should be maintained on site. This equates to a tank farm of four 45,000 L tanks.

The propane delivery strategy to site is as follows:

- An 18,000 L propane delivery truck drives from Terrace, BC to the Bowser Transfer Station.
- The truck will transfer its load into a 24,000 L tank. The ISO tanks will be modularized and skid mounted, and sized to meet the footprints and weights capable of being transported on the tracked vehicles.
- At the mine site, the tanks will transfer propane into the site tank farm.
- The tanks will supply propane to the heaters by means of a buried pipeline.

The frequency of propane delivery is dependent upon the air temperature and airflow volume required for the mine. During the coldest month of the year–January–at the

maximum anticipated airflow volume, the mine air heaters will consume approximately 4,000 L of propane each day of the month. At this rate, to maintain supply on site two delivery truck loads will be required each week.

16.9.14 DIESEL STORAGE

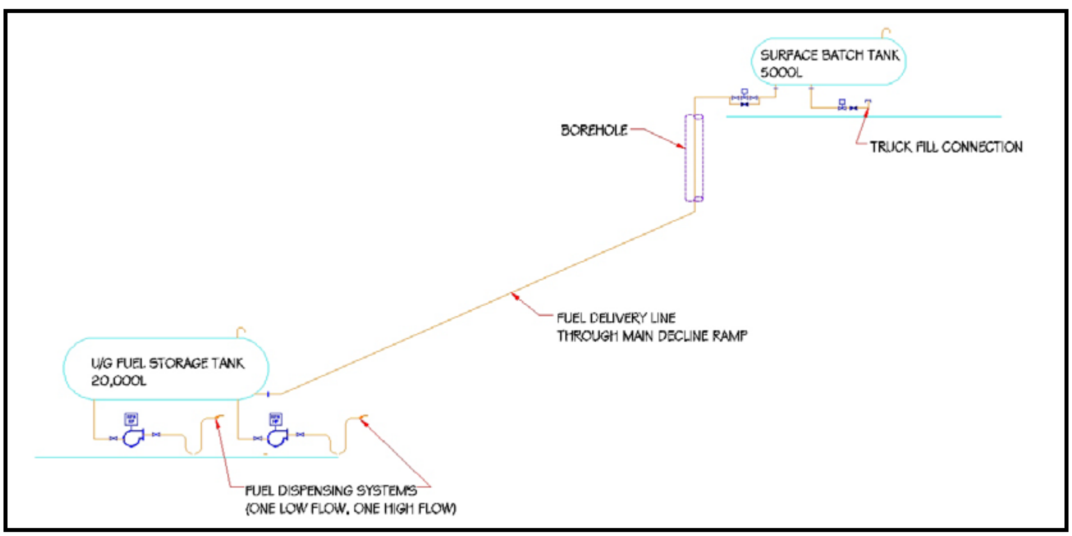
A storage and delivery system will provide fuel underground for vehicles and equipment. The system will consist of a 20,000 L storage tank located underground in the fuel bay area. This storage tank will be filled from a 2 in diameter line that connects the storage tank to a 5,000 L batch tank located near the portal. The fuel line will be routed from the surface batch tank through a bore hole connected to a pipe in the main access tunnel, where it will lead into the fuel storage bay.

To fill the underground storage tank, the surface tank will be filled by tank truck and an automatic valve located on surface is opened, emptying the batch tank by gravity. The underground storage tank and batch tank will be fitted with vents to allow air to escape during the fuel transfer process. The drain on the surface batch tank will stay open long enough to drain the tank of fuel and allow the connecting 2 in diameter drain line to completely empty into the underground storage tank. The fuel line will remain empty until the next fuel transfer is initiated.

The surface batch tank will have a volume of 5,000 L, as opposed to the underground storage tank volume of 20,000 L. A completely empty storage tank will require four complete batch tank emptying cycles to fill. A level indicator on the underground storage tank will show the level on an indicator located next to the surface batch tank. The indicator will be interlocked with the surface valve, allowing the valve to be opened only if the level in the underground storage tank is low enough to allow the complete volume of the surface batch tank to be emptied into it.

A level alarm, located in the underground storage tank, will sound if the level reaches a high-high level. Figure 16.32 shows a schematic of the fuel delivery system.

Figure 16.32 Fuel Line Schematic



16.10 PASTE FILL DISTRIBUTION

P&C completed a feasibility level paste backfill distribution design for the Project. The proposed paste fill distribution system transports the paste from the surface plant to the underground stopes through a pipeline system. The paste was characterized through laboratory rheology testing on un-cemented paste samples. A summary of the paste distribution system follows:

- The paste fill distribution will require a two-stage pumping system. A positive displacement pump in the paste fill plant will provide paste to all of the West Zone (West Zone Upper and West Zone Lower) and the lower zones of the VOK Zone (VOK, below the 1,330 Level). The paste plant pump will also feed a booster pump located near the crusher station at the bottom of the conveyor ramp. This booster pump will pump paste up to the Galena Hill and the upper VOK Zone (1,330 Level and above).
- The paste pumps will both be positive displacement piston pumps of 98 m^3/h peak capacity with a pressure rating of 120 bar. The nominal flow rate for the system will be 79 m^3/h, with a nominal design supply rate of 112 dry tonnes per hour.
- An underground booster pump station will be required to house the positive displacement pump, the pump hopper, and a water tank with a high-pressure pump for pipeline flushing. A smaller cubby adjacent to the station will be required for the pump hydraulic packs.
- Two large flush-out areas situated at low points in the system will be necessary for pipeline diversion during regular shutdown procedures and operation upsets. These will be sumps that can be mucked out regularly.
- Instrumentation required to ensure controlled operation will include 10 permanent pressure gauges; three permanent cameras and four mobile cameras for the pour points; automated diversion valves at the sumps, and integrated process control with the paste fill plant.
- Hydraulic modelling shows that this system will provide paste to the stopes at a nominal yield stress of 248 Pa with a range of 109 to 375 Pa. This equates to an un-cemented paste percent solids of 75.5% solids by weight (ranging from 74 to 77% solids by weight).
- The piping specified for this distribution system is 8 in API 5L X52. The schedule of the pipe varies with the pressure rating of the area: borehole casing and loops in the upper VOK Zone levels will be Schedule 120, while the lower VOK Zone and all the West Zone casing and loops will be Schedule 80. The main drift piping (trunk) and level piping to the stopes will be Schedule 80 and Schedule 40, respectively. Victaulic couplings will be used as the connection method.

16.10.1 Distribution System Design

The pipe routing for the underground distribution system (UDS) was developed with consideration given to site conditions and client preferences in combination with pipeline operation experience and hydraulic modelling. Some of the conditions that were taken into account in the design include:

- the availability of the conveyor ramp down to the 1,300 Level, which is isolated from vehicle traffic
- the difficulty foreseen in accessing any trenched pipelines on surface due to site conditions, especially during winter months
- the mining schedule, which defines that the VOK Zone will be developed in the early years while the West Zone will only be developed in the second half of the mine life
- the long distance from the paste fill plant to the underground workings (more than 800 m)
- the location of the paste fill plant below the elevation of the top third of the VOK ore zone

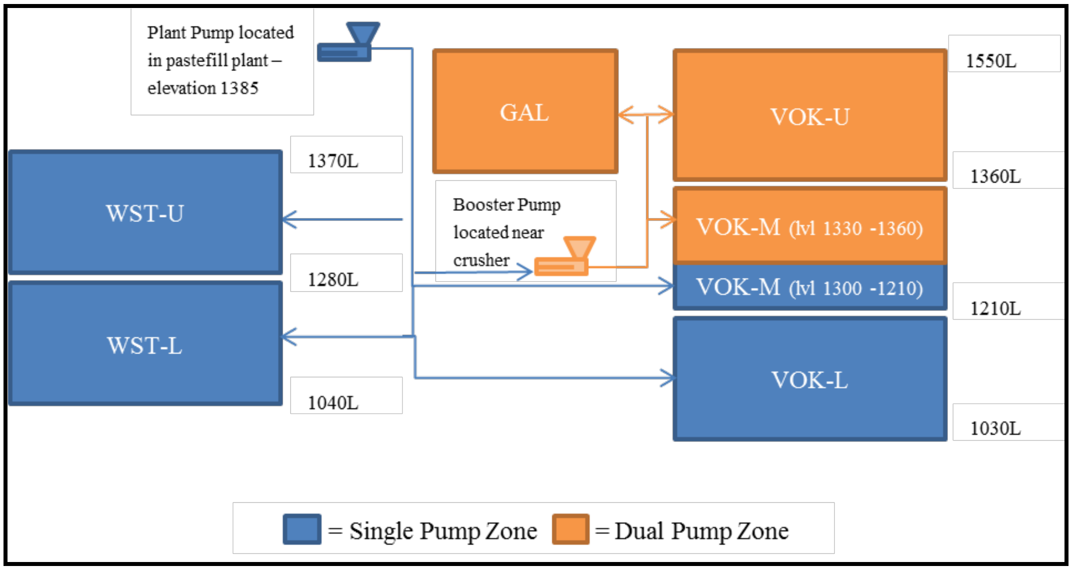
The mining schedule breaks down the Brucejack orebody into six areas: VOK-U, VOK-M, VOK-L, GAL, WST-U, and WST-L, as shown with their respective elevations in Figure 16.7 and Figure 16.8. The first areas to be mined are the VOK-U, GAL and VOK-M; coming on-line in Years 1, 2 and 3, respectively. Production in VOK-L will start in Year 6, while the WST Zones will only come on-line after Year 15. The VOK zones have continuous production scheduled until Year 20. The paste fill distribution system was designed with this schedule in mind.

The main challenge for the Brucejack paste fill distribution system is that a portion of the orebody is located above the elevation of the paste fill plant. A balance in strategy is required to ensure that paste can be pumped to this section of the orebody without compromising the quality and proper flow distribution to the rest of the mine.

16.10.2 Distribution Approach

The philosophy developed for the paste fill distribution system is a dual pumping system. This will optimize the pumping capacity and minimize wear on the paste pumps. A positive displacement pump in the paste fill plant will provide paste to all of the West Zone (WST-U and WST-L) and the lower VOK (below the 1,330 Level). The paste plant pump will also feed a booster pump located near the crusher station at the bottom of the conveyor ramp, and near to the main entrance to the VOK Zone area on the 1,330 Level. This booster pump will pump paste up to the Galena Zone and the upper VOK Zone (1,330 Level and above). Figure 16.33 shows the breakdown of the Brucejack ore zone by the paste pumps feeding them: single pump zone and dual pump zone.

Figure 16.33 Paste Fill Distribution System Schematic Showing Paste Pumping Zones



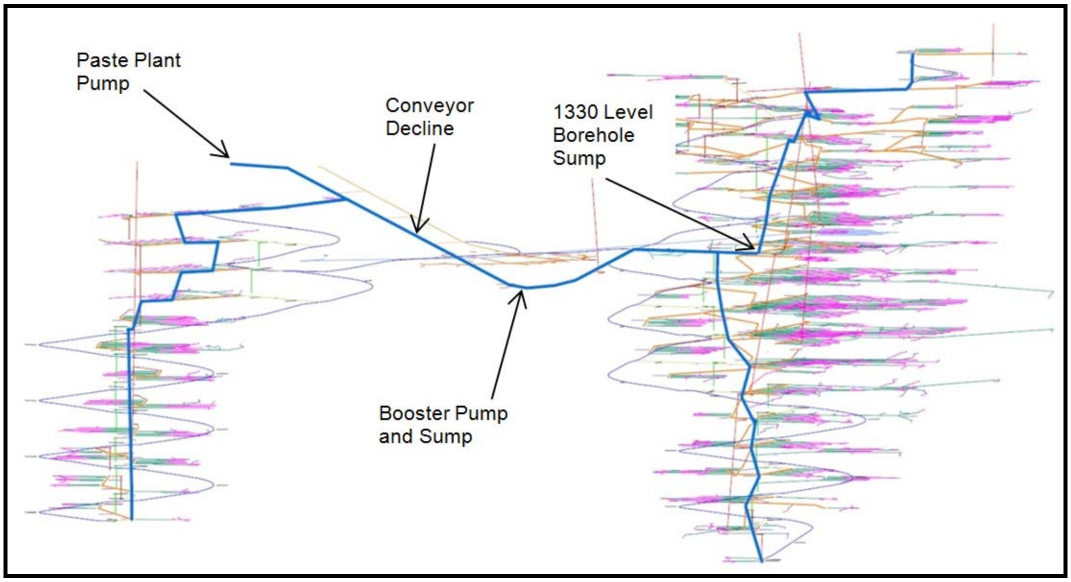
16.10.3 Distribution System Layout

The underground perspective view of the paste fill distribution system is provided in Figure 16.34.

Key points of the piping strategy are:

- one pump plus installed spare at the pastefill plant
- one booster pump plus installed spare near the crusher station, 1,300 Level
- main distribution pipeline in the conveyor decline
- paste access drift off the conveyor ramp on the West Zone 1,370 Level
- two sumps to divert paste from the pipeline during operation upsets.

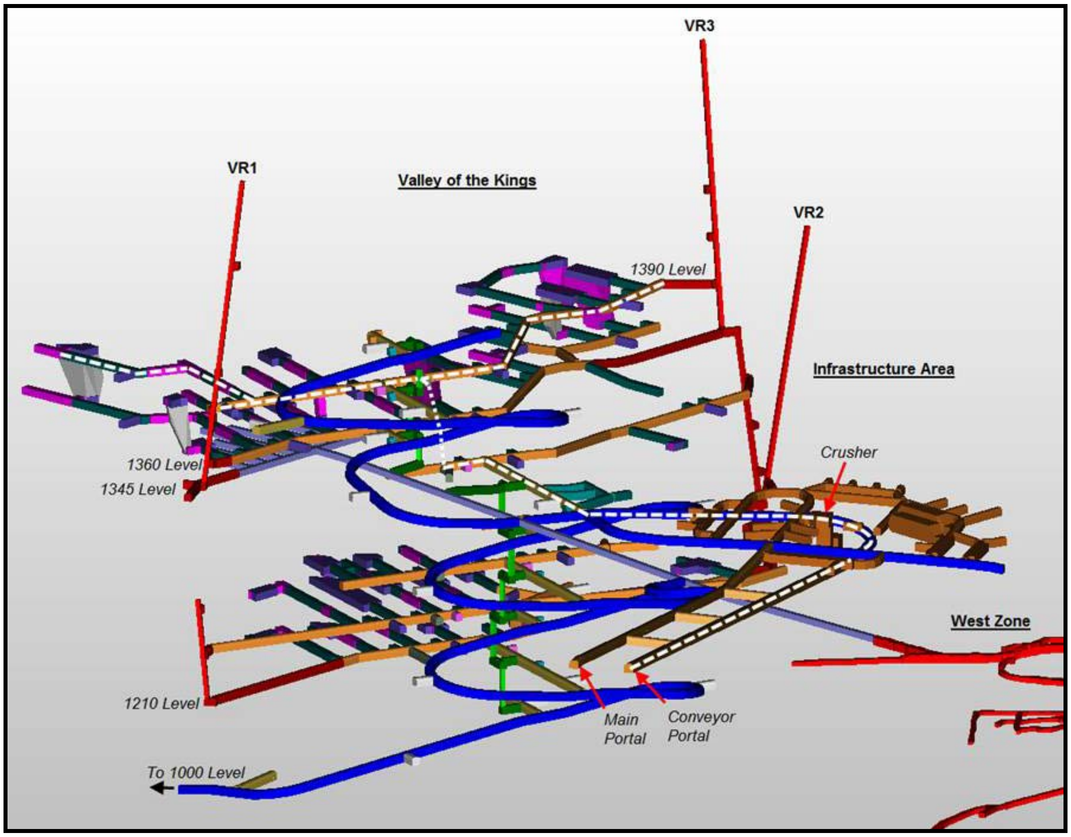
Figure 16.34 Paste Fill Distribution System Schematic



16.10.4 PRE-PRODUCTION REQUIREMENTS

The permanent paste fill line will be installed in the pre-production phase in the development of the mine, as it will be required in order to commence stoping. Figure 16.35 illustrates the placement of the paste fill line during this phase.

Figure 16.35 Pre-production Paste Fill Line Requirement



16.11 MANPOWER REQUIREMENTS

16.11.1 SCHEDULE

During the pre-production phase, lateral development and raising will be completed by a contractor. Ongoing lateral development will be completed by the Owner's workforce shortly after production commences.

As the Project site is remote, a reasonable crew rotation is required to attract the skilled labour that will be necessary for operations. A two-week-in, two-week-out rotation was chosen for the Project. The working time per day is based on an 11-hour shift. This will allow one hour for smoke to clear after end-of-shift blasting. However, the effective working time per day is less than 11 hours considering travel time, daily safety briefs, and pre-start safety checks. AMC estimates the effective working time per shift during production operations to be 8.25 hours.

To operate an 11-hour shift, a variance must be in place with the BC government (to allow work over 8 hours per shift). The current mine contractor has obtained such a variance for the completion of the bulk sample program. AMC believes this to be effectively a formality, given a number of recent examples of BC mines that operate with longer daily shifts.

16.11.2 Organization and Manpower

The underground mining team will be organized into operational groups consisting of mining, logistics, maintenance, and technical support. The mining group is further broken down into mining supervision, production, development, and raising. Table 16.19 shows the total hired personnel by operational group when the mine has reached full steady state production.

Table 16.19 Manpower by Operational Group

Role	Head Count
Mining Supervision (7)	
Underground Superintendent	1
Safety/Training/First Aid	4
Mine Captain	2
Development Crew (72)	
Development Shift Boss	4
Jumbo Operators	12
Bolter Operators	8
Cable Bolter Operators	8
LHD Operators	8
Truck Operators	8
Blasters	8
Service Installers	16
Production Crew (96)	
Production Shift Boss	4
Long Hole Drillers	16
Blasters	8
LHD (Electric) Operators	20
Truck Operators	12
Crusher Operator	4
Crusher Labourer	4
Backfill Leader	4
Timber Men	16
Backfill Operator	8
Raising (5)	
Raise Leader (Contract)	1
Raise Miner (Contract)	3
Raise Mechanic (Contract)	1
Logistics (25)	
Underground Chief of Logistics	1
Underground Warehouse Manager	4
Boom Truck/Grader Operators	12
Clerk/Labourer/Forklift Operator	8

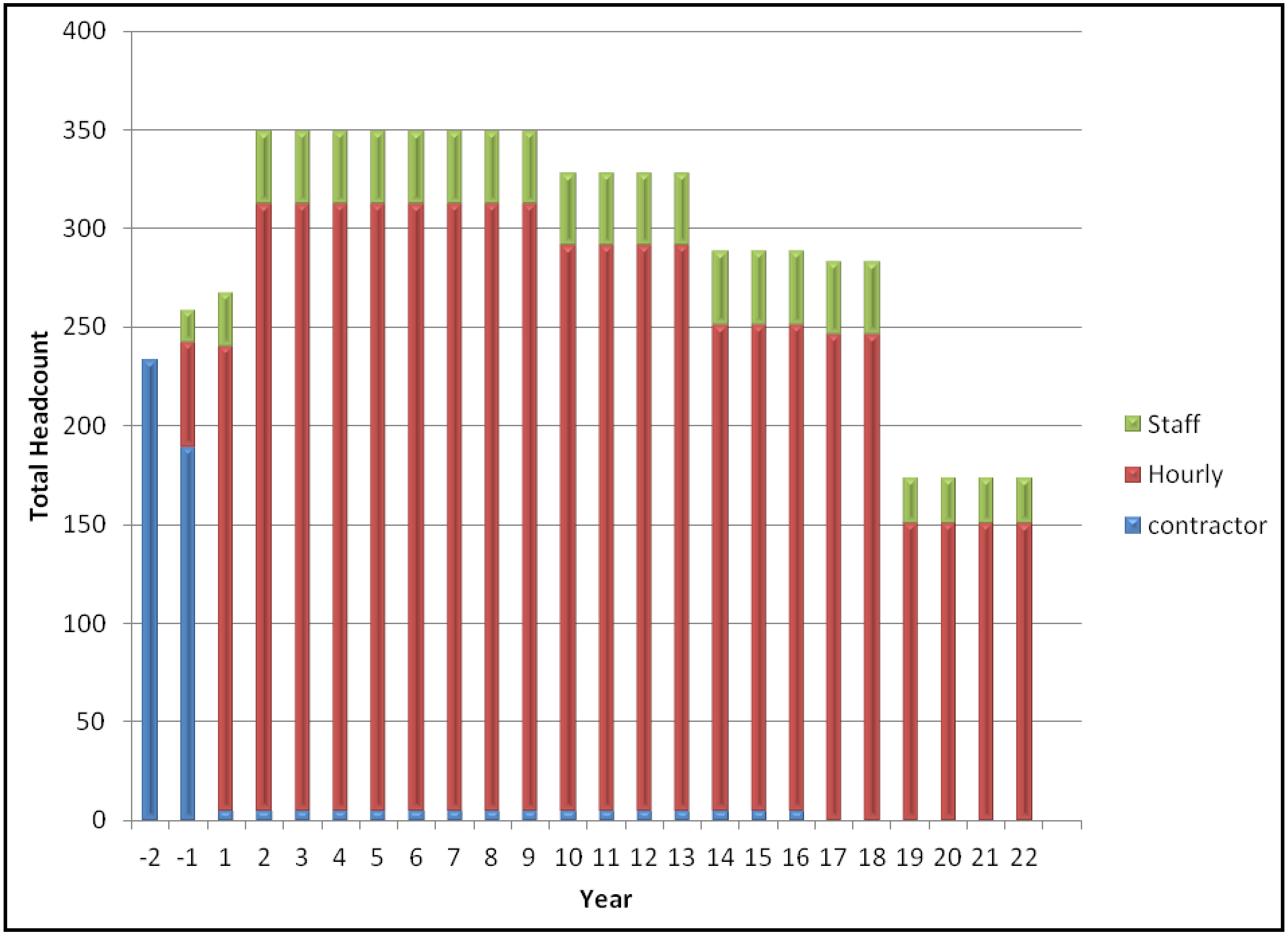
table continues…

Role	Head Count
Maintenance (72)	
Maintenance Superintendent	1
Master Mechanic	2
Mechanics	20
Mill Wrights	8
Apprentices/Labourers	12
Welders	8
Chief Electrician	1
Lead Electrician	1
Electricians	15
Electrical Apprentices	4
Technical Services (36)	
Technical Services Manager	1
Manager Secretary	1
Senior Geologist	1
Production Geologist	2
Geological Technologist	8
Production Diamond Drillers	8
Senior Engineer	1
Engineer	3
Mine Planning & Scheduling	2
Surveyors	8
Geotechnical Engineer	1
Additional Hires (36)	
Total Personnel	**349**

Figure 16.36 shows the manpower loading through the mine life. Years -1 and -2 are periods of construction and early production (mill operations begin in month 18).

Initial loading will primarily be provided by the contractor, with technical support provided by the Owner. During months 18 through 24, mine forces will transition from contractor to Owner.

Figure 16.36 Manpower Loading by Year



17.0 RECOVERY METHODS

17.1 MINERAL PROCESSING

17.1.1 INTRODUCTION

The Brucejack deposit mineralization typically consists of quartz-carbonate-adularia, gold-silver bearing veins, stockwork and breccia zones, along with broad zones of disseminated mineralization. Gold and silver are the major economical metals contained in the mineralization. There is a significant portion of gold and silver present in the form of nugget or metallic gold and silver.

The proposed concentrator will be conventional and will process gold and silver ore at a nominal rate of 2,700 t/d with an equipment availability of 92% (365 d/a). The concentrator will produce gold-silver doré from the gold and silver recovered by gravity concentration and smelting at the mine site. A gold-silver bearing flotation concentrate will also be produced, which will be sold and shipped off site. The mill feed will be supplied from the underground mine using conventional mining methods.

17.1.2 SUMMARY

The process flowsheet developed for the Brucejack mineralization is a combination of conventional bulk sulphide flotation and gravity concentration to recover gold and silver. The process plant will produce a gold-silver bearing flotation concentrate and gold-silver doré from melting the gravity concentrate produced from the gravity concentration circuits. Based on the LOM annual average, the process plant is estimated to produce approximately 4,300 kg of gold and 1,500 kg of silver as doré and 42,000 t of gold-silver bearing flotation concentrate from the mill feed, grading 12.0 g/t gold and 57.9 g/t silver. The estimated gold recoveries to the doré and flotation concentrate are 41.6% and 54.9%, respectively, totalling 96.5%. The estimated silver recoveries reporting to the doré and flotation concentrate are 3.0% and 86.6%, respectively, totalling 89.6%. The LOM average gold and silver contents of the flotation concentrate are anticipated to be approximately 130 g/t gold and 1,000 g/t silver. The flotation concentrate will be shipped off site to a smelter for further treatment to recover the gold and silver.

The process plant will consist of one stage of:

- crushing located underground
- a surge bin with a live capacity of 2,500 t on surface
- a primary grinding circuit integrated with a gravity concentration
- rougher flotation followed by rougher flotation concentrate regrinding

- cleaner flotation processes.

A gravity concentration circuit will also be incorporated in the bulk concentrate regrinding circuit. The final flotation concentrate will be dewatered, bagged, and trucked to the transload facility in Terrace, BC. It will be loaded in bulk form into rail cars for shipping to a smelter located in eastern Canada. The gravity concentrate will be refined in the gold room on site to produce gold-silver doré.

A portion of the flotation tailings will be used to make paste for backfilling the excavated stopes in the underground mine, and the balance will be stored in Brucejack Lake. The water from the thickener overflows will be recycled as process make-up water. Treated water from the water treatment plant will be used for mill cooling, gland seal service, reagent preparation, and make-up water.

17.1.3 FLOWSHEET DEVELOPMENT

The process flowsheet was developed based on test work conducted mainly from 2009 to 2013, as well as engineering experience. The comminution circuit design was based on the topography of the proposed plant site and operability of the system. The size selection of the grinding mills was based on the amenability of the ore to grinding, as determined through test programs performed by different laboratories. Grindability tests were performed to determine the following hardness parameters:

- BWi
- RWi
- CWi
- Ai
- SAG mill comminution breakage.

The gravity circuit was selected based on the laboratory gravity concentration tests and GRG test results and related simulations.

Flotation cell sizing was based on optimum flotation times, which were determined by test work and using scale-up factors from similar operations.

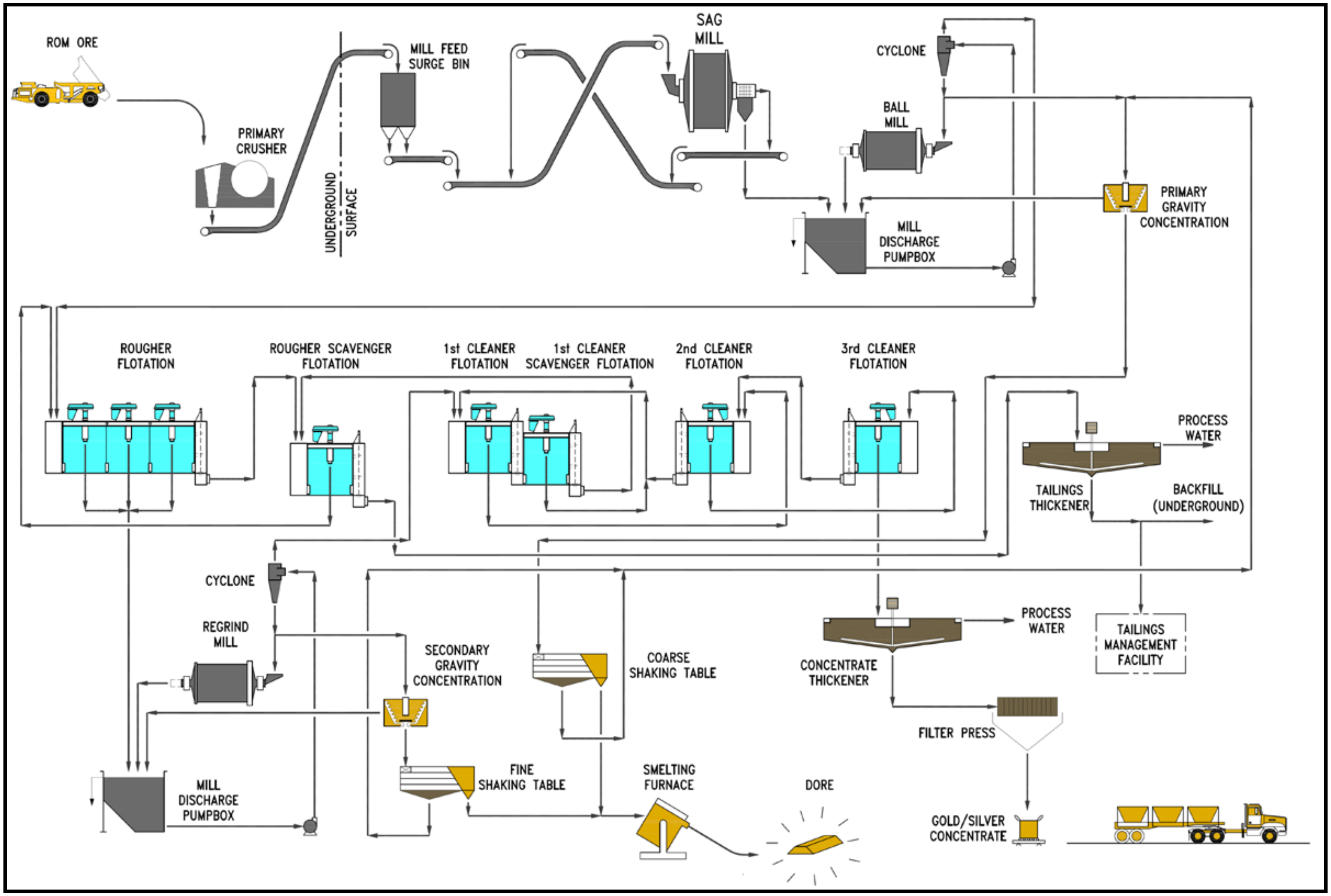
Various test programs evaluated the cyanidation process to extract the gold and silver from the test head samples and flotation concentrates. Although the mineralization responds reasonably well to the process, a few of the samples showed poor cyanidation responses, which may be a result of the presence of preg-robbing constituents (graphite), potential refractory components (arsenopyrite), and slow leaching kinetics from electrum or other silver bearing minerals. However, the cyanidation process has not been used for this study, including direct cyanide leach and carbon-in-leach procedures. Further studies are required to evaluate the feasibility of the cyanidation process to recover the gold and silver from the flotation concentrate, including further test work to more fully assess the metallurgical response of the mineralization to cyanidation.

The process plant will consist of the following:

- primary crushing underground
- a conveying system for crushed ore
- primary grinding and gravity concentration
- rougher/scavenger flotation
- bulk flotation concentrate regrinding and gravity concentration
- cleaner flotation
- gravity concentrate smelting to produce doré
- flotation concentrate dewatering, bagging, and load out
- tailings disposal to the tailings impoundment or to the underground mine for backfilling.

The simplified flowsheet for the operation is shown in Figure 17.1.

Figure 17.1 Simplified Process Flowsheet



17.1.4 PLANT DESIGN

MAJOR DESIGN CRITERIA

The process plant is designed to process 2,700 t/d, equivalent to 985,500 t/a. The major criteria used in the design are outlined in Table 17.1.

Table 17.1 Major Design Criteria

Criteria	Unit	Value
Daily Processing Rate	t/d	2,700
Operating Days per Year	d/a	365
Operating Schedule	-	two shifts/day; 12 hours/shift
Mill Feed Grades – Average	g/t Au	10 to 15
	g/t Ag	40 to 100
	% S	3
Metal Recovery – Doré	% Au	30 to 50
	% Ag	1 to 20
Metal Recovery – Flotation Concentrate	% Au	50 to 70
	% Ag	70 to 92
Primary Crushing (Underground)		
Crushing Availability	%	65
Crushing Product Particle Size, 80% passing	mm	120 to 150
Grinding/Flotation/ Gravity Concentration		
Availability	%	92
Milling and Flotation Process Rate	t/h	122
SAG Mill Feed Size, 80% passing	mm	120 to 150
SAG Mill Grinding Particle Size, 80% passing	µm	1,070
Drop Weight Breakage Parameter	Axb	41.4
Ball Mill Grinding Particle Size, 80% passing	µm	74
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index	kWh/t	16.0
Nugget Gold Recovery from Primary Grinding Circuit	-	Centrifugal and Tabling Gravity Concentration
Rougher Flotation Concentrate Regrinding Particle Size, 80% passing	µm	35 to 40
Nugget Gold Recovery from Reground Concentrate	-	Centrifugal and Tabling Gravity Concentration
Upgrading of Gravity Separation Concentrates	-	Direct Smelting

OPERATING SCHEDULE AND AVAILABILITY

The process plant is designed to operate on the basis of two 12-hour shifts per day, 365 d/a. The overall availability for the underground primary crusher circuit will be 65%. The grinding, flotation, and gravity concentration availability will be 92%. The gold room will be operated during the day shift only. These availabilities will allow for a potential increase in crushing rate, downtime for scheduled and unscheduled maintenance of the crushing and process plant equipment, and potential weather interruptions.

17.1.5 PROCESS PLANT DESCRIPTION

PRIMARY CRUSHING (UNDERGROUND)

The primary crushing facility with an average process rate of 173 t/h will be located underground. A jaw crusher is proposed for primary crushing.

The major equipment and facilities for the underground primary crushing facility include:

- a hydraulic rock breaker
- a stationary grizzly
- a jaw crusher (160 kW)
- a vibrating grizzly feeder
- associated dump pocket and belt conveyor
- belt scales
- a dust collection system.

The run-of-mine (ROM) material will be trucked from the underground mine to the underground primary crushing facility. The particle size of the jaw crusher feed will be less than 500 mm. The jaw crusher will reduce the ROM material to 80% passing 120 to 150 mm. A rock breaker will be installed to break any oversize rocks.

The crusher product will be transported by a conveyor system from the underground primary crushing facility to the SAG mill feed surge bin located on surface. The primary crushing and conveying facilities will be equipped with a dust collection system to control fugitive dust generated during crushing and conveyor loading. The crushing and conveying system will be monitored by closed-circuit television (CCTV) and can be controlled from the process central control room, located in the process plant.

MILL FEED SURGE BIN

The SAG mill feed surge bin is designed to have a live capacity of 2,500 t. The crushed product from the underground primary crushing facility will be conveyed to the transfer tower and then further transported to the SAG mill feed surge bin. The transfer tower, which is part of the process/administration/truck shop complex, will receive the crushed material from the underground primary crushing facility and transfer the ore onto the surge feed conveyor.

The ore from the mill feed surge bin will be reclaimed by four belt feeders onto the SAG mill feed conveyor at a nominal rate of 122 t/h.

The stocking and re-handling system for the crushed ore will include:

- a 1,150 mm wide jaw crusher discharge belt conveyor
- a transfer tower located at the surface

- a belt conveyor, 914 mm wide by 97.7 m long, to feed the SAG mill feed surge bin
- a mill feed surge bin with a live capacity of 2,500 t
- four belt feeders, 920 mm wide by 6,25 m long
- three local dust collection systems.

Each of the crushed ore transfer points will be equipped with a dust collection system to control fugitive dust that is generated while transporting the crushed material.

Primary Grinding, Classification and Primary Gravity Concentration

A SAG mill/ball mill (SAB) circuit is proposed for primary grinding. The circuit will be equipped with a centrifugal gravity concentrator to recover gold/silver nugget grains that are liberated or partially liberated from their host minerals.

The primary grinding circuit will consist of a SAG mill and a ball mill in a closed circuit with classifying hydrocyclones and a centrifugal gravity concentrator. Grinding will be conducted as a wet process at a nominal rate of 122 t/h of ore.

The grinding/gravity concentration circuit will include:

- one SAG mill, 5,790 mm diameter by 2,950 mm long (19 ft by 9.7 ft) (EGL), driven by a 1,300 kW variable frequency drive (VFD)
- one ball mill, 3,960 mm diameter by 7,260 mm long (13 ft by 23.8 ft) (EGL), powered by a 1,600 kW fixed speed drive
- two hydrocyclone feed slurry pumps
- four 500 mm hydrocyclones
- one centrifugal gravity concentrator
- one particle size analyzer
- one online sampler.

The crushed ore from the surge bin will be reclaimed onto the belt conveyor that feeds the ore to the SAG mill. The SAG mill will be equipped with 40 mm pebble ports to discharge the fine fraction from the SAG mill. The SAG mill discharge will be screened by a trommel screen that is integrated with the SAG mill. The trommel screen will have an opening of 9.5 mm (slot wide). The oversize from the trommel screen will be transported by three 500 mm-wide belt conveyors back to the SAG mill feed conveyor. The screen undersize will discharge by gravity to the hydrocyclone feed pump box in the grinding circuit.

The ball mill will be operated in closed circuit with hydrocyclones and a centrifugal gravity concentrator. The product from the ball mill will be discharged into the hydrocyclone feed pump box where the ball mill discharge will be combined with the SAG mill trommel screen undersize slurry and the gravity concentration tailings. The blended slurry in the

pump box will be pumped to the hydrocyclones for classification. Approximately 67% of the hydrocyclone underflow will return by gravity to the ball mill, while 33% of the hydrocyclone underflow will flow by gravity to the centrifugal concentration circuit. The circulating load to the ball mill will be approximately 300%. The particle size of the hydrocyclone overflow, or the product of the primary grind circuit, will be 80% passing 74 µm. The pulp density of the hydrocyclone overflow slurry will be approximately 33% solids. Steel balls will be manually added into the mills on a batch basis as grinding media.

The gravity concentration process will recover nugget gold particles from the hydrocyclone underflow. Tailings from gravity concentration will return to the hydrocyclone feed pump box by gravity. The gravity concentrate will be pumped to the gold room for further upgrading by tabling. Tailings from the tabling will be recycled back to the centrifugal gravity concentrator feed well, while the concentrate from the table will be further upgraded by smelting. The gravity concentration circuit will have a security enclosure and CCTV cameras; access will be restricted to authorized personnel only.

Dilution water will be added to the grinding circuit as required. A particle size analyzer will be installed to monitor and optimize the operation efficiency, in conjunction with an automatic sampling system and the required instrumentation such as solid density, pressure, and flow rate meters.

Rougher and Scavenger Flotation

The pulp from the primary grinding circuit will be subjected to conventional flotation to recover the free gold, silver, and their bearing minerals from the materials being processed. The feed rate for the flotation circuit will be 122 t/h of ore. Flotation reagents will be added to the flotation circuits as defined through testing. The flotation reagents include PAX as collector and MIBC as frother. The mass recovery of the rougher concentrate is approximately 20% of the flotation feed. The concentrates produced from the rougher flotation circuit will be sent to the regrind circuit and subsequently to the cleaner flotation circuit. The rougher flotation tailings will be further floated by scavenger flotation, along with the tailings from the first cleaner flotation circuit. The scavenger concentrate will be returned to the preceding rougher flotation head. Rougher and scavenger flotation will be carried out at the natural pH level (without slurry pH adjustment). The rougher/scavenger flotation circuit will consist of:

- four 100 m^3 rougher flotation tank cells
- two 100 m^3 scavenger flotation tank cells.

The tailings from the flotation circuit will be discharged to the tailings thickener. Depending on the mining operation requirement, the thickener underflow will be pumped to the paste backfill plant for excavated stope backfilling underground and/or to Brucejack Lake for storage.

Automatic sampling systems will be installed to collect the samples required for process optimization and metallurgical accounting.

CONCENTRATE REGRINDING

The flotation concentrate from the rougher flotation circuit will be forwarded to the regrinding circuit. The major equipment in the circuit includes:

- one 2,440 mm diameter by 4,270 mm long (EGL) ball mill, driven by a 265 kW motor
- one centrifugal gravity concentrator
- two cyclone feed pumps
- four 250 mm hydrocyclones.

The rougher flotation concentrate will be reground to 80% passing 35 to 40 µm in the ball mill, in closed circuit with hydrocyclones and one centrifugal gravity concentrator. The discharge from the ball mill and the tailings from the gravity concentrator will report to the hydrocyclone feed pump box, from where the slurry will be pumped to the hydrocyclones for classification. Approximately 50% of the hydrocyclone underflow will return by gravity flow to the ball mill, while the balance will report by gravity flow to the centrifugal gravity concentration circuit. The ball mill circulating load will be approximately 150%. The pulp density of the hydrocyclone overflow slurry will be approximately 22% solids. Steel balls will be manually added into the mills on a batch basis as grinding media.

The gravity concentrator will recover metallic gold grains from the hydrocyclone underflow. The tailings from the gravity concentration will return to the hydrocyclone feed pump box by gravity flow. The gravity concentrate will be sent to the gold room for further upgrading by tabling. Similar to the gravity concentration in the primary grinding circuit, this gravity concentration area will also be in a secure area and monitored by CCTV cameras.

The particle size of the hydrocyclone overflow will be automatically sampled and monitored. An instrumentation system, similar to that used for the primary grinding circuit, will be installed for the regrinding circuit to optimize the grinding efficiency and control the regrind particle size.

CLEANER FLOTATION

The reground concentrates will undergo three stages of cleaning by flotation in order to produce a final gold-silver bearing concentrate.

The major equipment in the cleaner flotation circuit includes:

- four 30 m^3 tank cells for the first cleaner flotation
- two 30 m^3 tank cells for the first cleaner/scavenger flotation
- one 30 m^3 tank cell for the second cleaner flotation
- one 30 m^3 tank cells for the third cleaner flotation.

The reground rougher concentrate will be initially upgraded in the first cleaner tank cells. The resulting concentrate will be pumped to the second cleaner circuit, while the tailings will report to the cleaner scavenger flotation cells for further flotation. The cleaner scavenger flotation concentrate will be recycled to the head of the first cleaner flotation cell bank, together with the tailings from the second cleaner flotation. The first cleaner scavenger flotation tailings will be pumped to the rougher scavenger flotation feed box.

The concentrate from the second cleaner flotation stage will be further upgraded by the third cleaner flotation, and the second cleaner tailings will be pumped to the first cleaner flotation. The concentrate from the third cleaner flotation cells, which will be the final concentrate, will be pumped to the concentrate thickener. The third cleaner tailings will be recycled back to the head of the second cleaner flotation circuit.

The reagents used in the primary bulk flotation circuits will also be added to the three stages of cleaner flotation to float the target minerals. The cleaner flotation processes will be carried out at the natural slurry pH level as well.

Gravity Concentrate Upgrading

The gravity concentrates produced from the primary grinding circuit and the regrinding circuit will be upgraded by conventional tabling followed by smelting to produce gold-silver doré. Upgrading will be conducted in a secure facility with security entrances and 24-hour CCTV surveillance. Operations in the secured gold room will be conducted during day-shift only, and access to the gold room will be restricted to authorized personnel only.

Key equipment that will be installed in the gold room includes:

- two gravity concentration tables – one for the coarse centrifugal gravity concentrate, and the other for the fine centrifugal gravity concentrate
- one table concentrate dryer
- one flux mixer
- one 175 kW induction melting furnace
- one vault for storing doré and table concentrate
- one off-gas and dust scrubbing system
- ancillary equipment, including slag treatment devices.

The coarse and fine centrifugal gravity concentrates from the primary grinding circuit and the regrinding circuit will be pumped to separate table feed stock bins located in the gold room. The two concentrates will be then upgraded by two different tables on a batch basis. The concentrates from the tables will be dewatered, dried in a dryer, then weighed and stored in the vault prior to smelting. The tabling middling products will be recycled back to their respective table feed stock bins. The table tailings will be pumped to the feed well of the centrifugal gravity concentrator in the primary grinding circuit for scavenging any potentially recoverable free gold and silver grains.

The dried tabling concentrate will be calcined and then mixed with flux, which consists of borax ($Na_2B_4O_2$), sodium nitrate ($NaNO_3$), silica (SiO_2), and fluorspar (CaF_2). The concentrate and flux mixture will be charged into a 175 kW induction furnace and melted at approximately 1,150°C. The metal melt and slag will be poured into bar molds in a cascade-casting arrangement. The gold doré will be weighed, sampled, and stored in the vault prior to being shipped to refiners.

The slag from the melting will be crushed by a jaw crusher and ground by a rod mill on a batch basis. The ground slag will be pumped to the fine gravity concentrate stock bin and tabled to recover gold-silver alloy grains entrained in the slag. The table tailings will be sent to the concentrate thickener to be blended with the flotation concentrate.

A wet scrubbing system will be installed for ventilating and cleaning the off-gas generated during the drying, calcination, mixing, melting, and slag crushing operations. The equipment used for these processes will be equipped with hooks. Sufficient ventilation will be provided in the gold room to protect the operators. All clothes, gloves, and other safety equipment necessary for high-temperature protection, will be provided to the operators working in the secure area.

Concentrate Handling

The concentrate from the third cleaner flotation will be thickened, filtered, and bagged prior to being transported to off-site smelter(s). The concentrate handling facility will have the following equipment:

- one 5 m diameter high-rate thickener
- two slurry pumps
- one concentrate filter feed stock tank (4,000 mm diameter by 5,000 mm high)
- one 75 m^2 pressure filter
- one bagging system.

The final flotation concentrate will be pumped to the concentrate thickener. Flocculant will be added to the thickener feed well to aid the settling process. The thickened concentrate will be pumped to the concentrate stock tank. The underflow density of the thickener will be approximately 65% solids. The concentrate stock tank will be an agitated tank, which serves as the feed tank for the concentrate filter. A plate-frame type press filter will be used for further concentrate dewatering. The filter press will reduce the moisture content of the thickener underflow to approximately 10 to 12%. The filter press solids will be discharged onto a conveyor that transports the filter cake to the bagging system feed surge bin. The filter cake will then be bagged in 2 t bags and stacked prior to being loaded into containers for shipping. The bagged concentrate will be transported by Tracked Vehicles to the Knipple Transfer Station, then to Terrace, BC, where the concentrate will be transported in bulk by rail to a smelter in eastern Canada. The plant will provide sufficient on-site storage capacity for up to 10 days of production in the event of unexpected transportation disruption. Additional secured storage will also be provided at the Knipple Transfer Station.

The filtrate from the pressure filter will be circulated back to the concentrate thickener feed well as dilution water. The overflow from the thickener will be pumped to the process water tank for re-use as process water.

Tailings Disposal

The final tailings from the bulk rougher/scavenger flotation will be thickened prior to being pumped either to the backfill plant for underground backfilling or to the Brucejack Lake for storage. Tailings management is further discussed in Section 18.0.

The tailings handling equipment and facility will include:

- one 20 m diameter high-rate thickener
- one 2,000 mm diameter by 3,000 mm high mixing tank for the tailings that will be discharged to Brucejack Lake
- one backfill plant to produce the tailings paste for underground backfilling (tailings paste production is detailed in Section 18.0).
- two slurry pumps.

When the tailings are discharged to Brucejack Lake, the tailings thickener underflow will be pumped to the tailings mixing tank where the tailings will be diluted to a solid density of 35% w/w. When the tailings are backfilled to the excavated underground stopes, the water from the water treatment plant or from Brucejack Lake will be sent to the mixing tank with the intent to maintain fluidization of the deposited tailings bed. The overflow of the thickener will be sent to process water tank.

Reagent Handling and Storage

PAX and MIBC will be added to the flotation process slurry stream to modify the chemical and physical characteristics of mineral particle surfaces, and to enhance the floatability of the valuable mineral particles into the concentrate products. Flocculant will be used as a settling aid for the flotation concentrate and tailings thickening. Anti-scalant will be added as required to protect pipelines and process equipment. Hydrated lime will be used to prepare an alkaline solution for scrubbing.

PAX will be shipped to the mine site in solid form. The reagent will be diluted to 10% solution strength in a mixing tank, and stored in a 1.50 m diameter by 1.50 m high holding tank. The solution will be added to the various addition points by metering pumps. Fresh water will be used to make up the required solution strength.

MIBC will be shipped to the plant as liquid in bulk tankers. The reagent will be stored in a holding tank and pumped in undiluted form to the points of addition using metering pumps.

Solid flocculant will be used for the Project. The flocculant will be prepared in the standard manner in a wetting and mixing system to a dilute solution of less than 0.2%

solution strength. The solution will be stored a holding tank prior to being pumped by metering pumps to the thickener feed wells.

Anti-scalant chemicals will be delivered in liquid form and added to the process water tank as required to minimize scale build-up in the water pipelines and process equipment. This reagent will be added in undiluted form.

A mixing, holding, and dosing system will be provided to occasionally test any new reagents that may improve the metallurgical performance for better metal recovery. These reagents will be handled in accordance with Material Safety Data Sheet (MSDS) requirements, and any unused test reagents will be returned to the suppliers for disposal.

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 MSDS stations will be provided in the area.

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.

Assay and Metallurgical Laboratory

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, process plant, and the environmental department. The most important assay laboratory instruments include:

- fire assay related furnaces and devices
- one atomic absorption spectrophotometer (AAS)
- two ICP, including one ICP-mass spectrometer (MS) for environmental sample analysis
- one Leco furnace.

The metallurgical laboratory will undertake all the necessary test work to monitor metallurgical performance and, more importantly, to improve process flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory crushers and ball mills, particle size analysis sieves, bench scale flotation cells, centrifugal gravity concentrators, leach units, filtering devices, balances, and pH meters. The personal protection devices and items will be provided to protect the workers.

Water Supply

Two separate water supply systems will be provided to support the operations for the process plant – one fresh water supply system, and one process water supply system.

Fresh Water Supply System

Fresh water will be supplied to a fresh/fire water storage tank from the water treatment plant or from Brucejack Lake. Fresh water will primarily be used for:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems
- gland water for the slurry pumps
- reagent make-up
- process water make-up.

The fresh/fire water tank will be equipped with a standpipe, which will ensure that the tank is always holding at least a 2 hour supply of fire water.

There will be two potable water supply systems – one located in the process/administration/truck shop complex, and one at the camp. Potable water will be supplied from wells, and will be treated (chlorination and filtration) and stored in the potable water storage tanks prior to delivery to various service points.

Process Water Supply System

The overflow solutions from the concentrate thickener and tailings thickener will be re-used in the process circuit. The balance of the process water will be supplied from the water treatment plant, which will treat water from the mine (underground water) and water collected from the plant site or from Brucejack Lake, as required. All process water required will be distributed to the process plant from an 8.0 m diameter by 8.0 m high process water tank.

AIR SUPPLY

Air service systems will supply air to the following service areas:

- Crushing circuit – high-pressure air will be provided by an air supply system located underground for dust suppression and equipment services.
- Dust collection at the transfer tower and SAG mill surge bin – high-pressure air will be provided by dedicated compressors for dust suppression.
- Flotation – low-pressure air for flotation cells will be provided by air blowers.
- Filtration circuit – high-pressure air will be provided by dedicated air compressors for filtration and drying.
- Plant air service – high-pressure air will be provided by dedicated air compressors for the various services.
- Instrumentation – the service air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

Process Control and Instrumentation

Plant Control

The type of plant control system will be a distributed control system (DCS) that will provide equipment interlocking, process monitoring and control functions, supervisory control, and an expert control system. The DCS will generate production reports, provide data and malfunction analyses, and produce a log of all process upsets. All process alarms and events will be logged by the DCS.

Operator interface to the DCS will be via programmable computer (PC) at operator workstations located in the following area control rooms:

- underground crushing facility
- process plant
- paste backfill plant.

The plant control room will be staffed by trained personnel 24 h/d.

Using the operator workstations in the main process plant, it will be possible to monitor the entire plant site process operations, view alarms, and control equipment within the plant. Supervisory workstations will be provided in the offices of the plant superintendent and the mill maintenance superintendent.

Field instruments will be microprocessor-based “smart” type devices. Instruments will be grouped by process area, and wired to field instrument junction boxes for each respective area. Signal trunk cables will connect the field instrument junction boxes to DCS input/output (I/O) cabinets.

Intelligent-type motor control centres (MCCs) will be located in electrical rooms throughout the plant. A serial interface to the DCS will facilitate the MCCs remote operation and monitoring.

A fiber optic backbone will be installed throughout the plant site for site wide infrastructure (i.e. telephone, internet, security, fire alarm, and control system).

A dedicated security system will be installed with multiple CCTV cameras in the gold room to monitor operations and security. The system will connect with the overall site security monitoring systems in the plant control room and the offices of the plant superintendent and security.

Control Philosophy

Primary Crushing Control System

The control system for the primary crushing facility in the underground mine will be connected with the plant control room, which will have a PC workstation. The plant control system, together with the underground control system, will monitor the

underground crushing operations and crushed ore conveying operations that will transport the crushed ore to the coarse ore surge bin. Data collected from the crushing and conveying operations will be provided to both the mill process control system and the underground control system via a serial Ethernet gateway.

The crushing control systems will control:

- SAG mill feed conveyors, including the conveyors in the primary crushing facility (zero speed switches, side travel switches, emergency pull cords, and plugged chute detection)
- surge bin levels (radar level, plugged chute detection)
- the primary crusher when the emergency stop is activated.

Plant Control System

To control and monitor all mill building processes, three PC workstations will be installed in the building's central control room.

The PC workstations will control and monitor the following:

- grinding conveyors (zero speed switches, side travel switches, emergency pull cords, and plugged chute detection)
- SAG and ball grinding mills, including the regrinding mill (mill speed, bearing temperatures, lubrication systems, clutches, motors, and feed rates)
- particle size monitors (for grinding optimization and cyclone feed)
- pump boxes, tanks, and bin levels
- variable speed pumps
- hydrocyclone feed density controls
- thickeners (drives, slurry interface levels, underflow density, and flocculant addition)
- flotation cells (level controls, reagent addition, and airflow rates)
- samplers
- gravity concentrators
- pressure filters and load out
- reagent handling and distribution systems
- tailings disposal to the paste backfill plant or tailings storage in Brucejack Lake
- water storage and distribution, including tank level automatic control
- air compressors
- paste backfill plant (vendor control system)

- vendors' instrumentation packages
- gold room operations.

An automatic sampling system will collect samples from selected product streams for daily metallurgical accounting and operation optimization.

Particle size-based computer control systems will be used to maintain the optimum grind sizes for the primary grinding and rougher/scavenger concentrate regrinding circuits. Particle size analyzers will provide the main inputs to the control system.

Online Sample Analysis

The plant will rely on the on-stream or in-stream particle size analyzer and various flowrate and solid density meters for process control. The analyzer and meters will analyze the various slurry streams in the circuit. The on-stream particle size monitor will determine the particle sizes of the hydrocyclone overflows in the primary grinding circuit and regrinding circuit. A sufficient number of samples will be taken in order to control hydrocyclone overflow particle size and optimize the grinding circuit operations. Specific samples that will be taken for metallurgical accounting purposes include the flotation feed to the circuit, the final tailings, the final concentrate sample, and occasionally the middling products. These samples will be collected on a shift-basis and will be assayed in the assay laboratory.

Remote Monitoring

CCTV cameras will be installed at various locations throughout the plant, including the primary crushing facility, the conveyor discharge point, the SAG mill surge feed conveyer, the SAG and ball mill grinding area, the flotation area, the regrind area, the paste plant, the gold room, the concentrate handling building, and the tailings handling facilities. The cameras will be monitored from the plant control room. Fuel storage facilities will be remotely monitored by level controls and CCTV cameras.

17.2 ANNUAL PRODUCTION ESTIMATE

The process plant will generate two products: gold-silver doré and gold-silver bearing concentrate. The annual metal production shown in Table 17.2 has been projected based on the mining production plan outlined in Section 16.0 and the metallurgical performance outlined in Section 13.0. Based on the LOM annual average, the process plant is estimated to produce approximately 4,300 kg of gold and 1,500 kg of silver contained in doré, and 42,000 t of gold-silver bearing flotation concentrate. On average, the flotation concentrate will contain approximately 135 g/t gold and 1,028 g/t silver. The arsenic content of the flotation concentrates to be shipped to the smelter(s) is expected to be marginally higher than the penalty thresholds outlined by most smelters and will require further review.

Table 17.2 Projected Gold and Silver Production

Year	Mill Feed					Doré				Concentrate					Doré and Concentrate	
		Feed Grade				Tonnage* (kg)		Recovery (%)			Recovery (%)		Grade (g/t)		Total Recovery (%)	
	Tonnage (t)	Au (g/t)	Ag (g/t)	As (ppm)	S (%)	Au	Ag	Au	Ag	Tonnage (t)	Au	Ag	Au	Ag	Au	Ag
1	811,376	15.4	12.2	326	2.5	5,490	1,033	44.0	10.4	40,499	52.8	74.7	162	183	96.8	85.1
2	937,169	13.8	11.1	313	2.3	5,603	1,137	43.2	10.9	43,413	53.4	73.9	159	178	96.7	84.8
3	979,037	13.1	11.0	307	2.6	5,511	1,180	42.9	11.0	50,579	53.7	73.8	137	157	96.6	84.8
4	980,631	15.8	11.7	319	2.2	6,863	1,222	44.2	10.6	43,506	52.6	74.3	188	196	96.8	84.9
5	982,603	17.1	13.5	326	2.3	7,519	1,318	44.8	9.9	44,157	52.1	75.4	198	227	96.9	85.3
6	985,759	12.7	8.8	288	2.2	5,342	1,058	42.6	12.2	43,046	53.9	71.9	157	145	96.6	84.1
7	985,324	15.5	11.4	263	2.1	6,714	1,209	44.1	10.8	40,932	52.7	74.1	196	203	96.8	84.9
8	984,974	14.0	9.9	290	2.1	5,953	1,124	43.3	11.5	40,295	53.4	73.0	182	177	96.7	84.5
9	980,427	14.0	11.3	283	2.0	5,941	1,191	43.3	10.7	39,290	53.4	74.1	186	209	96.7	84.8
10	990,705	11.2	17.8	271	2.1	4,590	1,319	41.4	7.5	40,702	55.0	78.9	150	342	96.4	86.4
11	978,466	12.6	14.1	300	2.2	5,212	1,233	42.3	8.9	42,863	54.2	76.4	156	246	96.5	85.3
12	978,883	12.1	54.6	258	2.5	4,912	1,441	41.6	2.7	48,892	54.8	85.9	132	938	96.4	88.6
13	981,517	12.8	19.1	295	2.3	5,317	1,164	42.4	6.2	45,895	54.1	80.5	148	329	96.6	86.7
14	987,498	11.9	20.0	281	2.2	4,941	1,194	41.9	6.1	42,354	54.5	81.2	152	378	96.5	87.2
15	986,651	9.6	60.5	279	2.5	3,673	1,485	38.9	2.5	50,266	57.2	86.2	108	1,023	96.1	88.7
16	979,339	12.3	36.1	322	2.4	5,087	1,457	42.2	4.1	45,947	54.3	84.9	143	652	96.5	89.0
17	981,683	8.1	97.7	182	2.7	2,814	2,135	35.4	2.2	54,479	60.3	86.8	88	1,527	95.7	89.0
18	949,013	6.2	237.1	133	3.3	1,724	3,538	29.2	1.6	63,623	66.0	89.0	61	3,146	95.2	90.6
19	501,236	5.2	289.4	124	3.4	639	2,196	24.7	1.5	35,335	69.6	89.3	51	3,668	94.4	90.9
20	495,290	5.4	333.1	106	3.3	673	2,474	25.2	1.5	33,925	69.5	89.7	55	4,363	94.7	91.2
21	404,038	4.1	282.4	102	3.1	320	1,711	19.5	1.5	25,815	73.8	89.2	47	3,944	93.2	90.7
22	144,317	3.2	268.7	118	3.2	71	582	15.4	1.5	9,572	76.6	89.1	37	3,609	92.0	90.6
LOM	18,985,936	12.0	57.9	266	2.4	94,908	32,399	41.6	3.0	925,384	54.9	86.6	135	1,028	96.5	89.6

Note: *Gold and silver content in the doré.

18.0 PROJECT INFRASTRUCTURE

18.1 OVERVIEW

The Project will require the development of a number of infrastructure items, both on-site and off-site. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, avoid avalanche hazards, and ensure efficient and convenient underground crew shift change by locating the mine dry in close proximity to the mine portal. An overall site layout is shown in Figure 18.1 and a mill site layout is shown in Figure 18.2.

Project facilities and infrastructure at the Brucejack site will include:

- an upgraded 79 km access road at Highway 37 and travelling westward to Brucejack Lake with the last 12 km of access road to the mine site traversing the main arm of the Knipple Glacier
- a 138 kV power supply line from the substation at Long Lake Hydro Substation to the Brucejack site substation
- site roads and pads
- water management infrastructure, including diversion ditches for both contact and non-contact water, interceptor ditches, and water storage ponds and pumps to direct water to a water treatment plant
- water treatment infrastructure, to treat underground infill water to a water treatment plant to treat and distribute process and fire water
- potable water and sewage treatment infrastructure
- waste management systems, including sewage disposal and domestic waste disposal
- ancillary facilities including:
  - on site fuel storage
  - on site explosive storage
  - detonator magazine storage
  - temporary and permanent camp accommodations with recreation area, commissary and laundry facilities
  - heli-pad
  - laydown area

- process facilities such as warehouse, truck shop/wash bay, mine dry/wicket and lamp room, first aid/emergency response, cold storage/laydown, administration building, assay laboratory, metallurgical laboratory, maintenance shop/tool crib, and covered parking

- power distribution including:
  - power distribution from the mine site substation to all the facilities
  - process control and instrumentation
  - communication systems.

Additionally, an operating gatehouse is currently located at the intersection of Highway 37 and the Project access road. This gatehouse will continue to operate during the construction phase and mine life of the Project.

At the Bowser Airstrip, project facilities and infrastructure will include upgrades to the current airstrip.

At the Knipple Transfer Station, project facilities and infrastructure will include:

- a transfer station
- on site fuel storage
- potable water
- a septic field
- a laydown area
- a communications system
- a 30-person camp
- a temporary covered storage.

Figure 18.1 Brucejack Overall Site Layout

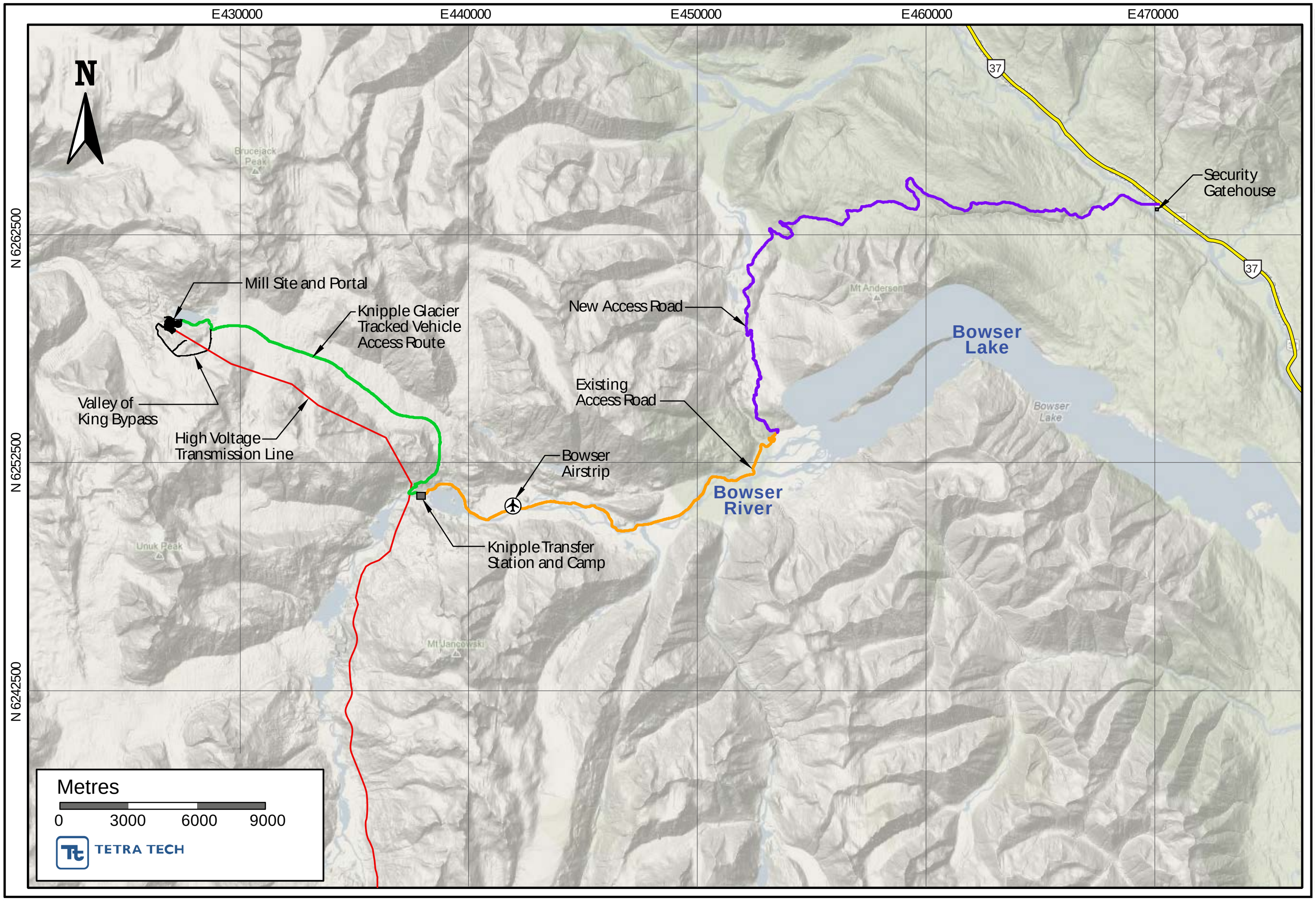
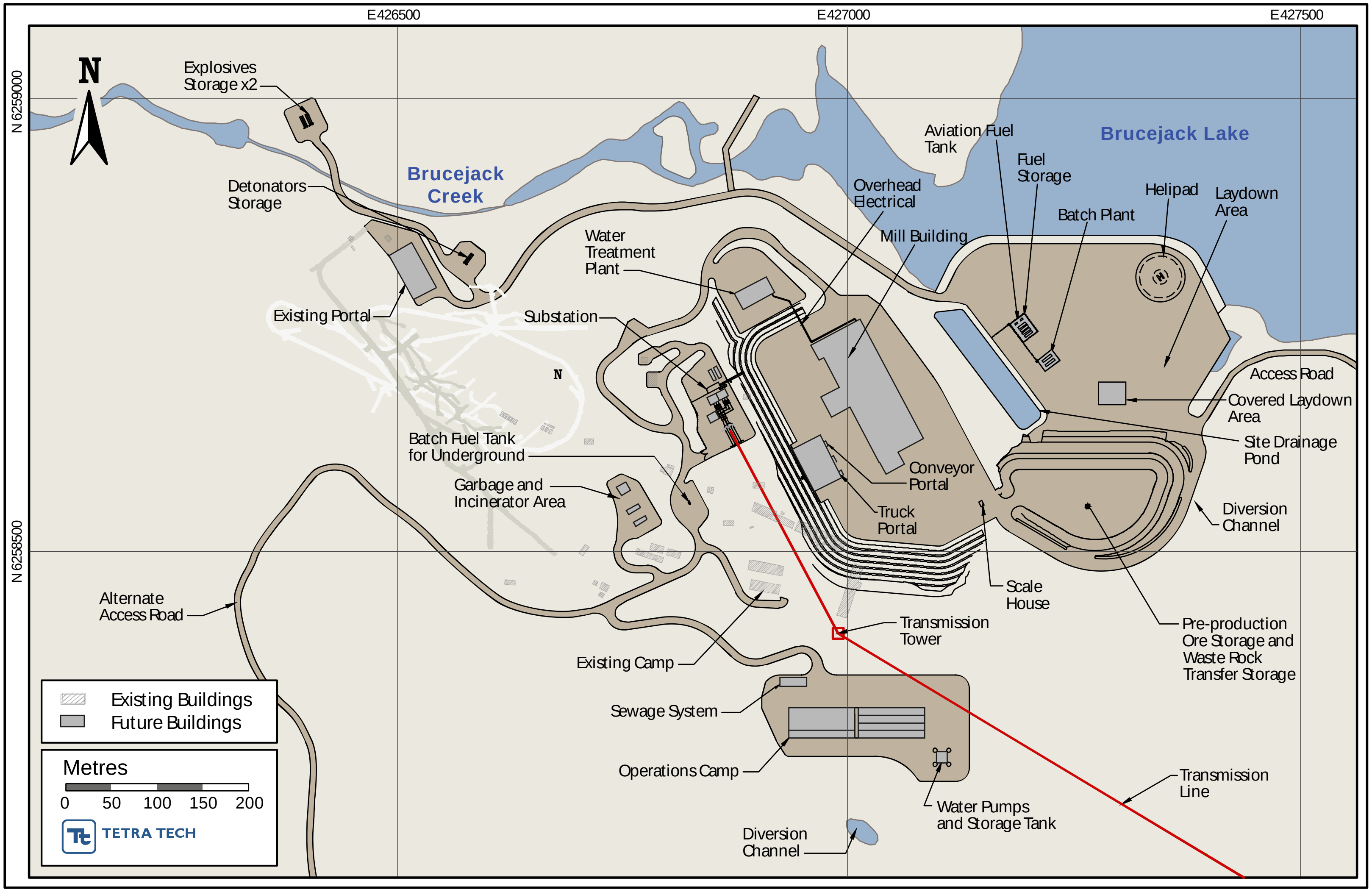


Figure 18.2 Brucejack Mill Site Layout



18.2 GEOTECHNICAL

18.2.1 OVERVIEW

The geotechnical designs and recommendations in this section are based on the results of site investigations and geotechnical assessments. Geotechnical site investigations were undertaken in 2011 and 2012 to evaluate the subsurface conditions in the area of the proposed mill building. These investigations included geotechnical drilling and logging, test pit excavations, laboratory testing of rock core and soil samples, installation of piezometers to measure groundwater pressures, and geophysical surveys. Details regarding the investigations are provided in BGC's 2013 design report, "Geotechnical Analysis and Design of the Plant Site Foundations". In general, the materials encountered during the site investigations consisted of glacial till, ranging in thickness from 1 to 8 m, overlying fair to good quality bedrock.

18.2.2 FOUNDATIONS

For all buildings and facilities located within or adjacent to the mill building, foundations will consist of conventional spread footings, strip footings, or other mass concrete foundation elements founded on good quality bedrock. For ancillary buildings located outside of this area, foundations will consist of conventional spread footings founded on either structural fill or bedrock. Recommendations were provided for allowable bearing pressures based on the results of the 2011 and 2012 site investigations, and the anticipated foundation types provided by Tetra Tech (BGC 2013). Allowable bearing pressures for foundation stratum were estimated – assuming a factor of safety of 3 – against bearing capacity failure and to limit total and differential settlements to those specified by Tetra Tech. Table 18.1 provides a summary of the recommended allowable bearing pressures for specific facilities.

Table 18.1 Recommended Allowable Bearing Pressures for Specific Facilities

Facility	Foundation		Foundation Stratum	Allowable Bearing Pressure (kPa)
	Type	Dimensions		
Surge Bin	Mat	Up to 8 m by 8 m	Bedrock	600
	Spread Footings	Up to 7 m by 3 m	Bedrock	600
Mill Building	Mat	Up to 13 m by 8 m	Bedrock	600
	Strip Footings	2.5 m wide	Bedrock	600
	Spread Footings	Up to 5 m by 3.5 m	Bedrock	600
Tailings Thickener	Mat	Up to 29 m by 29 m	Bedrock	600
	Spread Footings	Up to 5 m by 5 m	Bedrock	600
Ancillary Buildings	Spread Footings	Up to 3 m by 3 m	Structural Fill or Bedrock	200

Note: Foundations on structural fill assume a minimum embedment of 1 m below the surrounding grade.

For the design cold year, the depth of frost penetration is estimated to be up to 5 m in fill materials and 4 m in bedrock. Exterior building foundations and interior footings of

partially-heated buildings will either be founded below the design maximum frost penetration depth, or embedded a minimum depth of 1 m and insulated with sufficient rigid polystyrene insulation, as specified in BGC's design report (2013). Foundations will also be setback a minimum of 10 m from the crest of fill slopes.

Additional investigations (e.g. geotechnical drilling and test pit excavations) are recommended for subsequent stages of design to further evaluate subsurface conditions within the footprints of the site facilities. Laboratory testing of rock and overburden samples are also recommended to further evaluate the properties and behaviour of these materials. The recommendations provided in Table 18.1 will be re-evaluated based on the results of the investigations.

18.2.3 SITE GRADING

Recommendations were provided for excavations and fills necessary to bring the site to its design elevations (BGC 2013). It is expected that the overburden can be excavated with conventional earth moving construction equipment. As excavations extend into the bedrock, more aggressive excavation equipment and/or blasting will likely be required. Based on the results of the site investigations, it is estimated that the upper 1 to 2 m of bedrock may be rippable with a D9 bulldozer. Therefore, large excavations in bedrock will be conducted primarily by controlled blasting.

A summary of the recommended cut slope and fill slope angles is provided in Table 18.2

Table 18.2 Recommended Permanent Cut-and-Fill Slope Angles

Item[1]	Material[2]	Maximum Height (m)	Maximum Slope	Comments
Fill Slope	Structural fill	15	2H:1V	-
Fill Slope	Rock fill	15	2H:1V	-
Fill Slope	General Fill	5	2H:1V	-
Fill Slope	General Fill	15	2.5H:1V	-
Cut Slope	Overburden	5	2H:1V	May have to flatten below the water table and/or provide drainage
Cut Slope	Overburden	10	2.5H:1V	May have to flatten below the water table and/or provide drainage
Cut Slope	Rock - Unbenched	15	0.5H:1V	Spot bolting may be required based on field engineer's review
Cut Slope	Rock - Benched	60 (refer to comments)	Refer to comments	6 m bench height, 5 m bench width, 75° bench face angle; spot bolting may be required based on field engineer's review

Note: [1]All cut and fill slopes should be reviewed in the field by a qualified geotechnical engineer.
[2]Materials are as per descriptions provided by BGC (2013).

Final site grading will maintain positive drainage in the direction of natural drainage and will direct water away from the structures. Permanent surface water control will also be provided at the base of all excavation slopes to direct water away from the proposed facilities and to allow the slopes to drain effectively.

18.3 ACCESS

18.3.1 ACCESS ROADS

The Brucejack access road was constructed as a temporary, all-season exploration access road that commences at Highway 37 at km 216 and travels generally westward to Brucejack Lake, a distance of 73.5 km. The road route is shown in Figure 18.1. The access road is separated into the following sections (km 0 is located at the Highway 37 intersection):

- Section 1: km 0 to 35: new road construction
- Section 2: km 35 to 59: existing road upgrade
- Section 3: km 59 to 71: Knipple Glacier road
- Section 4: km 71 to 74: Brucejack Lake road upgrade.

The road is generally designed and constructed as follows:

- L100 minimum design loading, with consideration of equivalent D9 track loading
- 5.0 m road width, with 0.3 m wide by 0.8 m deep ditches between km 0 and 35
- 6.0 m road width, with 0.3 m wide by 0.8 m deep ditches between km 64.6 and 67
- 30 m right-of-way width
- 30 km/h design speed, 35 m minimum turning radius
- maximum 12% sustained grade, 18% pitches less than 150 m (with exceptions)
- maximum 500 m turnout spacing, optimum 300 m spacing
- 7% maximum grade break per 15 m travel
- Q100 flow culvert design:
  - drains are usually a 500 mm diameter pipe, used at low spots to redirect ditch line flows and wet depression accumulations and to broadcast these flows to natural forest floor below the road where filtration will occur
  - non-classifiable drainage, 500 mm or larger diameter pipe, used at very small drainages that do not meet the criteria to be a classifiable stream
  - S5 and S6 stream classifications, 600 mm or larger diameter pipe, used at drainages that can be classified
  - 50 m buffer to wildlife habitat.

Upon receipt of construction permits, these road sections–with the exception of the Knipple Glacier section–will require additional upgrading to improve safety and to handle the higher traffic loadings from both construction and operations activities. The work will

include minor re-alignments of the sharper curves, reductions of the steeper grades, and additional surfacing of some sections.

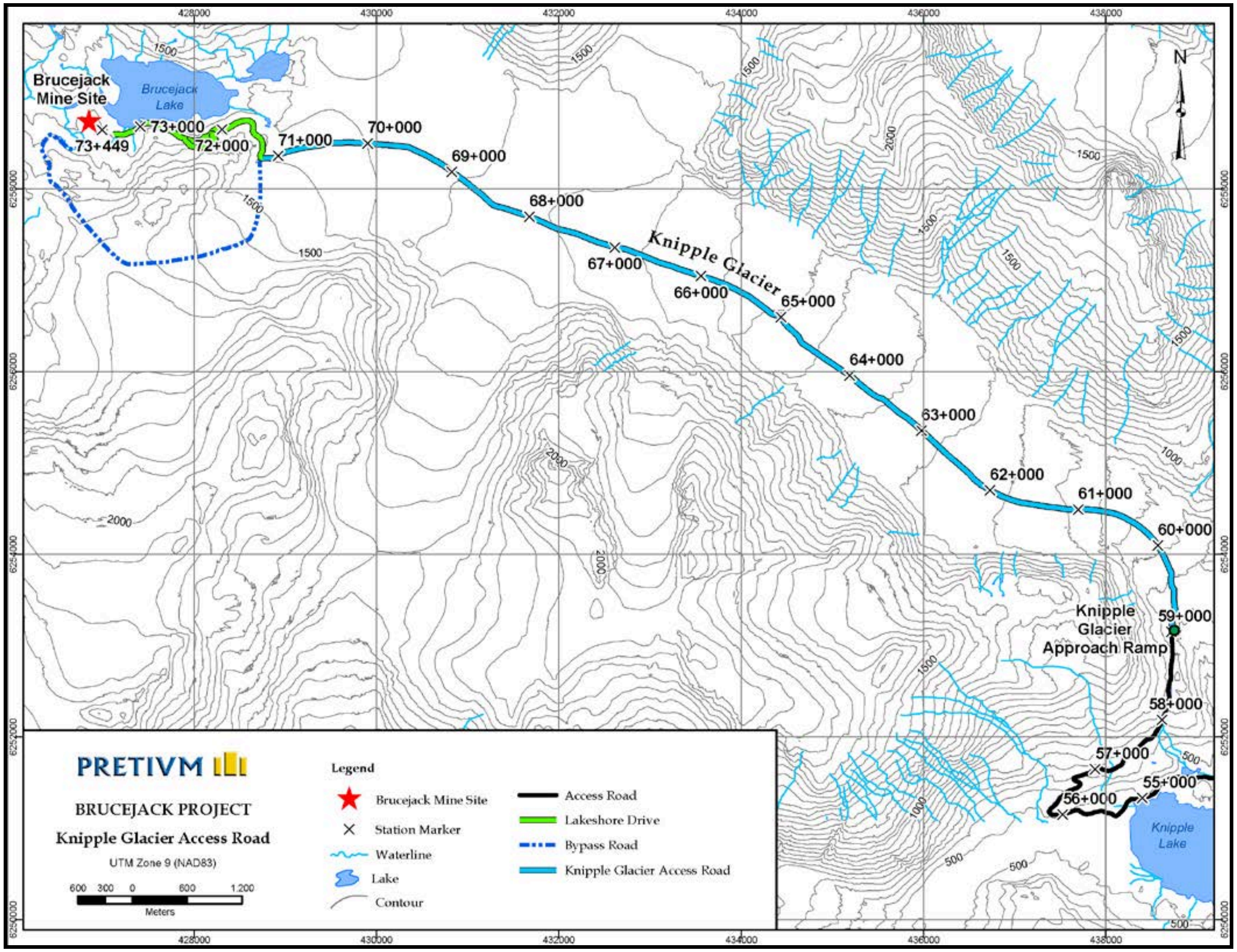
During mine operation, the road will be maintained throughout the year by road grooming equipment and snow plows. Regular patrols will be conducted in potential avalanche areas and avalanche control measures will be utilized.

18.3.2 Glacier Crossing

The last 12 km of access road to the mine site (km 59 to 71) traverses the main arm of the Knipple Glacier. During winter months the route is a groomed snow surface, but is an ice surface during the summer months, as illustrated in Figure 18.3.

The Knipple Glacier road route was used by Newhawk in the late 1980s and early 1990s to move personnel and materials to the mine site. Pretivm reactivated the route in 2012.

Figure 18.3 Knipple Glacier Access Road



Knipple Glacier Approach Ramp

The toe of the Knipple Glacier is not only receding, but the top surface of the glacier is melting vertically. Anecdotal reports suggest that the surface has melted some 90 m vertically since the early 1990s when Newhawk ceased operations.

Due to dropping surface, Pretivm established a new access ramp at km 58.6 of the road. This ramp provides access to the ice earlier, eliminating the need to traverse through an area of high-avalanche activity and rock falls. It is expected that this approach will require additional excavation every two years to maintain a safe gradient as the glacier melts further.

Winter Operation

During the winter months, the Knipple Glacier Road is covered in many meters of snow. Pretivm has experience using ski resort type snow cats to prepare the running surface for other tracked equipment. The snow cats are equipped with blades on the front to distribute the snow across the road surface and break through snow drifts. A hydraulically powered tiller on the rear of the snow cat then mixes the new snow with previous layers, reduces chunks, and beats the air out of the snow. A plastic comb then compresses the snow into a ribbed or “corduroy” running surface. The end result is a snow surface that is dense enough for other tracked equipment to run on.

The road surface is maintained as high above the ice level as possible to maintain a snow running surface well into summer.

Summer Operation

Summer maintenance of the Knipple Glacier Road consists mainly of leveling the snow surface as it melts in the warmer months. It is vital to keep running water off the road surface as much as possible to avoid channeling on the road. Excavators are used to channel the water into existing crevasses.

The melting snow exposes the crevasses and mill holes, which pose a hazard to small vehicles and personnel on foot. The avalanche and glacier technicians survey the ice conditions daily, marking safe travel areas to avoid the larger ice hazards. Ice bridges are constructed over the larger crevasses from ice harvested in other areas of the glacier.

The road route is checked during the summer months and altered to avoid particularly large hazards as needed. A crevasse survey is completed each summer so that a safe route can be planned for the winter when the hazards are obscured by snow bridging.

Alternate Bypass Route

The length of road on the south side of Brucejack Lake between the Knipple Glacier and the mine site is called Lakeshore Drive, and often has high avalanche risks. During time when it is unsafe to travel on Lakeshore Drive, an alternate snow route over the Valley of the Kings is available. This VOK bypass road traverses around to the south of the Property eventually meeting up at km 71 of Knipple Glacier Road. This road is only

available in the winter and also provides access to the upper elevations of the Property for avalanche control measures.

GLACIER SAFETY

Unlike other glaciers in the area, the Knipple Glacier is generally free from large crevasses that present a hazard to equipment. It does contain many crevasses and mill holes (moulins) that present hazards to all-terrain vehicles or personnel on foot. Seracs, or ice cliffs, are not present along the immediate travel route.

Glacier travel guidelines and glacier emergency response plans have been developed and implemented by Pretivm.

The road route is demarcated with closely spaced bamboo delineators that provide a visual reference for operators at night and in low-visibility weather. Personnel on foot must not wander from the safe zones.

Personnel operating on the glacier receive additional safety training and are issued additional personal protective equipment such as rescue harnesses, avalanche beacons, rope rescue equipment, and avalanche rescue equipment.

18.4 INTERNAL SITE ROADS AND PAD AREAS

Tetra Tech completed the internal site road and pad area design in accordance with the recommendations outlined in BGC's 2013 report entitled Geotechnical Analysis and Design of the Plant Site Foundations. The geometric design of the site roads meets the standards set out in the Transportation Association of Canada's (TAC) Geometric Design Guide for Canadian Roads (TAC 1999).

Internal site roads will connect various on-site facility pad areas. Internal site roads will be 8.0 m wide to accommodate two opposing lanes for unconstrained two-way traffic. They will be crowned gravel roads with ditch drainage, and will include a safety berm where required. See Figure 18.2 for internal site road locations.

A haul road (Haul Road 01) will be constructed for underground trucks to travel from the underground mine south portal to the pre-production ore storage area, the waste rock transfer storage area, or the Brucejack Lake waste rock dump. Waste rock will be dumped at the waste rock transfer storage area during poor weather conditions, due to the limited traction capabilities of the underground trucks and a haul road grade of approximately 7%. Underground trucks will haul waste rock directly to the Brucejack Lake waste rock dump when weather conditions permit.

The frequency of the standard axle load and subgrade modulus have been considered in the design of all road and gravel surfacing structures. Subgrade was assumed to be either rock or good soil. Road travelling surfaces, safety berms, and drainage channels will be regularly maintained.

The proposed development consists of several pad areas for the following:

- mill building
- water treatment plant
- substation
- garbage and incinerator
- batch fuel tank
- laydown
- operations camp
- detonator storage
- explosives storage
- Knipple Transfer Station.

Earthworks, including rock blasting, will be required to create the pads. There may be potentially acid generating (PAG) rock in the mill pad cut. Drainage from this area will be collected and treated. Rock excavated from this area is proposed to be subaqueous fill in the lower laydown area or waste deposited into Brucejack Lake. The pads will be finished with a gravel surface, and will drain to ditches. Despite the mountainous terrain, only minor retaining walls are proposed at this time.

Pretivm provided Light Detection and Ranging (LiDAR) data on October 19, 2012, that Tetra Tech deemed to be within ±0.5 m vertical accuracy, which is appropriate for costing at this phase of development. Consideration was given to optimizing operations, providing sound structural foundation bearing capacity, and minimizing earthworks operations. Proposed pad and road areas were designed and modelled in 3D to obtain approximate final grades. Cut-and-fill volumes were then derived using computer software, and verified by model and cross-section checks. The cut-and-fill volumes accounted for topsoil removal as well as shrinkage and swell factors. Overall, there is a surplus of material with the current design, which is largely attributed to the mill building pad area. The overall surplus of approximately 1.1 Mm^3 will be waste rock that is deposited into Brucejack Lake.

At this feasibility-level stage, Tetra Tech used the available geotechnical information for the earthworks design and material take-offs. Tetra Tech assumed an average of 100 mm of topsoil over 20% of the total disturbed area. Tetra Tech assumed that 25% of common excavation was not suitable as fill and was taken to waste. The other 75% was assumed acceptable for re-use if required. The rock surface was modelled but with very sparse data. As such, the rock surface is thought to be within ±2 to 3 m accuracy. Tetra Tech assumed that only 5% of the rock encountered will be rippable, with the remaining rock requiring blasting. BGC provided shrink and swell factors that range from a 7.5% shrinkage factor (bank cut of common excavation to compacted fill) to a 35% swell factor (bank cut of rock to loose stockpile or waste dump). Further geotechnical investigation and ground survey (to total station survey accuracy) should be undertaken as required for detailed design and scope/price certainty.

18.5 GRADING AND DRAINAGE

The site will be positively drained at all times. Existing drainage courses will be preserved as much as possible as this typically leads to the most economical drainage design. Any minor drainage systems will be designed for the 10-year storm event, but the site will always have a major storm outlet route to prevent any area flooding. Design flows were derived from a master drainage plan that was developed for the site. Site specific historical hydrological information was not available; therefore, intensity duration frequency (IDF) information was estimated using the Stewart Airport rain gauge for flow calculations.

Detailed grading of the facility pads has not been completed for the feasibility stage. Additional earthworks required for such grading can be significant because some of the pads are large areas. As such, an approximated additional volume of earthworks has been included in the capital cost.

Ditches are designed for the 10-year storm event and grades range from 0.3 to 8.0%. Culverts are designed to discharge a 10-year storm event without static head at the entrance, and to discharge a 100-year storm event utilizing available head at the entrance. Culvert corrugation and wall thicknesses were specified in accordance with structural load-carrying capacity under H-20 highway loading and depth-of-cover requirements, respectively. Rip-rap will be used for erosion protection at culvert inlets and outlets, and along diversion/drainage channels where required.

Storm drainage from PAG rock areas or ore storage areas will be collected and pumped to the water treatment plant. Currently, this provision has been included for the mill pad area and the pre-production ore storage area.

Sediment control during construction will be handled on site as needed by covering exposed cuts and stockpiles, diversion ditches, silt fences, etc. Maintenance of these items will be important. Tetra Tech assumes that these proven measures are acceptable to the regulatory authorities, and that a sediment pond is not required.

18.6 AVALANCHE HAZARD ASSESSMENT

18.6.1 BACKGROUND ON SNOW AVALANCHES

Snow avalanches generally occur in areas where there are steep open slopes or gullies, and deep (more than 50 cm) mountain snow packs. Risks associated with avalanches are normally due to exposure to the high impact forces that occur, as well as the effects of extended burial for any person caught in an avalanche. Impact forces vary significantly depending on avalanche size. Although the smallest avalanches can be insignificant to a human, larger avalanches may produce impact forces capable of destroying trucks, buildings, or several hectares of mature forest.

Characteristics of Snow Avalanches

Avalanches may initiate in either dry or wet snow. Although an avalanche may start in dry snow, it could become moist or wet during its descent. Wet snow avalanches can be deflected and often channelled by terrain features, including gullies. Conversely, large, fast-flowing dry avalanches tend to flow in a straighter path, and may overrun terrain features.

Most large, dry avalanches consist of a dense component that flows primarily along the ground, and a less dense powder component that travels above and sometimes ahead of the flowing component. In some cases these components can separate and move independently. The dense-flowing component and powder component may reach speeds up to 60 m/s (200 km/h). Impact pressures from dense flows are much greater than the powder component due to the density of the snow.

Avalanche terrain is usually associated with steep, open slopes in the mountains that allow an accumulation of snow before it releases in a destructive event. In addition to the steep slopes that the snow accumulates on, any area exposed to this release of snow is also considered avalanche terrain. Terrain is often subdivided into features that are connected, which generally contain or channel the volume of avalanche events into a common deposition area. These features are called avalanche paths.

Avalanche season is the time of year when avalanches may occur, and is dependent on when the ground roughness in starting zones is covered by snow, and the threshold for avalanches is exceeded. For the Brucejack area, avalanche season below 1,000 m generally occurs between November and May. For elevations above 1,200 m, avalanche season can extend into October and June, or even summer months if cool, wet conditions persist.

Avalanche Path

An avalanche path generally consists of a starting zone, a track, and a runout zone. Avalanches start and accelerate in the starting zone, which typically has a slope incline greater than 30°. Downslope of the starting zone, most large avalanche paths have a distinct track in which the slope angle is typically in the range of 15 to 30°. Large avalanches decelerate and stop in the runout zone where incline is usually less than 15°. Smaller avalanches may decelerate and even stop on steeper slopes (15 to 24°).

Within forested terrain, larger avalanche paths are often discernible as vertically oriented swaths of open forest terrain, bordered by trim lines (mature forest on either side of the swath). Smaller avalanches, however, can occur in more subtle paths, and can occur on large cut banks in a road cut.

Runout zones generally have vague trim lines, and analysis is required by an experienced avalanche specialist to determine estimates of maximum avalanche extent (often extends into mature forest). In terrain around cliffs, some avalanche paths can be much more subtle to observe, and can be confused with rock fall and/or geotechnical events.

Avalanche Frequency

Avalanche frequency is the reciprocal of avalanche return period and is typically referred to as an order of magnitude ranging from 1:1 (annual) up to 1:300 (1 in 300) years. Each winter, the probability of an avalanche with a specified return period is constant.

Avalanche frequency is dependent upon snow supply and terrain. Frequency decreases with distance downslope in the track and runout zone. Snow supply is determined by:

- the frequency of snowfalls and amount of snow
- the wind transport of snow into the starting zone.

Snow and weather conditions vary from year to year; therefore, the frequency of avalanches is not uniform.

The primary terrain factors in avalanche formation are incline, slope orientation (aspect) with respect to wind and sun, slope configuration and size, and ground surface roughness. Slope configuration is important because features such as gullies will often have more frequent and larger avalanches than open slopes. Ground roughness determines the threshold snow depth for avalanches to occur, which is particularly important in light snow climates where snow may not exceed threshold depths during some winters.

Avalanche Magnitude

Avalanche magnitude relates to the destructive potential of an avalanche and is defined according to the Canadian avalanche size classification system. This classification system is summarized in Table 18.3, which provides a general description of destructive potential, magnitude, and typical path length.

Table 18.3 Canadian Classification System for Avalanche Size

Size	Destructive Potential	Typical Mass (t)	Typical Path Length (m)	Typical Impact Pressures (kPa)
1	Relatively harmless to people	<10	10	1
2	Could bury, injure or kill a person	10^2	100	10
3	Could bury a car, destroy a small building, or break a few trees	10^3	1,000	100
4	Could destroy a large truck, several buildings, or a forest with an area up to 4 ha	10^4	2,000	500
5	Largest snow avalanches known; could destroy a village or a 40 ha forest	10^5	3,000	1,000

Source: McClung & Schaerer 2006

Magnitude is often related to frequency. In general, large destructive avalanches occur less frequently, while smaller ones occur on a more regular basis. Magnitude and frequency are also co-related to a specific location in an avalanche path. For example, a road location near the toe of an avalanche path will be affected by avalanches on a less

frequent basis, but they will be larger avalanches. Both low-frequency large avalanches and higher-frequency small avalanches may affect a road crossing that is higher up in the avalanche path.

18.6.2 BRUCEJACK AVALANCHE HAZARD

Avalanche paths and hazard areas that affect the Project were identified by reviewing topographic relief and vegetation features on maps and aerial photos, as well as available Google Earth™ ortho-imagery and digital elevation models (DEM). In addition, field reconnaissance (helicopter overview flights and ground based survey) was completed on March 19, 2012, and from April 28 to 29, 2013.

Approximately 15 avalanche paths or hazard areas reach (or potentially reach) project infrastructure or access roads, and many locations are estimated to be affected on an annual basis. Drawings for the avalanche paths and hazard areas are illustrated in the Alpine Solutions technical report entitled “Brucejack Project Avalanche Hazard Assessment” (Alpine Solutions 2013). Avalanche paths are labelled according to Table 18.2, referring to the element at risk, with the exception of paths along the Knipple Glacier, which may affect both the transmission line and access road.

Table 18.4 Avalanche Path or Area Label and Corresponding Element at Risk

Avalanche Path or Area Label	Main Facility at Risk
TL1, TL2, ..., TLx	Preferred transmission line alignment
AR1, AR2, ..., ARx	Access road and Knipple Transfer Station
MS1, MS2, ..., MSx	Facilities at or near the mine site
KG1, KG2, ..., KGx	Access road and transmission line corridor on glacier

Details of avalanche hazards and potential consequences are outlined in the following sections for the mine site, access road, Knipple Transfer Station, and transmission line.

MINE SITE

The mine site is located on a broad alpine plateau in undulating terrain on the southwest side of Brucejack Lake. The area is bounded by the Knipple Glacier to the east and south, the Sulphurets Glacier to the west, and rising alpine slopes to the north. Elevation of avalanche terrain at the mine site area ranges from 1,350 m to over 2,000 m. The proposed facilities assessed in the mine site area near Brucejack Lake include:

- two explosives and storage facilities – preliminary position
- detonator storage – preliminary position
- topsoil stockpile
- substation
- water treatment plant

- portal fuel storage
- conveyor portal and truck portal
- overhead electrical
- mill building including administration, mine dry, truck shop and warehouse
- tailings pipeline
- helipad
- upper and lower laydown areas
- batch plant, fuel storage, and aviation fuel tank
- pre-production ore storage
- diversion channel
- waste rock transfer storage
- garbage and incinerator area
- operations camp
- sewage system, water pumps, and storage tank
- proposed new transmission tower location
- scale house
- four air raise locations
- site access roads (not including mine access road).

These facilities are located away from avalanche paths and areas, with the exception of the operations camp, some sections of the site access roads, and the pre-production ore storage and diversion channel area. Short slopes that currently exist (ranging from 10 to 40 m in height) or will be created during construction, may be expected to affect other facility areas; however, the hazard and consequences would normally be assessed on a site-specific basis during construction and operations.

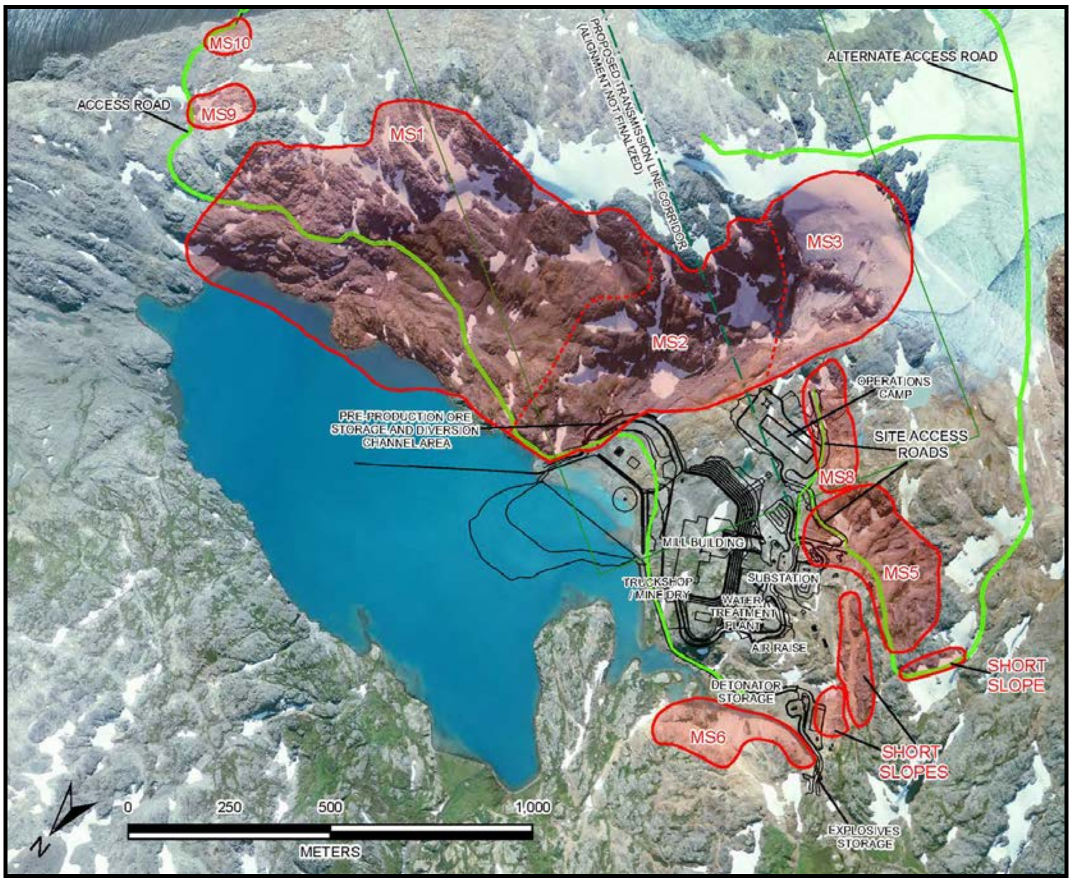
Table 18.5 provides a summary of avalanches reaching the mine site area and Figure 18.4 illustrates the approximate hazard locations.

Table 18.5 Mine Site Avalanche Paths or Areas

Path or Area ID	Avalanche Atlas Polygon Label	Facility Affected	Approximate Elevation of Facility (m)	Facility Position in Path	Length of Facility Affected (m)	Estimated Return Frequency (events:years)		
						Size 2	Size 3	Size 4
Mine Site 2	MS2	Pre-production ore storage and diversion channel area	1,390 to 1,370	RZ	300	-	1:10	-
Mine Site 5	MS5	Site access roads	1,460 to 1,420	RZ	800	1:1	1:3	-
Mine Site 8	MS8	Operations camp and site access roads	1,450	RZ	300	1:1	-	-

Note: RZ = Runout Zone

Figure 18.4 Mine Site Area Avalanche Hazards



The nature of the hazards to facilities at the mine site and the hazards associated with avalanches reaching Brucejack Lake are described in the following sections.

Operations Camp

Size 2 avalanches from Path MS8 are estimated to reach the southwest end of the operations camp with an annual return frequency. Potential consequences of avalanches reaching this area include damage to vulnerable infrastructure (e.g. windows or non-structural components of building, if built within the runout area) and worker injury or fatality if workers are in the runout area when the avalanche occurs.

Site Access Roads

Size 2 and 3 avalanches from Path MS5 and Size 2 avalanches from Path MS9 are estimated to reach site access roads annually. Potential consequences include damage to infrastructure and vehicles, and worker injury or fatality if workers are in the runout area when the avalanche occurs.

Pre-production Ore Storage and Diversion Channel Area

The pre-production ore storage and diversion channel area is exposed to Size 3 avalanches from Path MS2, approximately once every 10 years. Potential consequences are limited to damage to any vulnerable materials stored in this area during avalanche season, as well as worker injury or fatality if workers are in the runout area when the avalanche occurs. The diversion channel is expected to be buried underneath the snowpack during avalanche season.

Avalanches Reaching Brucejack Lake

Avalanches up to Size 3 may reach Brucejack Lake from Path MS1, and short steep slopes on the north side of Brucejack Lake may produce avalanches up to Size 3 reaching the lake. If avalanches reach the lake when the surface is not frozen, waves may develop. As a result of the small size and/or slow speed of the avalanches when they reach the lake, these waves are not expected to be destructive.

Access Road

The mine access road begins at Highway 37 near the confluence of Wildfire Creek and the Bell Irving River, approximately 30 km south of Bell II. The road extends northwest following the Wildfire Creek drainage for approximately 12 km before heading west to Scott Pass (677 m elevation) and then down Scott Creek drainage to the Bowser River valley (400 m elevation), 35 km from Highway 37. The access continues west along the Bowser River valley for approximately 15 km to the Knipple Transfer Station.

From the Knipple Transfer Station, the road ascends to the northwest to reach the south side of the toe of the Knipple Glacier, and along a short ramp to the Knipple Glacier. From here, a glacier road extends up the centre of the Knipple Glacier for approximately 15 km to the mine site at approximately 1,400 m. The glacier road is proposed to be located near the centre of the Knipple Glacier, although the location may vary depending on crevasse restrictions.

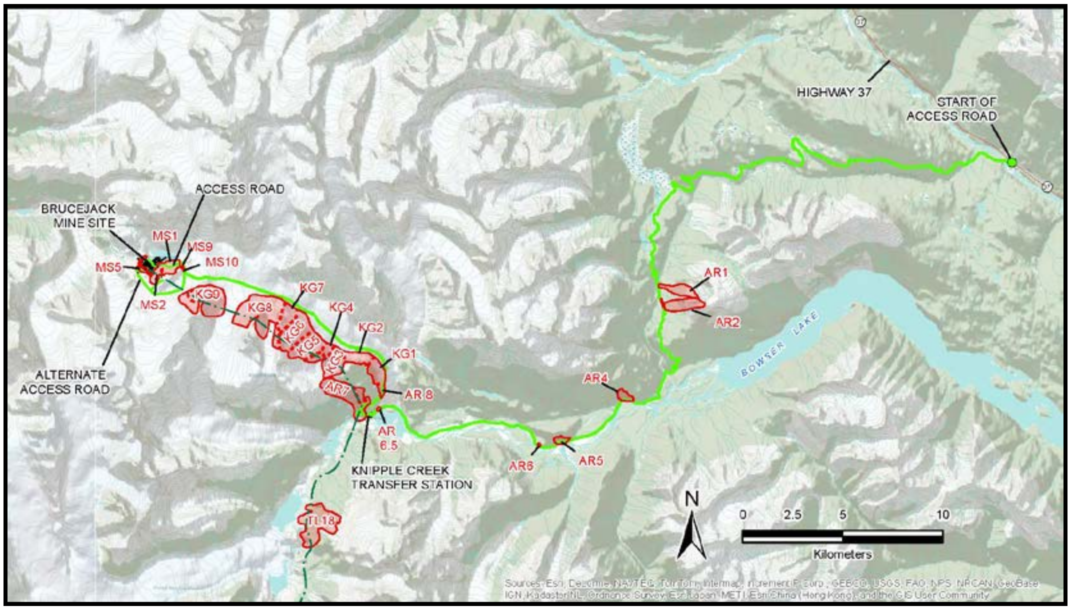
Fourteen avalanche paths or areas are estimated to affect the access road, and two paths approach within 50 m (Table 18.6).

Table 18.6 Access Road Avalanche Paths or Areas

Path or Area ID	Avalanche Atlas Polygon Label	Facility Affected	Approximate Elevation of Facility (m)	Facility Position in Path	Approximate Length of Facility Affected (m)	Estimated Return Frequency (events:years)		
						Size 2	Size 3	Size 4
Access Road 1	AR1	Access Road	580	-	-	-	-	P
Access Road 2	AR2	Access Road	580	-	-	-	-	P
Access Road 4	AR4	Access Road	400	RZ	600	1:1	1:3	-
Access Road 5	AR5	Access Road	440	RZ	540	>1:1	1:1	-
Access Road 6	AR6	Access Road	420	RZ	140	>1:1	-	-
Access Road 6.5	AR6.5	Access Road	470	RZ	250	1:1	-	-
Access Road 7	AR7	Access Road	600 to 470	RZ	1,000	-	1:3	1:10
Access Road 7	AR7	Knipple Transfer Station (west end only)	470	RZ	100	-	-	1:100
Access Road 8	AR8	Access Road	730 to 660	RZ	700	1:1	1:3	-
Knipple Glacier 1	KG1	Access Road	730 to 650	RZ	700	1:1	1:3	-
Mine Site 1	MS1	Access Road	1,440 to 1,370	RZ	2,000	>1:1	1:1	-
Mine Site 2	MS2	Access Road	1,370	RZ	300	1:1	1.10	-
Mine Site 5	MS5	Access Road	1,420 to 1,440	RZ	600	>1:1	1:3	-
Mine Site 9	MS9	Access Road	1,455	RZ	100	1:3	-	-
Mine Site 10	MS10	Access Road	1,455	RZ	150	1:3	-	-

Note: P = potential to reach access road or facility

Figure 18.5 Access Road Avalanche Hazards



One area (Path AR8) is affected by ice fall (blocks falling off bluffs above the road). Several avalanche paths on the southwest and northeast side of the Knipple Glacier could affect the road if it is realigned to avoid crevasses. Potential consequences of avalanches reaching the access road include damage to vehicles, occupant injury or fatality, and traffic delays for avalanche debris clean up. Avalanche path characteristics for the Knipple Glacier segment are expected to change as the glacier changes over time, so this segment will be re-assessed regularly.

Areas within Paths AR4, AR8, and KG1 have increased hazard and consequences due to the high frequency of avalanches and ice falls reaching the affected areas, as well as magnitudes large enough to severely damage vehicles, injure occupants, and delay the flow of traffic during storms, when avalanche control is not feasible.

Knipple Transfer Station

The Knipple Transfer Station is located at the valley bottom near the confluence of the Salmon and Knipple valleys. Extreme avalanches to Size 4 occurring in Path AR7 (Table 18.6) are estimated to reach the west end (approximately 20%) of the Knipple Transfer Station pad with an estimated return period of at least 100 years. Avalanches are not expected to reach the eastern side of the Knipple Transfer Station pad where primary fixed facilities (camp) are located. Potential consequences of avalanches reaching the site include damage to infrastructure, and injury or fatality for any personnel located in the runout area.

TRANSMISSION LINE

Avalanche Hazard

The preferred transmission line route begins at Long Lake, where a hydroelectric generation facility is currently being built, approximately 14 km north of Stewart, BC. Although the exact alignment has not been finalized, the line is proposed to follow a route on the east side of the Salmon Glacier Valley to the proposed Knipple Transfer Station area. The line is then proposed to follow a route along a ridge on the southwest side of the Knipple Glacier to the mine site. An optional transmission line route for the Project follows the access road alignment from the BC Hydro Northern Transmission Line (NTL) at Highway 37.

Initial analysis indicates that there is approximately 20 to 25 avalanche paths that affect the preferred transmission line route, although they would only pose a hazard if supporting structures (towers) were built in avalanche paths, or conductors were low enough to the ground. Potential consequences include damage to towers or conductors, and interruption of service to the mine. In addition, worker injury of fatality may occur if the line is built, or if maintenance if undertaken in avalanche hazard areas during avalanche season. The final alignment of the transmission line (including specific structure locations) is expected to be detailed during the next phase of the Project, and will be assessed further for avalanches at that time.

The optional transmission line alignment parallels the access road from the NTL adjacent to Highway 37, and is potentially affected by the same avalanches that affect the access road. There may also be additional paths that affect the line depending on the final alignment. Potential consequences to the Project would be the same as the consequences of avalanches reaching the preferred transmission line alignment.

Static Snow Forces

In addition to avalanche hazards, transmission line towers may be subject to forces of snow creep and glide, depending on their location on slopes. Although snow creep and glide are not fast moving events, they may generate forces that can exceed the bending strength of the tower or the strength of the foundation. If towers built on slopes are not designed to withstand these forces, potential consequences may include damage to towers and associated impact to conductors resulting in interruption of services to the mine.

18.7 TRANSMISSION LINE

The Project will be powered by electricity from the BC Hydro system. After reviewing potential transmission routes, it was determined that the preferred route is from the recently constructed Long Lake Hydro Substation to the Project site. The feasibility of this 50 km route was investigated by Valard, under contract to Pretivm, to provide a feasibility-level design, cost estimate, and an assessment of constructability for the transmission line route. The full analysis of the transmission line is presented in Valard's report entitled "Brucejack Project – Transmission Feasibility Study and Cost Estimates"

and dated June 2013. As requested by Pretivm, Valard also studied a contingency transmission line route from the NTL, which is currently under construction by Valard on behalf of BC Hydro.

18.7.1 Transmission Line Interconnection and Route

In fall 2011, a study was conducted of the potential points of interconnection on the BC Hydro grid and associated transmission routes to the Project site. As a result of this study, the route from the substation for the Long Lake Hydro (LLH) Project to the Project site was selected as the preferred route (Figure 18.1). Starting at LLH, the transmission line would follow the bedrock slopes on the east side of the Salmon Glacier to the terminus of the Knipple Glacier. From the Knipple Glacier, the preferred transmission line follows the upper crest of the bedrock slope south of the glacier to the Project site.

The route follows bedrock-dominated terrain that is characterized by gentle to moderate slopes, bedrock hummocks, and discrete debris flow/snow avalanche tracks (Figure 18.7). As shown in the photo, the high elevation and prolonged snow cover have limited both tree growth and stand density throughout the area. Snow avalanche areas are clearly evident by exposed soil tracks and/or areas devoid of trees and characterized in Section 18.6. For the portion of the route from the Knipple Glacier to the Project site, a feasible route exists over the bedrock slopes to the south of the glacier to avoid both the engineering challenges due to glacier movement and the snow avalanches and snow creep prevalent on the slopes to the north of the glacier.

The conditions along the preferred route are substantially different than the NTL contingency route. Specifically, the NTL contingency route would have a point of interconnection near Wildfire Camp and then follow along or near the Brucejack Mine Road to the foot of the Knipple Glacier. From this point, the NTL route would be the same as the preferred route.

Figure 18.6 Map of Transmission Line Route

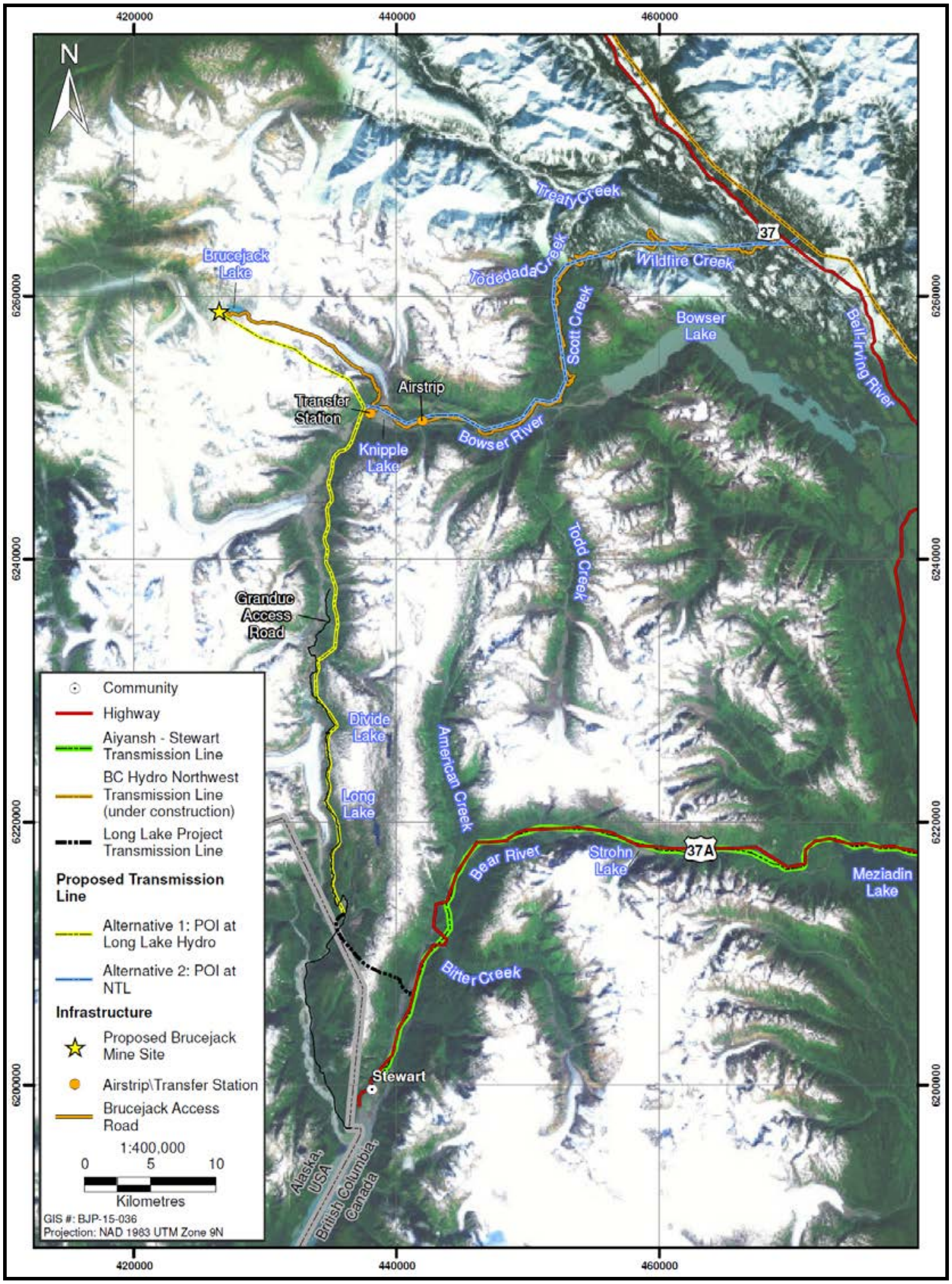


Figure 18.7 Photo of Typical Slopes in the Salmon River Valley

18.7.2 TRANSMISSION LINE DESIGN AND CONSTRUCTION

Given the preferred route and terrain for the proposed Brucejack Transmission Line, various options were investigated for conductor selection, tower design, and construction approach. While the vast majority of transmission lines in BC and elsewhere are constructed using wood poles, the use of steel towers were considered more suitable to the terrain and construction requirements for the Project.

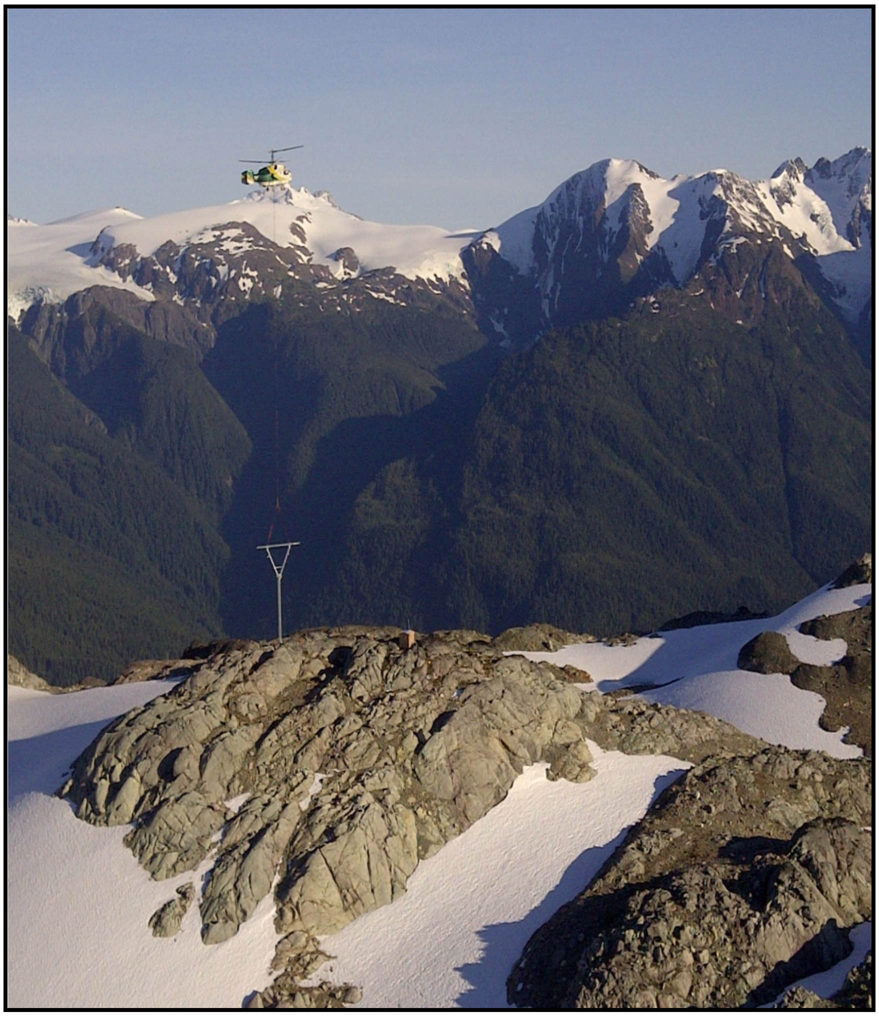
Engineering and construction constraints for the Project include:

- the moderate to steep slopes immediately above the existing road to the Granduc Mine, limiting the suitability of this corridor for a transmission due to impacts and significant risk of upslope hazards (tree fall, debris slides, etc)
- the lack of road access from the Granduc Mine site to the Knipple Glacier, significantly increasing the access costs (on a per-structure basis)
- the snow avalanches on many slopes in the area, limiting the technically viable routes – particularly in the Knipple Glacier area
- the movement of the Knipple Glacier, presenting foundation conditions not suitable for transmission towers

- the bedrock-dominated terrain along the proposed transmission route, which is favourable for rockbolt-type foundations.

It is important to note that the design constraints and construction conditions are similar to a nearby project currently being completed. This project, constructed almost entirely with helicopters, consists of 37 towers to carry a special 300MCM conductor and operates at 138 kV. Figure 18.8 shows a completed transmission tower for the Project.

Figure 18.8 Helicopter Placing Steel Transmission Structure for the LLH Project



Based on the constraints previously noted, the following key criteria for the initial design of the Brucejack Transmission Line include:

- the selection of 138 kV as the operating voltage to eliminate the need for a substation at LLH interconnection
- the use of two conductors, selected to accommodate corona effects due to elevations above 3,000 m and provide the necessary tensile strength to span the snowfields on the south slopes above the Knipple Glacier

- the design of special towers to span the snowfields and larger snow avalanche areas
- the use of single-steel monopole towers with helicopter placement, to lengthen the spans between structures and eliminate the need for an access road or track along the transmission route
- limited tree clearing with no removal (trees bucked and left in place along the corridor).

Valard's report on the Brucejack Transmssion Line contains additional information and a discussion of these constraints.

18.7.3 Transmission Line Operations, Maintenance and Emergency Response

Once constructed, the transmission line will be controlled out of the Brucejack Substation at the Project site with the main switches located at the interconnection at LLH. It is understood that the operation of the Brucejack Substation and Brucejack Transmission Line will very likely be carried out under a joint operating order with BC Hydro, as it is for other industrial customers. The joint operating order establishes the procedures and communication protocols for operation of the Brucejack Substation to protect any transmission line workers and the integrity of the BC Hydro system.

Maintenance of the Brucejack Transmission Line will consist of visual inspections along the transmission line, as well as periodic infrared surveys to look for potential deterioration in splices or other energized components. This will be complemented with a periodic inspection of the transmission line towers, with climbing inspections to ensure the functionality of all conductors, guy wires, crossarms, and other transmission tower components. Emergency response will also be important to manage the risk to the transmission line, with temporary wood poles kept at a central location to facilitate rapid response and restoration in the event of extreme weather damaging the transmission line.

18.7.4 Transmission Line Feasibility Budget Estimates

Capital and the operations/maintenance costs were developed for the transmission line based on engineering and construction criteria previously described. In both cases Valard included the available project information as well as its experience and expertise as a specialty transmission line contractor to estimate the costs.

The capital cost estimates are based on the following:

- The estimate includes the direct costs for procurement, construction, and commissioning of the transmission line for the preferred route.
- The estimate includes the indirect costs for the Project including engineering, project management/administration, camps, and other common costs for large transmission or construction projects.

- This estimate does not contain permitting costs or other development costs. Additionally, costs for management and engineering are based on an engineer-procure-contract (EPC) delivery model.
- The clearing estimate is based on the following parameters:
  - a 30 m wide right-of-way and average clearing width, with tree sizes and stand density as observed during aerial reconnaissance of the proposed route
  - very limited new and upgraded road, as almost all of the towers will be placed with helicopters
  - clearing along forested portions of the route, to be confirmed by forest engineering as part of detailed cost estimate; this estimate also assumes that the felled trees will be left in place with some bucking and limbing, as they are small and likely not merchantable.
  - these costs also assume that Pretivm will provide access (or permits for access) along the road to the Granduc Mine and the use of the tide staging area prior to line construction mobilization.
- Foundation conditions were estimated based on 30% soil, and 70% rock. The foundation estimates also include four specialized foundations for the longer spans across the snowfields south of the Knipple Glacier.
- Transmission structure assumptions include the assumption that single steel poles will be placed on suitable bedrock foundations for virtually the entire transmission line route.
- Helicopter stringing support will be along the entire route.
- Note that this estimate assumes construction over two field seasons, as per the proposed construction schedule. In the event the Project schedule stretches over a longer period of time, or compressing of the schedule is necessary due to delayed construction start, increased costs are likely compared to the costs in the estimate.

For the operation and maintenance costs, Valard estimated the annual costs for activities commonly included as part of transmission line operations and maintenance for lines comparable to the Brucejack Transmission Line. These include aerial inspections on an annual basis (including thermal imaging every second year), detailed ground inspections every four years, and some vegetation management. The operation and maintenance costs will also need to include an active snow avalanche control program for the slopes above the transmission line.

18.8 Ancillary Surface Facilities

18.8.1 Architectural Design Basis

The architectural design criteria outlines the cost estimation parameters of structures and facilities for the Project. The ancillary facilities design used pre-engineered and

modular construction where possible to minimize cost and site construction. The preliminary design of the buildings took into consideration local climate and site conditions.

Tracked vehicles will be used to transport components across Knipple Glacier to site. As such, vehicle limitations on shipping sizes and weights were considered in the cost estimation.

Pre-engineered Mill Building

The mill building was designed as one large structure to minimize issues with high snow drifts and snow removal between multiple buildings. This structure will include the following infrastructure and equipment under one roof:

- ore transfer tower and SAG mill feed bin
- process equipment including grinding mills, flotation tanks, reagent storage, thickeners, and concentrate handling and load out
- paste backfill plant
- administration
- mine dry
- truck shop
- emergency response vehicles and first aid
- warehouse
- assay and metallurgical laboratories
- wicket room for underground.

Budget prices received from bidders included the building envelope only. Three buildings will be constructed with a structural steel frame, steel girts and purlins, and intermediate structural members. The walls and roof will be constructed of insulated metal panels. The envelope package comes complete with doors and all other envelope-related items such as man doors, overhead doors, canopies, etc. High-bay lighting will also be included where applicable.

Modular Buildings

The new additional permanent camp (330 person capacity) and the scale house will be constructed using modular building design. Each building will include heating, ventilation, and air-conditioning (HVAC), electrical, piping, and 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 roof. Once the modules are in place and connected together, the complex will be weather tight. Roofs will be designed to minimize snow accumulation.

INTERNAL ARCHITECTURAL ITEMS

Many of the buildings that will be pre-engineered or constructed with a post and beam design will need a certain amount of architectural material within each structure, such as washroom, offices, mine dry, etc. These items were quantified and priced separately on a building by building basis.

18.8.2 MILL SITE INFRASTRUCTURE FACILITY DESCRIPTION

MILL BUILDING

The mill building will be a pre-engineered steel building with insulated roof and walls. It will be supported on a concrete spread footing with concrete grade walls along its perimeter. The building floor will be a concrete slab-on-grade, and will be sloped towards sumps for cleanup operation. Heavy equipment with dynamic loads housed in the mill building will be supported on a concrete foundation isolated from other building components.

The ground-floor level will house the process equipment, fresh and fire water tank, assay laboratory, warehouse, mine dry, wickets and lamp room, truck shops, first aid, fire truck and ambulance parking, and maintenance shop. A lean-to will be provided for the tailings thickener and covered parking.

Interior steel platforms on multiple levels will be provided to support process equipment and to meet ongoing operation and maintenance needs. Elevated concrete floors will be provided to house administration, mine dry, offices, and control rooms.

ASSAY AND METALLURGICAL LABORATORIES

The assay and metallurgical laboratories will house all necessary laboratory equipment for metallurgical grade testing and control. The lab will be equipped with all appropriate HVAC and chemical disposal equipment as needed. The facility floor will be reinforced as needed to accommodate specialized equipment.

WAREHOUSE FACILITY

The warehouse facility will house a mill shop, electrical and instrumentation shop, mechanical room, and tool crib.

MINE DRY

The mine dry will be constructed to accommodate 450 people, each with individual lockers and hanging baskets. The wicket and lamp rooms will be adjacent to the mine dry and alongside the covered parking, where underground personnel will be transported to and from the underground mine.

TRUCK SHOP

The truck shop will include bays for heavy and light equipment, a welding bay, and a wash bay complete with pressure washer. Sumps and trenches will be constructed to

collect waste water during maintenance operations. A floor hardener will be applied to concrete surface on high-traffic areas. Steel inserts will be embedded into the concrete in areas where tracked vehicles will be driven. A 5 t overhead crane will service this facility.

FIRST AID

The first aid and emergency response facility will include parking for a fire truck and an ambulance. A helicopter pad will be located close to the facility for any medical evacuation requirements.

ADMINISTRATION

The administration will be constructed on the second floor above the mine dry.

CONVEYING

Conveyors will be vendor-supplied systems and will include all structural support frames, trusses, bents, and take-up structures. Elevated conveyor systems will be supported on vendor supplied steel trusses spanning between steel bents on concrete spread footings, or supported from building trusses.

CAMP FOR CONSTRUCTION PHASE AND OPERATIONS

The current exploration camp includes a kitchen and dormitories, and can accommodate 180 people. Two dormitories will be demolished to allow for construction of the new mill pad, reducing the current camp capacity to 120. The new permanent camp will accommodate 330 people and will include an additional kitchen, and recreation and exercise facilities. The new camp and remaining exploration camp will have a combined capacity of 450 people during construction. Once construction is complete, the permanent camp will be refurbished for use during operations.

ON-SITE EXPLOSIVES STORAGE

Two pre-fabricated Sea Can-type structures will house the explosives storage and will be locked at all times to prevent unauthorized access. This storage facility will be located 1.5 km northeast of the mill building. Access to the explosives storage will be by road and the facility will be controlled by a locked gate.

DETONATOR MAGAZINE STORAGE

A pre-fabricated Sea Can-type structure will house the detonator magazine storage and will be locked at all times to prevent unauthorized access. This facility will be located 500 m southeast of the explosive storage facility, and 1.0 km from the mill building. Access to the detonator magazine storage will be by road, and the facility will be controlled by a locked gate.

TEMPORARY FACILITIES

A metal covered structure will be constructed at the laydown area to store equipment temporarily during construction.

18.9 WATER SUPPLY AND DISTRIBUTION

18.9.1 MILL SITE FRESH WATER SUPPLY INFRASTRUCTURE

The fresh water system will supply fresh water from the main water treatment plant and/or from Brucejack Lake to a common fresh and fire water holding tank located inside the mill building. For process use, water will then be pumped from the upper portion of the tank to gland water and other fresh water distribution such as paste plant, reagents, and other process equipment. For fire protection, water will be pumped from the holding tank to a fire main that will be routed along the inside perimeter of the building to service the fire protection system.

18.9.2 POTABLE WATER

MILL BUILDING

In the mill building, potable water will be drawn from groundwater wells and pumped to the potable water treatment system located inside the mill building. From here, treated water will be pumped to the potable water distribution system, which supplies water to washrooms, safety shower head tank, mine dry, etc.

CAMP

At the camp, potable water will be drawn from groundwater wells and pumped to the potable water treatment system located near the camp facility. The water treatment will be a chlorine disinfection based system similar to the mill building system. From here, treated water will be pumped to the camp for distribution.

18.10 WATER TREATMENT PLANT

18.10.1 UNDERGROUND MINE AND SURFACE WATER TREATMENT PLANT

The mine water treatment plant will treat underground inflows and surface water from the collection pond. For purposes of this study, it is estimated that the operations phase of the mine life will cease at the end of Year 22 and that the mine water treatment plant will continue to operate until Year 25.

To meet throughput needs, the mine water treatment plant will initially be constructed as two independent trains, each capable of treating up to 3,720 m^3/d for a total capacity of 7,440 m^3/d. It is scheduled to be constructed in Year -1 of the mine life so it will be operational to treat flows in Year 1. In Year 13 a third train will be added to the mine

water treatment plant to increase the total capacity to 11,160 m^3/d, as it is estimated that in Year 14 there will be an increase in underground inflow.

For underground development in Years -2 and -1, there is an existing water treatment plant that will be used to treat water pumped from underground.

18.10.2 POTABLE WATER TREATMENT PLANT

MILL BUILDING

The potable water treatment plant will be a vendor package installed inside the mill building. A hypochlorite solution storage and feed system will be provided to dose chlorine into the water pipeline as water is pumped from the groundwells to the storage tank. A packaged booster pumping system will be provided to pump water from the storage tank to the mill building distribution system.

The water requirements for the mill building will be approximately 84 m^3/d during operations, based on an average usage rate of 300 L/d per person and a crew of 280 people. The potable water system is not intended to provide process water or be a source of firewater.

CONSTRUCTION/OPERATIONS CAMP

The potable water treatment plant will be a vendor package installed on the camp pad. A hypochlorite solution storage and feed system will be provided to dose chlorine into the water pipeline as water is pumped from the groundwells to the storage tank. A packaged booster pumping system will be provided to pump water from the storage tank to the camp water system.

The camp water requirements will be approximately 99 m^3/d during construction (and less during operation) based on an average usage rate of 300 L/d per person and a camp population of up to 330 people. The potable water system is not intended to provide process water or be a source of firewater.

The existing camp has a potable water treatment package sized to service 180 people. This camp will be used during the construction period in addition to the new camp.

18.10.3 SEWAGE TREATMENT PLANT

A sewage treatment plant will be installed at the construction/operations camp for a camp population of up to 330 people. The mill building will have a sewage lift station inside and a heat-traced holding tank outside. A truck will transfer sewage from the holding tank to the sewage treatment plant at the camp.

The existing camp has sewage treatment facilities sized to service 180 people. This will augment the new camp capacity during the construction period.

18.11 WASTE ROCK DISPOSAL

It is anticipated that approximately 2.28 Mm^3 of excess waste rock generated from construction of the mine site and general mining activities will be disposed of in Brucejack Lake. Waste rock not disposed of in the lake will be used as backfill in the underground workings. A conceptual layout was developed for disposal of waste rock in the lake (BGC 2013). It is assumed that all of the waste rock disposed of in Brucejack Lake will be PAG and, therefore, must be placed more than 1 m below the low water elevation of the lake. This minimum depth will be reviewed at future stages of design. The waste rock will be placed in the lake by advancing a platform or causeway out into the lake. Waste rock will be end dumped from haul trucks onto the platform/causeway and then, either a dozer will be used to push it over the side or an excavator will be used to cast it over the side. As previously noted, all PAG waste rock disposed of in the lake must be placed more than 1 m below the low water elevation of the lake. Therefore, any waste rock placed above this minimum depth must be non-acid generating (NAG). Construction of the platform/causeway will require NAG material to be advanced out over the submerged PAG waste rock; a source of NAG rock will be quarried and stockpiled for this use.

18.12 TAILINGS DELIVERY SYSTEM

Thickened tailings slurry will be discharged to the bottom of Brucejack Lake (80 m deep) when not used for paste backfill (approximately 50% of the time). For discharge to the lake, the tailings slurry will be pumped to an agitated slurry mixing tank at approximately 65% w/w solids, and then diluted to 35% w/w with water at the nominal solids throughput rate of 112 t/h. The diluted slurry will be pumped overland at a distance of 525 m and then underwater along the lake bed another 445 m to the discharge point.

There will be one duty pump and one standby pump to permit an immediate switch over when necessary. The pumps will discharge the diluted slurry at a constant flow rate of varying concentration, which will depend upon the throughput and concentration of tailings slurry entering the mixing tank. The mixing tank will be maintained at a constant level by the addition of water through a control valve.

The diluted slurry will be pumped through a 10" high-density polyethylene (HDPE) DR 6.3 pipeline from the mill building to the bottom of Brucejack Lake. Much of the pipeline alignment may be subject to damage from avalanches. Accordingly, the pipeline overland will be trenched and backfilled in most locations to protect the pipe. The pipeline will also have a continuous downward slope from the mill building to the lake shore to ensure that the line drains during shutdowns.

At the lake shore, the pipeline will divide into two parallel pipelines. The primary pipeline will discharge at 80 m and the secondary pipeline will discharge at 60 m. The pipelines will be switched if and when the back-pressure associated with the growing deposit approaches the upper operating range of the discharge pump. This design increases system flexibility. The pipelines can also be switched if there is some unforeseen circumstance that causes a pipeline to be unusable.

Air/vacuum valves will be installed at critical points along the pipeline to prevent the possibility of air entering the underwater section. A large volume of air entering the underwater section could potentially float sections of the pipeline. The valves will function primarily during start up and shut down.

Figure 18.9 shows the pipeline route in plan and elevation. A small quantity of coarse sand and gravel will be placed at the terminus of the outfall prior to tailings discharge. Tailings solids will further accumulate and cover the discharge point through operation. This feature will act as a filter to prevent suspended solids from entering the upper layers of the lake's water column and from subsequently being discharged into Brucejack Creek, potentially in violation of strict receiving water quality regulations.

There will be a constant flow through the pipeline to keep the deposit at the end of the outfall fluidized. When the thickened tailings are used in the backfill plant, flow will be maintained with water.

Earlier Rescan designs for discharging tailings through a deposit (at Minahasa in Indonesia and at Compañía Minera del Pacífico in Chile) have been demonstrated to be effective in reducing TSS. This configuration was also recently adopted for discharge in a lake at a project in the US.

During lake turn over, there may be some fine deposited tailings particles transported upward toward the lake surface, potentially resulting in elevated suspended solids concentrations in the surface layer and at the outlet.

Figure 18.10 shows the area on the lake bottom that the tailings would occupy at the end of the 22-year project life, assuming that 9.5 Mt of tailings are deposited over the LOM (Keogh 2013 pers. comm.) and that the density of the tailings deposit increases with accumulation and consolidation. (Tailings dry unit weight estimates were taken from the lab report completed by Golder (2013)). The overall footprint of the tailings at the end of 22 years occupies most of the lake bottom, to a depth of approximately 48 m at its edge and a depth of 38 m at the apex of the deposition cone.

A complete bathymetric survey of the lake, with detailed data collected along the subaqueous pipeline alignment, will be accomplished in the detailed design stage. In addition, tailings slurry rheology, at a broad range of concentrations, will be definitively measured in the laboratory.

Figure 18.9 Plan and Profile Tailings Discharge Pipelines

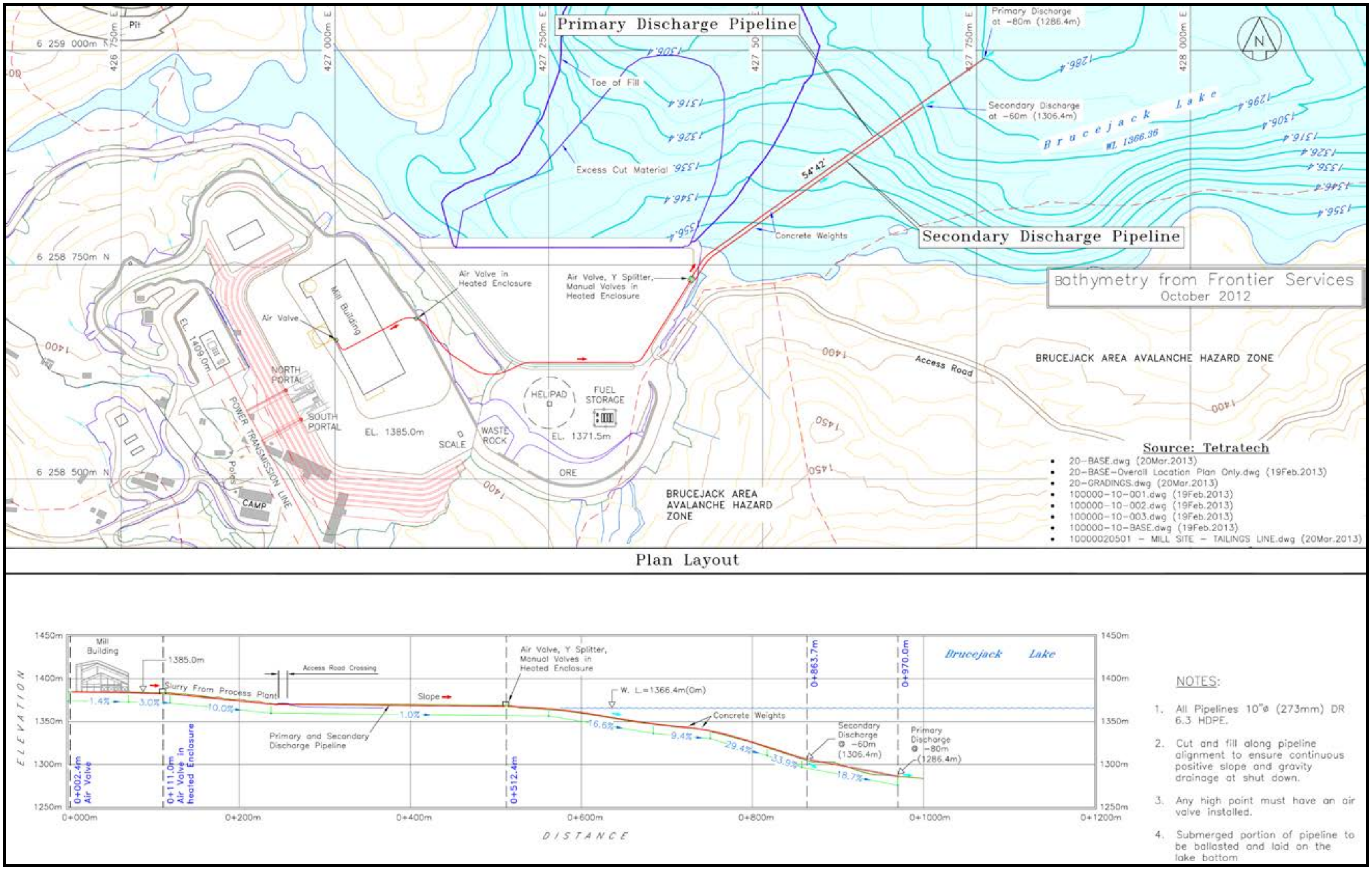
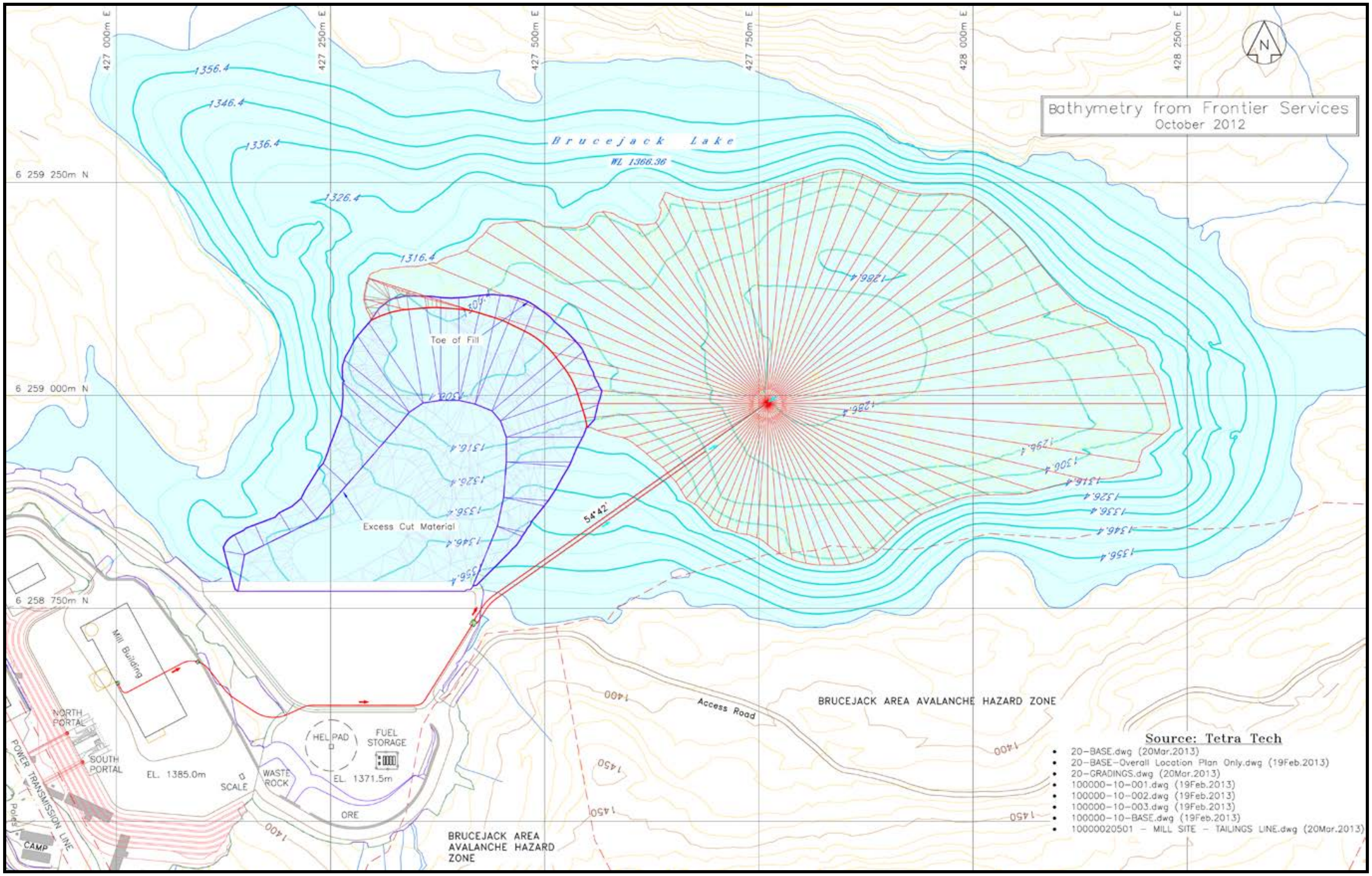


Figure 18.10 Foot Print of 3% Deposition



18.13 BRUCEJACK LAKE SUSPENDED SOLIDS OUTFLOW CONTROL

Approximately 2.3Mt of waste rock and 9.5 Mt of tailings are anticipated to be deposited in Brucejack Lake over the projected 22-year mine life. The estimated final footprint of the waste rock and tailings is shown in Figure 18.10. Stringent discharge criteria (based on the Metal Mining Effluent Regulations (MMER)) state that the TSS concentrations in the outflow at Brucejack Creek must be less than 15 mg/L.

The tailings deposition system has been developed to minimize the concentration of fine suspended solids in the outflow to Brucejack Creek by discharging near the bottom of the lake (at 80 m depth) and under the accumulations of tailings solids. On the other hand, waste rock with a wide range of particle sizes is to be deposited in the lake by surface dumping from causeways raising the possibility that fine granular material will be introduced to the surface layer of the lake and to the outflow.

It will be necessary to control the TSS concentrations at the outlet of Brucejack Lake to meet the MMER regulations. The following three possibilities were proposed and discussed, and an allowance has been included in the capital cost estimate:

- washing waste rock before depositing in the lake
- turbidity curtain surrounding tailings deposit and waste rock
- outlet control structure for temporary control of the lake outflow.

Further work regarding the TSS mitigation strategy is required during subsequent stages of design.

18.14 COMMUNICATIONS

18.14.1 SITE TELECOMMUNICATION SYSTEM

A complete site-wide telecommunications system will be installed in two phases. The first phase will include the base installation of the communication system during the construction phase. The second phase will allow for expansion of the system to include the operating plant. Major subsystems include:

- a VoIP telephone system for buildings, camps, and offices
- satellite communications for critical voice and data needs
- Ethernet cabling for site infrastructure and wireless internet access
- very-high frequency (VHF) two-way radio system with eight public channels
- four remotely located VHF repeaters
- satellite TV and Internet for the camp at the mill site and the camp at the transfer station, including a wireless access tower for communications to the transfer station and airport location.

A pre-manufactured trailer will be used as a central equipment enclosure (CEE) to house all communications equipment for both phases. The CEE will include all HVAC equipment and an uninterruptable power supply (UPS). The site telecommunications will be linked to the site fibre optic backbone via the CEE. A separate existing satellite communications system is provided and is isolated in a separate building from the CEE cabinet. This system will handle emergency off-site contact in the unlikely event that the CEE and its vital equipment are compromised.

18.14.2 Process Plant Control

Overview

Plant Control

A control system will provide equipment interlocking, process monitoring and control functions, supervisory control, and an expert control system. The control system will generate production reports and provide data and malfunction analyses, as well as a log of all process upsets. All process alarms and events will be also logged by the control system.

Operator interface to the DCS will be via PC-based operator workstations located in the following area control rooms:

- underground crushing
- process plant
- paste plant.

The plant control rooms will be staffed by trained personnel 24 h/d.

Operator workstations will be capable of monitoring the entire plant site process operations, viewing alarms, and controlling equipment within the plant. Supervisory workstations will be provided in the offices of the Plant Superintendent and the Mill Maintenance Superintendent.

Field instruments will be microprocessor-based “smart” type devices. Instruments will be grouped by process area, and wired to local field instrument junction boxes in each respective area. Signal trunk cables will connect the field instrument junction boxes to the control system I/O cabinets.

Intelligent-type MCCs will be located in electrical rooms throughout the plant. A serial interface to the control system will facilitate the MCCs remote operation and monitoring.

For site-wide infrastructure (i.e. telephone, Internet, security, fire alarm, and control systems), a fiber optic backbone will be installed throughout the plant site.

CONTROL PHILOSOPHY

Primary Crushing Control System

A PC workstation will be installed in the main control room to monitor the underground and crushing operations, and the crushing and conveying operations to the coarse ore stockpile. The information will be provided to the mill process control system via serial or Ethernet gateway to the underground control system.

The control system will control:

- SAG feed conveyors (zero speed switches, side travel switches, emergency pull cords, and plugged chute detection)
- surge bin levels (radar level, plug chute detection).

Concentrator

To control and monitor all mill building processes, three PC workstations will be installed in the mill building's central control room.

The PC workstations will control and monitor the following:

- grinding conveyors (zero speed switches, side travel switches, emergency pull cords, and plugged chute detection)
- SAG and ball grinding mills (mill speed, bearing temperatures, lubrication systems, clutches, motors, and feed rates)
- particle size monitors (for grinding optimization and cyclone feed)
- pump boxes, tanks, and bin levels
- variable speed pumps
- cyclone feed density controls
- thickeners (drives, slurry interface levels, underflow density, and flocculent addition)
- flotation cells (level controls, reagent addition, and airflow rates)
- samplers (for flotation optimization)
- gravity concentrators, pressure filters, and load out
- reagent handling and distribution systems
- tailings disposal to paste backfill or tailings storage
- water storage, reclamation, and distribution, including tank level automatic control
- air compressors
- paste plant (vendor control system)

- fuel storage
- vendors' instrumentation packages.

An automatic sampling system will collect samples from various product streams for online analysis and daily metallurgical balance.

Particle-size-based computer control systems will be used to maintain the optimum grind sizes for the primary grinding and concentrate regrinding circuits. Particle-size analyzers will provide main inputs to the control system.

Remote Monitoring

CCTV cameras will be installed at various locations throughout the plant, such as the crusher conveyor discharge point, the SAG surge feed conveyer, the SAG and ball mill grinding area, the flotation area, the regrind area, the paste plant, the gold room, the concentrate handling building, and the tailings handling facilities. The cameras will be monitored from the plant control rooms.

18.15 POWER SUPPLY AND DISTRIBUTION

A BC Hydro 138 kV overhead power line will supply power to the Project site.

The main site power will be stepped down from 138 kV to 4.16 kV via two 15/20/25 MVA oil-filled transformers, complete with neutral grounding resistors, located in the main substation yard. Each transformer is capable of carrying the entire site load. A pre-fabricated electrical house (e-house), containing 5 kV class switchgears, will distribute power to various points on the site.

The main mill and underground loads will be fed via power cables in cable tray. The substation location is at a significant elevation above the mill; therefore, the proposed power cable routing will descend several benches from the substation then follow one bench to both the portal and mill building areas. A cable bridge will be constructed to span from the bench to the mill building allowing for safe clearance and free traffic flow below.

Within the mill, large loads will be powered at 4.16 kV. Smaller loads will be powered at 600 V via switchgear and MCCs. Variable frequency drives and soft starters will be employed strategically to optimize process and energy performance.

Approximately 1.6 km of 4.16 kV single wood-pole overhead power lines will be constructed to provide power to outlying buildings such as the water treatment plant, fuel storage, operations camp, etc.

The emergency power strategy will employ two elements:

- Four existing 500 kW, 600 V diesel generators will be installed at the main substation and will connect to the main power distribution bus. Although the

primary function of these units is to power critical loads underground and in the mill, select critical loads throughout the site can be powered as well.

- One of the 500 kW, 600 V diesel generators purchased for construction activities will be re-deployed as a dedicated back-up power supply for the permanent camp.

A dedicated power system PLC will be included in the e-house. This PLC will connect to 4.16 kV and 600 V switchgear as well as mine heating systems using fiber optic communication. An UPS will back up the PLC and communications to ensure reliable operations under all circumstances.

The power system PLC will perform two important functions:

- load optimization/load shedding to ensure line limits are not exceeded, while maximizing electricity use for mine heating
- power control during emergency power operations to ensure correct sequencing and operations of critical loads.

Although soil resistivity tests have not been performed, soil resistivity is expected to be very poor because of bedrock and mine waste. As a result, remote ground(s) will be constructed in addition to substation yard grounding.

18.16 Fuel Supply and Distribution

Diesel

Diesel fuel primarily for mobile equipment will be stored in four 50,000 L double-walled tanks located at the laydown area. The storage is estimated for a 10-day capacity, including allowance for auxiliary equipment. The fueling station will include a receiving pump, a strainer and delivery pumps, and filters.

One 5,000 L double-walled diesel fuel batch tank will be located on the surface above the vicinity of the underground fuel storage, and will serve to fill the 20,000 L underground mine tank by gravity.

Aviation Fuel

Aviation fuel for helicopters will be stored in one 5,000 L double-walled fuel tank located adjacent to the helicopter landing pad.

Gasoline

Gasoline for mobile equipment will be stored in one 5,000 L double-walled fuel tank located adjacent to the diesel fuel tanks.

Propane

Three 5,000 gal propane tanks will be located adjacent to the permanent camp facilities.

18.17 OFF-SITE INFRASTRUCTURE

18.17.1 KNIPPLE TRANSFER STATION SITE PREPARATION

The Knipple Transfer Station facility will be located along the access road approximately 5 km west of the Bowser airstrip. It is in a relatively flat terraced area understood to be above historical flood levels, and positioned away from an existing creek. Previous activities in the area used this location as a camp. Site preparation will include topsoil stripping, cut and fill, and pad surfacing. Site drainage will include surface drainage to the perimeter and outlet to connect with existing drainage courses.

18.17.2 KNIPPLE TRANSFER STATION FACILITIES

The Knipple Transfer Station facility layout will include a camp, transfer station, fuel dispensing system, helipad, and laydown area as shown in Figure 18.11. All deliveries to and from the mill site will report to this facility. Loads from highway trucks will be transferred onto tracked vehicles that will transport the load across the glacier and to the mill site. Similarly, loads from the mill site will be managed in reverse order.

CAMP

The camp will be sized to accommodate 30 people, complete with kitchen, recreation, dormitories, and a sewage treatment plant. Offices will be included in the camp to manage the shipping and receiving of goods. A diesel generator with backup will provide power to the camp and a potable water system will installed to distribute water. A wireless system will be installed for communications. An incinerator will be installed within a fenced area.

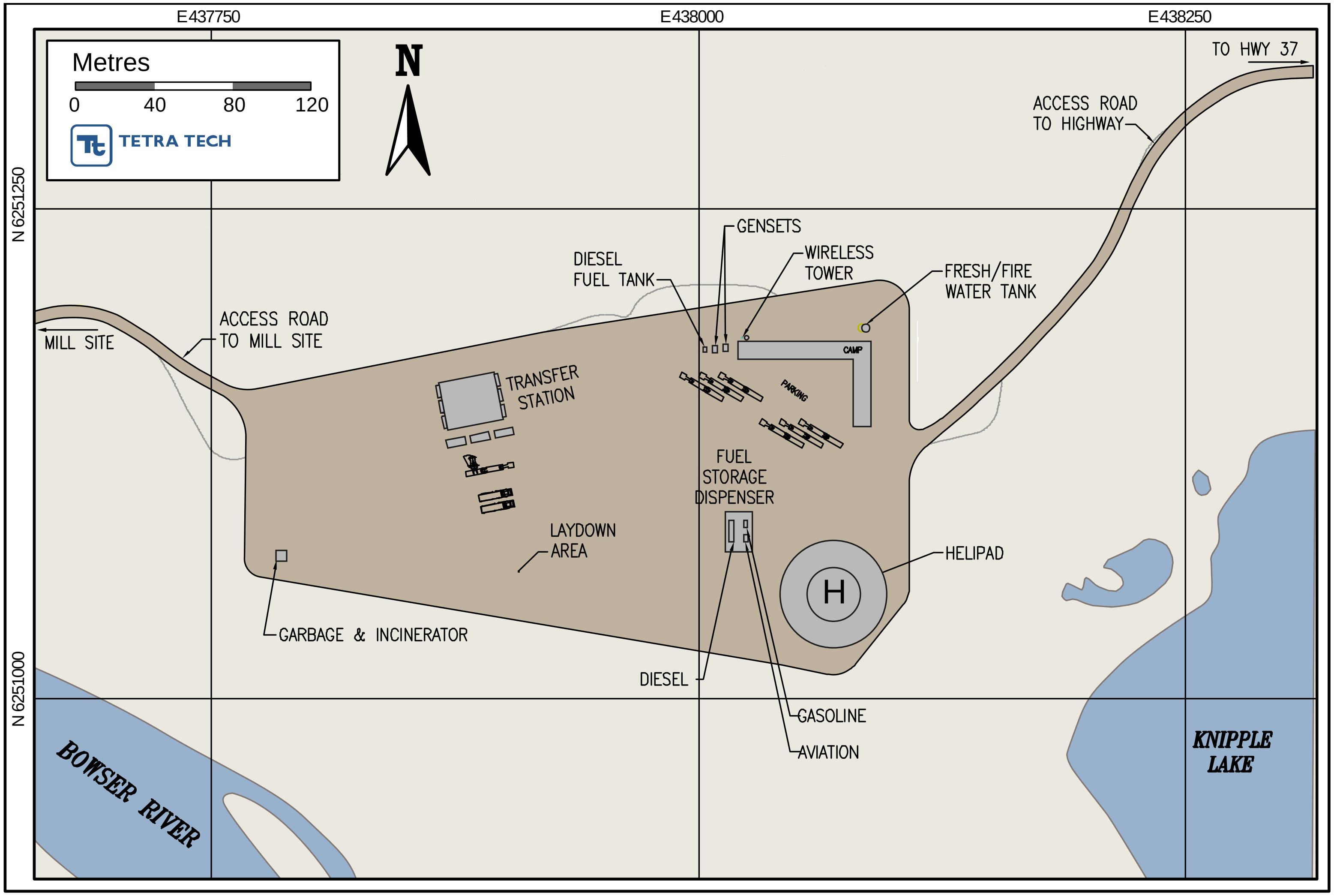
TRANSFER STATION

A two-bay, pre-engineering building, complete with a 5 t overhead crane will be constructed on this site for use as a transfer station. This station will be used to transfer smaller loads and to maintain vehicles on a temporary basis. For larger loads, a mobile crane will be used to load/unload shipments at the laydown area.

TEMPORARY FACILITIES

A metal covered structure will be constructed at the laydown area to store equipment temporarily during construction.

Figure 18.11 Knipple Transfer Station Facility Layout



18.17.3 BOWSER AIRSTRIP

Regular chartered flights will transport mine personnel to and from the Project site from the point of origin to an aerodrome located west of Bowser Lake. Personnel will then be transported from the aerodrome to the mine camp by bus.

The new airstrip will be constructed at the site of the current gravel airstrip, which will be improved and expanded to provide a safe and maintainable facility for the chartered air traffic. The new airstrip is shown in Figure 18.12. This site was chosen without the benefit of meteorological information to confirm the direction of prevailing winds, or sufficient topographical or geotechnical information to confirm precise pavement structures or earthwork quantities.

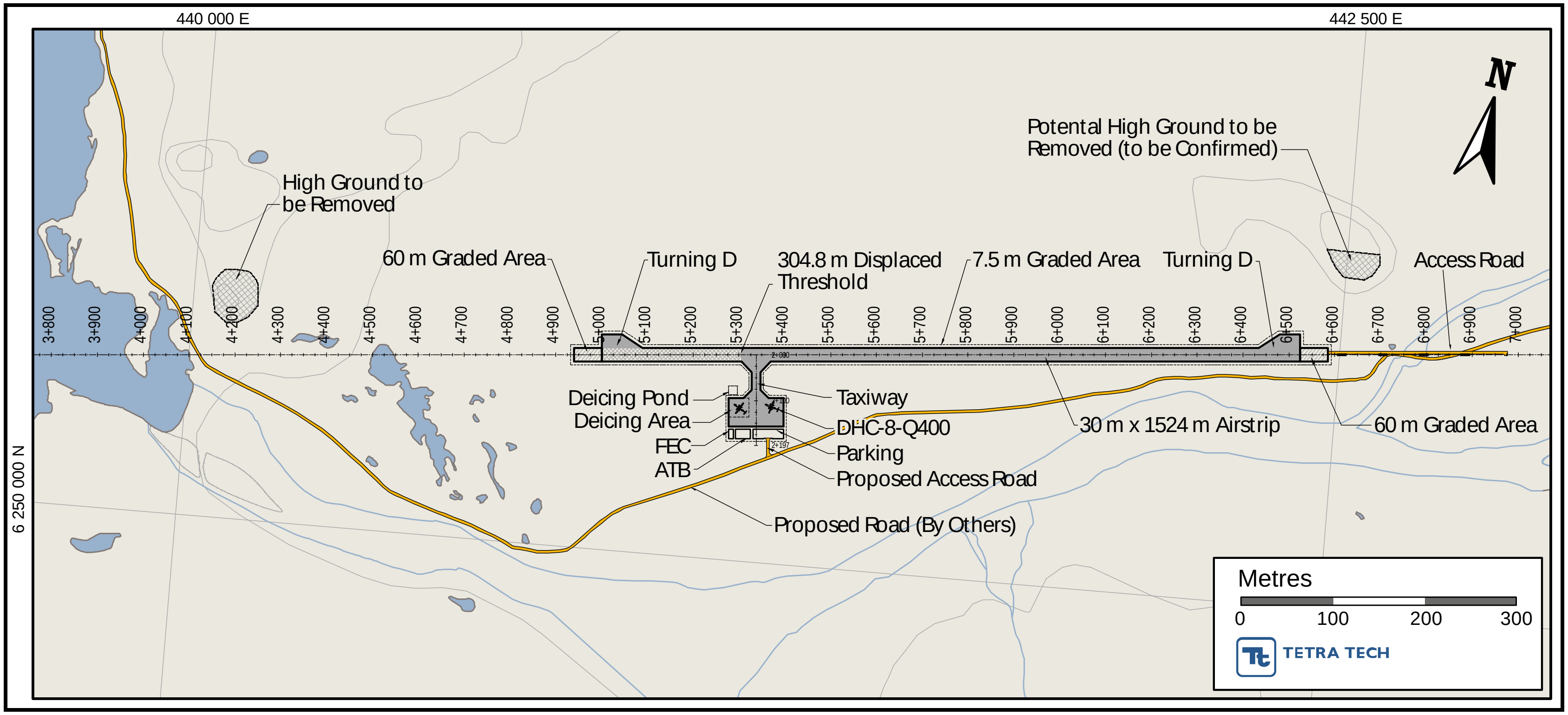
The passenger aircraft used in the design of the aerodrome is the Beechcraft 1900 however, the aerodrome facilities are sized sufficiently to allow DE Havilland Dash 8 turboprops and C-130 Hercules aircraft upon acceptance of the sites by the aircraft operators.

The aerodrome will be designated as “Registered”, which will allows service with an approved chartered aircraft without having to meet and comply with all of Transport Canada’s standards and operational requirements. The aerodrome will be designed to allow future “certification” for Beechcraft 1900 aircraft, should Transport Canada regulations change regarding charter flights into registered aerodromes. Upgrading to a “Certified” status will require additional earthworks on the west approach and additional grading and earthworks along the edges and ends of the runway. To certify for Dash 8 aircraft in the future will require even greater earth removal from the hill west of the runway. Future certification will also require significant additional administrative, reporting, safety management, wildlife management, and documentation duties on an ongoing basis.

The aerodrome will be supported by Instrument Approach Procedures (IAP) allowing Instrument Flight Rules (IFR) approaches and departures under suitable meteorological conditions.

The aerodrome will have a granular surfaced runway, taxiway, and apron and will be maintained to operate year round.

Figure 18.12 Bowser Airstrip



Instrument Approach Procedures

An IAP assessment has been undertaken with tentative approach limits established for flights under IFR. The assessment indicates the approach limits will remain quite high and require 3 miles forward visibility

Weather conditions would have to be such that for the approach to be safe, the cloud ceiling at the airport would be a minimum of 4,698 ft above the airstrip with a horizontal visibility of at least 3 miles. The limits for a Category D aircraft (faster and larger) would be 5,038 ft and 3 miles.

Accordingly, the east approach could operate in weather with a ceiling of not less than 5,838 ft above the airstrip and 3 miles horizontal visibility.

For aircraft departures, the site was reviewed for viability and found that due to surrounding terrain, a standard maximum rate of climb departure is not possible. For an IFR departure, the Visual Climb Area (VCA) limit would be 9,100 ft above the airstrip. This will likely result in Visual Flight Rules (VFR) departures whenever the ceilings are below 9,100 ft, but above legal VFR limits.

The restrictions on the proposed IAPs will likely result in poor reliability when weather conditions result in low ceilings or poor visibility. The airlines providing service for the Project should be provided this information to judge the reliability of flights based on a combination of VFR and IFR flights to the site. The airline should also be consulted on whether they could provide service to a lit runway at night based on the likely IFR limits and challenges of the surrounding terrain.

Obstacle Limitation Surfaces

The Obstacle Limitation Surfaces (OLS) for this site have been based on utilizing a Beechcraft 1900 aircraft. This does not mean the aerodrome cannot be used by larger aircraft, but operators must be made aware of the limitations of the site and the obstacles they will encounter.

The approach from the west (runway 07) encounters a hill which is the primary obstacle to the site. Earthworks will be required to reduce the impacts of the hill on the take-off approach surface areas. The threshold for runway 07 has been displaced to a position 300 m from the end of the runway to reduce the quantity of excavation required at the obstructing hill. This will reduce the landing length for runway 07 but likely not sufficiently to reduce the runway usability.

The approach from the east (runway 25) is relatively unobstructed and the threshold will remain at the runway end.

Both runway directions will retain the full length for take-offs, which are generally more critical for runway length.

MANOEUVRING AND MOVEMENT SURFACES (RUNWAY, TAXIWAY, AND APRON)

The runway, taxiway, and apron surface will be granular and suitable for turbo-prop aircraft. The runway will be 1,524 m (5,000 ft) long and 30 m (100 ft) wide and oriented magnetically to correspond to the runway designations 07-25. The runway will include a 7.5 m (25 ft) graded area long each runway edge and 60 m (200 ft) long graded area beyond each runway end. The taxiway will be constructed to a width of 18 m (60 ft) with a 6 m (20 ft) wide graded area along each edge. The aircraft parking apron has been sized to allow two Dash 8 sized aircraft to manoeuver and park.

All granular surfaces will be treated for dust reduction.

AERODROME LIGHTING

Due to the relatively high IFR approach limits and surrounding terrain, it will be important to confirm with the air service provider that night flights will operate at this site. For the purposes of this study we have included aerodrome lighting. Obstruction lights and hazard beacons on surrounding terrain have not been included.

The aerodrome will include runway, taxiway, and apron edge lighting. Illuminated signage and wind socks are included to provide pilots with clear directional cues. An Omni Directional Approach Lighting System (ODALS) will guide aircraft on the east approach. The west approach will have Runway End Identifier Lights (REIL) due to the displaced threshold. Precision Approach Path Indicators (PAPI) will be installed along the edge for both runway approaches to provide the aircraft with visual vertical guidance.

The lighting controls will be set up to allow pilots to switch the lighting on automatically as well as direct control by ground personnel.

AERODROME BUILDING INFRASTRUCTURE

A pre-fabricated ATPO-type trailer Air Terminal Building (ATB) will be located adjacent to the apron. The ATB will be equipped with sufficient windows to view the aerodrome and surrounding area. The ATB will contain radio equipment for Pretivm and ground to air communication. It will also be adjacent a weather station and will be equipped to give altimeter readings to the incoming and outgoing pilots. The ATB will be heated and contain washroom facilities. It is expected that all passengers will be loaded from the aircraft directly to a bus and that the ATB will not be sized to contain the passengers. The ATB will support several apron flood lights to illuminate aircraft loading and unloading.

A pre-fabricated modular Field Electrical Centre (FEC) will be located adjacent to the ATB. The FEC will contain all of the regulators and controls for the aerodrome and ATB electrical service. The FEC will be adjacent to another pre-fabricated modular enclosure containing the site electrical generator.

At this time there are no plans for aircraft re-fuelling facilities. Emergency supplies of jet fuel could be kept in barrels on-site with portable dispensing equipment.

AERODROME PERSONNEL AND MAINTENANCE EQUIPMENT REQUIREMENTS

Mine equipment can be used for much of the aerodrome surface maintenance.

A grader, dump trucks, and loaders from the mine or the mine road maintenance crews can be used for aerodrome snow removal and surface re-grading. A self-propelled compaction roller should be available on site.

A water tanker/distributer and operator should be available to supplement the chemical dust control measures.

A service pick-up truck with equipment for measuring the runway surface friction index will be required to allow the operator to relay the information to pilots.

Mine trucks and a loader equipped with a baggage box can be used to load and transfer passenger baggage and air freight.

A trained radio operator should be at site prior to the flight leaving its originating airport and remaining until the flight has landed and departed. This will allow the operator to relay weather and altimeter information to the pilots prior to, during, and after departure (in case an emergency return is required).

19.0 MARKET STUDIES AND CONTRACTS

19.1 MARKETS

The final products that will be produced at Brucejack will be gold and silver doré and a gold-silver flotation concentrate. The gold and silver doré will likely be transported to a North American-based precious metals refinery or sold to precious metals traders, most likely located in Asia, Europe, and North America. The flotation concentrate, will likely be sold to a base metal smelter or metal traders. Based on the LOM average, the gold-silver flotation concentrate is expected to contain approximately 130 g/t gold and 1,000 g/t silver.

19.2 SMELTER TERMS

Pretivm contacted the metal trader Transamine for information regarding concentrate sales, and subsequently received indicative smelting terms based on the assay data of the concentrate that was produced from the 2012 test work. According to the terms received from Transamine, it is anticipated that the concentrate will be trucked to Terrace, BC, and then transported by rail to a smelter in eastern Canada.

Tetra Tech recommends conducting further marketing studies for shipping concentrate to smelters located in Asia for a potential reduction in the shipping costs.

19.3 LOGISTICS PLAN

19.3.1 EQUIPMENT AND MATERIALS TRANSPORTATION

A major logistical initiative is required during the construction phase of the Project as a large amount of mining, construction and processing equipment, and consumables will be transported to the site. The costs and preferred modes of transport will be dependent upon the size and weight of the cargo and the origins of the shipping locations. The following modes of transport are available for the Project:

- truck
- rail
- barge
- ship transport
- air freight.

TRANSPORT MODE OPTIONS

Truck

There are a number of regional, long-haul and heavy-haul trucking companies that are capable of providing various services from all shipping points in North America and Mexico. All equipment and supplies by roads will be shipped by highway truck along Highway 37 to the mine site via an access road that begins at km 215 of Highway 37. The access road extends to the Knipple Transfer Station, which will be approximately 5 km west of the Bowser Creek air strip. The distance from Kitwanga (located at the junction of Highway 37 and Highway 16) to the entrance at the access road (located at the intersection of Highway 37 at Wildfire Creek) is approximately 215 km. The distance along the access road from Highway 37 to the transfer station is approximately 55 km.

Rail

Rail transport could be a viable option, particularly for cargo sourced from locations in the eastern regions of North America or Mexico. Rail service is available by Canadian National Railway to Kitwanga, Smithers, or Terrace. Cargo would then be unloaded for subsequent transport to the site via truck. Canadian National Railway operates in eight Canadian Provinces and 16 US states with connections to numerous points in North America. Canadian National Rrailway crosses the continent east-west and north-south serving ports on the Atlantic, Pacific and Gulf coasts with links to all three North American Free Trade Agreement (NAFTA) nations.

Barge

Barge service is available from either Port Metro Vancouver or Prince Rupert to the Port of Stewart. In addition to the current Stewart Bulk Terminal (SBT) operation, there are plans underway to construct and operate a new multi-purpose facility called Stewart World Port which will include a roll-on/roll-off cargo ramp capable of accommodating 6,000 t barges and 200 t loads. The advantages of using barge service include the opportunity to consolidate and ship large amounts of cargo, as well as the potential for moving oversized or heavy components in order to minimize highway travel and bypass any limitations due to bridges, tunnels, or overpasses.

Ship Transport

Cargo arriving from Asia could be directed to terminals in either Port Metro Vancouver or Prince Rupert. Consideration will be given to whether the cargo arrives in containers or in break-bulk form. The closest container terminal to the Project site is the Fairview Terminal in Prince Rupert. The 24 ha terminal is strategically located to receive cargo from Asia and has an operational capacity to handle 750,000 twenty-foot equivalent units (TEUs) per annum. There are expansion plans for the terminal to increase the capacity of the facility to 2,000,000 TEUs in order to meet demands of continued growth in the Asia-Pacific traffic trade.

Air Freight

Scheduled and chartered cargo service is available to nearby communities such as Terrace, Smithers, and Dease Lake. Cargo could also be delivered to the Bowser Creek air strip.

OVERSIZE AND HEAVY EQUIPMENT

For truck shipments within BC, the legal gross vehicle weight (GVW) limit is 63,500 kg (cargo and transport vehicle combined). Dimension limits of the combination of cargo and transport vehicle are 26 m length by 4.14 m height by 2.6 m width. Any shipments that exceed these dimensions or weight are classified as “overloads” and require applications to the Province of BC to obtain permits for travel.

TRANSPORT LIMITATIONS

In addition to any transport permits required, bridges along Highway 37 from Kitwanga to the junction of the Brucejack Access Road will need to be evaluated to ensure that the structures are capable of handling the legal vehicle weight and dimension requirements.

One bridge in particular that may be a limiting factor is the Nass River Bridge. This bridge is located approximately 141 km north of Kitwanga. It is a single-lane bridge with yield signs at either end. Built in 1972, it was constructed with wooden glue-laminated beams and has a capacity of 90 tons (or 180,000 lb) GVW. The bridge is 323 ft in length and has a horizontal restriction of 14 ft, 10 in.

In an effort to bypass the Nass River Bridge, barging cargo to the Port of Stewart is a viable option. Cargo would travel via barge to Stewart, unloaded, and transferred to truck for delivery to site along Highway 37A to Meziadin Junction, then north on Highway 37 to the Knipple Transfer Station. Permits and bridge evaluations may also be required for the truck haul portion of the trip.

The Port of Stewart has the capability of handling equipment via barge on a roll-on/roll-off basis. Currently, barge unloading takes place at SBT in the north eastern area of the terminal where the ground is sloped from the road level to the waters edge. Inbound movements would require mobile equipment such as a Bulldozer to assist in pulling the equipment.

Cargo that has been either received or loaded on barges includes:

- pipes
- paving equipment
- camp trailers
- gravel
- bags of concentrate.

SBT has the ability to mobilize cranes with up to a 200 ton capacity and can handle barges with up to a 5,000 t capacity. Barges must have permanent or portable ramps available. Currently, Wainwright Marine Services provides charter barge service and operates out of Prince Rupert.

TRANSPORT VIA TRACKED VEHICLES

At the Knipple Transfer Station, all equipment and supplies will be received and transferred on husky tracked vehicles. These vehicles will transport the equipment and supplies along the Knipple Glacier to the mine site. All cargo must fit within the following guidelines in order to accommodate transport to the site via tracked vehicles:

- length not to exceed 11,600 mm
- width not to exceed 2,600 mm
- weight not to exceed 36,000 kg.

Vendors are being informed of these limitations so equipment can be manufactured or modularized to allow for transport. Due to limited loads on tracked vehicles, large equipment will need to be delivered to the site in manageably sized sections and assembled on site.

19.3.2 CONCENTRATE TRANSPORTATION

Based on the current marketing arrangements, the concentrate will be destined for the Horne Smelter operation which is located in Noranda, Québec. This facility receives concentrate via rail in open top gondola railcars. The concentrate will be loaded in 2 t bags for shipment from the Brucejack site. The bags will be being transported via tracked vehicles to the Knipple Transfer Station. The bags will be unloaded and transferred to standard Highway B-Train flat-decks for shipment via the Brucejack Access Road, Highway 37, and then Highway 16 to Terrace. The bags will be received in Terrace, inventoried, and then broken to allow for the bulk loading of railcars. The estimated transit time from Terrace to the Horne Smelter is 13 days. A built for purpose transload facility will need to be constructed for the Project. Back-haul opportunities for hauling reagents and consumables will be explored in an effort to reduce transport costs during the operating period.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 SUSTAINABILITY AND ENVIRONMENTAL MATTERS

Pretivm is committed to operating the Project in a sustainable manner and according to their guiding principles.

20.1.1 GUIDING PRINCIPLES AND CRITERIA

PROJECT DEVELOPMENT PHILOSOPHY

Every reasonable effort will be made to minimize long-term environmental impacts and ensure that the Project provides lasting benefits to local communities while generating substantial economic and social advantages for shareholders, employees, and the broader community.

Pretivm is committed to sustainable resource development which balances environmental, social, and economic interests. Pretivm will comply with regulatory requirements and apply technically proven and economically feasible methodologies to protect the environment throughout exploration, mining, processing and closure activities.

Environmental management is a corporate priority that is integrated into all aspects of the organization which includes risk management, efficiency in development, design, and operation of facilities and that is implemented on a basis of continual improvement.

PRECAUTIONARY PRINCIPLE

Pretivm will use appropriate and cost-effective actions in all aspects of the Project to prevent serious or irreversible damage. The lack of full scientific certainty regarding the probability of such effects occurring will not be used as a reason for postponing such mitigation.

TRADITIONAL KNOWLEDGE

Pretivm respects the traditional knowledge of the Aboriginal peoples who have historically occupied or used the Project area. Pretivm recognizes that it has significant opportunity to learn from people who may have generations of accumulated experience regarding the character of the plants and animals, and the spiritual significance of the area. Traditional knowledge will guide aspects of the Project, including any future changes once the mine is approved.

Pretivm is committed to a process that invites and considers input from people with traditional knowledge of the Project area towards the design and EA of the Project. Pretivm is striving to establish a cooperative working relationship with all relevant Treaty and First Nations and Metis people to ensure opportunities to gather traditional knowledge.

Ecosystem Integrity

The Project area ecosystem is relatively undisturbed by human activities, although it is not static. Glacier retreat and relatively recent volcanic activity (within the last 10,000 years) along with landslides, debris flows, and snow avalanches, continue to modify the landscape. Pretivm's objective is to retain the current ecosystem integrity as much as possible during construction, operation, and closure of the Project. This objective will be met by:

1. avoiding adverse impacts, where feasible
2. mitigating unavoidable adverse impacts
3. compensating for adverse impacts that cannot be mitigated.

Upon closure and reclamation of the Project, the intent will be to return the disturbed areas to a level of productivity equal to or better than that which existed prior to project development, and for the end configuration to be consistent with pre-existing ecosystems to the extent possible.

Biodiversity and Protected Species

Pretivm is committed to making every reasonable effort toward maintaining biodiversity in the Project area. Biodiversity is defined by the BC Ministry of Forests, Lands and Natural Resource Operations as, "the diversity of plants, animals, and other living organisms in all their forms and levels of organization, and includes the diversity of genes, species, and ecosystems, as well as the evolutionary and functional processes that link them" (BC Ministry of Forests, Lands and Natural Resource Operations 1995).

Maintenance of biodiversity is not an isolated effort but an integral part of project planning (mitigation and monitoring), environmental effects analyses, and achievement of sustainability goals. This approach will be implemented throughout project development and the EA process.

20.1.2 Consultation

Pretivm recognizes the importance of carrying out consultation and will meet all regulatory requirements.

Consultation Policy Requirements

Both the BC *Environmental Assessment Act* (BCEAA) and the *Canadian Environmental Assessment Act* (CEAA) 2012 contain provisions for consultation with Nisga'a Nation, First Nations, and the public as a component of the EA process. All engagement and

consultation measures will comply with federal and provincial regulations, best practices, and internal company policies.

CONSULTATION PROGRAM

Community engagement and consultation are fundamental to the success of the proposed Project. Since 2011, consultation has been ongoing with the Nisga'a Nation, Tahltan Nation, Skii km Lax Ha, as well as other First Nations, and will continue to take place. Pretivm will participate in all BC Environmental Assessment Office (EAO) technical working group meetings, which are comprised of government agencies, First Nations, and Nisga'a Nation. Pretivm has and will continue to undertake engagement and consultation activities with government agencies (provincial, federal, and local), First Nations, and the Nisga'a Nation, as well as the public and other interested parties during the EA process as well as each phase of the Project lifecycle. As part of the BC EA process, consultation plans for both Nisga'a Nation and First Nations will be defined and developed for the EA pre-application and post-application periods. Aboriginal, public, and government consultation efforts will include a variety of engagement methods including private meetings, community meetings and open houses, information distribution activities (i.e. communications and outreach materials), and site tours.

Since 2011, Pretivm has initiated project and company introductions, a series of follow-up meetings, and regularly disseminated relevant information with the potentially affected Nisga'a nation and First Nation groups. Engagement, information sharing, and consultation will continue during the planning and regulatory review as well as the construction, operations, and closure phases. A consultation record has been developed, which is being maintained and reviewed to enable and strengthen ongoing relationship building and issue tracking.

CONSULTATION GROUPS

Nisga'a Nation and First Nations

Pretivm has established a relationship and will continue to engage and consult with Nisga'a Nation and First Nations, as identified by the provincial government's Section 11 Order as well as the Metis Nation of BC as indicated in the federal government's environmental impact statement (EIS) guidelines. Pretivm has also provided opportunities for First Nations employment during exploration and environmental baseline studies through Tsetsaut Ventures Ltd., a First Nations-owned contracting company. Ongoing consultation efforts will aim to engage both the leadership and community membership and attempt to resolve potential issues and concerns as they arise. No substantive issues have been raised to date regarding the Project.

Government

Pretivm will engage and collaborate with the federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as: land and resource management, protected areas, official community plans (OCPs), environmental and social baseline studies, and effects assessments. Pretivm will also

engage with the Nisga'a Lisims Government as noted above regarding consultation with the Nisga'a Nation.

Public and Stakeholders

Pretivm will consult with the public and relevant stakeholder groups, including land tenure holders, businesses, economic development organizations, businesses and contractors (e.g. suppliers and service providers), and special interest groups (e.g. environmental, labour, social, health, and recreation groups), as appropriate.

20.1.3 ENVIRONMENTAL SETTING

INTRODUCTION

The Property is situated within the Sulphurets District in the Iskut River region. The Property is located in the Boundary Range of the Coast Mountain physiographic belt along the western margin of the Intermontane tectonic belt. The climate is typical of northwestern BC, with cool, wet summers and relatively moderate but wet winters. The optimum field season is from late June to mid-October.

The widely varying terrain hosts a broad range of ecosystems. Its rivers are home to all five species of Pacific salmon, as well as trout and Dolly Varden char. Black and grizzly bears frequent the forests; moose and migratory birds can be found in the wetlands. Mountain goats are common in the alpine areas. The tree line is at approximately 1,200 masl. The Project is centred on the VOK Zone deposit, which is located southwest of Brucejack Lake at 1,400 masl.

Pretivm has undertaken baseline studies of the regional project area's atmosphere/climate, surface hydrology, aquatics, water and sediment, limnology, and fish habitat. Pretivm will also initiate comprehensive baseline studies of rock geochemistry, soils, vegetation, and wildlife to characterize the local and regional ecosystem prior to major disturbances. Archaeology, heritage, land use, cultural, Traditional Knowledge, and socioeconomic baseline studies have also been carried out to characterize the regional human environment. The methodologies for the baseline studies have been developed based upon standard procedures recommended by government agencies and professional experience.

CLIMATE

The climate of the Iskut region is relatively extreme and daily weather patterns are unpredictable. Prolonged clear sunny days can prevail during the summers. Precipitation in the region is approximately 1,600 to 2,100 mm annually. The majority of precipitation falls during the autumn and winter months, from October to April. Estimates show that Brucejack Lake receives approximately 70% of its annual precipitation on average during this period. The months of October through to January typically have the highest monthly precipitation amounts, while late spring or early summer months are typically much drier. Snowpack typically ranges from 1 to 2 m deep, but high winds can

create snowdrifts up to 15 m deep. Permanent icefields are present in the upper reaches of the Brucejack Lake watershed.

A full meteorological station was established near the Brucejack Lake camp in mid-October 2009 to collect site-specific weather data. The station measures wind speed and direction, air temperature and pressure, rainfall, snowfall, relative humidity, solar radiation, net radiation, and snow depth.

Table 20.1 presents the estimated average monthly climate data for the Project site (BGC 2013). Precipitation data reported at the Unuk River Eskay Creek (#1078L3D) Meteorological Service of Canada (MSC) climate station were used to characterize the Project site. Data from this station are available for the period of September 1989 to September 2010. The MSC station is located at an elevation of 887 m, approximately 30 km north of Brucejack Lake. Precipitation at the Project site is currently estimated to be similar to that observed at the Unuk River Eskay Creek station given their close proximity (30 km), similar basin physiography, and correlation of coincident monthly rainfall data. Using the MSC station as a proxy, the estimated average annual precipitation at Brucejack Lake is 2,034 mm.

This annual value compares favourably with the Climate BC dataset and Environment Canada (2012). Mean annual precipitation generated by the Climate BC climate data model, using the 1981 to 2009 climate normals (Wang 2012), exceeds 2,000 mm at an elevation of 1,400 masl, while Environment Canada (2012) estimated an average annual precipitation of 2,100 mm at an elevation of 1,400 masl. Climate design estimates by Environment Canada were developed by interpolating from calculated values at surrounding locations.

Table 20.1 Average Monthly Climate Data for the Project Site

Month	Average Temperature (°C)	Average Precipitation (mm)	Average Evaporation/ Sublimation (mm)
January	-10.1	249	2
February	-8.0	214	2
March	-6.1	181	2
April	-1.5	97	4
May	1.7	88	10
June	4.4	67	28
July	6.9	83	61
August	7.5	139	51
September	4.6	207	31
October	-1.3	247	5
November	-6.2	215	2
December	-8.2	247	2
Average/Total	**-1.2**	**2,034**	**200**

Source: BGC (2013).

Average monthly temperature data used at the Project site are based on scaling the Unuk River Eskay Creek data to an elevation of 1,400 m assuming an adiabatic lapse rate of -0.4°C per 100 m. These estimated temperature values compare reasonably well with site data collected since 2009.

Annual evaporation at the site was estimated using local climate data from the on-site climate station for the period 2010 to 2012, and Reference Evapotranspiration (REF-ET) calculation software (Version 3.1.14). Climate inputs required for the model include air temperature, wind speed, incoming solar radiation (or sunshine hours), relative humidity, dew point temperature, and atmospheric pressure. Monthly evaporation and sublimation totals are summarized in Table 20.1.

Terrain and Soils

The Project is located in a rugged area with elevations ranging from approximately 500 m at the lower elevations, along the access road and transmission line corridor, to 1,400 m at the mine site. Surrounding peaks are up to 2,200 m in elevation. Glaciers and icefields surround the mineral deposits to the north, south, and east.

Recent and rapid deglaciation has resulted in over-steepened and unstable slopes in many areas. Recently deglaciated areas typically have limited soil development, consisting of glacial till and colluvium. Lower elevation areas with mature vegetation may have a well-developed organic soil layer. Avalanche chutes are common throughout the area, and management of avalanches will be a concern for the development and operation of the Project. Avalanche hazards are being actively managed by on-site professional avalanche forecasters; operational avalanche planning is described in Section 18.6. Similarly, the potential for debris flows in some areas should be considered in the Project design.

Ecosystems and Vegetation

The proposed mine site is situated above the treeline and contains alpine ecosystems, as well as an abundance of unvegetated and sparsely vegetated terrain. Alpine ecosystems, including tundra, heather, and fellfield classes, are common around the mine site. The access road travels through old valley bottom forests, subalpine stands of subalpine fir and Engelmann spruce, and along dry glaciofluvial terraces supporting early seral pioneer ecosystems. The proposed transmission line from Long Lake substation is situated in both mature forest and recently deglaciated terrain, dominated by scoured rock, eroding moraine, and glaciofluvial deposits.

Wetlands

Wetland ecosystems are distributed throughout the local study area, though they are limited in extent in the Project area. Wetlands are valued ecosystem components. Wetlands are conserved and managed through federal initiatives, such as the Federal Policy on Wetland Conservation. Baseline studies were conducted in 2012 to map and classify wetlands, and to identify the primary wetland functions. These baseline data will

allow for the identification of areas where project modification may limit negative impacts.

WILDLIFE

The region encompassing the proposed Project is home to many terrestrial and aquatic wildlife species including black and grizzly bears, mountain goats, moose, bats, furbearers, small mammals, birds of prey, migratory songbirds, waterfowl, and herptiles. These include several species at risk as well as species of substantial cultural and economic importance. Pretivm will evaluate the potential impacts on representative species that are identified as being at risk or of concern within the area. Wildlife baseline studies have been conducted for the Project in 2010, 2011, and 2012. Species at risk that were encountered during baseline studies include wolverine, fisher, grizzly bear, western toad, barn swallow, rusty blackbird, olive sided fly catcher, and little brown myotis.

Species of concern include those that may not be of conservation concern but are of regional importance for other reasons identified in the Cassiar Iskut-Stikine LRMP such as moose, mountain goat, black bear, American marten, harlequin duck, and trumpeter swan. Grizzly bears and black bears have been observed close to the Project study area and will use a wide range of forage producing habitats. Moose are known to occupy and be harvested along low elevation areas associated with the larger river valleys. Mountain goats occur throughout the mountainous terrain of the Project area.

Bats, including the at-risk little brown myotis, are known to use suitable habitat in the Project area. A range of birds, including waterbirds, terrestrial breeding birds, and raptors, occupy a wide range of niches from high alpine areas to valley bottoms. Finally, the low elevation wetlands support herptiles including the at-risk western toad. Documenting numbers and distribution for representative species or groups have included a range of survey methods adhering to appropriate provincial standards, including those used for aerial surveys. A selection of key habitats for important species are currently being mapped using provincial standards to inventory sensitive areas across the Project area in support of an EA for the Project.

FISHERIES

The Unuk River is a large river system that provides important habitat for the five species of Pacific salmon, as well as habitat for resident trout (cutthroat, rainbow), and resident Dolly Varden. The Bell-Irving River system provides habitat for:

- sockeye, coho, and chinook salmon
- resident and anadromous trout (rainbow and steelhead)
- resident char (Dolly Varden and bull trout)
- mountain whitefish
- coarse fish species.

The fisheries resources and fish habitat of potentially affected rivers and their tributaries were assessed from 2010 to 2012 as part of the baseline program. Results from two seasons of sampling have shown that fish do not occur in Brucejack Lake. Fish are also absent downstream of Brucejack Lake in all waterbodies, including Sulphurets Creek, upstream of a barrier located 300 m upstream of the confluence of Sulphurets Creek and the Unuk River. Additional barriers have been identified in many other streams in the area, including all but the Bowser River crossing on the proposed transmission line route.

Visual Quality and Aesthetics

The Project is located in a relatively remote and undisturbed area; the area is characterized by rugged mountains, glaciers, forests, and rivers. The nearest major road is Highway 37. The controlled-access Eskay Creek Mine Road terminates approximately 25 km north of the proposed adit and mill. The Granduc Access Road is used by mineral exploration traffic and tourists during the summer accessing the viewpoint to the Salmon Glacier. The Granduc Access Road terminates at the Granduc staging area, approximately 20 km south of the Project site. The mine will be located in an isolated area that is not visible from either the Eskay Creek Mine Road or Highway 37. The southern portion of the transmission line will roughly parallel the Granduc Access Road, but will not interfere with viewscapes of the Salmon Glacier from the currently used viewpoint of the glacier along the Granduc Access Road.

Environmental Management Systems

Pretivm will develop and implement a comprehensive Environmental Management System (EMS) for the construction, operation, and closure phases of the Project. The EMS will comprise a series of written plans outlining the scope of environmental management to ensure compliance with both regulatory requirements and Pretivm's environmental policy.

Environmental management and mitigation measures will be provided for each of the following areas:

- air emissions and fugitive dust
- water management
- tailings and waste rock
- diesel and tailings pipelines
- concentrate load out
- ARD/ML containment
- materials management
- erosion control and sediment
- spill contingency and emergency response
- fish and fish habitat

- wildlife management
- waste management
- archaeological and heritage site protection.

Two employees, on a rotational basis, will be required for environmental monitoring including:

- federal MMER monitoring requirements
- permit and license compliance monitoring
- environmental effects monitoring
- reclamation research and monitoring.

Pretivm's environmental staff, supported by specialist consultants, will also research and advise the Mine Manager on alternative mitigation strategies as part of the mine's process of continual improvement. Outside laboratories will be required for some analyses while more routine analyses will be done in-house, such as conventional water sample analysis. Resources will be required for ongoing equipment upgrades and replacement, specialized equipment procurement, helicopter support, and mitigation and reclamation research.

20.1.4 Acid Rock Drainage/Metal Leaching

BGC completed an assessment of the ARD/ML characteristics of waste materials (waste rock and tailings) that will be generated at the Property. This assessment was required to determine if waste materials will generate ARD and cause ML, so that appropriate waste management strategies can be developed for the site. The assessment included:

- a review of drill core logs to select waste rock samples
- static tests (acid base accounting (ABA) data and elemental analysis) on the collected samples
- kinetic tests (humidity cells and subaqueous columns) on the collected samples
- analysis and reporting of the test results.

The following sections provide a brief summary of the key components to the ARD/ML assessment. BGC (2013) describes an in-depth discussion of the Brucejack ARD/ML testing program.

Acid Base Accounting

ABA data are used to assess the neutralization potential ratio (NPR) of waste materials in order to determine which materials are classified as PAG. Static tests were carried out in 2011 (using 150 samples) and 2012 (using 241 samples) on material collected from drill cores across the Project site. In addition, ABA tests were conducted on flotation tailings samples generated from the VOK Zone and West Zone ore. A frequency analysis

was conducted on the NPR values of the waste rock samples, which indicated that approximately 77% all of the samples analyzed are PAG material (an NPR value of less than two). This implies that a high percentage of waste rock material generated at the mine site will likely be PAG. The flotation tailings are considered non-PAG material (an NPR value of more than two).

Percentage of PAG Materials in Units of Pretivm's Geological Model

The geological model developed by Pretivm consists of seven units. Each of these units is characterized by several lithologies. By comparing the location of drillholes with the collected samples relative to the spatial extent of geological model units, the samples collected for testing could be assigned to the different units of the geological model. The frequency distribution of the NPR values associated with each geological unit was tested and the results of the statistical analysis are summarized in Table 20.2.

Except for the Office P1 unit, the percentage of PAG material from each unit exceeded 70%. Samples representing the P2 unit were almost entirely PAG material (97%). The Office P1 unit had the lowest percentage of PAG material among its waste rock samples (25%) compared to all other units of the geological model.

Table 20.2 NPR Characterization of Geological Model Units and their Spatial Extent Underground

	Bridge P1	Conglomerate	Fragmentation	Office P1	P2	Silicified Cap	Volcanic Sedimentary Facies
NPR	-	-	-	-	-	-	-
Mode	0.51	0.03	0.02	1.23	0.67	0.02	0.13
Median	1.07	0.64	0.37	4.56	0.82	0.34	0.67
Mean	1.5	3.2	1.6	8.8	0.9	1.3	1.5
Standard Deviation	1.6	15.2	7.0	14.4	0.4	4.7	3.1
PAG (%)	75	72.5	82	25	97	80	78
Underground Working							
Crusher	-	-	-	-	-	-	-
Volume (%)		0.12	23.75	39.01			37.12
West Zone	-	-	-	-	-	-	-
Volume (%)	-	17.47	35.74	10.35	7.58	0.01	28.86
VOK Zone	-	-	-	-	-	-	-
Volume (%)	14.76	8.97	22.12	4.02		1.00	49.14

Table 20.2 also provides rough estimates of volume percentages of the underground workings (Crusher, VOK Zone, and West Zone areas) occupied by the various geological model units. The data indicate the bulk of the waste rock generated from the underground workings will be from the Fragmental, Office P1, and Volcanic Sedimentary Facies units. Except for waste rock from the Office P1 unit, waste rock from all other units of the geological model can be considered PAG. Segregation of PAG and non-PAG materials in each unit will be very difficult because the distinction of the geological model

units is not exclusively based on lithology. Therefore, all units of the geological model are considered to be entirely PAG, except unit Office P1.

KINETIC TESTING

Kinetic tests are used to predict the long-term weathering characteristics of mine waste under accerlerated weathering conditions (simulated with humidity cells) and under subaqueous conditions (simulated with custom column experiments).

In total, 21 humidity cells (19 for waste rock and 2 for tailings solids) and 5 subaqueous columns (2 for waste rock and 3 for tailings) were constructed. The results from the kinetic tests indicate that, under optimal weathering conditions, some waste rock material at the site has the potential to generate ARD within one year; however, the onset to ARD for the majority of the waste rock will take tens of years before the available acid neutralization potential in the material is consumed. At lower pH values (e.g. a pH value of less than 5) the mobility of most metals (copper, zinc, iron) should increase. Concentrations of most dissolved metals should remain low under a circumneutral pH. The weathering of waste stored beneath a water cap (subaqueous columns) is expected to be significantly less due to the overlying water column. Waste material deposited subaqueously is generally characterized by a neutral pH and low dissolved metal concentrations, except dissolved arsenic. The tailings solids are not expected to generate ARD.

SUMMARY

The preceding discussion of the ARD/ML assessment indicates that site waste rock is primarily PAG material and, although ARD and ML processes can be significant under optimal weathering conditions, these processes are reduced in a subaqueous environment. The selected management strategies for the Project mine waste should prevent the potential for ARD, thereby reducing the potential for ML at the site. This includes the subaqueous disposal of waste rock in Brucejack Lake and storage of waste rock material in the underground workings. Although ML can be minimized by appropriate management strategies, its impact on the downstream receiving environment has been evaluated by water quality predictions outlined in Section 20.1.5.

20.1.5 WATER QUALITY

BGC developed a site-specific water quality model for the Project using GoldSim simulation software, an environmental and engineering statistical program. The water quality model was used to predict the water quality of Brucejack Lake and the downstream receiving environment during mine operations. The predictions were calibrated and validated against baseline data and compared to regulation standards and other criteria developed for similar mine discharge waters.

The predictions represent the probable observed water qualities during mining operations and were calculated based on information provided to BGC at the time of modelling and reporting. Where possible, BGC (2013) used conservative assumptions to estimate dissolved parameters in an attempt to account for data gaps. However, as with

all modelling exercises, the accuracy of the predictions was limited by the information available. The model will continue to be updated as new information becomes available and updated water quality predictions will be provided in the EA report for the Project.

Lorax Environmental Services Ltd. is currently developing a hydrodynamic model to assess the stability of the water column. The results of this model will be used to predict total metal and TSS concentrations. As a result, total metal concentrations and TSS estimates have not been evaluated at this time.

Predicted concentrations were compared to the MMER, the Canadian Council of Ministers of the Environment's (CCME) Canadian Water Quality Guidelines for the Protection of Aquatic Life, and the BC Ministry of Environment (MOE) Contaminated Site Regulation (CSR) Standard for the Protection of Freshwater Aquatic Life. Brucejack Lake and the immediate downstream environment are not considered fish habitat (Price 2005). Guidelines for aquatic life are likely considered stringent; however, site-specific guidelines have not yet been developed for the Project. Dissolved parameters that could exceed guidelines at the outlet of Brucejack Lake are outlined in Table 20.3.

Table 20.3 Summary of Predicted Potential Exceedances of Water Quality Guidelines for the Project

	CCME		BC CSR		MMER	
Parameter	Minimum Guideline (mg/L)	Potential Exceedance	Minimum Guideline (mg/L)	Potential Exceedance	Minimum Guideline (mg/L)	Potential Exceedance
Dissolved Aluminum	0.005	X	-	-	-	-
Dissolved Arsenic	0.005	X	0.05	-	0.5	-
Dissolved Cadmium	0.000017	X	0.0001	X	-	-
Dissolved Copper	0.002	X	0.02	-	0.3	-
Dissolved Iron	0.3	X	-	-	-	-
Dissolved Lead	0.001	X	0.04	-	0.2	-
Dissolved Mercury	0.000026	X	0.001	-	-	-
Dissolved Selenium	0.001	X	0.01	-	-	-
Dissolved Silver	0.0001	X	0.0005	-	-	-
Dissolved Zinc	0.03	X	0.075	-	0.5	-

Note: "X" indicates parameter exceed lower limit of guidelines. If a guideline is provided as a range, the results are compared against the minimum value of that range. Dissolved metal concentrations are compared to total concentrations if no guideline for totals exists.

All of the parameters listed in Table 20.3 have previously exceeded guidelines at some point during baseline sampling events in mine seepage water. Water quality predictions will need to be re-evaluated once additional information (such as an updated water balance and mine plan) is incorporated into the water quality model and site-specific guidelines are established. For additional details on the water quality model development and methodology, please refer to BGC's report entitled "Draft Brucejack Project Feasibility Study – Geochemistry Report" and dated June 2013.

20.1.6 SOCIAL SETTING

SOCIO-ECONOMIC SETTING

Northwest BC is a sparsely populated and relatively undeveloped region of the province. Many of the smaller communities have predominantly Aboriginal populations that are isolated from one another as well as from the main regional centres of Smithers and Terrace. Approximately one-third of the 40,000 to 45,000 people in the region are Aboriginal, which is a far higher proportion than for the province as a whole.

Primary resource industries, principally mining and forestry, are the mainstay of the economy. The forest industry in particular has been in decline in recent decades, which has significantly weakened the economy and lead to a steady decline in the regional population. Since the mid-1990s, the regional population has dropped almost 15% although, in recent years, the rate of decline has begun to slow.

Transportation and communication is limited; the region is intersected by Highway 37 (north to south) and Highway 16 (east to west).

Strong commodity prices and the global boom in mining have led to increased mineral exploration activity. The mining industry is widely expected to represent an increasingly important source of investment and employment. Communities in the region are accustomed to cycles of "boom and bust" associated with mining. Nevertheless, extractive industries and related energy projects are expected to continue to form the basis of the regional economy.

Community and socio-economic impacts of the Project can potentially be very favourable for the region, as new long-term opportunities are created for local and regional workers. Such opportunities could reduce and possibly reverse the out-migration to larger centres. Pretivm will continue to work with Treaty Nation and First Nations groups and members of local communities to maximize benefits through employment and business opportunities, training, and skills development programs.

Socio-economic baseline studies were carried out through much of 2012 and Q1 2013. The studies have covered a regional study area that encompasses the Regional District of Kitimat-Stikine including all communities from Terrace to the north as far as Dease Lake, and from the Town of Smithers in the east to the Port of Stewart in the west.

The following sections on the Highway 16 and Highway 37 corridors are compiled from the Northwest BC Mining Projects Socioeconomic Impact Assessment, prepared in 2005 for the Ministry of Small Business and Economic Development, and updated using data from the 2006 Census of Canada.

Highway 16 Corridor

Highway 16 extends from the Prince Rupert port eastwards to Terrace, Hazelton, Smithers, and Prince George. The Canadian National Railway also follows this corridor. Most of the communities along this corridor are discussed in this section. The Highway

16 corridor is recovering from the economic downturn of the 1990s, and has excess capacity with respect to social service infrastructure. The respective communities are incorporated providing a framework and capacity to plan for, finance, and deliver services that might be required, and meet incremental growth from new mine developments.

Highway 37 Corridor

Highway 37 connects with Highway 16 at Kitwanga and runs northwards to the Yukon border. Highway 37A branches off from Highway 37 at Meziadin and connects to the Port of Stewart. Highway 37 communities include Iskut, Dease Lake, and Good Hope Lake.

With the exception of Stewart, the majority of the population belongs to First Nations (e.g. Good Hope Lake). These communities rely heavily on public sector and mining employment. Since 1996, Highway 37 communities have experienced an overall decline in population. Stewart is located 60 km west of Meziadin Junction on the west coast of BC, at the head of the 145 km-long Portland Canal and the terminus of Highway 37A. The Stewart Bulk Terminals are used by the mining and forestry industries to ship products from northern BC and Yukon to international destinations. Much of the town of Stewart was built for the development of the Granduc Mine. The town's population has fallen dramatically in the past 20 years, coinciding with the closure of the Granduc and Premier mines.

Traditional Knowledge and Traditional Use

The Project site is located on Crown land in an area historically used by several Aboriginal groups. A desk-based ethnographic overview for the potentially affected First Nations and Treaty Nations was implemented between May 2012 and March 2013. In addition, a Traditional Knowledge/Traditional Use (TK/TU) study was completed for the Skii km Lax Ha and will also be pursued for the Tahltan Nation. These studies will identify areas and seasons where Aboriginal groups have engaged in traditional interests and activities including hunting, fishing, gathering, and spiritual activities. The outcomes of these studies will be used to inform the overall design and operation of the Project.

Non-aboriginal Land Use

The western part of the Project area is included in the Cassiar Iskut-Stikine LRMP, which was approved by the province in 2000. The LRMP is a sub-regional integrated resource plan that establishes the framework for land use and resource management objectives and strategies, and provides a basis for more detailed management planning. The LRMP outlines the management direction, research and inventory priorities, and economic strategies for the Cassiar Iskut-Stikine area, and presents an implementation and monitoring plan to reach the established objectives. Detailed planning initiatives and resulting outcomes are expected to be guided by, and be consistent with, the LRMP management direction. Part of the Project area lies within the boundaries of the South Nass Sustainable Resource Management Plan (SRMP) area, finalized in June 2012. The SRMP is a landscape-level plan that addresses the sustainable management of land, water, and resources while considering economic interests.

The Project area has been the focus of mineral exploration for many years. There are indications that prospectors explored the area for placer gold in the late 1800s and early 1900s. Placer gold production has been reported from Sulphurets Creek in the 1930s, and a large log cabin near the confluence of Mitchell and Sulphurets Creeks was reportedly used by placer miners until the late 1960s. The whole region surrounding the Project is heavily staked and several other mining companies have active exploration programs nearby.

The Kerr and Sulphurets deposits have been extensively explored on an intermittent basis since the 1960s. Intensive underground exploration adjacent to Brucejack Lake in the 1980s was supported by an exploration road from Bowser Lake over Knipple Glacier. Results of the 2012 non-traditional land use baseline research program indicate that a limited number of people access the area. Those who do access the area include trappers, guide outfitters, resident and non-resident hunters, and those who participate in commercial recreation activities such as heli-skiing, guided freshwater recreation, and guided mountaineering. Activities in the area of the transmission line are generally similar to those in the Project area. Other individuals with interests in the area include those who hold forestry licences, mineral claims, and placer claims, which are typically linked to resource development and industry, as well as water licences, which may be linked to commercial recreation businesses. Overall, land use the Project area is minimal and seasonal in natural.

Archaeology and Heritage Resources

Archaeological assessments were conducted around the mine site and along the access and transmission corridors. Two small prehistoric archaeological sites were identified in proximity (within 1 km) to planned infrastructure but are located outside all currently planned areas of disturbance.

Social and Community Management Systems

Pretivm will develop and implement broad Social and Community Management Systems (SCMS) for the construction, operation, and closure phases of the Project. The SCMS will comprise an ongoing engagement plan and Impact Benefit Agreements (IBAs) to be developed through a series of written agreements and relationship-building initiatives with First Nations. Monitoring and oversight of the SCMS will require a team of staff responsible for coordinating community development initiatives, training, communications and commitment tracking, and fund management.

Social, community management, and relationship-building measures will be provided for each of the following areas:

- IBAs
- community engagement meetings
- training
- participation in community events

- reporting and feedback mechanisms.

20.1.7 Water Management

General

Water management will be a critical component of the Project design in this high runoff environment. The most likely avenue for transport of contaminants into the natural environment will be through surface or groundwater. As such, through its consultants, Pretivm has developed a water management plan that applies to all mining activities undertaken during all phases of the Project. The goals of this management plan will be to:

- provide a basis for management of freshwater on site, especially with the changes to flow pathways and drainage areas
- protect ecologically sensitive sites and resources, and avoid harmful impacts on fish and wildlife habitat
- provide and retain water for mine operations
- define required environmental control structures
- manage water to ensure that any discharges are in compliance with the applicable water quality levels and guidelines.

Strategies for water management include:

- protecting disturbed areas from water erosion, and collecting surface water from disturbed areas and treating it to meet discharge standards prior to release
- minimizing the use of fresh water through recycling of water whenever possible
- monitoring the composition of release water and treating it to remove or control contaminants as required to meet discharge standards
- constructing diversion channels to direct undisturbed runoff away from mining activities.

Water Management Overview

The Brucejack deposit, located west of Brucejack Lake on the east side of the Sulphurets Valley, is proposed to have an underground mine and associated facilities. The underground facility will be mined over a 22-year period. As mining progresses, 2.4 Mt of waste rock that was excavated from the underground mine prior to operations and approximately 2.2 Mt of blasted rock from plant site excavations during construction will be deposited into Brucejack Lake, along with 9.5 Mt of flotation tailings during operations. An additional 8.6 Mt of tailings paste backfill and 2.0 Mt of waste rock will be deposited in the underground mine during operations. Of the total processed mineralized material, 8% (approximately 1.6 Mt) will be trucked to an off-site facility as concentrate for secondary processing.

Contact Water

There are three expected sources of contact water during construction and operations:

- the upper laydown area where the waste rock transfer and pre-production ore will be stored
- the mill building and portal site, which require an extensive cut into bedrock (some of which is currently assumed to be PAG)
- groundwater seepage to the underground mine tunnels.

Runoff from the former two sources will be managed by storage and treatment. Contact water ponds will be sized to contain runoff from the 24-hour, 10-year return period rainfall event (102 mm). The contained runoff will be pumped to the water treatment plant for treatment prior to release into Brucejack Lake. The water treatment plant has been designed with a maximum capacity of 7,440 m^3/d.

Groundwater seepage into the underground workings is expected to vary from approximately 2,140 to 4,080 m^3/d throughout the LOM. This water will initially be sent to the water treatment plant for treatment before being sent to the process plant, where its use will be maximized in process. Excess treated groundwater will be used as fluidizing water and discharged to Brucejack Lake at depth. As noted in Section 18.12., there will be a constant flow through the pipeline at all times to keep the subaqueous tailings deposit at the end of the outfall fluidized. When the thickened tailings are used in the backfill plant, flow will be maintained with water.

It is assumed that outflows from Brucejack Lake will be of suitable water quality for discharge to Brucejack Creek following water treatment.

Diversion Channels

Fresh water diversion channels will be constructed around the plant site. The channels will discharge directly to Brucejack Lake.

Process Water Requirements

The average water requirement for the Brucejack process plant is 3,134 m^3/d, based on a mill throughput of 2,700 t/d. This water is required for the tailings slurry to the lake, the underground paste backfill, the concentrate slurry, and minor evaporative losses within the plant (approximately 7 m^3/d). The process water will be sourced from:

- treated underground seepage water
- ore moisture (approximately 3% by weight)
- reclaim from the lake.

Reclaim from the lake is required because there are periods when the groundwater inflows are predicted to be less than the process requirement.

Approximately 47% of the tailings will be deposited underground as paste backfill, while 53% will be discharged to the bottom of Brucejack Lake at a maximum depth of 80 m. Additional details of the subaqueous deposition plan are provided in Section 18.12. Tailings will either be diverted to the paste backfill plant or diluted and sent to Brucejack Lake, but never concurrently. A constant flow is required through the pipeline at all times to keep the deposit at the end of the outfall fluidized; however, the tailings line to the lake will be operational less than 50% of the time. Therefore, when the thickened tailings are used in the backfill plant, flow will be maintained with fluidizing water, which will be sourced from excess underground seepage water and reclaim water from the surface of Brucejack Lake. The average fluidizing water requirement is 3,447 m^3/d.

WATER BALANCE MODEL

A water balance model (WBM) for the Brucejack site was constructed using a monthly time-step. The following assumptions were used as input to the WBM:

- a final tailings settled dry density of 1.6 t/m^3 for the lake deposition and of 1.46 t/m^3 for the underground mine deposition
- a solids specific gravity of 2.68 is assumed for tailings and 2.71 for paste backfill
- a nominal mill throughput of 2,700 t/d with:
  - 219 t/d (8.1% of total production) sent to an off-site facility as concentrate for secondary processing in a slurry of 88% solids by weight (30 m^3/d of slurry water)
  - 1,307 t/d (48.4% of total production) will be deposited at depth in Brucejack Lake in a slurry of 35% solids by weight (2,427 m^3/d of slurry water)
  - 1,245 t/d (43.5% of total production including 5 to 6% bonder) will be deposited in the underground mine in a backfill paste of 65% solids by weight (670 m^3/d of slurry water)
- an average mill loss of about 7 m^3/d
- an average annual precipitation of 2,034 mm and potential lake evaporation and sublimation losses of 200 mm
- annual average runoff of about 1,820 mm from undisturbed ground.

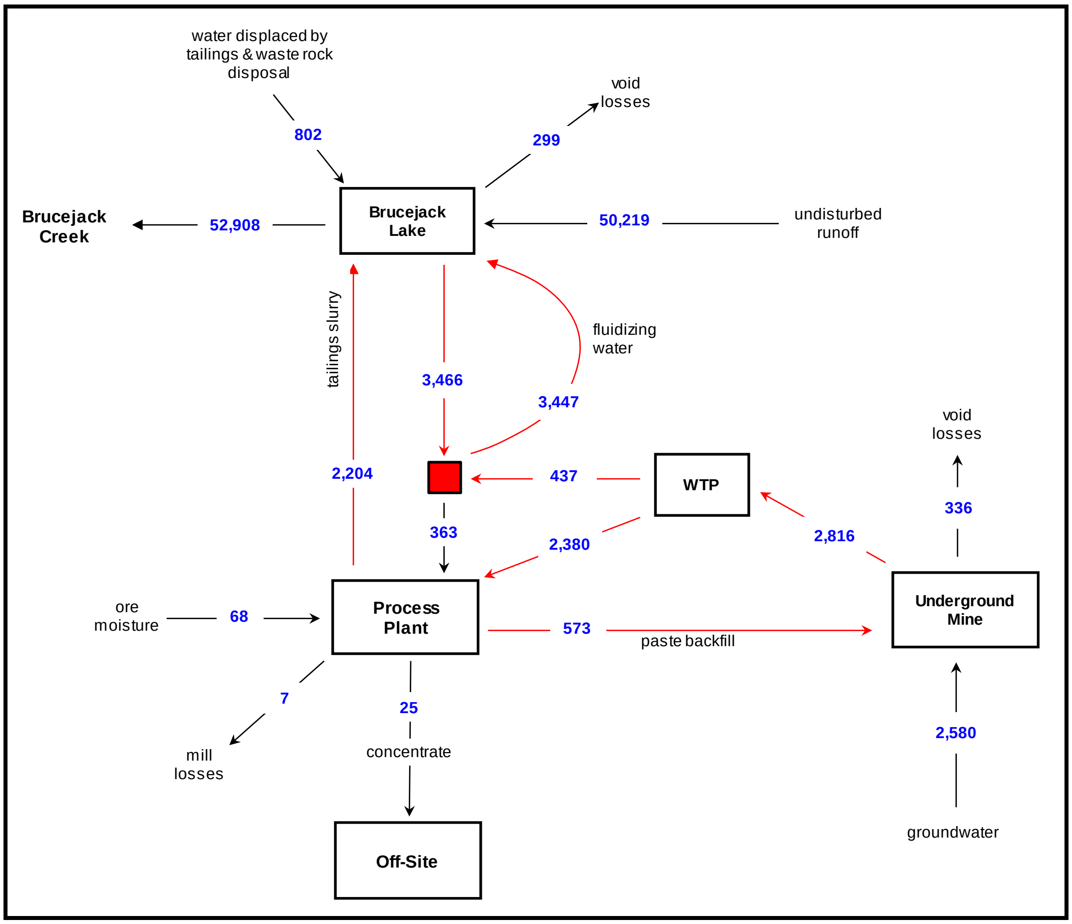
A water balance schematic for the mine during operations is shown in Figure 20.1. Values shown are average flows (m^3/d) over the LOM and account for the annual variations in ore production. The following items should also be noted in Figure 20.1:

- The model accounts for the displacement of lake water resulting from tailings and waste rock deposition.
- Numerical groundwater modelling of the site indicates that during mine operations, the natural groundwater flow pattern will be altered and a cone of depression will form around the underground workings, as seepage water is

pumped from the underground and used in process. In response, the baseflow inputs to Brucejack Lake will also be altered during this period. The undisturbed runoff value in the flow schematic accounts for these reduced baseflows.

- With a settled dry density of 1.46 t/m³ and a slurry consisting of 65% solids by weight, the paste backfill will exude some water during the curing phase. It is assumed that this additional water will be pumped out with the seepage water and sent to treatment.

Figure 20.1 Brucejack Lake Water Balance Model Schematic – Operations (Average Conditions)



Note: Units are m³/d. Red arrows indicate pumping routes.

An average annual outflow of 2,205 m³/h from Brucejack Lake has been estimated for the LOM, an average increase of approximately 5% above existing conditions (2,103 m³/h). The increase in flow results from the introduction of tailings slurry water and the displacement of water by the deposition of tailings and waste rock.

20.1.8 WASTE MANAGEMENT

Mine Wastes

Pretivm has initiated dialogue with the BC MOE and the BC MEMNG as well as Environment Canada at a federal level. All agencies' have had a favourable response to the mine waste management plans.

Mine wastes, including waste rock and tailings, will be backfilled underground and subaqueously deposited into Brucejack Lake to provide reducing conditions to prevent potential ARD development. This method was previously used to dispose of waste rock into Brucejack Lake in 1999, following underground development completed by Newhawk Gold Mines.

Brucejack Lake is approximately 85 m deep; tailings and waste rock will be stored within the bottom 45 m of the lake. When not being directed as paste backfill to the underground, tailings will be delivered to the lake via a pipeline from the process plant. Lake and water quality modeling studies are ongoing. Mitigation measures may be required to ensure compliance with discharge and receiving environment water quality criteria. Mitigation measures currently being considered to address potential elevated levels of TSS include installation of a water control structure, turbidity curtains, and washing of waste rock of fines before being placed in the lake.

Backfilling of tailings and waste rock will reduce the amount of waste materials placed in the lake and avoidance of sub-aerial deposition of PAG waste rock material will minimize the potential to generate ARD. This will also reduce the visual signs of the mine following closure, and eliminate the need for long-term waste rock stability monitoring. Long-term water quality monitoring is anticipated. These disposal methods are described in more detail in Section 18.0.

Non-hazardous Waste

Waste management will also involve the segregation of industrial and domestic waste into appropriate management streams. Project waste collection and disposal facilities will include one or more incinerators, a permitted landfill, waste collection areas for recyclable and hazardous waste, and sewage effluent and sludge disposal. Waste collection areas will have provisions to segregate waste according to disposal methods and facilities to address spills, fire, and wildlife attraction.

Hazardous Waste

Hazardous waste materials, such as spoiled reagents and used batteries, will be generated throughout the life of the Project, from construction to decommissioning. These materials will be anticipated in advance; they will be segregated, inventoried, and tracked in accordance with federal and provincial legislation and regulations, such as the federal *Transportation of Dangerous Goods Act* (1992). A separate secure storage area will be established with appropriate controls to manage spillage. Hazardous waste will be labelled and stored in appropriate containers for shipment to approved off-site disposal facilities.

20.1.9 AIR EMISSION CONTROL

Since most of the mining will be underground and most of the tailings will be stored subaqueously, air emissions will not represent a significant component of contaminant dispersion for the Project. Baseline studies, utilizing on-site meteorological stations and wind monitoring stations, have collected atmospheric data in the Project area to allow for air dispersion modelling, which is to be completed as part of the EA process. Mitigation procedures will then be developed to minimize adverse impacts from emissions. Regular monitoring of emissions will assess the success of the mitigation methods, and warn of any need to adjust the current approach.

Pretivm will implement an air emissions plan to ensure that the levels of air emissions generated by project activities are below the regulatory requirements of the Canada and BC Ambient Air Quality Objectives.

Adverse effects from air emissions and fugitive dust will be minimized through the implementation of mitigation measures such as:

- the use of clean, high-efficiency technologies for diesel mining equipment
- the use of appropriate emissions control equipment such as scrubbers
- the use of low-sulphur diesel fuel when practical
- the use of a vehicle fleet powered by diesel engines with low emissions of nitrous oxide and hydrocarbons (greenhouse gases)
- preventative maintenance to ensure optimum performance of light-duty vehicles, diesel mining equipment, and incinerators
- the implementation of a recycling program to reduce the amount of incinerated wastes, and hence CO_2 emissions
- the segregation of waste prior to incineration to minimize toxic air emissions.

The use of electricity as primary source of power during operations, as opposed to on-site generators, will also have a substantial impact on the reducing project-related air emissions.

20.1.10 CLOSURE PLAN AND COSTS

The Project will be closed at the end of mining. This will involve the removal of all structures and equipment, closure of the portals, and rehabilitation of site disturbances. The goal is to minimize the long-term effects on the environment and return the site to as close to its pre-disturbance condition, as practical.

Closure of the underground will include the removal of all loose material and mobile equipment such as ventilation fans and safety equipment. These will be removed from the site for reuse or will be recycled. Structures will be removed from the underground. Oil will be drained from all equipment and the oil disposed of in a regulated facility. All fuel will be removed from the underground storage and distribution system and removed

from the site. Fixed equipment, such as electrical cables and pipes, will be left underground.

The underground workings will be progressively backfilled with tailings and waste rock throughout mine operations and, once mining is completed, the underground will be allowed to flood. The ventilation shafts and underground portals will be sealed with concrete plugs. The plugs will be equipped with outlets in the event the water table rises in the underground workings and some seepage occurs, which is not expected in the two new mine portals but may occur in the existing portal. The seepage water from the existing portal will be monitored post-closure and directed to Brucejack Creek.

Closure of the above-ground facilities will include the removal of all buildings and structures on site, including the camp and mill. The buildings will be dismantled and the materials will be taken off-site for reuse or recycling. All oil, fuels, and processing fluids will be drained from equipment before the equipment is removed, and disposed of in a regulated facility off-site. The processing equipment will be removed from site and sold or recycled. Any concrete foundations will either be left in place or broken up and possibly used as backfill. All gravel surfaces will be ripped, such as the helicopter pad and the roads, to increase water infiltration and reduce the potential for surface erosion and instability. The above-ground pipes that carry tailings to the lake and the turbidity curtain will be removed and disposed of off-site.

The transmission line will be dismantled. The wooden poles will be left in the transmission line right-of-way to degrade naturally, and will be strategically located across the right-of-way to discourage vehicle access. The cables and transformers will be removed off-site and sold or recycled.

The access road will be decommissioned. The culverts will be removed and natural drainage will be restored. The bridges will be dismantled and the materials will be removed from site. Wood material may be burned under controlled conditions. Some wood will be strategically placed on the roadway to limit vehicle access. The road surface will be ripped to increase water infiltration and reduce the potential for surface runoff. Soils will be spread on the surface where soil is available and the areas will be re-vegetated. All other structures related to the mine will be removed and sold or recycled.

The cost to dismantle the above structures and to close the various facilities has been developed.

20.2 CERTIFICATION AND PERMIT REQUIREMENTS

Mining projects in BC are subject to regulation under federal and provincial legislation to protect workers and the environment. This section discusses the principal licences and permits required for the Project.

20.2.1 ENVIRONMENTAL ASSESSMENT PROCESS

Major mining projects in BC are subject to environmental assessment and review prior to certification and issuance of permits to authorize construction and operations. Environmental assessment is a means of ensuring the potential for adverse environmental, social, economic, health, and heritage effects or the potential adverse effects on Aboriginal interests or rights are addressed prior to project approval. Depending on the scope of a project, assessment and permitting of major mines in BC will proceed through the BC EA process pursuant to the BCEAA and the CEAA (2012).

At a provincial level, proposed mining developments that exceed a threshold criterion of 75,000 t/a (or 205 t/d) as specified in the Reviewable Project Regulations, are required under the BCEAA to obtain an Environmental Assessment Certificate from the MEMNG and MOE before the issuance of any permits to construct or operate. The Project will thus require a provincial Environmental Assessment Certificate, because its proposed production rate exceeds the specified threshold.

At a federal level, proposed gold mine developments (other than placer mines) that exceed a threshold criterion of 600 t/d as specified under the Regulations Designating Physical Activities, are required to complete an EIS pursuant to the CEAA (2012). Thus completion of an EIS will be necessary for the Project since the proposed production rate exceeds the specified threshold.

Pretivm has formally entered both the provincial and federal EA processes. While the provincial and federal decisions are made independently, the two levels of government work together to allow for a coordinated effects assessment process. In relation to the provincial EA process, Pretivm has submitted a Project Description and has received a Section 10 order under the BCEAA. Federally, Pretivm has submitted a Project Description and has received final EIS guidelines from the Canadian Environmental Assessment Agency. Pretivm is targeting completion of a combined application for an Environmental Assessment Certificate and EIS by the end of 2013. Provincial and federal decisions on the EA process are expected in Q4 2014. Provincial approval of the Environmental Assessment Certificate and federal approval of the EIS will then allow for the issuance of the necessary statutory permits and authorizations to commence construction of the Project.

20.2.2 REGULATORY REQUIREMENTS

Pretivm will design, construct, operate, and decommission the Project to meet all applicable BC and Canadian environmental and safety standards and practices. Some of the pertinent provincial and federal legislation that establish or enable these standards include:

- *Environment and Land Use Act* (BC)
- *Environmental Management Act* (BC)
- *Health Act* (BC)

- *Forest Act* (BC)
- *Forest and Range Practices Act* (BC)
- *Fisheries Act* (BC)
- *Land Act* (BC)
- *Mines Act* (BC)
- *Soil Conservation Act* (BC)
- *Water Act* (BC)
- *Wildlife Act* (BC)
- *Canadian Environmental Protection Act*
- Canada Transportation
- *Transportation of Dangerous Goods Act*
- WHMIS
- *Safety Act*.

Lists of the major federal and provincial licences, permits, and approvals that are required to construct, operate, decommission, and close the Project are summarized in the following sections. These lists cannot be considered comprehensive due to the complexity of government regulatory processes, which evolve over time, and the large number of minor permits, licences, approvals, consents, authorizations, and potential amendments that will be required throughout the LOM.

British Columbia Authorizations, Licences and Permits

Provincial permitting, licensing, and approval processes (statutory permit processes) may proceed concurrently with the BCEAA review or may, at the proponent's option, follow the Environmental Assessment Certificate. No statutory permit approvals may be issued before an Environmental Assessment Certificate is obtained. Pretivm will seek to pursue concurrent permitting for all permits, for which engineering information will be available at a sufficient level of detail with respect to the concurrent permitting process. Statutory permit approval processes are normally more specific than the EA level of review, and will require detailed and possibly final engineering design information for certain permits. Table 20.4 presents a list of provincial authorizations, licences, and permits required to develop the Project. The list includes only the major permits and is not intended to be comprehensive.

Table 20.4 List of BC Authorizations, Licences, and Permits Required to Develop the Brucejack Project

BC Government Permits and Licences	Enabling Legislation
Environmental Assessment Certificate	BCEAA
Permit Approving Work System and Reclamation Program (mine site – initial development and preproduction)	*Mines Act*
Reclamation Program (bonding)	*Mines Act*
Amendment to Permit Approving Work System and Reclamation Program (mine plan production)	*Mines Act*
Permit Approving Work System and Reclamation Program (gravel pit/wash plant/rock borrow pit)	*Mines Act*
Mining Lease – Mine Area and Mine Site Facilities	*Mineral Tenure Act*
Water Licence – Changes in and about a stream	*Water Act*
Water Licence – Storage and Diversion	*Water Act*
Water Licence – Use	*Water Act*
Licence to Cut – Transmission Line, Gravel Pits, Borrow Areas, Construction Laydown Areas	*Forest Act*
Licence of Occupation – Transmission Line	*Land Act*
Waste Management Permit – Effluent (tailings and sewage)	*Environmental Management Act*
Waste Management Permit – Air (crushers, concentrator, incinerators)	*Environmental Management Act*
Waste Management Permit – Refuse	*Environmental Management Act*
Camp Operation Permits (drinking water, sewage, disposal, sanitation and food handling)	*Drinking Water Protection Act/Health Act/Municipal Wastewater Act*
Special Waste Generator Permit (waste oil)	*Environmental Management Act* (Special Waste Regulations)

FEDERAL APPROVALS AND AUTHORIZATIONS

Applications for federal approvals can be completed concurrently with or following the EA process. Statutory permits and authorizations cannot be obtained until federal approval of the EIS. Table 20.5 lists some of the federal approvals that may be required. Notably, it is expected that neither fisheries authorizations nor a Schedule II amendment to the MMER will be required for the Project.

Table 20.5 List of Federal Approvals and Licences that May be Required to Develop the Brucejack Project

Federal Government Approvals and Licences	Enabling Legislation
CEAA Approval	CEAA 2012
Alteration of flow on international river	*International Rivers Improvement Act*
MMER	*Fisheries Act/Environment Canada*
Navigable Water: stream crossings authorization	*Navigable Waters Protection Act*
Navigable Water: sub-aqueous disposal of waste rock and tailings	*Navigable Waters Protection Act*
Explosives Factory Licence	*Explosives Act*
Ammonium Nitrate Storage Facilities	*Canada Transportation Act*
Radio Licences	*Radio Communication Act*
Radioisotope Licence (nuclear density gauges/x-ray Analyzer)	*Atomic Energy Control Act*

20.2.3 Financial Assurance

Pretivm will provide financial assurance, in the form of bonding, that the Project will be closed and reclaimed according to the reclamation and closure plan. Bonding or security amounts generally increase as disturbance and infrastructure increase such that, if the Project is stopped, there will be sufficient funds to close the Project according to the reclamation and closure plan.

The construction period for the Project will be between 18 and 24 months, at which stage, the portals will be installed as well as the mill and other structures. As no increase in disturbance above-ground and no additional infrastructure will be developed during the operations stage, the amount of the security amount should be close to the maximum amount by the end of the construction period. As there will be little opportunity to progressively close and reclaim the Project areas during the operations stage, the maximum security amount will be held by the MEMNG until the end of operations. All interest on the security amount will belong to Pretivm.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $663.5 million, and includes all direct costs, indirect costs, Owner's costs, and contingency. A summary breakdown of the initial capital cost is provided in Table 21.1

Table 21.1 Summary of Initial Capital Costs

Major Area	Area Description	Capital Cost ($ million)
Direct Costs		
11	Mine Site	32.7
21	Mine Underground	174.5
31	Mine Site Process	80.1
32	Mine Site Utilities	23.7
33	Mine Site Facilities	43.7
34	Mine Site Tailings	3.5
35	Mine Site Temporary Facilities	10.2
36	Mine Site (Surface) Mobile Equipment	14.3
84	Off Site Infrastructure	69.1
Subtotal Direct Costs		**451.8**
91	Indirect Costs	125.0
98	Owner's Costs	22.3
99	Contingencies	64.4
Total Initial Capital Costs		**663.5**

Note: Numbers may not add due to rounding.

21.1.1 PURPOSE AND CLASS OF ESTIMATE

PURPOSE

The purpose of this capital cost estimate is to advance the Project toward EPCM execution, commissioning, and operations.

CLASS OF ESTIMATE

This estimate is a Class 4 feasibility cost estimate prepared in accordance with the standards of the AACE. There was no deviation from the AACE's recommended practices in the preparation of this estimate.

21.1.2 ESTIMATE BASE DATE AND VALIDITY PERIOD

This estimate was prepared with a base date of Q2 2013 and does not include any escalation beyond this date. The quotations used for this feasibility study estimate were obtained in Q2 2013, and have a validity period of 90 days.

21.1.3 ESTIMATE APPROACH

CURRENCY AND FOREIGN EXCHANGE

The capital cost estimate uses Canadian dollars as the base currency. Foreign exchange rates, noted in Table 21.2, were applied as required. All costs presented in this section are in Canadian dollars unless otherwise stated.

Table 21.2 Foreign Exchange Rates

Base Currency	Foreign Currency
Cdn$1.00	US$1.00
Cdn$1.00	AUD$1.00
Cdn$1.00	€0.81

Most pricing was submitted to Tetra Tech in US dollars. All foreign currency quotations received from vendors were converted to Canadian dollars using the exchange rates listed in Table 21.2.

DUTIES AND TAXES

Duties and taxes are not included in the estimate.

MEASUREMENT SYSTEM

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

21.1.4 PROJECT EXECUTION SCHEDULE

The capital cost estimate is based on the following key milestone dates:

- Q3 2013: start of basic engineering
- Q1 2014: EPCM award
- Q3 2016: underground development completion
- Q3 2016: production start.

21.1.5 Responsibility Matrix

A team of engineers, procurement specialists, and cost estimators developed the following areas of the capital cost estimate:

- Tetra Tech – processing, infrastructure, process operating cost estimate, financial analysis, and overall preparation of the capital cost estimate
- AMC – mining including mine capital and operating cost estimates
- Rescan – environmental aspects including closure and tailings delivery system design
- BGC – tailings impoundment facility, waste rock and water management, and geotechnical design
- Valard – transmission line.

21.1.6 Work Breakdown Structure

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

- Level 1 = Major Area
- Level 2 = Area
- Level 3 = Sub-Area.

21.1.7 Elements of Cost

This capital cost estimate consists of the following four main parts.

Direct Costs

AACE defines direct costs as:

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

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

The total direct cost for the Project is estimated to be $451.8 million.

Indirect Costs

AACE defines indirect costs as:

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

The total indirect cost for the Project is estimated to be $125.0 million.

OWNER'S COSTS

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

The Project execution strategy involves an EPCM organization, supervising one or more general contractors. The allowance for Owner's costs was provided by Tetra Tech and confirmed by Pretivm. Some items included in the Owner's costs are home office staffing, home office travel, home office general expenses, field staffing, field travel, general field expenses, and Owner's contingency.

The total Owner's cost for the Project is estimated to be $22.3 million.

CONTINGENCY

When estimating costs for a project, there is always uncertainty as to the precise content of all items in the estimate, how work will be performed, what work conditions will be like when the project is executed, etc. These uncertainties are risks to a project. These risks are often referred to as "known-unknowns"; the estimator is aware of them and, based on experience, can estimate the probable costs. The estimated costs of the known-unknowns are referred to by cost estimators as "cost contingency."

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

The total allowance contingency for the Project is $64.4 million.

The overall contingency for the Project is 14.3% of the direct costs.

21.1.8 METHODOLOGY

This estimate was developed based largely on first principles. The work to complete the estimate can be broken down into three categories:

- design basis
- planning basis
- cost basis.

21.1.9 Design Basis

The following items were referenced during preparation of this estimate:

- equipment list and process flow sheets provided by Tetra Tech
- site layout drawings
- equipment data sheets
- quantity take-offs for civil bulk materials
- quantity take-offs for concrete, steel, architectural, HVAC, piping and electrical, instrumentation/controls provided by Tetra Tech
- costs for preproduction mining and mining equipment provided by AMC
- quantities and costs for tailings, water management, waste rock, and geotechnical design provided by BGC.

21.1.10 Planning Basis – Execution Strategy

The project execution plan (PEP) has been considered in the preparation of this capital cost estimate.

Key elements of the PEP are described in Section 24.0.

21.1.11 Cost Basis

This section describes the methods and sources used to determine material, labour, and subcontract pricing. The section is organized to be consistent with the Level 2 Area headings.

Labour Rate Development

The construction schedule and labour cost are based on shifts of 10 h/d for 7 d/wk. The work rotation has been assumed as 3-weeks-on/1-week-off.

A blended labour rate of $110/h was calculated for the Project and used throughout the estimate. The rate is based on a typical crew consisting of a lead hand, certified tradesman, uncertified tradesman, skilled labourer, and helper.

The blended labour rate of $110/h includes:

- base rate
- vacation and statutory holiday pay
- fringe benefits and payroll burdens
- overtime and shift premiums

- small tools
- consumables
- contractor's personal protective equipment
- non-productive time (such as tool box briefing, breaks, etc.)
- supervision
- overhead and profit.

Travel and a living out allowance have been calculated separately and are included in the construction indirects section.

21.1.12 PRODUCTIVITY FACTOR

A productivity factor of 1.20 has been applied to the labour portion of the estimate to allow for inefficiencies, based on historical data for similar projects in this region.

The six different classes of production elements affecting work efficiencies are:

- General Economy/Competing Projects – an allowance of 5% has been added to allow the Project to attract labour from other areas or projects.
- Project Supervision – the quality and experience of available supervision will have a significant impact on productivity for the Project as well as the ratio of supervisors to workers.
- Labour Relationship – it is envisioned that the Project would be an open shop environment.
- Job Conditions – the size of the Project is the largest factor in this category. In general, very small or very large projects experience a significant loss of productivity. This includes a general allowance, job preparation allowance, personal care allowance, extra man allowance and moving time allowance. Other condition factors include walking/driving distance, height, underground or low rooms, enclosed spaces, dust, temperature, noise, protective clothing requirements and congestion.
- Construction Equipment and Tools – it is assumed that all equipment will be new; however, weather and local conditions will impact the typical operation of equipment.
- Weather – temperature is one of the largest factors when assigning a productivity factor.
- Estimate Details – Level 1 Major Area

MAJOR AREA 11 – OVERALL SITE

The largest component in this area is bulk earthworks/site preparation cost of $28.2 million. Other significant components in this section include the site roads

($1.6 million), the site control system ($1.0 million), and the site communication and security system ($1.6 million).

The total cost of Major Area 11 – Overall Site is $32.7 million.

Major Area 21 – Mine Underground

The largest component in this section is the lateral ($73.7 million) and vertical ($3.4 million) development costs followed by ventilation shaft (no raise bore) ($20.3 million), and the pre-production mine power/propane ($12.8 million).

Other major components in this section are the underground mobile mining equipment ($28.9 million) and underground material handling system ($6.6 million), which are followed by infrastructure development ($21.6 million). Infrastructure development costs include the underground electrical substation and distribution, pump station, workshop, refuge chambers, portal infrastructure, primary and secondary ventilation fans, fuel storage, and distribution safely equipment.

The total cost of Major Area 21 – Mine Underground is $174.5 million.

Major Area 31 – Mine Site Process

The major cost in the mill feed and storage area is the SAG mill feed surge bin ($5.7 million).

The mill building is the largest single component ($35.1 million) in the grinding, flotation and dewatering area. Other significant contributors to the cost are primary grinding ($16.3 million), concentrate re-grinding ($2.6 million), flotation ($8.7 million) and tailings thickening ($4.7 million), process reagent preparation/storage ($2.2 million), concentrate dewatering ($2.6 million), and smelting/gold room ($1.5 million).

The total cost of Major Area 31 – Mine Site Process is $80.1 million.

Major Area 32 – Mine Site Utilities

The total cost for power and electrical is $13.8 million which includes the high voltage switchyard and substation is ($6.0 million), site power distribution ($5.7 million) and emergency power ($2.0 million).

Other major components in the Mine Site Utilities are fuel storage and distribution ($1.3 million), water systems ($7.3 million) and plant and instrument air is $1.2 million.

The total cost for Major Area 32 – Mine Site Utilities is $23.7 million.

Major Area 33 – Mine Site Facilities

The largest single cost component in this area is the supply and installation of the 330-person permanent camp ($25.9 million). The other major cost component in this area is the paste plant at a cost of $14.5 million.

The total cost for Major Area 33 – Mine Site Facilities is $43.7 million.

Major Area 34 – Mine Site Tailings

The largest cost component in this area is the outlet control system at a cost of $3.3 million.

The total cost for Major Area 34 – Mine Site Tailings is $3.5 million.

Major Area 35 – Mine Site Temporary Facilities

Construction catering ($10.2 million) is the only item contained in this cost area.

The total cost for Major Area 35 – Mine Site Temporary Facilities is $10.2 million.

Major Area 36 – Mine Site (Surface) Mobile Equipment

The largest components in this area are four Foremost Husky 8 tracked vehicles at a quoted price of $1.5 million each, for a total cost of $6.0 million.

The total cost for Major Area 36 – Mine Site (Surface) Mobile Equipment is $14.3 million.

Major Area 61 – Off-site Infrastructure

The largest cost component in off-site infrastructure is the power transmission line at a total cost of $46.1 million. The total cost for the Knipple Transfer Station (including the station, services, utilities, and mobile equipment) is $11.1 million. The access road is estimated to cost $5.0 million. The airstrip is estimated to cost $4.9 million.

The total cost for Major Area 61 – Off-site Infrastructure is $69.1 million.

Major Area 91– Indirects

All construction indirect costs, contractor indirect costs and costs for equipment required for construction are estimated to be $61.7 million.

Other cost components in this area are:

- spares ($7.6 million)
- initial fills ($2.2 million)
- freight and logistics ($13.3 million)
- commissioning and start-up ($6.1 million)
- EPCM ($32.6 million)
- vendors assistance ($1.7 million).

The total costs for Major Area 91 – Indirect Costs is $125.0 million.

Major Area 98 – Owner's Costs

Items included in the Owner's costs are the Project team (salaries), travel, recruitment, training, human relations, community relations, health and safety, security, environmental considerations, permits/licenses/fees, insurance, legal/audits, satellite office, communications, and orientation.

The total allowance for Major Area 98 – Owner's Costs is $22.3 million.

Major Area 99 – Contingency

A contingency allowance was established for each area/discipline/task depending on the level of engineering effort and risk.

The largest single contingency allowance is for mining at $19.1 million, followed by architectural ($8.0 million), power line ($7.5 million), and bulk earthworks ($6.2 million).

The total allowance for contingency for the Project is $64.4 million.

21.2 Operating Cost Estimate

21.2.1 Summary

The total LOM average operating cost for the Project is estimated at Cdn$156.46/t ore milled, which includes costs for:

- mining
- process
- material re-handling in Year 1 for the stockpiled ore produced during pre-production
- general and administrative (G&A)
- surface services
- backfill, including paste preparation
- water treatment.

The operating costs exclude sustaining capital costs, off-site costs (such as shipping and smelting costs), taxes, permitting costs, or other government imposed costs, unless otherwise noted.

A total of 542 personnel are projected to be required for the Project, including an average of 316 personnel for mining operations, 95 personnel for process, 43 personnel for G&A, 78 personnel for surface services, and 10 personnel for the backfill plant and water treatment plant. The mining personnel requirement is expected to gradually reduce after Year 15 to approximately 167 personnel in Year 22.

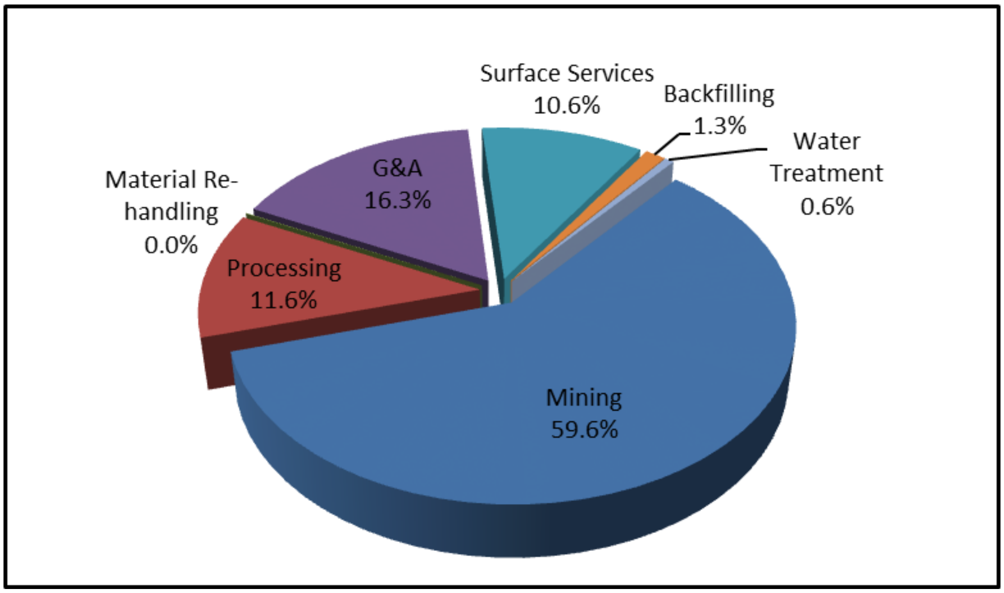
The unit cost estimates are based on the LOM ore production and a mine life of 22 years. The currency exchange rate used for the estimate is 1:1 (Cdn$:US$). The operating cost for the Project has been estimated in Canadian dollars within an accuracy range of ±15%. A summary of the overall operating cost is presented in Table 21.3. The cost distribution is illustrated in Figure 21.1.

Table 21.3 Overall Operating Cost

Area	Personnel	Unit Operating Cost ($/t milled)
Mining*	316**	93.18
Processing	95	18.16
Material Re-handling***	Contract	0.07
G&A	43	25.47
Surface Services	78	16.53
Backfilling	6	2.10
Water Treatment	4	0.95
Total	**542**	**156.46**

Notes: *Average LOM mining cost including crushing cost and cement cost for backfill; if excluding the ore mined during preproduction, the estimated unit cost is $94.40/t
**316 workers during year 1 to 14 and then reduce to 167 workers at the end of the mine life.
***Material re-handling cost is the LOM average cost, which will occur in year 1 only. The operation is assumed to be contracted with approximately eight workers required.

Figure 21.1 Overall Operating Cost Distribution



21.2.2 MINING OPERATING COSTS

Mining operating costs were assembled from first principle work-ups of key activities.

Operating costs for major mobile equipment and ancillary mine equipment were generated from vendor rates and estimates of equipment utilization. These estimates were derived directly from mucking, hauling, ground support installation, and drill and blast activities necessary for development and stoping. Ancillary equipment utilization was based on expected usage to complete general mine tasks. Consumable costs were generated in a similar manner where unit costs from suppliers were applied to the calculated use and quantities derived from the mining activity.

Labour is a significant contributor to operating costs but was calculated independently from the first principle work-ups. The annual labour requirements are considered to be largely fixed once the mine attains near full production in Year 2. Labour costs were reduced to 60% of the peak value in Years 19 to 22 to coincide with reduced ore production near the end of the mine life.

Mine personnel will be phased-in during the second half of Year -1, in consideration of the handover from contract personnel and growing operational requirements. AMC anticipates some labour overlap and general inefficiency during this period.

The following workforce assumptions were made for the second half of Year -1:

- technical services and supervisory: 60% of full complement
- production crew: 30% of full complement
- development crew: personnel count based on scheduled metres.

In Year 1, ore production attains approximately 58% of the 980,000 t/a target. AMC assumes that 70% of the full labour complement would be required to achieve this target. From Year 2 (approximately 94% of full production achieved), full labour numbers have been applied.

As the mine matures and development requirements decline, the following adjustments were made in relation to development labour.

- Years 10 to 13: 67% of the full complement
- Years 14 to 21: 20% of the full complement
- Year 22: 5% of the full complement.

All labour rates were developed from the nature of the job classification. Burdens were calculated at 30.4% of the base salary plus annual bonuses, statutory holidays, and vacations.

Mine Operating Costs

The total underground LOM operating cost is estimated at $1,769 million, as shown in Table 21.4.

Table 21.4 LOM Underground Operating Costs

Area	$ million	$/t*
Development	296.9	15.84
Stoping	765.2	40.83
Maintenance**	281.2	15.01
Supervisory and Technical Services	153.5	8.19
Diamond Drilling	56.0	2.99
Mine General	216.2	11.54
Total* **	**1,769.0**	**94.40**

Notes: *Average cost over the LOM.
**Maintenance costs include maintenance labour, electrical and fixed plant maintenance only.
***if including the ore mined during preproduction, the estimated unit cost is $93.18/t.

The Mine General area of the underground operating costs is further detailed in Table 21.5.

Table 21.5 Underground Operating Costs – Mine General Area

Item	$ million	$/t*
Power	40.1	2.14
Propane	3.6	0.19
Crushing and Conveying	18.7	1.00
Ancillary Services	56.2	3.00
Ancillary Equipment	29.0	1.55
Logistics Labour	68.6	3.66
Total	**216.2**	**11.54**

Note: *Average cost over the LOM.

Table 21.6 shows the annual mine operating cost breakdown over the LOM.

Table 21.6 Annual Mine Operating Costs

Year	Development ($ million)	Stoping ($ million)	Maintenance ($ million)	Supervisory and Technical Services ($ million)	Diamond Drilling ($ million)	Mine General ($ million)	Ore (t)	$/t
-2	-	-	-	-	-	-	5,124	-
-1	-	-	-	-	-	-	240,599	-
1	14.2	26.5	9.8	5.7	1.7	8.2	565,653	116.8
2	20.0	38.9	14.2	7.7	2.8	10.8	937,169	100.7
3	18.8	40.3	14.3	7.7	2.9	11.0	979,037	97.2
4	21.8	37.4	14.3	7.7	2.9	11.1	980,631	97.2
5	22.8	37.4	14.3	7.7	2.9	11.1	982,603	97.9
6	22.9	37.1	14.3	7.7	2.9	11.2	985,759	97.6
7	18.7	41.5	14.3	7.7	2.9	10.9	985,324	97.5
8	17.4	38.9	14.3	7.7	2.9	10.8	984,974	93.6
9	15.1	42.2	14.3	7.7	2.9	11.1	980,427	95.2
10	16.4	41.1	14.3	7.7	3.0	11.1	990,705	94.4
11	14.4	38.2	14.3	7.7	2.9	11.2	978,466	90.8
12	16.8	38.3	14.3	7.7	2.9	11.2	978,883	93.2
13	17.0	39.5	14.3	7.7	2.9	11.3	981,517	94.4
14	11.8	40.4	14.3	7.7	3.0	11.4	987,498	89.6
15	9.3	38.5	14.3	7.7	3.0	11.0	986,651	84.9
16	7.5	39.4	14.3	7.7	2.9	11.0	979,339	84.6
17	8.7	39.1	14.3	7.7	2.9	10.3	981,683	84.6
18	6.8	38.2	14.2	7.7	2.8	10.1	949,013	84.2
19	5.5	21.5	8.4	5.0	1.5	6.6	501,236	96.7
20	6.0	21.4	8.4	4.9	1.5	6.5	495,290	98.4
21	3.5	21.3	8.2	4.9	1.2	5.5	404,038	110.3
22	1.3	8.2	3.4	1.9	0.4	2.8	144,317	125.3
Total	**296.9**	**765.2**	**281.2**	**153.5**	**56.0**	**216.2**	**18,740,213**	**94.4**

21.2.3 Process Operating Costs

The estimated operating cost for process operations is shown in Table 21.7, including grinding, gravity concentration, bulk flotation, concentrate dewatering, flotation tailings delivery, and doré production. The LOM average process operating cost is estimated to be $18.16/t milled or $16.3 million per year excluding the operating cost for crushing, which has been covered in the mining operating cost. The estimate is based on a total mill feed of 18,985,936 t and a mine life of 22 years, or an average annual process rate of 897,000 t.

Table 21.7 Summary of Process Operating Cost

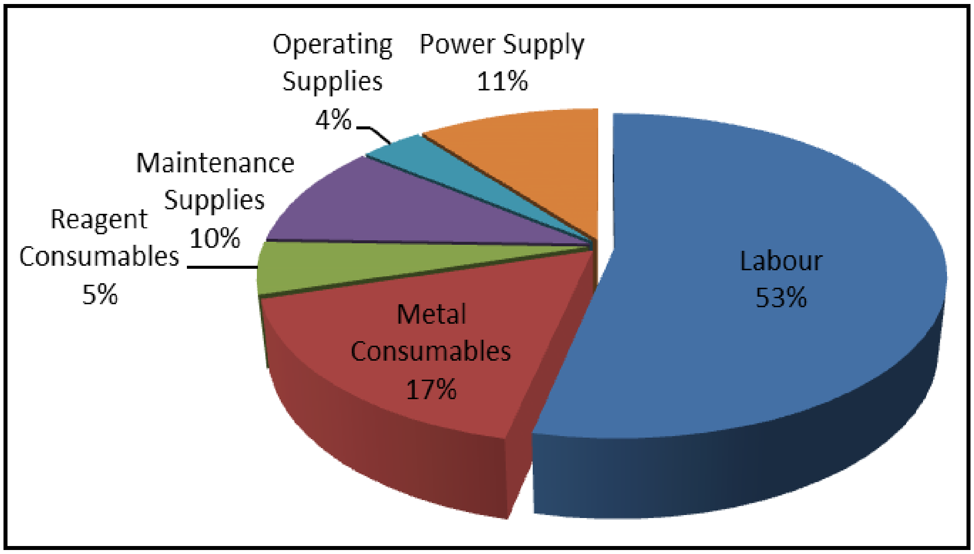
Description	Labour Force	Annual Cost ($)	Unit Cost ($/t Milled)
Labour Force			
Operating Staff	18	2,057,000	2.29
Operating Labour	38	3,163,000	3.53
Maintenance Labour	39	3,487,000	3.89
Subtotal Labour Force	**95**	**8,707,000**	**9.71**
Major Consumables			
Metal Consumables	-	2,750,000	3.06
Reagent Consumables	-	842,000	0.94
Subtotal Major Consumables	**-**	**3,592,000**	**4.00**
Supplies			
Maintenance Supplies	-	1,669,000	1.86
Operating Supplies	-	589,000	0.66
Power Supply	-	1,732,000	1.93
Subtotal Supplies	**-**	**3,990,000**	**4.45**
Total (Process)	**-**	**16,289,000**	**18.16**

All process operating costs are exclusive of taxes, permitting costs, or other government imposed costs unless otherwise noted. The following items have been included in the estimate:

- labour force requirements, including supervision, operation, and maintenance, salary/wage levels based on current labour rates in comparable operations in BC. The annual salary includes holiday and vacation pay. A benefit burden of 30% for the labour includes Registered Retirement Savings Plans (RRSPs), various life and accident insurances, extended medical benefits, BC Medical Service Plan (MSP), Canadian Pension Plan (CPP), Employment Insurance (EI), Workers' Compensation Board (WCB) insurance, tool allowance, and other benefits.
- mill liner and mill grinding media consumptions, estimated from the Bond grinding media/liner consumption estimate equations and the Tetra Tech database; the steel ball and mill liner prices based on the quotations from the potential suppliers.
- maintenance supply costs, based on approximately 7% of major equipment capital costs.
- laboratory supplies, building maintenance and other costs, based on Tetra Tech's in-house database and industry experience.
- reagent costs, based on the consumption rates from test results and quoted budget prices or the Tetra Tech database.
- service vehicle costs, including fuel consumables and maintenances, are included in the surface service cost estimates.

Figure 21.2 shows the cost distribution in the different areas.

Figure 21.2 Process Operating Cost Distribution



The estimated labour force cost is $9.71/t milled. A total of 95 personnel are estimated for the process operation, including 18 staff for management and professional services, 38 operators for operating and assaying, and 39 personnel for maintenance. The estimate is based on 12 hours per shift, 24 h/d, and 365 d/a. The operator rotation is based on a schedule of two-weeks-in/two-weeks-out. Travel time will occur during the employee's time off.

The operating cost for the major metal consumables is estimated to be $3.06/t milled. The metal consumables include mill and crusher liners, and mill grinding media.

The estimated reagent cost is $0.94/t milled. Reagent consumptions are estimated from laboratory test results and comparable operations. The reagent costs are from current budget prices received from potential suppliers.

The maintenance supply cost is $1.86/t milled while the operating supply cost is $0.66/t milled.

The power cost is estimated at $1.93/t milled. Electricity will be supplied from the transmission line connected to the mine site. The power cost is based on a unit electric energy price of $0.050/kWh.

21.2.4 Backfilling Operating Costs

The estimated operating cost for the backfilling plant is $2.1/t and is shown in Table 21.8. The estimate includes costs for tailings filtration, paste generation, and paste delivery but excludes the cement cost, which is the major cost for backfilling and is included in the mining cost estimates.

Table 21.8 Summary of Backfilling Operating Cost

Description	Labour Force	Annual Cost ($)	Unit Cost ($/t Milled)
Labour Force			
Operating Labour	4	360,000	0.40
Maintenance Labour	2	196,000	0.22
Subtotal Labour Force	**6**	**556,000**	**0.62**
Major Consumables (Included in Mining Operating Cost)			
Supplies			
Maintenance Supplies	-	319,000	0.36
Operating Supplies	-	760,000	0.85
Power Supply	-	252,000	0.28
Subtotal Supplies	**-**	**1,331,000**	**1.48**
Total (Backfilling)	**-**	**1,887,000**	**2.10**

The estimated labour force cost is $0.62/t milled with 6 personnel required to operate the plant. The annual labour cost includes the benefit burden outlined in Section 21.2.3. The maintenance supply cost is $0.36/t milled while the operating supply cost is $0.85/t milled. The power cost is estimated at $0.28/t milled. The estimates are based on the LOM average mill feed rate.

21.2.5 WATER TREATMENT OPERATING COSTS

The total cost for the water treatment is estimated at $0.95/t milled. Four personnel are required to operate the plant. The annual labour cost includes the benefit burden outlined in Section 21.2.3. Water treatment personnel, together with personnel from the backfill plant, will undertake surface services duties as required. A summary of the water treatment plant operating cost is shown in Table 21.9.

Table 21.9 Summary of Water Treatment Operating Cost

Description	Labour Force	Annual Cost ($)	Unit Cost ($/t Milled)
Labour Force			
Operating Labour	4	276,000	0.31
Subtotal Labour Force	**4**	**276,000**	**0.31**
Major Consumables			
Reagent Consumables	-	361,000	0.40
Subtotal Major Consumables	**-**	**361,000**	**0.40**
Supplies			
Maintenance Supplies	-	180,000	0.20
Operating Supplies	-	10,000	0.01
Power Supply	-	28,000	0.03
Subtotal Supplies	**-**	**218,000**	**0.24**
Total (water treatment)	**-**	**855,000**	**0.95**

21.2.6 General and Administrative, and Surface Services

Tetra Tech and Pretivm developed the G&A costs, which are estimated to be $25.47/t milled. Personnel transportation and catering expenses are the major components of the G&A cost.

The G&A costs include:

- labour costs for administrative personnel, including the benefit burden outlined in Section 21.2.3
- expenses for the services related to travel, human resources, safety and security
- site communications, including technical services support and spare parts
- allowances for insurance, regional taxes and licenses allowance
- sustainability including environment, community liaison, and engineering consulting
- transportation of personnel, including air transportation and travel time allowances
- camp accommodation costs.

A summary of the G&A costs is provided in Table 21.10.

Table 21.10 G&A Operating Cost

Description	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
G&A Labour Force			
G&A	27	3,216,000	3.59
G&A Hourly Personnel	16	1,422,000	1.58
Subtotal Labour Force	**43**	**4,638,000**	**5.17**
G&A Expense			
General Office Expense	-	300,000	0.33
Computer Supplies including Software	-	61,000	0.07
Communications	-	702,000	0.78
Travel	-	150,000	0.17
Audit	-	85,000	0.10
Consulting/External Assays	-	150,000	0.17
Environmental	-	500,000	0.56
Insurance	-	1,500,000	1.67
Regional Taxes and Licenses Allowance	-	430,000	0.48
Legal Services	-	350,000	0.39
Warehouse	-	200,000	0.22
Recruiting including Relocation Expense	-	100,000	0.11
Entertainment/Memberships	-	50,000	0.06

table continues…

Description	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
Medicals and First Aid	-	100,000	0.11
Training/Safety	-	250,000	0.28
Accommodation/Camp Costs	-	6,297,000	7.020
Crew Transportation (Flight and Bus)	-	6,633,000	7.39
Liaison Committee/Sustainability	-	200,000	0.22
Others	-	150,000	0.17
Subtotal Expense	**-**	**18,208,000**	**20.30**
Total	**30**	**22,846,000**	**25.47**

There are 78 personnel required to provide surface services. The total surface services cost is estimated to be $16.53/t milled. The estimate is summarized in Table 21.11 and includes the following costs and operations:

- labour costs for surface service personnel, including transport drivers, site and access road maintenance operators, surface equipment maintenance operators, mine dry cleaners, and the operators at the Knipple Transfer Station; the annual labour cost includes the benefit burden outlined in Section 21.2.3.
- surface mobile equipment and light vehicle operations, including transportation between the mine site and the Knipple Transfer Station, as well as snow removal.
- portable water and waste management
- general maintenance including yards, roads, fences, and building maintenance
- off-site operation expenses
- building heating
- access road maintenance
- avalanche control
- airstrip maintenance and operations
- maintenance of the power line supplying power to the mine and process plant.

Table 21.11 Surface Services Operating Costs

Surface Service	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
Surface Service Labour Force			
Surface Service Personnel	78	6,599,000	7.36
Subtotal Labour Force	**78**	**6,599,000**	**7.36**
Surface Service Expense			
Small Vehicles/Equipment	-	100,000	0.11
Potable Water and Waste Management	-	320,000	0.36
Supplies	-	150,000	0.17
Building Maintenance	-	300,000	0.33
Building Heating	-	400,000	0.45
Road Maintenance	-	1,320,000	1.47
Avalanche Control	-	375,000	0.42
Power Line Maintenance	-	164,000	0.18
Mobile Equipment - Maintenance	-	1,400,000	1.56
Mobile Equipment - Fuel	-	3,375,000	3.76
Airstrip Instrumentation Maintenance	-	20,000	0.02
Off-site Operation Expense	-	300,000	0.33
Subtotal Expense		**8,224,000**	**9.17**
Total	**78**	**14,823,000**	**16.53**

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

Tetra Tech prepared an economic evaluation of the Project based on a pre-tax financial model. For the 22-year LOM and 18.99 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 42.9% IRR
- 2.1-year payback on the US$663.5 million initial capital
- US$2,687 million NPV at a 5% discount rate.

A post-tax economic evaluation of the Project was prepared with the inclusion of applicable taxes (Section 22.6).

The following post-tax financial parameters were calculated:

- 35.7% IRR
- 2.2-year payback on the US$663.5 million initial capital
- US$1,763 million NPV at a 5% discount rate.

The base case metal prices used for this study are as follows:

- gold – US$1,350/oz
- silver – US$20.00/oz
- exchange rate – 1.00:1.00 (US$:Cdn$).

Sensitivity analyses were carried out to evaluate sensitivity of the Project economics to the key parameters.

22.2 PRE-TAX MODEL

22.2.1 FINANCIAL EVALUATIONS

The production schedule has been incorporated into the pre-tax financial model to develop annual recovered metal production. The annual at-mine revenue contribution of each metal has been determined by deducting the applicable treatment, refining, and transportation charges (from mine site to market) from gross revenue.

Unit operating costs were multiplied by annual milled tonnages to determine the total mine operating costs. The total mine operating costs were then deducted from the at-the-mine-revenues to derive annual operating cash flows.

Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the operating cash flows to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of doré and concentrate; sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and tailings embankment construction. The total LOM sustaining capital is $328.54 million.

The mine closure and reclamation cost is $25.68 million.

Working capital has been calculated based on a three-month operating cost in Year 1 of the mine operation and will be recovered at the end of the mine life.

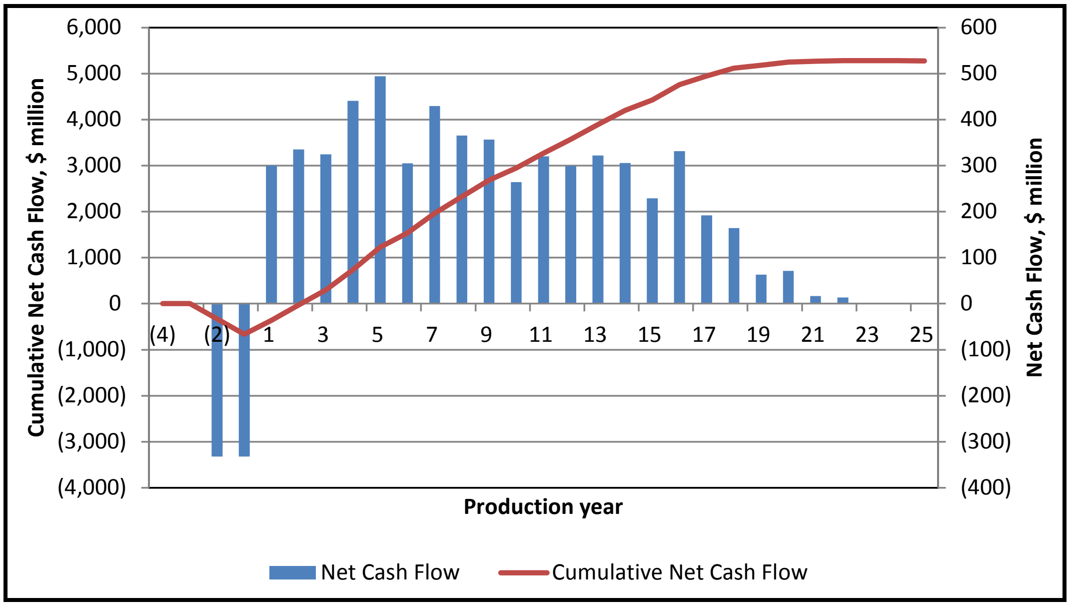
NPV was estimated at the beginning of the four-year construction period.

Metal production quantities are presented in Table 22.1. The annual pre-tax net cash flows (NCFs) and cumulative net cash flows (CNCFs) are presented in Figure 22.1.

Table 22.1 Metal Production Quantities

	Years 1 to 10	LOM
Total Tonnes to Mill ('000)	9,618	18,986
Annual Tonnes to Mill ('000)	962	863
Average Grade		
Gold (g/t)	14.234	12.011
Silver (g/t)	11.882	57.869
Total Production		
Gold ('000 oz)	4,257	7,073
Silver ('000 oz)	3,126	31,641
Average Annual Production		
Gold ('000 oz)	425.668	321.493
Silver ('000 oz)	313	1438

Figure 22.1 Pre-tax Cash Flow



22.2.2 Metal Price Scenarios

The financial outcomes for the different metal price scenarios have been tabulated for NPV, IRR, and payback of capital. A discount rate of 5% was applied to all cases identified by the following metal price scenarios:

- base case
- lower prices case
- spot metal prices as of June 6, 2013.

The summary of pre-tax project economic evaluation is presented in Table 22.2.

Table 22.2 Summary of Pre-tax NPV, IRR, and Payback by Metal Price

Economic Returns	Unit	Base Case	Lower Price	Spot Prices*
NCF	US$ million	5,278	1,408	5,898
NPV at 5.0% Discount Rate	US$ million	2,687	602	3,014
Project IRR	%	42.9	16.6	47.0
Payback	Years	2.1	4.7	1.9
Exchange Rate	US$:Cdn$	1.00	1.00	0.98
Gold Price	US$/oz	1,350	800	1,416
Silver Price	US$/oz	20.00	15.00	22.70

Note: *Spot prices as at June 6, 2013

The summary of post-tax project economic evaluation is presented in Table 22.3.

Table 22.3 Summary of Post-tax NPV, IRR, and Payback by Metal Price

Economic Returns	Unit	Base Case	Lower Price	Spot Prices*
NCF	US$ million	3,499	964	3,913
NPV at 5.0% Discount Rate	US$ million	1,763	384	1,984
Project IRR	%	35.7	13.7	39.2
Payback	Years	2.2	4.8	2.0
Exchange Rate	US$:Cdn$	1.00	1.00	0.98
Gold Price	US$/oz	1,350	800	1,416
Silver Price	US$/oz	20.00	15.00	22.70

Note: *Spot prices as at June 6, 2013

22.2.3 Royalties

There are royalties applicable to the Project. "Royalty" means the amount payable by the Owner, calculated as 1.2% of the at-mine-revenue, with the following exemptions:

- gold: the first 503,386 oz produced from the Property
- silver: the first 17,907,080 oz produced from the Property.

22.3 Smelter Terms

In the absence of letters of interest or letters of intent from potential smelter or buyers of doré, in-house database numbers were used to benchmark the terms supplied by the owner.

Contracts will generally include payment terms as follows:

- doré:
  - gold and silver – pay 99.8% of gold and silver content. A refining and transport charge of $2.00/troy oz will be deducted from the metal price.
- concentrate:
  - gold and silver – pay 95% of gold and silver content. A treatment charge is $200.00/dmt of concentrate is applied. A penalty charge of $10 per each 0.1% of arsenic is also applied.

22.4 Markets and Contracts

22.4.1 Markets

The Project will produce gold and silver doré and concentrates. Doré will be trucked to the smelter. Concentrates will be loaded in bags and transported from the mill site to the transfer station for storage and transfer to flat-deck trucks. The flat-deck trucks will

transport the concentrate to a built for purpose rail trans-load facility in Terrace, BC. The bags will be stored and loaded in gondola type railcars and shipped via rail to the Horne Smelter in Noranda, Quebec.

22.4.2 CONTRACTS

There are no established contracts for the sale of the doré or the concentrate currently in place for the Project.

22.4.3 TRANSPORT INSURANCE

Doré transportation cost is $1.00/oz. Concentrate transportation cost is $193.24/wmt of concentrate. An insurance rate of 0.5% will be applied to the provisional invoice value of doré and concentrate to cover transport from the mine site to the smelter.

22.5 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters by changing one parameter at a time between ±30% at 10% intervals while holding the rest of the following parameters constant:

- gold price
- silver price
- exchange rate
- operating cost
- capital cost.

The analyses are presented as financial outcomes in terms of NPV in Figure 22.2, IRR in Figure 22.3 and payback period in Figure 22.4. The Project NPV (at a 5% discount rate) is most sensitive to the gold price and less sensitive to the rest of the parameters. The Project IRR and payback are most sensitive to exchange rate and gold price followed by operating costs, capital costs, and silver price.

Figure 22.2 Pre-tax NPV (5%) Sensitivity Analysis

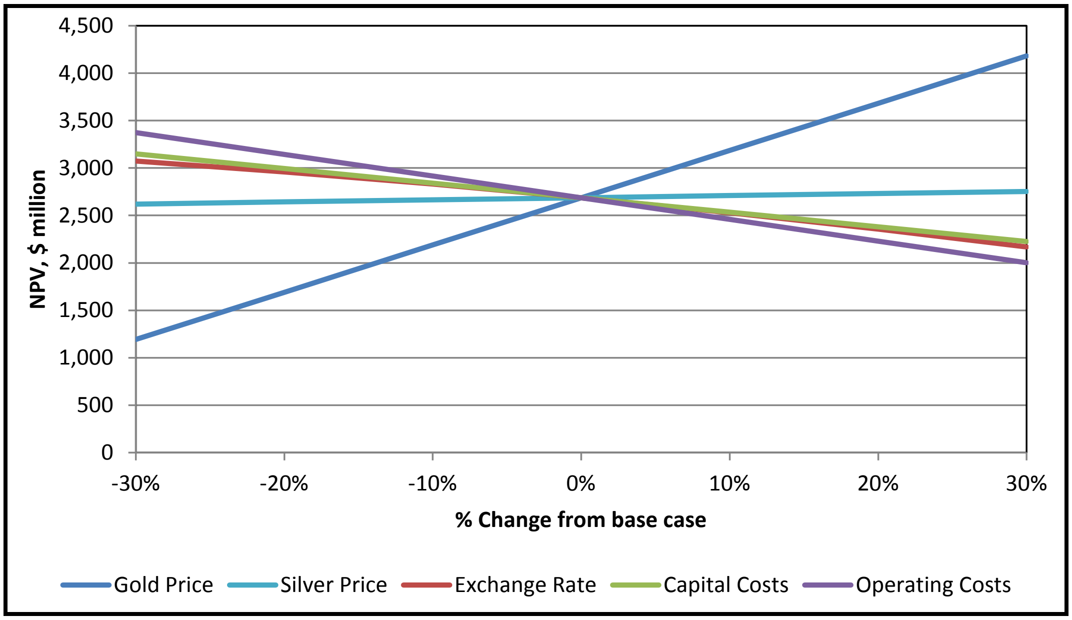


Figure 22.3 Figure 22.3 Pre-tax IRR Sensitivity Analysis

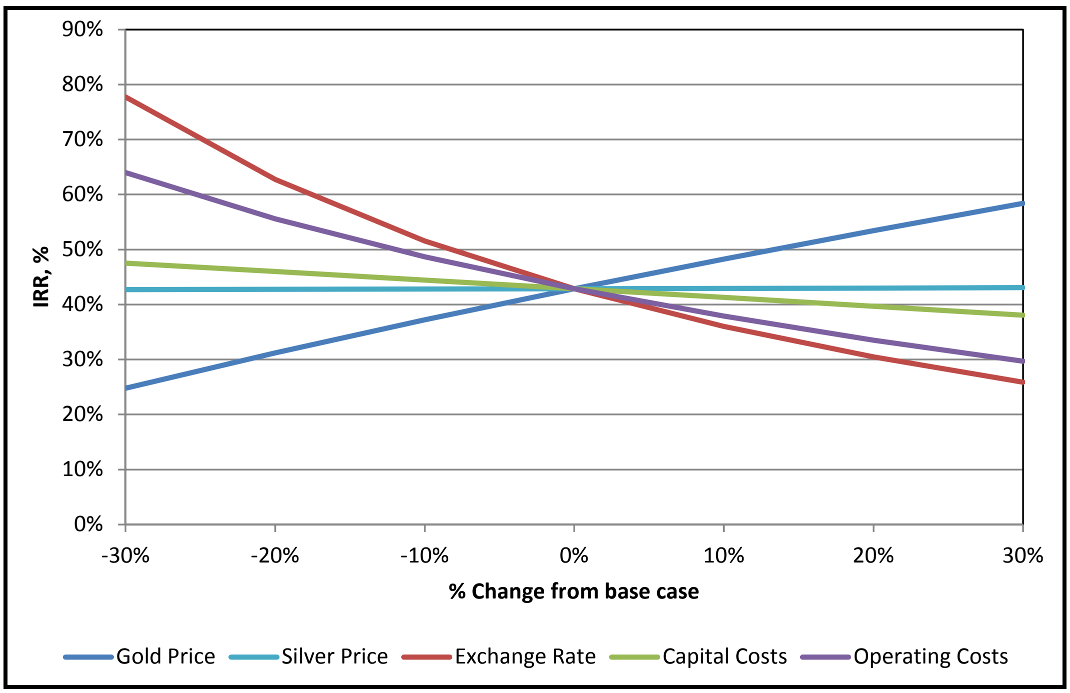
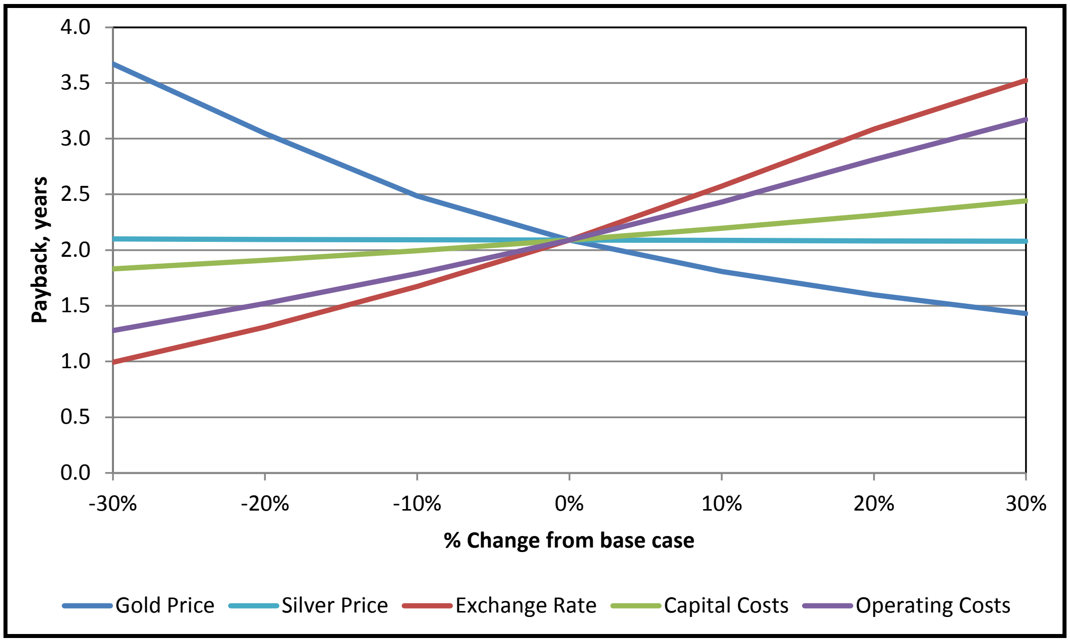


Figure 22.4 Pre-tax Payback Period Sensitivity Analysis



22.6 TAXES

Pretivm commissioned Sadhra & Chow LLP in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes.

Based on the long-term metal prices used for this study, the total estimated taxes payable on Brucejack profits are $1,778 million over the 22-year mine life. The components of the various taxes that will be payable are shown in Table 22.4.

Table 22.4 Components of the Various Taxes

Tax Component	LOM Amount ($ million)
Corporate Tax (Federal)	671.910
Corporate Tax (Provincial)	457.244
Provincial Resource Tax	649.124
Total Taxes	**1,778.278**

The following general tax regime was recognized as applicable at the time of report writing.

Canadian Income Tax System

Federal Income Tax Rate:	15%
Provincial (BC) Income Tax Rate:	10%
Machinery and Equipment:	Prior to 2021, assets purchased prior to commercial production are added to a Class 41(a) pool and are deducted at an accelerated rate, at up to 100% of the balance, to the extent of taxable income from the mine. Recent proposed tax amendments in the 2013 federal budget will phase out the accelerated deduction over the 2017 to 2020 years. One hundred percent of the accelerated rate will be permitted in 2013 to 2016, 90% in 2017, 80% in 2018, 60% in 2019, and 30% in 2020. Assets purchased after commencement of production are added to a Class 41(b) pool and are deducted at up to 25% of the balance.
Mine Acquisition Costs:	Including costs of land, exploration and mining rights, licenses, permits and leases. Costs are added to a Canadian development expense pool and can be deducted at up to 30% of the balance of a year.
Pre-production Mine Expenditures:	This includes both exploration and mine development costs. Prior to 2015, exploration and mine development are added to a Canadian exploration expense pool. One hundred percent of the balance can be deducted in a year, but the deduction is also limited to the income from the mine. Under recent proposed tax amendments in the 2013 federal budget, pre-production mine development costs incurred subsequent to 2017 will be treated as a Canadian development expense instead of a Canadian exploration expense. The transition will be phased-in beginning in 2015, with 20% of costs being allocated proportionately to a Canadian development expense and 80% to a Canadian exploration expense in 2015, 40% to a

	Canadian development expense and 60% to a Canadian exploration expense in 2016, and 70% to a Canadian development expense in 2017 and 30% to a Canadian exploration expense in 2017.

Provincial (BC) Mining Tax System

Net Current Proceeds (2%) Tax:	Two percent is levied on amount by which gross revenues exceed current operating costs. Hedging income and losses, royalties and financing costs are excluded. Capital costs including exploration, pre-production development costs and leasing costs are excluded. Capital costs are relevant for net revenue tax (below). Net current proceeds tax is added to a cumulative tax credit account (CTCA) and is available to offset net revenue tax payable.
Net Revenue (13%) Tax	Tax is levied at 13% of net revenue. All capital expenditures, both mine development costs and fixed asset purchases, are accumulated in a cumulative expenditures account (CEA). Net revenue is defined as 13% of gross revenues less the current operating costs for the year, less any accumulated CEA balance. Therefore, for net revenue tax, all current and capital expenditures are fully deductible in the year they are incurred or in the following year. Net revenue does not become assessable until the costs of all pre-production capital expenditures have been recovered. A “New mine allowance” is also provided to encourage new mine development in BC. The allowance allows a mine operator to add 14% of its capital expenditures incurred prior to commencing production to the CEA account. Under current legislation, the new mine allowance is scheduled to expire on January 1, 2016. The

model is calculated on the assumption that the mine allowance will be continued.

BC mineral taxes are deductible for federal and provincial income tax purposes.

23.0 ADJACENT PROPERTIES

Snowden notes:

- This information was publicly disclosed by the Owner or Operator of the adjacent property and was sourced as per the notes in the relevant section below.
- The QP has been unable to verify the information provided here, except against what has been publicly reported.
- The information is not necessarily indicative of the mineralization at Brucejack.

23.1 KERR-SULPHURETS-MITCHELL

Within the adjacent KSM Property there are four copper-gold mineral deposits, namely Kerr, Mitchell, Sulphurets, and Iron Cap. All of these occurrences are situated within the claim holdings that are, at the time of writing this report, owned and operated by Seabridge Gold.

Seabridge Gold acquired the KSM Property from Placer Dome in June 2000.

In May 2012, Seabridge published a revised prefeasibility study, which resulted in a mineral reserve of 2.2 Bt (2.4 billion tons) of gold, copper, silver, and molybdenum ore (Table 23.1). Seabridge Gold reported that all ore will be mined using open pit methods for the first 25 years, and will switch to underground block caving in Year 26. Over the entire 55-year mine life, ore will be fed to a flotation mill, which will produce a combined gold/copper/silver concentrate. The concentrate will be transported by truck to the nearby deep-water sea port at Stewart, BC, for shipment to a Pacific Rim smelter. Extensive metallurgical testing confirmed that KSM could produce a clean concentrate with an average copper grade of 25%, making it readily saleable. Separate molybdenum concentrate and gold-silver doré will be produced at the KSM processing facility. All information for this section has been taken from the Seabridge Gold website (www.seabridgegold.net).

The QP for this section of the report has not verified the information concerning the KSM Deposits, and the information is not necessarily indicative of the mineralization at the Brucejack Project.

Table 23.1 Mineral Reserve Estimates for the KSM Property

Zone	Mining Method	Reserve Category	Tonnes (Mt)	Average Grades				Contained Metal			
				Gold (g/t)	Copper (%)	Silver (g/t)	Moly (ppm)	Gold (Moz)	Copper (Mlb)	Silver (Moz)	Moly (Mlb)
Mitchell	Open Pit	Proven	476	0.67	0.17	3.05	60.9	10.3	1,798	47	64
		Probable	497	0.61	0.16	2.78	65.8	9.8	1,707	44	72
	Block Cave	Probable	438	0.53	0.17	3.48	33.6	7.4	1,589	49	32
Iron Cap	Block Cave	Probable	193	0.45	0.20	5.32	21.5	2.8	834	33	9
Sulphurets	Open Pit	Probable	318	0.59	0.22	0.79	50.6	6.0	1,535	8	35
Kerr	Open Pit	Probable	242	0.24	0.45	1.2	0.0	1.9	2,425	9	0
		Proven	476	0.67	0.17	3.05	60.9	10.3	1,798	47	64
		Probable	1,688	0.51	0.22	2.65	40.1	27.9	8,090	144	149
		Total	2,164	0.55	0.21	2.74	44.7	38.2	9,888	191	213

Note: Cut-off values were defined based on NSR values and type of mining. The reader should refer to the information provided by Seabridge Gold to get an accurate appreciation of the definition of the cut-off values for reporting.

23.2 HIGH PROPERTY

The Teuton Resources Corporation (Teuton) High Property is located immediately to the south of the Brucejack Property. Teuton conducted limited preliminary exploration of the High Property in 2011 and 2012, including prospecting, collection of surface grab samples, and drilling. Results posted on Teuton's website (www.teuton.com) indicate the presence of porphyry-style gold and base metal sulphide mineralization on the High Property. A single drillhole through a hypabyssal porphyry body was reported as intersecting 222 m of 0.88 g/t Au (in the King Tut Zone).

The QP for this section of the report has not verified the information concerning the High Property, and the information is not necessarily indicative of the mineralization at the Brucejack Project.

23.3 TREATY CREEK PROPERTY

The Treaty Creek Property, the title for which is currently under litigation (www.teuton.com and www.americancreek.com) adjoins directly northeast of the Seabridge Gold's KSM gold-copper property and is underlain by a similar geology. Exploration work uncovered several zones, the most promising of which are the Copper Belle (porphyry-style), GR2 (feeder zone to a volcanogenic massive sulphide (VMS)), Eureka (porphyry-style with a gold-silver epithermal overprint), and Treaty Ridge (VMS/Sedex?) zones.

The QP for this section of the report has not verified the information concerning the Treaty Creek Property, and the information is not necessarily indicative of the mineralization at the Brucejack Project.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

24.1.1 INTRODUCTION

This project execution plan describes how the Project will advance towards EPCM and environmental activities.

24.1.2 HEALTH, SAFETY, ENVIRONMENTAL AND SECURITY

Health, safety, and environmental (HSE) programs and initiatives will be essential to the Project's success. A fully-integrated HSE program will be implemented to help achieve a "zero-harm" goal. To achieve this goal, all key project stakeholders will be responsible for providing leadership and committing to the highest HSE standards and values.

The development of HSE practices will require a high level of communication, motivation, and involvement 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. The Project team will incorporate HSE as key criteria in the design, constructability, and operability of each facility and major area.

To meet established capture and containment guidelines, all design and engineering stages incorporate criteria for the responsible management of process flows, effluent, and waste products. The design also incorporates basic clean plant design standards, including operational safety and maintenance access requirements. The Project design team will conduct a hazard and operability analysis (HAZOP) for each area of the plant during the detailed design stage, and will strive to eliminate any identified hazards. 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 of the boundaries of the plant.

24.1.3 EXECUTION STRATEGY

The project management organization chart is illustrated in Figure 24.1. The Project Team (the Team) will be led by the Owner and the engineering and construction

managers, and the mining manager, all of whom will assume responsibility for completing the Project successfully, using the following strategies.

For the surface facilities, the execution strategy reflects a single EPCM contractor approach to manage project execution. Under the direction of a construction management team (CMT), field construction contractors, will commence work after engineering and procurement tasks are well-advanced.

For underground development, the feasibility level design will be updated by a selected mining design firm. After substantial completion of the design update has been completed, a mining contractor will be engaged to perform all decline, drift, raise, and infrastructure development.

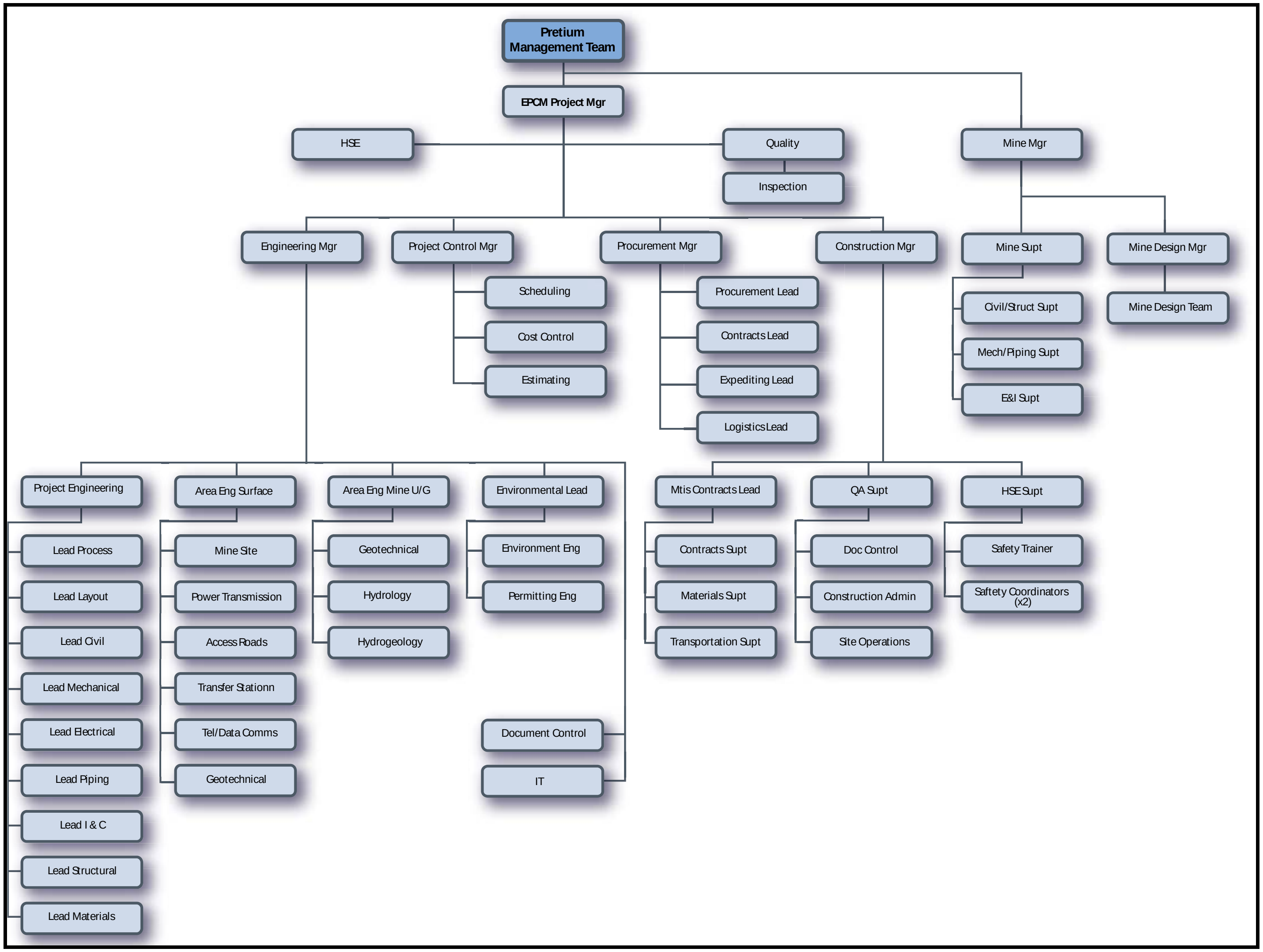
The Project will transition from the study phase to basic engineering in Q3 2013 and will move forward in the following phases:

- Stage I – early works including mine development, the EAC application, permitting, access road upgrades, preliminary power transmission line ROW, basic engineering, and the procurement of long-lead equipment.
- Stage II – full project execution (following permit approval), including detailed engineering, procurement, construction team mobilization, construction, and commissioning.

Upon completion of the feasibility study phase and project sanction, Stage I will begin and will focus on permitting, basic engineering, preliminary infrastructure works, and procurement of long delivery equipment to maintain the targeted project completion schedule. Stage II will focus on delivery of the operating mine, process facilities, and infrastructure in order to provide a fully operational facility by Q3 2016.

The following subsections discuss the framework for the execution of the Project during the EPCM phases, and into operations.

Figure 24.1 Project Management Organization Chart



Management Procedures

The EPCM contractor and the Mine Manager will develop a comprehensive set of project procedures, in conjunction with the Owner. These procedures will outline the requirements for the execution of the administrative activities, as well as the Owner, EPCM contractor rights, Mining contractor rights, authorities, and obligations to the Project.

The procedures will include:

- project organization, key names, and communication procedures
- reporting requirements including project systems, project meetings, minutes, and a communications matrix
- identification of the division of responsibilities among the Project stakeholders using a responsibility matrix format
- risk management procedures
- project data management, format, and distribution/filing requirements of project correspondence and documentation
- cost management and accounting procedures
- drawing and specification preparation including numbering, revision tracking, and transmittal procedures
- document control procedures
- equipment and materials procurement procedures
- project scheduling requirements, tools, formats, and frequency of delivery
- project accounting methods including 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

- project close-out and hand-over procedures
- other administrative matters and issues specific to the Project for use by the Team.

Project Scheduling and Progress Reporting

The overall project schedule (the Schedule) identifies the critical sequences and target milestone dates that need to be managed in order for the Project to be executed successfully. While executive-level reports will provide an overview of project 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, field management staff, and the Owner.

As detailed in Table 24.1, the 25-month project construction duration assumes that field activities will commence in July 2014, and Production Start will occur in July 2016.

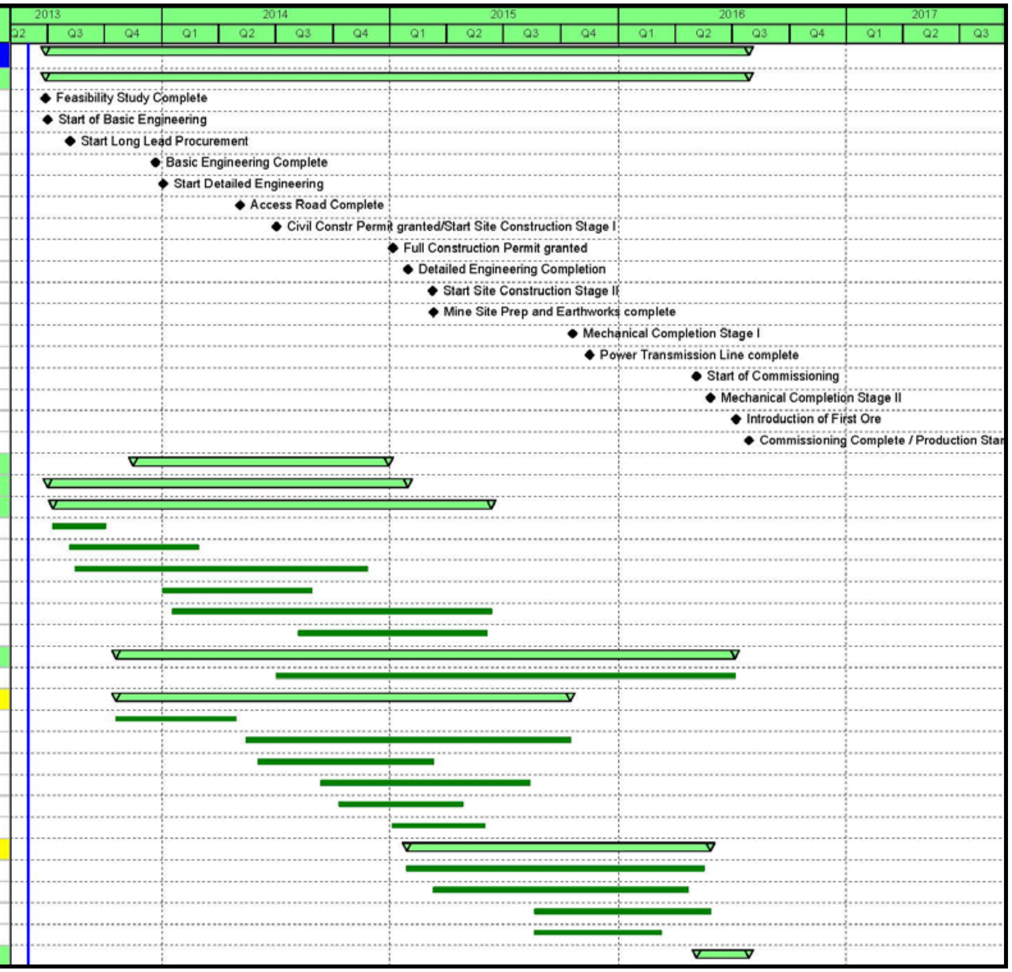
Table 24.1 Significant Activity Milestone Dates to Project Handover

Year	Quarter	Activity
2013	2	Feasibility Study Completion
2013	3	Start of Basic Engineering
2014	1	EPCM Award
2015	1	Detailed Engineering Completion
2014	3	Start of Stage I Early Infrastructure Construction Works
2015	1	Start of Stage II Mine Site Surface Construction
2015	4	Mechanical Completion Stage I Works
2016	2	Mechanical Completion Stage II Works
2016	3	Underground Development Completion
2016	3	Mine Site Commissioning Completion
2016	3	Production Start

A Level 1 execution schedule is provided in Figure 24.2. A Level 2 schedule was developed during the feasibility study stage.

Figure 24.2 Level 1 Execution Schedule

Activity ID	Activity Name	Orgn Dur	Start	Finish
Brucejack EPCM Schedule Level 1		752	28-Jun-13	28-Jul-16
Major Milestones		752	28-Jun-13	28-Jul-16
ML1000	Feasibility Study Complete	0		28-Jun-13
ML1100	Start of Basic Engineering	0	02-Jul-13	
ML1500	Start Long Lead Procurement	0	07-Aug-13	
ML2600	Basic Engineering Complete	0		20-Dec-13
ML2000	Start Detailed Engineering	0	02-Jan-14	
ML1900	Access Road Complete	0		05-May-14
ML2700	Civil Constr Permit granted/Start Site Construction Stage I	0	02-Jul-14	
ML1300	Full Construction Permit granted	0	05-Jan-15	
ML1200	Detailed Engineering Completion	0		28-Jan-15
ML2400	Start Site Construction Stage II	0	09-Mar-15	
ML2500	Mine Site Prep and Earthworks complete	0		11-Mar-15
ML2300	Mechanical Completion Stage I	0		20-Oct-15
ML2100	Power Transmission Line complete	0		15-Nov-15
ML1600	Start of Commissioning	0	06-May-16	
ML1400	Mechanical Completion Stage II	0		27-May-16
ML2200	Introduction of First Ore	0	08-Jul-16'	
ML1700	Commissioning Complete / Production Start	0		28-Jul-16
Permitting, Associated Studies and Licenses		409	16-Nov-13	29-Dec-14
Engineering		381	02-Jul-13	28-Jan-15
Procurement & Contracts		703	09-Jul-13	11-Jun-15
KO1300	Access Road Contracts	86	09-Jul-13	02-Oct-13
PO1000a	Procurement for Major Equipment (Basic Engineering)	205	07-Aug-13	27-Feb-14
KO1200	Design Supply & Construction Contracts	469	14-Aug-13	25-Nov-14
KO1000	Contracts - Early Construction Works	239	02-Jan-14	28-Aug-14
PO1000	Procurement for Major Equipment	511	17-Jan-14	11-Jun-15
KO1100	Major Construction Contracts	302	07-Aug-14	04-Jun-15
Construction		661	18-Oct-13	06-Jul-16
CN1600a	Underground Development	690	02-Jul-14	06-Jul-16
Phase 1 - Early Works		485	18-Oct-13	16-Oct-15
CN1000	Off site Access Road Upgrade	175	18-Oct-13	28-Apr-14
CN1900a	Power Transmission Line	492	15-May-14	16-Oct-15
CN1000b	Early Site Construction Works (incl mobilization)	237	04-Jun-14	11-Mar-15
CN2100	Knipple Transfer Station	311	13-Sep-14	11-Aug-15
CN1200	Temporary Facilities Development	179	11-Oct-14	26-Apr-15
CN1100	Initial Site Development	144	05-Jan-15	01-Jun-15
Phase 2 - Full Project Execution		459	28-Jan-15	27-May-16
CN1700	Process Plant Construction	449	28-Jan-15	16-May-16
CN1500	Site Power Distribution	383	12-Mar-15	22-Apr-16
CN1800	Ancillary Buildings Construction	261	20-Aug-15	27-May-16
CN2000	Paste Plant Installation	185	20-Aug-15	09-Mar-16
Commissioning		84	06-May-16	28-Jul-16



Once the Project has been approved to proceed, the following basic project tasks must be completed as early as possible in order to guarantee planning certainty and maintain a proper monitoring program for all long-lead items and engineering deliverables:

- Continue basic engineering to support permitting and Stage I activities.
- Select the general EPCM contractor, the mine design firm, and the mining contractor.
- Establish project procedures and standard forms.
- Establish the cost reporting system based on the approved capital cost estimate.
- Prepare procurement and contract documents to support Stage I activities.
- Order long delivery and early engineering information for capital equipment.
- Confirm that the Schedule coincides with the Project's actual start date.

24.1.4 ENGINEERING

The engineering groups will establish a list of drawings and specifications for both capital equipment and construction. These represent the "engineering deliverables" that are needed to construct the facilities, order the capital equipment and bulk materials, and control and commission the new plant.

The engineering groups will be responsible for identifying and scheduling their deliverables in accordance with the Schedule. Engineering progress will be based on budgeted versus actual man-hours, combined with the knowledge-based experience of the design team leaders.

24.1.5 PROCUREMENT AND CONTRACTS

PURCHASING AND EXPEDITING STRATEGY

The standards for purchasing will be determined during the initial stages of project establishment. The EPCM and Mining contractors will prepare the purchase requisition on behalf of the Owner, and the equipment will be purchased through the Owners' purchase order administered by the EPCM purchasing group

The EPCM purchasing group will provide capital equipment procurement, vendor drawing expediting and, when required, equipment inspection. The procurement department will package the technical and commercial documentation, and will manage the bidding cycle for equipment and materials to be supplied to the contractors by the Owner. Standard procurement terms and conditions that have been approved for the Project will be utilized for all equipment and materials purchase orders. Suppliers will be selected based on location, quality, price, delivery, and support services.

Capital equipment that will be purchased for the Project includes all equipment shown with an equipment number on the Project flowsheets and included on the equipment list.

Capital equipment purchases will be managed by the engineer-of-record for that package of work. The engineer will develop the equipment specification, solicit prices, prepare the technical and commercial analysis, and provide the Owner with the recommendation for purchase. Once approved for purchase, the engineer will issue the purchase order on behalf of the Owner and then continue to expedite the vendor drawings in order to complete the design drawings to an "issued for construction" level.

The EPCM contractor will purchase bulk materials such as piping, electrical cables, cable trays, and hi-bay lighting on behalf of the Owner, based on the bills of quantities provided by the engineer. The remaining materials will be purchased by the construction contractors.

The EPCM contractor will assemble contract tendering documents, establish qualified bid lists, tender the work, analyze and recommend contractors to the Owner, and prepare the executed contracts for issue.

A field procurement manager will be responsible for supporting ongoing construction needs for miscellaneous materials and services to be provided by the Owner. The EPCM contractor will provide expediting services and will also be responsible for the receipt, storage, and disbursement of purchased materials and equipment at the job site.

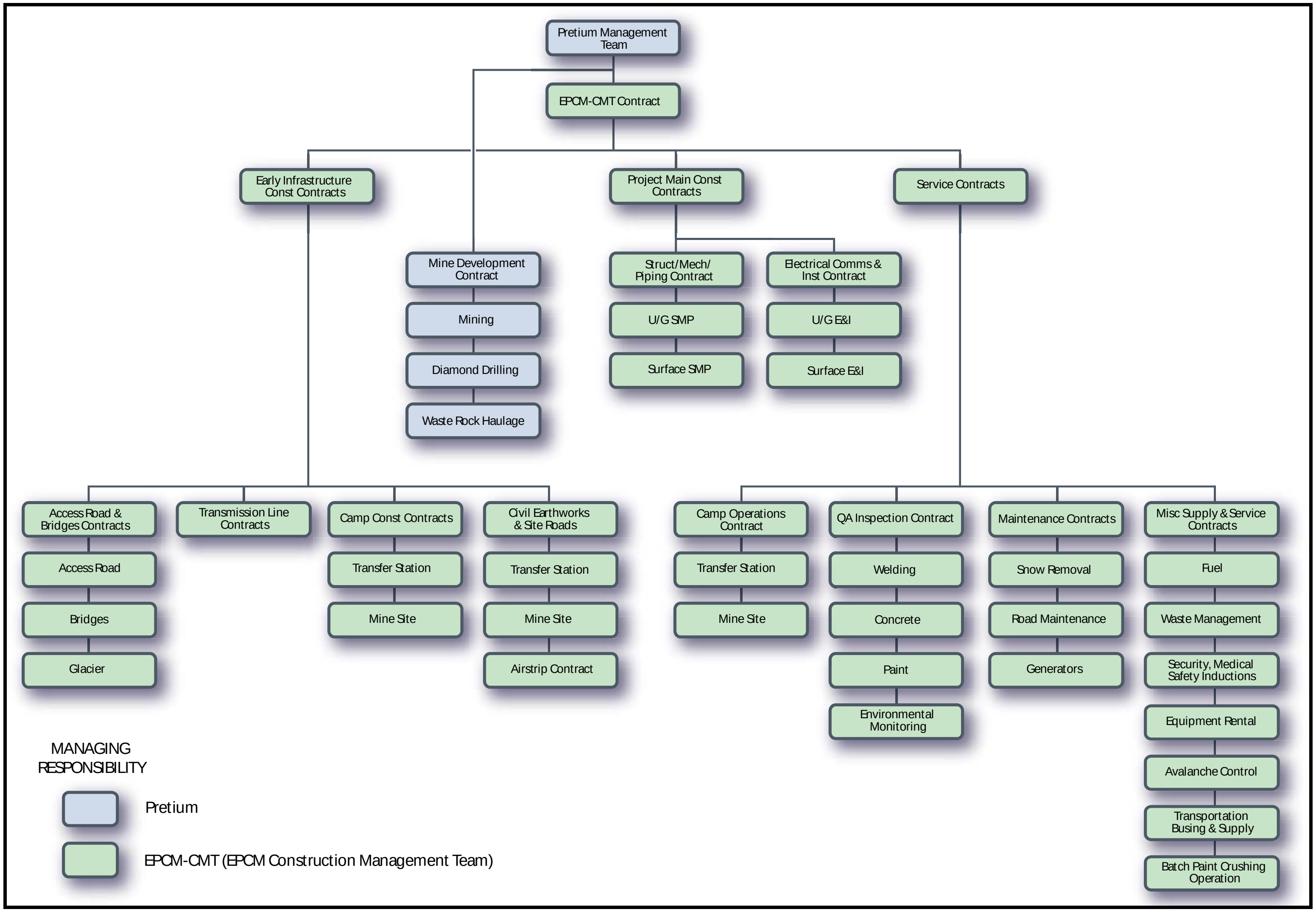
The EPCM contractor will prepare a plan for expediting equipment purchase orders based on the schedule and equipment list. Expediting will be coordinated by the EPCM contractor. Third-party resources may be used to inspect equipment being manufactured in various parts of the world. Purchase orders will be expedited based on complexity, manufacturing cycle time, and schedule criticality. Expediting Reports will be entered into the material control reporting system after each contact with suppliers.

Purchase orders will be used for the receipt of equipment at site and to provide support for vendor invoices. All invoices will be processed through the site-based project team; invoices will be matched against goods received and then cost coded and processed for payment by the Owner.

Construction Contract Strategy

Figure 24.3 illustrates a preliminary contracting structure.

Figure 24.3 Preliminary Contracting Structure



Construction work will be split into a number of construction contracts, determined by:

- the availability of detailed engineered information
- the availability of resources
- cost advantages
- scheduling issues
- permits and approvals
- cash flow.

24.1.6 Construction Labour Requirement

The construction contracting strategy and feasibility study cost estimate are based on a "managed open shop" construction program, which is not preferentially union or non-union. This approach takes advantage of the vast pool of skills available from all types of union and non-union shops, and allows for the use of local labour sources as well as contractors from anywhere in BC or Canada. The Schedule is based on a 60-hour work week, with some double shifts as required. Crew rotations are planned to be scheduled as three weeks on-site and one week off-site.

There are approximately 836,400 man-hours of total construction labour associated with Project construction, excluding mine pre-development and engineering. Construction manpower on site will peak at approximately 300 total construction workers.

In addition to the total construction labour, there will be approximately 100,700 man-hours of indirect labour associated with construction management, construction camp expansion, site survey, quality assurance, geotechnical support, start-up, and commissioning (including vendor assistance). Owner's hours have not been included in this estimate.

24.1.7 Construction Camp

A 330-bed camp will be constructed at the mine site to supplement an existing 120 bed camp. A 30-bed camp will be constructed at the transfer station at Knipple Lake. Full-service modular, propane-heated camps will be used for construction contractors; these camps will then be refurbished for operational use upon completion of construction. The construction camps will be built during Stage I; the existing exploration camp facilities will be utilized by both the EPCM and mining contractors during construction of the permanent camp facilities.

The mine site camp will be a multi-storey structure with a combination of single and multi-person dormitories, and complete with recreational facilities and a commissary. The transfer station camp is designed for short stays in the event of adverse weather conditions that prevent the transportation of personnel to the mine site. The camps will be designated as dry camps (i.e. no alcohol or non-prescription drugs); firearms will also be prohibited.

Construction crews working on the high-voltage power line to site and the access road upgrades will provide their own mobile camps, which will be situated in convenient locations along the route of their work. Manpower levels for these crews are not included in the construction camp sizing.

The EPCM contractor's CMT will manage the camp and catering contractor to ensure that quality service is provided in areas such as hygiene, food storage and handling, menus, nutrition, and staff qualifications.

24.1.8 Housekeeping and Hazardous Waste Management

Specific procedures will be implemented for waste management and spill response during the construction period. These procedures will be defined in the Project procedures and include compliance, auditing, and reporting requirements. Procedures will be established regarding ongoing clean-up and rubbish removal as well as the safe handling, storage, and disposal of batteries, fuels, oil, and hazardous materials during the construction phase. Waste will be recycled to the extent feasible. Ongoing dust suppression and rainwater management programs will also be established and observed for the duration of the construction phase. Specific procedures and storage areas will be designated for construction waste prior to recycling or removal from the site. Solid waste will be recycled or disposed off-site.

24.1.9 Construction Equipment

The supply, maintenance, and operation of construction equipment will be the responsibility of the individual construction contractors. No cranes are allowed to operate on-site without recent inspections. All heavy lifts shall be certified by rigging engineers. Any modifications to equipment have to be certified fit for operation, especially where welding is concerned.

The Owner will supply the large construction cranes to be managed by the CMT.

24.1.10 Communication

The Owner will determine the appropriate temporary (for construction) and permanent microwave telecommunications technologies for the Project, with input from the EPCM 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 during Stage I. The system will be supplemented with the installation of telephones in common areas and for individual room Internet access.

24.1.11 Construction Power

Power for construction will be provided by the contractors' modular generators. The transfer station will be powered by modular generators; both construction camps will be

provided with dedicated emergency power, which will be supplied by low-noise, low-emission temporary generator sets. Permanent power will be available by Q3 2015 for the commissioning phase of the Project.

24.1.12 Mechanical Completion

Mechanical completion is defined as the point when a contractor is considered to have completed his work such that commissioning activities may be initiated and that the Owner may operate the facility in a safe manner. The facility may not be completely finished at that time, and it could be only the systems within an overall project that are complete (e.g. a building or fresh water system, etc.). Mechanical completion is often descriptive of substantial completion, at which time the contractor, the CMT, and the Owner develop a full punch list of the remaining deficiencies, which is then used to measure progress to final completion.

Each process system or ancillary facility will be checked for compliance with 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 tested without any load in order 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 is a prelude to commissioning the overall plant. By this time, the Owner's operating personnel have completed a deficiency list for all equipment and facilities, and the CMT has worked with the contractors to ensure that the required state of completion has been reached. Critical utility features will have been completed before mechanical completion, such that:

- water is available for hydro testing of piping and tanks
- air is available to test the pneumatics
- permanent power is available to test the motors.

24.1.13 Commissioning

The sequence of system commissioning is vital to shifting the construction schedule from a general area completion to a more specific system completion that will enable commissioning and start-up of the entire facility. System identification and prioritization must be expedited to allow for any construction schedule adjustments and completion of the work, in order to satisfy the established commissioning sequence.

The commissioning sequence and plan will be developed at the start of the Project, and the Team will execute this plan during the latter part of construction. All systems will be identified and scheduled for commissioning by priority. Packages will be assembled for each system that will be commissioned, which will include all sign-off and test documentation, drawings, and vendor information.

The various completed systems will be transferred to the Owner's operations team, once the CMT has determined that these systems are free of deficiencies that would prevent safe operation. The Owner's team will consist of plant operators and maintenance staff who will enlist the help of vendors, contractors, and construction management personnel as needed to "dry-run" and then "wet–run" the systems until they are accepted by the Owner's operations management. The transfer of systems will be formally documented and will include all mechanical/electrical testing documents and vendors' information.

24.1.14 CONSTRUCTION METHODS

MINE DEVELOPMENT

Mine development methods are discussed in Sections 16.3 and 16.4.

CONSTRUCTION MANAGEMENT KEY OBJECTIVES

The key objectives for construction management are described as follows:

- 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.
- Implement the contracting and construction infrastructure strategies to support the project execution plan.
- 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. The EPCM contractor will be responsible for evaluating costs on an ongoing basis, and will provide comparisons of budgeted and actual project trending for the cost report on a monthly basis.
- Establish a field contract administration system to effectively manage, control, and coordinate the work performed by the contractors.
- Manage the catering contractor (by contract) to ensure that services meet the expected quality standards for the facilities, staff qualifications, hygiene, food handling, storage, and provision of meals.
- Apply an effective field constructability program as a continuation of the constructability reviews performed in the design office.
- Organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, tender the work, analyze and make recommendations to the Owner for the most suitably qualified contractors, and prepare the executed contracts for issue.
- Receive, inspect, and log all incoming materials, assign storage locations, and maintain a database of the status of all materials received and dispensed to the contractors. Ongoing reconciliation with the procurement system (including

reconciliation to the freight consolidation point) will confirm that the materials ordered for the Project were correctly received, and that the suppliers were paid. An allowance has been included in the construction budget for lease or purchase of offloading equipment, temporary structures, and other equipment required during construction.

- Develop a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.

CONSTRUCTION MANAGEMENT RESPONSIBILITIES

The construction management group will be responsible for the management of all field operations. The construction manager will be responsible to the Owner to effectively plan, organize, and manage construction quality, safety, budget, and schedule objectives.

The EPCM construction management field engineering team will employ independent quality assurance specialists to ensure the implementation and success of the contractor's quality control.

Detailed construction management responsibilities include, but are not limited to:

- project management:
  - camp management
  - camp catering and housekeeping
  - camp installation
  - insurance, WCB, general liability, third-party and auto
  - labour relations plan and site work rules–supported by Owner
  - freight logistics and deliveries
  - overall project cost control, monitoring and reporting system
  - scheduling
  - site offices
  - site topographical survey
  - site utilities for field offices
- design:
  - concrete batch plant requirements
  - commissioning–assisted by engineering and Owner
  - communications system for construction
  - document control–general project and construction
  - constructability reviews–with support from engineering and Owner
- purchasing and expediting:
  - equipment and consumables inventory management–construction and commissioning

   - spare parts inventory management–start-up and commissioning
   - vendor representatives–erection support and commissioning
- construction:
   - HSE policy implementation and enforcement
   - site construction management
   - warehouse and laydown area
   - security personnel (by contract)
   - contracting plan
   - contract bid documents
   - contract tendering
   - contract execution and administration
   - earthworks and civil site supervision
   - mechanical and piping site supervision
   - structural site supervision
   - electrical and instrumentation site supervision
   - commissioning–assist Owner, and engineering and procurement
   - on-site monitoring of construction equipment condition and safe operating capability
   - survey and layout (by contract)
   - site quality control (by contract)
   - cost reporting and controls–with engineering and procurement, and Owner support
   - as-built drawings (by contractors).

CONSTRUCTABILITY

A constructability review for surface and underground facilities was carried out during the feasibility study. The recommendations from this review will be implemented during the execution phase of the Project. Additional constructability reviews will be carried during the detailed design and construction phases of the Project.

CONSTRUCTION EQUIPMENT SUPPLY PHILOSOPHY

Generally, the construction contractors will be responsible for the supply of all equipment required for construction. However, the Owner will provide the use of their Snowcat and Husky fleet for the transportation of personnel and equipment across the Knipple Glacier. The EPCM Contractor will manage the scheduling of the Snowcat and Husky fleet to ensure the most efficient use of the equipment.

Owner’s equipment intended for miscellaneous purposes, such as snow removal, road grading, ditching, etc., will not be available for use by the construction contractors.

PROJECT TEAM RESPONSIBILITIES

It is essential for each team member to understand the relationships and responsibilities of the other team members during the engineering and construction phase of the Project. The overall responsibility matrix and the Project organization chart are key documents that provide an overview of the Project's communication and management structure, and indicate areas of responsibility.

The Project organization chart is illustrated in Figure 24.1 and the project responsibility matrix by WBS is summarized in Table 24.2.

Table 24.2 Project Responsibility Matrix

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
11	**Mine Site**				
111	General Development				
111100	Bulk Earthworks/Site Preparation		X		
111200	Site Roads		X		
111300	Helicopter Pad		X		
111400	Site Drainage		X		
111500	Fencing/Gates (Site Control)		X		
111600	Control System		X		
111700	Communication and Security System		X		
111800	Fire Alarm System		X		
111900	Yard Lighting		X		
21	**Mine Underground**				
211100	Lateral Development (Capital)			X	
211200	Vertical Development (Capital)			X	
211300	Infrastructure Development (Capital)			X	
211400	Ventilation Shafts			X	
212	Underground Mining Equipment				
212100	Mobile Equipment			X	
212200	Fixed Equipment			X	
212300	Ancillary Equipment (Survey, etc.)			X	
212400	Mine Rescue Team Equipment			X	
213	Underground Infrastructure				
213050	Underground Electrical Sub and Distribution			X	
213100	Underground Pump Station			X	
213150	Underground Workshop (and/or Service Bay)			X	
213200	Underground Explosive Magazine			X	
213250	Mobile Refuge Chambers			X	
213300	Underground Backfill Distribution			X	
213350	Portal Infrastructure			X	
213400	Controls and Instrumentation			X	

table continues...

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
213425	Primary Crushing		S	X	
213450	Primary Ventilation Fans (Main RAR Fans)			X	
213500	Secondary Vent Fans (Development Fans)			X	
213550	Mine Heating (Main FAR/FAW)			X	
213600	Backfill Piping and Distribution (Underground)			X	
213650	Washrooms			X	
213700	Canteen facilities			X	
213750	Safety Equipment, Lamps and Storage			X	
213800	Surface Works, Ventilation Fans, etc.		X	S	
213850	Backfill Plant		X	S	
213900	Backfill Piping and Distribution (Surface)		X	S	
213950	Underground Piping			X	
213960	Underground Fuel Storage and Distribution			X	
31	**Mine Site Process**				
311	Crushing				
311100	ROM Storage Emergency Stock Pile		S	X	
311300	Transfer Tower		X	S	
311400	Mill Feed Surge Bin		X	S	
312	Grinding, Flotation and Dewatering				
312100	Mill Building		X		
312200	Primary Grinding		X		
312220	Pebble Crushing		X		
312225	Concentrate Regrinding		X		
312250	Process Reagents Preparation/Storage		X		
312300	Flotation		X		
312400	Concentrate Dewatering		X		
312500	Concentrate Storage and Load Out		X		
312600	Tailings Dewatering		X		
312700	Smelting and Gold Room		X		
32	**Mine Site Utilities**				
321	Utilities – Power and Electrical				
321100	HV Switchyard and Substation		X		
321200	Power Distribution		X		
321300	Emergency Power		X		
321400	Lightning Protection		X		
321500	Area Lighting		X		
322	Utilities –Fuel, Storage and Distribution				
322100	Propane		X		
322200	Diesel		X		
322300	Gasoline		X		
322400	Lube Oil		X		

table continues…

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
322500	Waste Oil Disposal		X		
323	Utilities – Water Systems		X		
323100	Water Distribution System		X		
323200	Potable Water		X		
323300	Process Water		X		
323400	Fire Water		X		
323500	Site Drainage		X		
323600	Water Treatment		X		
323700	Water Management at Brucejack Lake		X		
323800	Gland Water		X		
323900	Glycol Piping		X		
324	Utilities – Waste Disposal				
324100	Solid Waste Disposal		X		
324200	Sewage – STP		X		
324300	Waste Oil Disposal		X		
324400	Incinerator		X		
325	Utilities - Other				
325100	Plant and Instrument Air		X		
33	**Mine Site Buildings**				
331	Ancillary Buildings				
331100	Administration and Mine Dry		X		
331200	Warehouse		X		
331250	Cold Storage		X		
331300	Permanent Camps		X		
331400	Gatehouse and Weigh Scale		X		
331500	Emergency Vehicles and Medical Clinic		X		
331600	Assay/Met/Environmental Lab		X		
331700	Truck Shop and Maintenance Shop		X		
331750	Paste Plant		X		
331950	Truck Wash/Tire Wash		X		
34	**Mine Site Tailings (Brucejack Lake)**				
341	Tailings				
341100	Tailings Delivery System		X		S
341200	Deposition		S		X
341300	Waste Rock Disposal		X		
341400	Reclaim Water Barge		X		
35	**Mine Site Temporary Facilities**				
351	Temporary Facilities				
351100	Construction Camp		X		
351200	Construction Catering		X		
351300	Construction Laydown Area		X		

table continues...

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
36	**Mine Site (Surface) Mobile Equipment**				
361	Surface Mobile Equipment				
361100	Surface Mobile Equipment		X		
61	**Off-site Infrastructure**				
611	Off-site Infrastructure				
611100	Off-site Power Transmission		X		
611200	Off-site Access Roads		X		
611300	Glacier Road and Transfer Station		X		
611400	Avalanche Control	X	S		
91	**Indirects**				
911	Indirects – Mine Site				
911100	Mine Site-Construction Indirects		X		
911200	Mine Site-Initial Fills		X		
911300	Mine Site-Spares		X		
911400	Mine Site-Freight and Logistics		X		
911500	Mine Site-Commissioning and Start-up		X		
911600	Mine Site-EPCM		X		
911800	Mine Site-Vendor Commissioning		X		
98	**Owners Costs**				
981	Owner's Costs	X			
981100	Owner's Costs	X			
981200	Owner's Risk and Contingency	X			
99	**Contingency**				
991	Contingency				
991100	Contingency	X			

Note: X = primary; S = supporting

Plant operation and performance testing will be the responsibility of the plant operations group, and will include operating the facility through the full range of design sizes and capacities specified in the design criteria. Equipment suppliers and process design engineers may be required to provide support to ensure that the optimal performance is achieved for all equipment.

Suppliers will oversee and assist with performance tests, which will be managed and carried out by the Owner's operational personnel.

Selected members of the Project commissioning team may be re-mobilized to assist as required.

24.1.15 Risk Management

A Risk Assessment Workshop was conducted during the Feasibility Study and a Risk Register developed which identified the following high level, medium level and low level risks.

- 49 high level risks
- 135 medium level risks
- 246 low level risks.

An additional four risks were identified but not assessed and required more information for an assessment to be made.

These risks will be further reviewed and mitigated throughout the design, construction, and commissioning stages. The 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.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCE

In November, 2012, an updated Mineral Resource estimate was prepared for VOK and West Zone at the Brucejack Property of Pretivm located near Stewart, BC. The Measured, Indicated and Inferred Mineral Resource estimates, effective November 2012 were intended for use in a feasibility study for a high-grade underground mining scenario.

25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

The test results from all the test programs indicate that the mineralization responded well to the conventional gravity and flotation combined concentration which are widely used to recover gold and silver from this type of ore, especially ones containing significant amounts of nugget gold. In general, the VOK Zone and West Zone mineralization is moderately hard.

The proposed process consists of one stage of crushing (located underground), a SAG and ball mill primary grinding circuit integrated with gravity concentration, rougher flotation, followed by rougher flotation concentrate regrinding and cleaner flotation processes. A gravity concentration circuit is also incorporated in the bulk concentrate regrinding circuit. By melting the gravity concentrate produced from the gravity concentration circuits a gold-silver bearing flotation concentrate and gold-silver doré are produced.

The equipment that has been incorporated into the design is widely operated in the industry.

25.3 MINING METHODS

The Project orebody can be extracted economically through underground mining employing long hole stoping methods in conjunction with paste backfill. Modern trackless mobile equipment will be employed for the majority of mining activities. The ore will be transported to surface by conveyor belt.

A two-year mine development phase is projected. The mineral reserves identified support a mine life of 22 years at 2,700 t/d, with a steady state reached in Year 2 of production operations. There is further mineralization at depth that may add to the total resource available for mining.

There may be opportunities to optimize the underground development design and schedule. Further geotechnical and hydrogeological studies will better define ground control and dewatering requirements.

The study should be advanced to the detail design and construction stage.

25.4 PROJECT INFRASTRUCTURE

25.4.1 AVALANCHE HAZARD ASSESSMENT

An avalanche hazard assessment was completed for the Project. Facilities and access routes are exposed to approximately 15 paths or areas. Avalanche magnitude varies between Size 2 and 4, and frequency varies between annual and 1:100 years. Potential consequences of avalanches reaching the Brucejack mine facilities, transmission line, worksites, and roads include damage to infrastructure, worker injury (or fatality), and project delays. Potential consequences of static snow loads on transmission towers include damage to towers and foundations, and potential loss of electrical service to the mine. Without mitigation to the effects of avalanches and static snow loading, there is a high likelihood of some of these consequences affecting operations on an annual basis.

Avalanche mitigation for the Project includes location planning in order to avoid placement of facilities in avalanche hazard areas. For areas where personnel and infrastructure may be exposed, an avalanche management program will be implemented for mine operations during avalanche season (October through June). The program will utilize an Avalanche Technician team to determine periods of elevated avalanche hazard and provide recommendations for closures of hazard areas. The options for reducing control include explosive control, or waiting for natural settlement. Areas that are expected to have increased frequency of hazard and consequences will be evaluated for the installation of the RACS in order to allow for avalanche explosive control during reduced visibility (darkness and during storms). An allowance has been made in the capital and operating cost estimates for six RACSs.

25.4.2 TRANSMISSION LINE

Valard reviewed potential routes and developed an initial design for the transmission line to the Project site. Based on Valard's considerable experience in construction of transmission lines, as well as the experience gained from the two significant transmission projects very near the Project site, Valard identified a transmission line route from LLH northwards up the Salmon Valley to the Knipple Glacier, and then high on the slopes above the glacier to the Project site. Based on this route, Valard has the following interpretations and conclusions:

- The preferred route for the transmission line is along the moderate slopes on the west side of the Salmon Glacier valley. These slopes have exposed or near surface bedrock along most of the route, along with short and sparse tree cover. No harvesting activities exist in the area.

- Although road access exists to the old Granduc Mine site, the best route for the transmission line is higher on the slopes. No road access exists from the Granduc Mine site to the terminus of the Knipple Glacier. Given the lack of road access along the route, helicopter construction is a means to eliminate the need for road access along the route.
- Helicopter construction and near-surface bedrock are favourable constraints for a steel monopole design for the transmission towers. Such a design will also lengthen the conductor span between towers and eliminate the need for road access, thus reducing the construction costs.
- A review of the site conditions along the route concluded that the subsurface conditions and upslope snow avalanche hazards can be mitigated through detailed design and construction measures, as well as operational requirements.

25.4.3 GEOTECHNICAL

BGC completed geotechnical designs and recommendations for the foundations of the proposed mill building and ancillary buildings based on site investigations in 2011 and 2012 (BGC 2013). Recommended allowable bearing pressures were provided for foundations on bedrock and on structural fill. Based on the foundation loading requirements for the mine site infrastructure, all facilities located within or adjacent to the mill building must be founded on bedrock, whereas lightly loaded ancillary buildings can be founded on either structural fill or bedrock.

A large cut, made primarily in bedrock, will be required to bring the mill building to its design elevation. Geotechnical recommendations have been provided for this cut, along with general recommendations for other excavations and fills required to bring the site to its design elevations.

Additional investigations (e.g. geotechnical drilling and test pit excavations) will be required for subsequent stages of design to further evaluate subsurface conditions within the footprints of the mine site facilities. Laboratory testing of rock and overburden samples will also be required to further evaluate the properties and behaviour of these materials.

25.4.4 WASTE ROCK DISPOSAL

A conceptual layout was developed for the disposal of approximately 2.28 Mm^3 of PAG waste rock in Brucejack Lake (BGC 2013). Waste rock will be end-dumped from haul trucks onto a platform/causeway and then, either a dozer will be used to push it over the side or an excavator will be used to cast it over the side. All PAG waste rock disposed in Brucejack Lake must be placed more than 1 m below the low water elevation of the lake. Construction of the platform/causeway will require NAG material to be advanced out over the submerged PAG waste rock. Therefore, a source of NAG rock will be quarried and stockpiled for this use.

The conceptual layout is preliminary in nature and reflects numerous assumptions. Additional investigations and assessments are required to address assumptions and uncertainties regarding the waste rock's layout and its impact on Brucejack Lake.

25.4.5 Brucejack Lake Suspended Solids Outflow Control

It will be necessary to control the TSS concentrations at the outlet of Brucejack Lake to meet the MMER regulations. The following three possibilities were proposed and discussed, and an allowance has been included in the capital cost estimate:

- washing waste rock before depositing in the lake
- turbidity curtain surrounding tailings deposit and waste rock
- outlet control structure for temporary control of the lake outflow.

Further work regarding the TSS mitigation strategy is required during subsequent stages of design.

25.4.6 Water Management Plan

Contact runoff is expected from three sources during construction and operations:

- the upper laydown area where the waste rock transfer and pre-production ore will be stored
- the mill building and portal site which requires an extensive cut into bedrock, some of which is currently assumed to be potentially acid-generating material
- groundwater seepage to the underground mine tunnels.

Runoff from the former two sources will be managed by storage and treatment. Contact water ponds will be sized to contain runoff from the 24-hour, 10-year return period rainfall event (102 mm). The contained runoff will be pumped to the water treatment plant for treatment prior to release into Brucejack Lake.

The average water requirement for the Brucejack process plant is 3,134 m^3/d based on a mill throughput of 2,700 t/d. This water is required for the tailings slurry to the lake, the underground paste backfill, the concentrate slurry, and minor evaporative losses within the plant (approximately 7 m^3/d). Process water will be sourced from:

- treated underground seepage water
- ore moisture (approximately 3% by weight)
- reclaim from the lake.

Groundwater seepage into the underground workings is expected to vary from approximately 2,140 to 4,080 m^3/d throughout the LOM. Seepage water will be sent to a water treatment plant, and then the process plant, where its use will be maximized in process. With a settled dry density of 1.46 t/m^3 and a slurry consisting of 65% solids by

weight, the paste backfill will exude some water during the curing phase. This additional water is assumed to be pumped out with the seepage water and sent to treatment.

Excess treated groundwater will be used as fluidizing water and discharged to Brucejack Lake at depth. Fluidizing water is required at an average rate of 3,447 m^3/d in order to maintain flow in the discharge line to Brucejack Lake during periods when thickened tailings are used for backfill paste. Reclaim from the lake is also required, as there are periods when the groundwater inflows are predicted to be less than the process requirement.

An average annual outflow of 2,215 m^3/h from Brucejack Lake has been estimated for the life of mine, an average increase of about 5% above existing conditions (2,103 m^3/h). The increase in flow results from the introduction of tailings slurry water and the displacement of water by the deposition of tailings and waste rock. Outflows from Brucejack Lake are assumed to be of suitable water quality for discharge to Brucejack Creek.

25.4.7 Hydrogeological Assessment

A calibrated 3D numerical hydrogeologic model was used to estimate the inflow of groundwater to the proposed underground mine workings at the Project. The rate of groundwater inflow to the underground workings is predicted to remain relatively stable throughout the development of the VOK Zone resource during the first 14 years of mine life, ranging between 2,250 m^3/d and 2,650 m^3/d. The rate of inflow to the underground workings is predicted to increase to a peak of approximately 3,750 m^3/d in Year 15, with the development of the West Zone resource. During Years 16 to 21 of mine life, predicted inflows range between 2,700 m^3/d and 3,000 m^3/d, before decreasing to approximately 2,400 m^3/d in the final two years of mine life.

The inflow estimates are most sensitive to the hydraulic properties of the bedrock represented in the model. Increasing the hydraulic conductivity by a factor of five resulted in the highest inflow estimates, with predicted inflows increasing by a factor of approximately two and average annual inflows of approximately 5,900 m^3/d. The peak inflow associated with the increased hydraulic conductivity scenario is predicted to be approximately 8,900 m^3/d in Year 15 of operations. Decreasing the hydraulic conductivity by a factor of five resulted in a commensurate decrease in inflow (factor of 0.5), with average annual inflows of approximately 1,394 m^3/d. The peak inflow associated with the decreased hydraulic conductivity scenario is predicted to be approximately 2,017 m^3/d.

Using the base case, or best estimate simulation, the water table elevation in the immediate vicinity of the underground workings is drawn down significantly during mine dewatering operations, by up to 120 m in the mine footprint at the end of mine life. The cone of depression associated with this dewatering drawn down has an areal extent of approximately 1.5 km by 2.5 km.

25.4.8 Tailings Delivery System

The tailings placement system for Brucejack Lake has been designed utilizing the best available technology for the confinement of fine particulates to the bottom layer of an impoundment.

25.5 Environmental

Pretivm is committed to operating the mine in a sustainable manner and according to their guiding principles. To this end, Pretivm has been carrying out baseline studies and aboriginal engagement and consultation for several years. Pretivm has good and on-going working relationships with the Skii km Lax Ha, Nisga'a Nation, the Tahltan Nation, as well as other First Nations in the region. Environmental baseline studies have been underway since 2009. Information from the baseline studies have provided a robust understanding of biophysical, social-economic and current land use conditions in the area. Fish do not occur within any immediate receiving environments and the occurrence and distribution of wildlife species of potential concern is well understood.

The goal is to develop the Project such that long-term environmental impacts are minimized. Waste rock and tailings management are being planned to minimize potential for water quality issues in Brucejack Lake and downstream receiving environments with predictive studies on-going. The closure and reclamation plan will focus on closing the adits, removing all site infrastructure, removing the transmission line, and closing the access road including removing the bridges and culverts and revegetation.

Pretivm has formally entered provincial and federal EA processes and are working towards submission of Application for the EAC under the BCEAA and the Environmental Impact Statement under the CEAA in Q4 2013.

25.5.1 Geochemistry

The development of proper waste management strategies requires an assessment of the ARD/ML characteristics of the mine waste. In addition, potential adverse effects on the downstream receiving environment from mine waste need to be assessed by water quality prediction. The two strategies selected for the long-term management of mine waste at the Project are disposal in the underground mine and subaqueous disposal in Brucejack Lake.

Underground disposal involves the storage of waste rock in the underground mine workings. Waste rock and flotation tailings will also be stored in Brucejack Lake. The underwater storage in the (flooded) underground mine and in the lake will prevent oxidation of pyrite and other metal sulphides that may cause ARD.

Considering the ARD/ML characteristics of the mine waste and its potential adverse effect on the downstream receiving environment, the following conclusions can be drawn regarding the selected strategies for mine waste disposal:

- The waste rock from the underground workings is predominantly PAG material that will be stored in Brucejack Lake and in the (flooded) underground mine. According to the established policy for ARD/ML at mine sites in BC, the selected underwater storage of waste rock is one of the preferred strategies to prevent acid rock drainage and metal leaching.
- Results from humidity cell tests indicate that the onset to ARD for the majority of the waste rock could potentially take tens of years before the available NP in the material is consumed. However, for a small proportion of the waste rock the onset of ARD may be within a year.
- The onset to ARD for waste rock stored within the underground mine will likely be delayed until the underground is flooded. This conclusion reflects NP depletion rates of waste rock and potentially the addition of NP afforded by the use of paste backfill (tailings with cement). Further investigation will be required to assess the NP of the paste backfill.
- Results of kinetic tests with subaqueous columns suggest that underwater storage of waste rock maintains neutral pH values in the pore water and the overlying water. Due to the reduced chemical mobility of most metals at neutral to alkaline pH values, the dissolved metal concentrations of most elements will likely remain low.
- Results from the kinetic tests with tailings in humidity cells and subaqueous columns show neutral pH values and low dissolved metal concentrations except dissolved arsenic due to its mobility at neutral pH. The flotation tailings will be NAG. This relates to their non-PAG properties and subaqueous placement.
- With the current assumptions and available information incorporated into the water quality predictions, approximately 10 parameters could exceed CCME guidelines for aquatic life at the Brucejack Lake discharge. These parameters include: dissolved aluminum, dissolved arsenic, dissolved cadmium, dissolved copper, dissolved iron, dissolved mercury, dissolved selenium, dissolved silver and dissolved zinc.

26.0 RECOMENDATIONS

26.1 GEOLOGY

The author makes the following recommendations:

- Complete mining and processing of a 10,000 t representative bulk sample from VOK.
- Continue infill drilling of the Inferred Resources along strike at VOK with the aim to upgrade the classification of the estimates to Indicated Resources.
- Continue to attempt to define high grade resources and their geological controls in the zones outside of VOK and West Zones.
- Continue to refine the geological model with the aim of improving the single integrated geological model.

The budgeted cost for the drilling and associated studies (including the resource updates) is $20 million, whilst the cost of the underground bulk sample is budgeted at $30 million. These costs are in Canadian dollars.

26.2 MINERAL PROCESSING AND METALLURGICAL TESTING

Although the test works indicate good metallurgical response to gravity and flotation concentration, a significant variation in metallurgical performance was observed. Tetra Tech recommends further metallurgical test work to confirm the findings. Pilot plant campaigns on a large size of samples should be conducted. The recommended test work includes:

- Additional metallurgical test work and mineralogical evaluations should be conducted to confirm the metallurgical response of the samples to the established process flowsheet, including locked cycle tests. The samples used for testing should include the samples representing the initial five years mill feeds based on the updated mine plan. The cost of the test work is estimated to be $200,000.
- Pilot plant campaigns to better understand the metallurgical performances. The cost of the test work is estimated to be $500,000.

Further cyanide leach tests should be conducted to improve gold and silver metallurgical performances if cyanidation is considered.

26.3 MARKET STUDIES AND CONTRACTS

Tetra Tech recommends conducting further marketing studies, including shipping concentrate to smelters located in Asia for a potential reduction in the shipping costs.

26.4 MINING METHODS

AMC has identified the following key mining risks and mining opportunities:

- Inferred Mineral Resource blocks in the resource model have been treated as waste and have been assigned zero grades. Stope shapes that incorporate Inferred blocks due to the practical constraints of mining may realize an increase in grade.
- The pre-production development plan is highly focused on establishing infrastructural waste development. Increased ore production during the preproduction phase may be realized through further optimization of scheduling and/or design.
- Notwithstanding the opportunity above, if development advance targets are not achieved during the preproduction phase, and also during the first few years of production, the ramp-up schedule to full production may be compromised.
- The ability to sustain full production in the early years of mine life is dependent on the development of the VOK middle block. The infrastructural waste development program required to begin production from the VOK middle block will be realized in an environment of competing priorities.
- The Brucejack orebody hosts multiple economic lenses in close proximity to each other, which creates both sequential and geotechnical complexity. Definition drilling and detailed modelling must precede mining in a timely manner or the loss of resources and/or an increase in ore dilution may occur.

AMC makes the following mining-related recommendations:

- Average LOM operational costs have been estimated at $157.7/t. The $180/t NSR cut-off grade used in estimating Mineral Reserves would thus give an average 22.3/t operational margin on ore mined. However, it is not currently known whether this cut-off grade is optimal in terms of value, or in terms of other key metrics such as cash flow. AMC recommends further study work on cut-off grade optimization at an estimated cost of $40,000.
- Opportunities to improve the ore production profile in the early years of operation should be investigated through further scheduling exercises at an estimated cost of $30,000.
- A detailed definition drilling program should be developed and scheduled to ensure that mine production targets are fully supported by a level of resource information that is consistent with the complexity of the Brucejack orebody. The estimated cost for this work is $15,000.

- Further test work to identify the benefit of light classification for paste backfill by cycloning is recommended; with a view to remove clays and production of a coarser paste fill PSD to improve the filtering process and give a higher-quality paste fill. The estimated cost for this work is $10,000.
- Further test work using locally available binders, preferably slag based cement, should be undertaken to determine the appropriate cement dosages. Local or imported cement suppliers that can supply high slag based cement as per the annual demand should be contacted. The estimated cost for this work is $35,000.

26.4.1 GEOTECHNICAL

BGC has identified the following key risks and opportunities with regards to the rock mechanics assessment:

- At the time of reporting, a structural geology model for the Project area had not been completely developed. BGC generated a structural model based on historic maps, a limited review of historic drillhole data, and a structural geologic report generated for an adjacent property. BGC assumed that the Brucejack Fault Zone is the only major structure that intercepts the proposed mining footprint. The geotechnical reliability of the underground designs could be improved if further geological modelling work could define the location, orientation, and geotechnical characteristics of major geologic structures.
- The lack of a West Zone geology model and lack of in situ stress data limits the potential of the MAP3D model. Additional refinement of inputs, and calibration to underground observations, are required before quantitative characterization of the rock mass response is possible.
- The stope dimensions provided in this report assume stope-scale geologic structures are present sub-parallel to the stope walls. As additional exploration drillholes are drilled, a structural model is developed for the Property, and particularly as additional underground developments are exposed, the structural database and subsequent structural domains should be reviewed to refine the assumptions inherent in the stope span recommendations. Tighter definition of structural domains may allow less-conservative assumptions with respect to stope-scale structure, with a subsequent increase in recommended maximum stope spans.

BGC makes the following recommendations for additional rock mechanics assessment work:

- Six to eight geotechnical drillholes should be completed, with the associated data used to confirm the geological interpretations and the geotechnical parameters of the rock mass for final design. The program should include packer testing above, below and across/within faults or geologic contacts. Key areas include the Brucejack Fault Zone, the proposed underground crusher excavation, exhaust raise developments, and the West Zone ramp. Laboratory

testing should focus on samples within the VOK D1 domain, the weathered rock zone, and fault zones, as these geotechnical units are under-represented in the current testing database.

- Additional refinements are required for the 3D geological model of the VOK Zone, and a 3D geological model should be developed for the West Zone. Further work should also be completed on the interpretation and modelling of large and intermediate scale faults. The presence of unknown major structures or splays off the Brucejack Fault Zone has the potential to significantly affect rock mass stability. The updated model should be reviewed to determine if updates to the geotechnical assessments are required.
- Numerical stress modeling has identified potential instability zones in stope clusters around the sill pillars and the crown pillar. The model should be updated with the West Zone geology model, in situ stress measurements, and a detailed stope-by-stope extraction plan, all of which are currently unavailable. The updated model should be calibrated using ground deformation observations recorded during the VOK Zone bulk sampling currently scheduled for the summer of 2013. The calibration data can then be used to increase confidence in the modelling results. This will facilitate a more detailed study of the mine sequencing effects on the rock mass stability, including pillars, stope hangingwalls, mine abutments, and excavations through the Brucejack Fault Zone.

The estimated cost for the above-mentioned recommended work is $1,800,000.

26.4.2 Hydrogeological Recommendations

BGC makes the following hydrogeological recommendations:

- Further investigation of hydraulic conductivities (K) in the area of the Brucejack Fault is recommended to support the current distribution of K in the model. The additional packer testing recommended above will be key to this investigation.
- Inflow estimates are sensitive to the hydraulic properties of the project area bedrock as well as recharge applied to the model. Further consideration should be given to, and potentially further investigation made of, glacial contributions to baseflow and groundwater recharge.
- The current model does not include the proposed lake outlet structure. The next phase of modeling should include the effects of this structure, which will include an increase in the lake water level during certain times of year.
- It will be important to continue the collection of hydraulic head data and pumping rate data from underground dewatering operations on a year-round basis at the project site, as these data will be important for future refinement of the conceptual hydrogeologic model and the numerical flow model calibration.

The estimated cost for the above-mentioned recommended work is $200,000. Site investigation activities associated with these recommendations have been included in the underground and surface geotechnical costs.

26.5 PROJECT INFRASTRUCTURE

26.5.1 AVALANCHE HAZARD ASSESSMENT

During the regular avalanche season (October through June), an avalanche management program will be implemented in order to reduce risk to project personnel and infrastructure. The program will include daily hazard and risk assessments by a qualified Avalanche Technician (or team of technicians) to forecast periods of elevated avalanche hazard so that closure of hazard areas can be implemented until hazard is reduced by avalanche explosive control or natural settlement. Avalanche explosive control methods may include hand charging, helicopter explosive control, and pneumatic explosive launchers (avalaunchers). The specific components and capital and operating cost estimate inputs required for the Brucejack avalanche management program are outlined in Table 7-1.

Table 26.1 Brucejack Avalanche Management Program Components

Component	Inputs to Capital Cost Estimate	Inputs to Operating Cost Estimate	Comments
Personal protective equipment	Avalanche transceivers	Maintenance and replacement as required	Numbers of PPE should include enough for all personnel that are working in or transiting avalanche hazard areas
Avalanche rescue equipment	Avalanche rescue caches Mobile avalanche rescue packs	Maintenance and replacement as required	Up to four avalanche rescue caches located in strategic locations
Explosives program	Two explosives magazines Two pneumatic explosive launchers (avalaunchers) mounted on mobile platforms	Explosives and associated materials	Magazines located in strategic locations – most likely one magazine at mine site, and one in Bowser Valley
Remote telemetry weather stations	Two weather stations (Bowser Valley and ridge top location)	Maintenance of all associated equipment	Optimal locations of weather stations to be determined before or during mine start up
Tracked snow vehicles	One tracked vehicle with enclosed cab	Maintenance	Snowmobiles optional

Sections of the access road affected by Paths AR4, AR8 and KG1 are exposed to high frequency events that may have high consequences to traffic. Considering the expected traffic volumes along the access road an allowance has been made in the capital and operating cost estimates for six fixed RACS to be installed in the starting zones of these paths. The RACS facilitate the ability to conduct avalanche control remotely during reduced visibility when helicopters cannot fly (darkness and during storms). Inputs into the capital cost estimate include the equipment and installation. Inputs into the

operating cost estimate include annual maintenance and replacement as required. It is recommended that a trade-off study be done during the next phase of the project to decide on the most appropriate method of control in these areas.

Alpine Solutions also recommends the following:

- The area affected by icefall hazard at Path AR8 should receive constant monitoring throughout the winter, and be regularly controlled using explosives to limit the chance of large icefall events impacting a vehicle.
- The segment of the access road which transits the Knipple Glacier should be re-assessed on a regular basis due to the effects of glacial recession on avalanche runout distance on the glacier.
- During winter, snow berms should be constructed in areas at the mine site affected by short slopes or avalanches to Size 2, in order to reduce the frequency of small avalanches reaching facilities.
- Transmission line structures (towers) should be located away from avalanche paths in order to reduce the requirement for avalanche mitigation. If this is not possible, additional analysis should be completed to determine the most optimal mitigation option. Mitigation may include designing towers for avalanche impact, diversion structures, or earthworks upslope of the tower.
- The final design of the transmission line should involve collaboration with an Avalanche Specialist in order to optimize structure (tower) and conductor locations.
- Construction of the transmission line during avalanche season should include an avalanche management program to reduce risk to personnel and infrastructure.
- Any changes to layout of facilities and roads should be re-assessed for avalanche hazard.

26.5.2 Transmission Line

Valard reviewed the potential for a transmission line to the Project site, as a means to provide electricity for mining operations. In comparing the transmission line to other projects in the area near the Project site, as well as other transmission line projects in the area, Valard has the following recommendations:

- Specialized steel structures should be used to allow for longer spans and limit the number of structures. This will reduce the overall cost of the transmission line, and allow for spanning of many snow avalanche areas along the route.
- An active snow avalanche program should be used to manage the operational risk around snow avalanches to the transmission line. Snow avalanches, particularly on the east side of the Salmon Glacier valley, pose a risk to the transmission line. An active snow avalanche program will likely be needed to control the size and timing of snow avalanches during operations.

- Careful planning as well as detailed design and construction will be required to maximize the relatively short construction season and the use of helicopters for construction of the transmission line. Receiving construction permits late and/or unfavourable weather will significantly affect the construction schedule and limit the ability to construct the line to meet the target in-service date for mine operations. Such planning and design are best carried out by staff who have considerable experience in the planning, design, and construction of transmission lines in the terrain of the BC Northwest.

26.5.3 Geotechnical

Limited or no subsurface information is currently available within the footprints of some of the mine site infrastructure. Therefore, BGC makes the following recommendations for subsequent stages of design:

- Additional investigations (e.g. geotechnical drilling and test pit excavations) to further evaluate subsurface conditions within the footprints of: all the mine site roads and development areas (i.e. the “pads”); all mine site facilities, including the mill building, operations camp, explosive storage, and detonator storage; the Knipple Transfer area; and any borrow sources. Further investigations may also be required if any changes are made to the proposed layout of the mine site infrastructure.
- Laboratory testing of rock and overburden samples to further evaluate the properties and behaviour of these materials.
- Laboratory testing on drill core from within the mill building, portal, and borrow areas to evaluate the potential use of rock in these areas as concrete aggregate and/or construction fill materials. Materials engineering advice should be acquired to guide assessments related to concrete aggregate, including trial mix designs possibly with additives to make use of local aggregates and trial mix designs for lean concrete.

The analyses and design recommendations currently provided for the mine site infrastructure should be updated based on the results of any future investigations. Additional analyses and designs are also recommended to develop detailed design recommendations and specifications. These recommendations should be completed in light of all information available including the results of any further investigations completed.

The estimated cost for the above-mentioned recommended work is $675,000.

26.5.4 Waste Rock Disposal

The conceptual layout for the disposal of waste rock in Brucejack Lake is considered preliminary in nature and reflects numerous assumptions that will require further assessment at subsequent stages of design. BGC makes the following recommendations for future evaluations:

- Samples of the lake bottom sediments should be collected so that laboratory analyses can be completed to assess this material's composition, strength, and consolidation behaviour. Quality sampling of lake bed sediments can be difficult to achieve, though, and may require specialized techniques/equipment. If possible, the thickness of the sediments should also be evaluated through drilling or geophysical methods.
- Stability analyses should be conducted on the waste rock pile. This will require the information on the lake bottom sediments previously recommended.
- Construction procedures for the waste rock pile should be re-evaluated based on the results of the stability analyses if considered necessary.
- Sequencing of the waste rock and tailings deposition should be completed to prevent waste rock from being placed on top of the tailings that will be deposited in the lake.
- The minimum depth that waste rock can be submerged in the lake should be further assessed.
- Reclamation and/or mine closure plans should be developed for the waste rock pile.
- The need for suspended sediment control measures should be evaluated.

The estimated cost for the above-mentioned recommended work is $150,000.

26.5.5 Tailings Delivery System

Rescan recommends the following work for the tailings delivery system:

- Numerical simulations predicting the effect of lake-turnover, wind mixing and flow-through on elevating suspended solids concentrations in the lake discharge could include 3D modelling to fully include the effects of lake bathymetry and horizontal circulation. Results of the simulations will inform the prescription of sediment control methods that may need to be taken. The estimated cost for this work is $50,000.
- A comprehensive survey accuracy bathymetric survey of Brucejack Lake should be completed before the detailed design of the tailings disposal system is finalized. The estimated cost for this work is $50,000.
- Tailings slurry rheology should be determined over a range of solids concentrations from at least as low as 35% solids by weight. The estimated cost for this work is $10,000.
- An attempt should be made to refine the area that can be impacted by avalanches.

26.5.6 BRUCEJACK OUTLET CONTROL STRUCTURE

BGC (2013) completed a scoping level study and provided material take-off quantities for an outlet control structure for Brucejack Lake. BGC recommends completing the ongoing study to evaluate the potential TSS issue, which includes:

- hydrodynamic modelling of the lake to evaluate stratification in the lake and turn over events for various conditions
- analyses of the potential for re-suspension of settled tailings during lake turnover events
- methodology of waste rock deposition in the lake and estimation of anticipated TSS concentrations released into the water column by the waste rock and from disturbance of the lake bottom sediments and potentially tailings during placement of waste rock
- estimation of potential TSS concentrations at the lake outlet based on the above inputs.

If the estimated TSS concentrations at the lake outlet are within the anticipated regulatory limits, no active TSS mitigation measures may be required. If otherwise, further design of appropriate TSS mitigation measures will be required. If, following the additional analyses outlined in the points above, an outlet control structure is deemed a required, practical and preferred method of TSS mitigation, a site investigation, bathymetric survey, geotechnical, structural, hydrotechnical and hydraulic design will be required to bring the proposed outlet structure to an appropriate design level. In addition to the TSS concentrations, the ongoing study to estimate the dissolved water quality should also be evaluated to confirm that parameter concentrations will remain within regulatory limits.

The estimated cost for the above-mentioned recommended work is $400,000.

26.5.7 WASHING WASTE ROCK PRIOR TO DEPOSITION IN BRUCEJACK LAKE

Waste rock deposition in Brucejack Lake presents the possibility of introducing fine suspended solids to the surface water and to the outflow from the lake. It is recommended that development of a washing mechanism to remove fine suspended solids from the waste rock be undertaken. Initial investigations suggest that waste rock could be fed to bar screens with the fine material (less than 1 in) directed to a screw classifier. Overflow from the classifier would be reintroduced to the tailings stream for disposal on the deep lake bed under the deposit of tailings. All other material could be dumped directly to the lake with the knowledge that the settling rates of these coarser solids will ensure that they are trapped in the lake. The estimated cost for this design work is $60,000.

26.5.8 WATER MANAGEMENT PLAN

Water management is considered to be a critical component of the Project design in this high precipitation environment. BGC makes the following recommendations for water management at the next level of design:

- Existing climate and hydrometric stations must continue to be monitored and maintained with an appropriate level of quality control. The data from the climate and hydrometric stations near Brucejack Lake should be reviewed during the next stage of engineering design to confirm assumptions being used for precipitation and runoff.
- The storage volume and pump requirements for the contact water pond should be re-visited during detailed design.
- Mitigations to address potential TSS exceedance during lake turnover need to be investigated in more detail.
- The water balance models and water management strategy need to be refined to account for staging of the various mine facilities. This work should include probabilistic water balance modeling for the EIA process.

The estimated cost for the above-mentioned recommended work is $100,000.

26.5.9 MILL SITE LAYOUT

Tetra Tech recommends completing a trade-off study to assess potential cost savings in reduced earth-work by relocating the mill building further north and raising the pad elevation. Information required to complete the trade-off study include:

- additional geotechnical drilling and test pit excavations to further evaluate subsurface conditions within the relocated mill building footprint
- coordination with mining to review re-alignment of the portals.

26.5.10 CONSTRUCTION CAMP CAPACITY

The estimated construction camp capacity is currently 450 beds. Tetra Tech recommends reviewing the construction manpower requirements in more detail to determine if there is an opportunity to reduce the required capacity during construction.

26.6 ENVIRONMENTAL

Rescan recommends additional work to further develop the Project. This work includes:

- Further water quality modelling of Brucejack Lake to provide direction on closure and post-closure monitoring and mitigation requirements (estimated cost at $50,000).

- Further groundwater quality and quantity assessment in the underground workings at closure is required to inform closure and post-closure monitoring and mitigation requirements.
- The completion of the EA process under the BCEAA and the CEAA 2012 and other permitting requirements including major permits under the *Mines Act* and *Environmental Management Act*, among others (estimated cost at $2 million).
- The development of a more detailed reclamation and closure plan in conjunction with the EA and *Mines Act* permitting. The detailed reclamation and closure plan will require a final site plan and a detailed description of the various facilities, as well as project scheduling to provide assurance to regulators that Pretivm has accounted for closure of the Project in mine project planning (estimated cost at $60,000).
- The continuation with the consultation and the development of a good working relationships with local First Nations including the Skii km Lax Ha and Tahltan First Nation as well as the Nisga'a Nation.

26.6.1 GEOCHEMISTRY

Given the preliminary conclusions and the data available to date, the following recommendations for the continuing study of the geochemistry at Brucejack are made:

- The composition of waste rock (PAG, non-PAG) generated from mining the underground workings will need to be determined once the mine plan has been finalized.
- The final mine plan will have to specify the composition of the run of mine (ROM) waste (tailings and waste rock) destined for (1) the underground mine and (2) Brucejack Lake.
- Ongoing kinetic tests need to be continued to provide better steady state chemistry data that will be scaled up for use in the water quality model to be updated in the detailed design and the EA.
- Ongoing column tests need to be modified to estimate more accurately the pore water chemistry (tailings and waste rock) and the diffusion of soluble constituents into the overlying water column.
- Paste backfill static tests and kinetic tests will need to be conducted to further evaluate the NP contribution of the paste.
- The water quality model will need to be updated with relevant information, as it becomes available.
- Site specific guidelines and a compliance point for the site should be developed to properly evaluate the impact mine operations will have on water quality.
- Nitrogen species should be incorporated into the model once a blasting schedule is finalized.

- The results of the hydrodynamic lake modelling (being completed by Lorax Environmental) will need to be reviewed to assess the potential for elevated TSS concentrations at the outlet of Brucejack Lake resulting from the deposition of waste rock and tailings in the lake.
- Total metal estimates should be incorporated into the model once TSS estimates have been made.

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28.0 CERTIFICATES OF QUALIFIED PERSONS

28.1 DAVID IRELAND, C.ENG., P.ENG.

I, David Ireland, C.Eng., P.Eng., of Richmond, British Columbia, do hereby certify:

- I am a Senior Project Manager with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the “Technical Report”).
- I am a graduate of John Moores University in Liverpool, UK (B.Sc. Mechanical Engineering (Hon.), 1977). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #23419. My relevant experience is consulting engineering and project management for more than 30 years. I have been involved on projects throughout Canada, the UK, the US and Australia including the successful development of two major mine projects with capital expenditures exceeding $2.5 billion and a port selection study for an iron ore marine export facility with capital expenditures exceeding $4.5 billion. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was August 7, 2012 for one day.
- I am responsible for Sections 1.1, 1.8, 1.13, 1.14, 2.0, 3.0, 18.1, 18.3, 18.8, 18.9, 18.10, 18.13, 18.17.2, 18.7.3, 24.0, 25.4.5, 26.5.6, 26.5.9, 26.5.10, and 28.1 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
David Ireland, C.Eng., P.Eng.

David Ireland, C.Eng., P.Eng.
Senior Project Manager
Tetra Tech WEI Inc.

28.2 IVOR W.O. JONES, M.SC., CP, FAUSIMM

I, Ivor W.O. Jones, M.Sc., CP, FAusIMM, of West Perth, Australia, do hereby certify:

- I am an Executive Consultant with Snowden Mining Industry Consultants Inc. with a business address at 87 Colin Street, West Perth, Western Australia, 6005.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of Macquarie University (Honours Degree in Bachelor of Science in Geology, 1986) and the University of Queensland (Master of Science degree in resource estimation, 2001). I am a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy (AusIMM). I have worked as a geologist continuously for a total of 25 years since graduation. I have been involved in resource evaluation for 20 years and consulting for 15 years, including resource estimation of primary gold deposits for at least 5 years. I have been involved in gold exploration and mining operations for at least 5 years. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was June 3 to 8, 2013. A prior visit to the property was completed on February 15 and 16, 2012 for two days.
- I am responsible for the preparation of Sections 1.2, 1.3, 1.4, 4.0 to 12.0, 14, 23, 25.1, 26.1, 27.11, and 28.2 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have had prior involvement with the Property that is the subject of the Technical Report. I have completed three prior technical reports dated April 30, 2012, September 18, 2012 and November 20, 2012. I have also reviewed a technical review prepared by Dr. W. Board of Snowden in 2010 and additional grade modelling work by Dr. Board during 2011.
- I have read the Instrument and the sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at West Perth, Australia

Original document signed and sealed by
Ivor W.O. Jones, M.Sc., CP, FAusIMM

Ivor W.O. Jones, M.Sc., CP, FAusIMM
Executive Consultant
Snowden Mining Industry Consultants

28.3 JOHN HUANG, PH.D., P.ENG.

I, Jianhui (John) Huang, Ph.D., P.Eng., of Burnaby, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of North-East University (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals (M.Eng., 1988), and Birmingham University (Ph.D., 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #30898. My relevant experience with respect to mineral engineering includes more than 30 years of involvement in mineral process for base metal ores, gold and silver ores, and rare metal ores. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was August 7, 2012 for one day.
- I am responsible for Sections 1.5, 1.11, 13.0, 17.0, 19.0, 21.2 (except 21.2.2), 25.2, 26.2, 26.3, 27.12, and 28.3 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Jianhui (John) Huang, Ph.D., P.Eng.

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech WEI Inc.

28.4 PIERRE PELLETIER, P.ENG.

I, Pierre Pelletier, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am the Division Managing Director of Rescan Environmental Services Ltd., an ERM company with a business address at #600 – 1111 West Hastings Street, Vancouver, British Columbia, V6E 2J3.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of Montana, Montana College of Mineral Science and Technology (Environmental Engineering, 1993). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #27928. My relevant experience is an environmental engineer with 20 years of experience in mining and the environment. Over the last 15 years, I have managed several environmental and social impact assessments. I have also permitted treatment plants and mine closure plans, led due diligences and environmental audits and I have been the "Qualified Person" for environmental and social aspects of several preliminary economic assessments, prefeasibility and feasibility studies. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was August 7, 2012 for one day.
- I am responsible for Sections 1.9, 20.0 (except 20.1.4, 20.1.5, and 20.1.7), 25.5 (except 25.5.1), 26.6 (except 26.6.1), 27.8 (except 27.8.1 and 27.8.2) and 28.4 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Pierre Pelletier, P.Eng.

Pierre Pelletier, P.Eng.
Division Managing Director
Rescan Environmental Services Ltd., an ERM company

28.5 HAMISH WEATHERLY, M.SC., P.GEO.

I, Hamish Weatherly, M.Sc., P.Geo., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Hydrologist with BGC Engineering Inc. with a business address at Suite 800 – 1045 Howe Street, Vancouver, British Columbia, V6Z 2A9.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia, (M.Sc., 1995). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #25567. My relevant experience is 18 years as a consultant specializing in water resources management. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was August 7, 2012 for one day.
- I am responsible for Sections 20.1.3 (the Climate section only), 20.1.7, 25.4.6, 26.5.8, 27.8.1, and 28.5 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the section of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 21st day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Hamish Weatherly, M.Sc., P.Geo.

Hamish Weatherly, M.Sc., P.Geo.
Senior Hydrologist
BGC Engineering Inc.

28.6 HARVEY WAYNE STOYKO, P.ENG.

I, Harvey Wayne Stoyko, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Manager of Estimating with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of Saskatchewan (B.Sc. Mechanical Engineering, 1985). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #17092. My relevant experience with respect to mine development and costing includes over 20 years in mine expansion, capital cost engineering for both green and brownfield construction, planning, costing and execution of mine/concentrate handling facilities including plant, road, rail and port and the preparation of studies. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.10, 21.1, and 28.6 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by Harvey Wayne Stoyko, P.Eng.

Harvey Wayne Stoyko, P.Eng.
Manager of Estimating
Tetra Tech WEI Inc.

28.7 SABRY ABDEL HAFEZ, PH.D., P.ENG.

I, Sabry Abdel Hafez, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (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 Association of Professional Engineers and Geoscientists of British Columbia, License #34975. My relevant experience is in mine evaluation. I have more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from the conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in the technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.12, 22.0, and 28.7 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Sabry Abdel Hafez, Ph.D., P.Eng.

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

28.8 COLM KEOGH, P.ENG.

I, Colm Keogh, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd with a business address at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia (BASc Mining Engineering, 1988). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #37433. My relevant experience is approximately 20 years in the mining industry, specifically underground base metal and precious metal operations in Canada and Europe. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was October 24, 2012 for one day.
- I am responsible for Sections 1.6, 1.7, 15.0, 16.0 (except 16.5, 16.6, 16.9 and 16.10), 21.2.2, 25.3, 26.4 (except 26.4.1 and 26.4.2), 27.9, and 28.8 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Colm Keogh, P.Eng.

Colm Keogh, P.Eng.
Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd

28.9 CATHERINE SCHMID, M.SC., P.ENG.

I, Catherine Schmid, M.Sc., P.Eng., of Kamloops, British Columbia, do hereby certify:

- I am a Senior Geotechnical Engineer with BGC Engineering Inc. with a business address at 234 St. Paul Street, Kamloops, British Columbia, V2C 6G4.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of Queen's University, Master of Science (Engineering), 2005. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, #33195. My relevant experience is 10 years of mining rock mechanics projects, including consulting and operations experience. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was February 2012 for 7 days.
- I am responsible for Sections 16.5, 26.4.1, 27.10, and 28.9 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the section of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Kamloops, British Columbia

Original document signed and sealed by
Catherine Schmid, M.Sc., P.Eng.

Catherine Schmid, M.Sc., P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

28.10 VIRGINIA CULLEN, M.ENG., P.ENG.

I, Virginia Cullen, M.Eng., P.Eng., of Gibsons, British Columbia, do hereby certify:

- I am a Senior Hydrogeological Engineer with BGC Engineering Inc. with a business address at Suite 800 – 1045 Howe Street, Vancouver, British Columbia, V6Z 2A9.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia, (M.Eng., 2005). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #31105. My relevant experience is 12 years as a consultant involved with mine reclamation and groundwater studies. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was July 6, 2012 for 3 days.
- I am responsible for Sections 16.6, 20.1.4, 20.1.5, 25.4.7, 25.5.1, 26.4.2, 26.6.1, 27.8.1, 27.8.2, and 28.10 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the section of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Virginia Cullen, M.Eng., P.Eng.

Virginia Cullen, M.Eng., P.Eng.
Senior Hydrogeological Engineer
BGC Engineering Inc.

28.11 BRENT MCAFEE, P.ENG.

I, Brent McAfee, P.Eng., of Kamloops, British Columbia, do hereby certify:

- I am a Geotechnical Engineer with BGC Engineering Inc. with a business address at 234 St. Paul Street, Kamloops, British Columbia, V2C 6G4.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia, (Bachelor of Applied Science, 2006). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #38494. My relevant experience is 7 years of geotechnical engineering design for mine development projects including the Ajax Project, B.C.; Eagle Gold Project, Yukon; and Donlin Creek Gold Project, Alaska. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was June 6 to 12, 2012 for 7 days.
- I am responsible for Sections 18.2, 18.11, 25.4.3, 25.4.4, 26.5.3, 26.5.4, 27.5, 27.6, and 28.11 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Kamloops, British Columbia.

Original document signed and sealed by
Brent McAfee, P.Eng.

Brent McAfee, P.Eng.
Geotechnical Engineer
BGC Engineering Inc.

28.12 MAUREEN MCGUINNESS, P.ENG.

I, Maureen McGuinness, P.Eng., of Sudbury, Ontario, do hereby certify:

- I am a Senior Process Engineer with Paterson & Cooke Canada Inc. with a business address at 1351-C Kelly Lake Road, Unit #2, Sudbury, Ontario, P3E 5P5.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of McGill University (Bachelors of Engineering (Metallurgical), 1995). I am a member in good standing of the Professional Engineers Ontario, License #100088420 and with l'Ordre des ingenieurs du Québec, License #117139. My relevant experience is 10 years as a metallurgical engineer for Xstrata with experience in commissioning, system expansion and operation of pastefill plants. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 16.10 and 28.12 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Sudbury, Ontario

Original document signed and sealed by
Maureen McGuinness, P.Eng.

Maureen McGuinness, P.Eng.
Senior Process Engineer
Paterson & Cooke Canada Inc.

28.13 MICHAEL CHIN, P.ENG.

I, Michael Chin, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Civil Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of Alberta (Bachelor of Science in Civil Engineering, 1985). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #17172. My relevant experience is 26 years of civil engineering design and construction for mines, power plants, highways, and other heavy civil project. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 18.4, 18.5, 18.7.1, 27.2, and 28.13 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Michael Chin, P.Eng.

Michael Chin, P.Eng.
Civil Engineer
Tetra Tech WEI Inc.

28.14 BRIAN GOULD, P.ENG.

I, Brian Gould, P.Eng., of Squamish, British Columbia, do hereby certify:

- I am a Senior Avalanche Specialist/Engineer with Alpine Solutions Avalanche Services with a business address at PO Box 417, Squamish, British Columbia, V8B 0A4.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia (B.A.Sc. in Civil Engineering, 1992). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License # 31663. My relevant experience is 21 years in the avalanche industry and 9 years as an engineer/planner for avalanche risk control projects. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was April 29, 2013 for two days.
- I am responsible for Sections 1.8.1, 18.6, 25.4.1, 26.5.1, 27.1, and 28.14 the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Squamish, British Columbia

Original document signed and sealed by
Brian Gould, P.Eng.

Brian Gould, P.Eng.
Senior Avalanche Specialist/Engineer
Alpine Solutions Avalanche Services

28.15 MICHAEL PAUL WISE, P.ENG., MBA

I, Michael Paul Wise, P.Eng., MBA, of Vancouver, British Columbia, do hereby certify:

- I am a Director, Project Development Valard LP with a business address at Suite 1790, 999 West Hasting Street, Vancouver, British Columbia, V6C 2W2.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia (B.A.Sc., Geological Engineering, 1989; and M.A.Sc. Civil Engineering, 1996) and Simon Fraser University (Executive MBA, 2007). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #18891. My relevant experience is over 20 years in resource roads and infrastructure projects, including transmission lines, resource roads, forestry activities, and other aspects of linear project development. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was August 5, 2012 for one day.
- I am responsible for Section 1.8.2, 18.7, 25.4.2, 26.5.2, and 28.15 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Michael Paul Wise, P.Eng., MBA

Michael Paul Wise, P.Eng., MBA
Director, Project Development
Valard LP

28.16 PAUL GREISMAN, PH.D., P.ENG.

I, Paul Greisman, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am an Engineering Discipline Manager of Rescan Environmental Services Ltd., an ERM company with a business address at #600 – 1111 West Hastings Street, Vancouver, British Columbia, V6E 2J3.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of The Cooper Union (B.E., 1968), New York University (M.S., 1969) and the University of Washington (Ph.D., 1976). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #13952. My relevant experience over the past 23 years is the design of subaqueous tailings placement systems and their effects on suspended solids concentrations in receiving waters. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I completed a personal inspection of the Property on August 17, 2010 for one day.
- I am responsible for Sections 1.8.3, 18.12, 25.4.8, 26.5.5, 26.5.7, 27.3, and 28.16 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which 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 21st day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Paul Greisman, Ph.D., P.Eng.

Paul Greisman, Ph.D., P.Eng.
Engineering Discipline Manager
Rescan Environmental Services Ltd., an ERM company

28.17 CLAYTON RICHARDS, P.ENG.

I, Clayton Richards, P.Eng., of Thunder Bay, Ontario, do hereby certify:

- I am a Chief Systems and Process Control Engineer with Tetra Tech WEI Inc. with a business address at 725 Hewitson Street, Thunder Bay, Ontario, P7B 6B5.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of Lakehead University (Bachelor of Engineering (Electrical, 1989). I am a member in good standing of the Association of Professional Engineers Ontario, License #90277922. I am an accomplished automation engineer with 23 years of experience in process control and DCS systems for the mining and pulp and paper industries. My experience includes designing, managing and commissioning various DCS and process control projects. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 18.14 and 28.17 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Thunder Bay, Ontario

Original document signed and sealed by
Clayton Richards, P.Eng.

Clayton Richards, P.Eng.
Chief Systems and Process Control Engineer
Tetra Tech WEI Inc.

28.18 WAYNE E. SCOTT, P.ENG.

I, Wayne E. Scott, P.Eng., of Thunder Bay, Ontario, do hereby certify:

- I am a Mining Divisional Manager, Electrical with Tetra Tech WEI Inc. with a business address at 725 Hewitson Street, Thunder Bay, Ontario, P7B 6B5.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of Lakehead University (Bachelor of Engineering (Electrical), 1985). I am a member in good standing of the Association of Professional Engineers Ontario, License #41302506. I am also a member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan, License #16893. My relevant experience is 25 years as an electrical engineer in engineering design, process optimization and mill operations. My expertise includes design and integration of control systems, power system design, LV/MV motor controls and switchgear. I have been the lead electrical engineer on major underground mining projects and acted as the project engineer. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 18.15 and 28.18 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Thunder Bay, Ontario

Original document signed and sealed by
Wayne E. Scott, P.Eng.

Wayne E. Scott, P.Eng.
Mining Divisional Manager, Electrical
Tetra Tech WEI Inc.

28.19 ALI FARAH, P.ENG.

I, Ali Farah, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Lead Mechanical Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of Shiraz University (B.Sc.Eng. in Mechanical Engineering, 1984). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #144443. My relevant experience includes 20 years of experience with hydraulic calculations, equipment design/selection and design of pumping systems. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 18.16 and 28.19 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Ali Farah, P.Eng.

Ali Farah, P.Eng.
Lead Mechanical Engineer
Tetra Tech WEI Inc.

28.20 KEN HALISHEFF, M.ENG., P.ENG.

I, Ken Halisheff, M.Eng., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Geotechnical Engineer with BGC Engineering Inc. with a business address at Suite 800 – 1045 Howe Street, Vancouver, British Columbia, V6Z 2A9.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia (M.Eng., 2001). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #29738. My relevant experience is 12 years as a consultant involved with mine development and operation. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 26.5.6, 27.4, and 28.20 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Ken Halisheff, M.Eng., P.Eng.

Ken Halisheff, M.Eng., P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

28.21 S. (KUMAR) SRISKANDAKUMAR, M.A.SC., P.ENG.

I, S. (Kumar) Sriskandakumar, M.A.Sc., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Geotechnical Engineer with BGC Engineering Inc. with a business address at Suite 800 – 1045 Howe Street, Vancouver, British Columbia, V6Z 2A9.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the “Technical Report”).
- I am a graduate of the University of British Columbia (M.A.Sc., 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #33087. My relevant experience is nine years as a consultant involved with mine development and operation. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 26.5.6, 27.4, and 28.21 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
S. (Kumar) Sriskandakumar, M.A.Sc., P.Eng.

S. (Kumar) Sriskandakumar, M.A.Sc., P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

28.22 MO MOLAVI, P.ENG.

I, Mo Molavi, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd with a business address at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.
- This certificate applies to the technical report entitled Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC, dated June 21, 2013 (the "Technical Report").
- I am a graduate Laurentian University (B. Eng. in Mining Engineering, 1979) and McGill University (M. Eng. in Mining Engineering specializing in Rock Mechanics and Mining Methods, 1987). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #37594 and a member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have worked as a Mining Engineer for a total of 31 years since my graduation from university and have relevant experience in project management, feasibility studies and technical report preparations for mining projects in North America. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 16.9 and 28.22 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the 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 has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of June, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Mo Molavi, P.Eng.

Mo Molavi, P.Eng.
Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd