Appendix 5-A

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

ERM

Report to:

Pretium Resources Inc.

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

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TETRA TECH

Report to:

PRETIUM RESOURCES INC.

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

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Prepared by David Ireland, C.Eng., P.Eng.
Lynn Olssen, MAusIMM (CP)
John Huang, Ph.D., P.Eng.
Pierre Pelletier, P.Eng.
Hamish Weatherly, M.Sc., P.Geo.
Harvey Wayne Stoyko, P.Eng.
Sabry Abdel Hafez, Ph.D., P.Eng.
Colm Keogh, P.Eng.
Catherine Schmid, M.Sc., P.Eng.
Brent McAfee, P.Eng.
Michael Chin, P.Eng.
Brian Gould, P.Eng.
Michael Wise, P.Eng.
Paul Greisman, Ph.D., P.Eng.
Wayne E. Scott, P.Eng.
Ali Farah, P.Eng.
George Zazzi, P.Eng.
Trevor Crozier, M.Eng., P.Eng.
Sharon Blackmore, M.Sc., P.Geo.

DI/jc

TETRA TECH

Suite 800, 555 West Hastings Street, Vancouver, British Columbia, V6B 4N6
Phone: 604.408.3788 Fax: 604.408.3722

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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

abrasion index	Ai
acid base accounting	ABA
acid rock drainage	ARD
acidifying potential	AP
Air Terminal Building	ATB
Alpine Solutions Avalanche Services	Alpine Solutions
ALS Chemex	ALS
ALS Global	ALS
AMC Mining Consultants (Canada) Ltd.	AMC
arsenic	As
Association for the Advancement of Cost Engineering International	AACE
atomic absorption spectroscopy	AAS
atomic absorption	AA
BC Environmental Assessment Act	BCEAA
BGC Engineering Inc.	BGC
Black Hawk Mining Inc.	Black Hawk
Bond ball mill work index	BWi
Bond crushing mill work index	CWi
Bond rod mill work index	RWi
borax	Na_2B4O_2
British Columbia	BC
Canadian development expense	CDE
Canadian Environmental Assessment Act	CEAA
Canadian exploration expense	CEE
Canadian Institute 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
Corona Corporation	Corona
counter current decantation	CCD
cumulative distribution function	CDF
Cumulative Expenditures Account	CEA
cumulative net cash flows	CNCFs
cumulative tax credit account	CTCA

diesel engine exhaust emissions	DEE
digital elevation models	DEM
distributed control system	DCS
drop weight index	DWi
effective grinding length	EGL
Employment Insurance	EI
engineering, procurement, construction management	EPCM
environmental assessment certificate	EAC
Environmental Assessment Office	EAO
Environmental Assessment	EA
environmental impact statement	EIS
Environmental Management System	EMS
Exploration & Metallurgical Testing Inspectorate America Corporation or Metallurgical Division, Inspectorate Exploration and Mining Services Ltd.	Inspectorate
extended gravity recoverable gold and silver	E-GRG
Field Electrical Centre	FEC
FLSmidth Dawson Metallurgical	FLS-DM
FLSmidth Knelson	Knelson
fluorspar	CaF_2
fly-ash	FA
Gekko Systems Pty Ltd.	Gekko
general and administration	G&A
General Purpose	GP
GeoSpark Consulting Inc.	GeoSpark
gold equivalent	AuEq
gold	Au
Granduc Mines Ltd.	Granduc
gravity recoverable gold	GRG
gravity recoverable silver	GRS
gross vehicle weight	GVW
half absolute relative difference	HARD
hazard and operability analysis	HAZOP
Hazen Research Inc.; Inspectorate	Hazen
health, safety, and environmental	HSE
high-density polyethylene	HDPE
humidity cell	HC
Impact Benefit Agreements	IBAs
inductively coupled plasma	ICP
inductively coupled plasma-atomic emission spectroscopy	ICP AES
input/output	I/O
Instrument Approach Procedures	IAP
Instrument Flight Rules	IFR
integrated communications cap lamp	ICCL
internal rate of return	IRR
International Organization for Standardization	ISO
Joe Zhou Mineralogy Ltd	JZM

joint venture	JV
Lacana Mining Corp.	Lacana
Land and Resource Management Plan	LRMP
lead nitrate	$Pb(NO_3)_2$
life-of-mine	LOM
Light Detection and Ranging	LiDAR
load-haul-dumps	LHDs
Local Area Network	LAN
long hole open stoping	LHOS
Long Lake Hydro	LLH
mass spectrometer	MS
Medical Service Plan	MSP
metal leaching	ML
Metal Mining Effluent Regulations	MMER
Meteorological Service of Canada	MSC
methyl isobutyl carbinol	MIBC
Met-Solve Laboratories Inc.	Met-Solve
Mineable Shape Optimizer	MSO
MineCem	MC
Mineral Titles Online	MTO
Ministry of Energy, Mines, and Natural Gas	MEMNG
Ministry of Environment	MOE
motor control centres	MCCs
Multiple Indicator Kriging	MIK
National Instrument 43-101	NI 43-101
net cash flows	NCFs
net neutralization potential	NNP
net present value	NPV
net smelter return	NSR
neutralization potential ratio	NPR
neutralization potential	NP
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
Omni Directional Approach Lighting System	ODALS
P&E Mining Consultants Inc.	P&E
parameters of concern	POCs
Particle Mineral Analysis	PMA
Paterson & Cooke	P&C
personal protective equipment	PPE
Placer Dome Inc.	Placer Dome
Pocock Industrial Inc.	Pocock
potassium amyl xanthate	PAX

potassium permanganate	$KMnO_4$
potentially acid generating	PAG
Precision Approach Path Indicators	PAPI
Pretium Resources Inc.	Pretivm
Process Mineralogical Consulting Ltd.	PMCL
programmable computer	PC
programmable logic controller	PLC
project execution plan	PEP
qualified person	QP
quality assurance/quality control	QA/QC
Quantitative Evaluation of Minerals by Scanning Electron Microscopy	QEMSCAN
Reference Evapotranspiration	REF-ET
Registered Retirement Savings Plans	RRSPs
remote avalanche control system	RACS
right-of-way	ROW
Risk Management Plan	RMP
rock quality designation	RQD
run-of-mine	ROM
Runout Zone	RZ
Runway End Identifier Lights	REIL
SAG mill/ball mill	SAB
scanning electron microscopy	SEM
Science-Based Effects Benchmarks	SBEBs
semi-autogeneous grinding	SAG
SGS Canada	SGS
shake flask extracts	SFEs
silica	SiO_2
Silver Standard Resources Inc.	Silver Standard
silver	Ag
Snowden Mining Industry Consultants Inc.	Snowden
Social and Community Management Systems	SCMS
sodium cyanide	NaCN
sodium nitrate	$NaNO_3$
sodium silicate	Na_2SiO_3
solids liquid separation	SLS
Standards Council of Canada	SCC
sulphur	S
Sunstate Slag Blend	SS
Sustainable Resource Management Plan	SRMP
Teuton Resources Corporation	Teuton
the Brucejack Project	the Project or the Property
Traditional Knowledge/Traditional Use	TK/TU
twenty-foot equivalent units	TEUs
unconfined compressive strength	UCS
underground distribution system	UDS

uninterruptable power supply	UPS
Universal Transverse Mercator	UTM
Valard Construction	Valard
Valley of the Kings	VOK or VOK Zone
vertical shaft impact	VSI
very-high frequency	VHF
Visual Climb Area	VCA
Visual Flight Rules	VFR
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
Volcanic Sedimentary Facies	VSF
volcanogenic massive sulphide	VMS
Wardrop Engineering Inc.	Wardrop
water balance model	WBM
Water Quality Guidelines	WQGs
Water Treatment Plant	WTP
West Zone Fault Zone	WZ FZ
West Zone Fresh Rock	WZ FR
West Zone Weathered Rock Zone	WZ WRZ
West Zone	WZ
work breakdown structure	WBS
Workers' Compensation Board	WCB
x-ray diffraction	XRD

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 18-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 2014, Pretivm commissioned a team of consultants to complete a feasibility study update 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 update:

- 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
- ERM Rescan: environmental studies, permits, and social or community impacts; and tailings delivery system
- BGC Engineering Inc. (BGC): geotechnical design, mine hydrogeological/groundwater; waste disposal; 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.

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, 2025.

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 (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.

Gold (± silver) mineralization is hosted in predominantly sub-vertical vein, vein stockwork, and subordinate vein breccia systems of variable intensity. The vein/stockwork systems display both parallel and discordant relationships to stratigraphy. These systems are relatively continuous along strike (several tens of metres to several hundreds of metres).

Several mineralization zones have been explored to varying degrees including (from south to north): Bridge Zone, Valley of the Kings (VOK or VOK Zone), West Zone, Gossan Hill, Shore Zone, and SG Zone.

High-grade gold mineralization in the VOK, the current focus of the Project, occurs in a series of west-northwest (and subordinate west-southwest) trending sub-vertical corridors of structurally reoriented vein stockworks and vein breccias. Stockwork mineralization displays both discordant and concordant relationships to the volcanic pile stratigraphy. Gold is typically present as gold-rich electrum within deformed quartz-carbonate (±adularia?) vein stockworks, veins, and subordinate vein breccias.

Recent underground exploration carried out as a part of the bulk sample program confirmed the location of corridors of stockwork-style mineralization and the lithological contacts in this part of the deposit (within the VOK).

The VOK deposit is currently defined over 1,200 m in east-west extent, 600 m in north-south extent, and up to 650 m in depth below the topographic surface. The West Zone appears to form the northern limb of an anticline that links up with the VOK in the south, and the southern limb of a syncline that extends further to the north. This zone, which is currently defined over 590 m along its northwest strike, 560 m across strike, and down to 650 m in depth, is open to the northwest, southeast, and at depth to the northeast.

The Brucejack deposit is considered to be a transitional to intermediate sulphidation epithermal stockwork vein system-hosted gold-silver deposit that was developed in a dynamic extensional basin. It is likely associated with a deeper porphyry system that developed within an active island arc tectonic setting.

1.4 Mineral Resource Estimates

In August 2013, Snowden was engaged by Pretivm to complete an update of the Mineral Resource estimate for the VOK Zone at the Project in compliance with NI 43-101 and Form 43-101F1. In addition, the West Zone estimate created as part of the April 2012 Mineral Resource (Jones 2012a) has been documented in this report for completeness. West Zone was not updated for this Mineral Resource as there has been very little additional drilling in this area.

1.4.1 Drilling, Sampling, Assaying and Data Verification

The input data for the VOK Mineral Resource estimate comprised 932 drillholes totalling 218,238 m. The drilling consisted of:

- 9 historic drillholes (579 m)
- 490 surface drillholes drilled between 2009 and 2012 (173,619 m)
- 24 surface drillholes drilled in 2013 (5,200 m)
- 409 underground drillholes drilled in 2013 (38,840 m).

The sample data for the West Zone estimate comprised 756 drillholes (63,208 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.

Historical drill core sizes for surface drillholes were generally NQ (47.6 mm diameter) or BQ (36.5 mm diameter). The core size for drillholes collared from an underground exploration ramp at West Zone was AQ (27 mm diameter).

Core sizes for Pretivm's surface collared drillholes were PQ (85 mm diameter), HQ (63.5 mm diameter) and NQ (47.6 mm diameter). Approximately 50% to 60% of the Pretivm core was HQ size. For drillholes less than 600 m length, core size was commenced at HQ and reduced to NQ when required. For drillholes greater than 600 m length the commencing core size was PQ which was run down to between 200 m and 300 m in order to minimize drill path deviation. All drillcore collected from the underground drilling in 2013 was HQ size. No significant bias was noted between the PQ and HQ drill core samples that intersected the VOK mineralization. No testing was required on the NQ drill tails as these were almost without exception at depths below the main mineralization zones.

The drill collars were surveyed by McElhanney Surveying from Terrace, BC. McElhanney Surveying used a total station instrument and permanent ground control stations for reference and have completed all the surveying on the Project since 2009. All underground drill collars were surveyed by Procon.

Drillhole paths were surveyed at a nominal 50 m interval using a Reflex EZ single shot instrument. All drillhole paths were checked in a mining software package for deviation errors, which, if present, were corrected on a real time basis. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results.

Split PQ samples weigh approximately 10 kg. HQ samples were around 6 kg and NQ 3 to 4 kg. These weights assume a nominal 1.5 m sample length. In general, the average sample size submitted to the primary analytical laboratory, ALS Chemex (ALS) was 6.5 kg. 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.

Gold was determined using fire assay on a 30 g aliquot with an atomic absorption spectroscopy (AAS) finish. In addition, a 33 element package was also analysed using a four acid digest with an inductively coupled plasma-atomic emission spectroscopy (ICP AES) finish, which included silver. Specific gravity determinations were done by ALS using the pycnometer method on pulps from the drilling program.

Sampling procedures (prior to dispatch) have been completed under the supervision and security of Pretivm's staff. Laboratory sample reduction and analytical procedures have been conducted by independent accredited companies in line with standard industry practices.

Pretivm ensured quality control was monitored throughout the drilling campaigns with the insertion of blanks, certified reference materials, and duplicates. All data was stored and managed by independent database managing company, Geospark Consulting Inc. (GeoSpark) who carried out real time quality assurance/quality control (QA/QC) analysis.

Snowden carried out several site inspections and reviewed Pretivm's procedures including:

- independent sampling to verify the grade tenor
- inspection of the underground workings to confirm the mineralization style
- review of diamond core
- review of site procedures
- independent QA/QC analysis
- independent data validation.

It is the author's opinion that the sample preparation, security, and analytical procedures are satisfactory and that the data is suitable for use in Mineral Resource estimates.

1.4.2 Bulk Sample Test Work

In 2013, Pretivm excavated a bulk sample from within the VOK to further evaluate the geological interpretation and provide a comparison with the results from the Mineral Resource estimate. The location of the proposed bulk sample was selected to be representative of the grade and character of the typical mineralization in the VOK.

The design of the bulk sample was limited by provincial legislation to a maximum allowable bulk sample size of 10,000 t. The bulk sample was collected as a series of nominal 100-t rounds in underground development. Pretivm elected to process the bulk sample both through a sample tower on site and at a custom mill (Contact Mill) in Montana, US. In Snowden's opinion, the results of assaying of the samples from the sample tower provided an unacceptable degree of variation in the results due to the coarse gold nature of the mineralization and this information was not used further.

Prior to the December 2013 Mineral Resource estimate, the mill results from the underground bulk sample processing were used to investigate the local accuracy of the November 2012 Mineral Resource estimate within the VOK, and to determine whether the estimation methodology could be improved for the December 2013 Mineral Resource estimate.

A series of statistical tests were run to determine whether any bias exists between the surface diamond drilling, underground diamond drilling, underground channel samples, and chip samples. No significant difference/bias, based on the statistical analysis, was evident between the different sample types.

However, additional test work in the estimation did display some bias caused by directional drilling in the area of the bulk sample. The underground drilling had been aligned in a north-south orientation which is consistent with the orientation of some high-grade mineralization identified in the bulk sample, resulting in under sampling of this mineralization. Removal of the underground drillholes resulted in an increase in the grade of the local estimate.

While there is no bias evident between the channel samples and the drilling, the location of numerous channel samples in the centre of some of the higher-grade mineralization does result in a local overestimation around the bulk sample crosscuts. Consequently, the decision was made not to use the channel samples for the final mineral resource estimate.

The final metal and tonnes from the mill accounting were compared to those predicted by the November 2012 Mineral Resource estimate for each drive to assess the effectiveness of the resource modelling process. This test work has in part relied on comparisons between the test estimates and results from the bulk sample processing. However, the reader should be warned that there is a significant difference in the sample support for the resource estimate (each block in the resource estimate represents

2,700 t whereas the bulk sample packages are around 100 t), and the grade is not homogenous throughout any block. In other words, the grade can vary from a high-grade side of the block to a low grade side of the block, whereas the block grade represents an average of the whole block. If the bulk sample happens to take a high-grade part of the block, then the comparison will look like the resource estimate under-estimated the grade, and conversely if the bulk sample takes a low grade part of the block, then the comparison will look like the resource estimate over-estimated the grade in the block. Whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate (given the different sample support) locally, it does provide the best opportunity to fine-tune the estimate to some hard data. The reader should be warned that the results are only used to give some local perspective to the grade estimates.

The results indicated that the November 2012 Mineral Resource underestimated the total metal content in the bulk sample by about 10%. In more detail, the November 2012 Mineral Resource estimated high-grade into lower-grade areas, and low-grade into the high-grade areas, a result of extrapolating the high-grade values around the high-grade core. This extrapolation of high-grade values was based on the nature of the mineralization and the interpreted continuity of the high grades.

Based on the bulk sample comparisons, Snowden concludes that the November 2012 Mineral Resource was a good representation of the contained metal within the VOK deposit and satisfactory for mining studies based around bulk underground mining, but that it was not locally accurate at the 10 m block scale. As a result further test work was undertaken to adjust the estimation method for the December 2013 Mineral Resource, to produce an estimate that is more responsive to the local scale grade variations.

The results of the test work included:

- Removal of channel samples as these caused local scale over-estimation.
- Local adjustment of domain boundaries and incorporation of the north-south mineralization domains into the main stockwork domain packages. Test work using separate domaining of the north-south mineralisation resulted in over-estimation of the grade in these areas.
- Adjustment of the estimation parameters and methodology to reduce smoothing, including the method for re-blocking the high-grade Multiple Indicator Kriging (MIK) estimates, the chosen parent cell size, and search neighbourhood parameters.

1.4.3 Mineral Resource Estimation

The mineralization in the VOK exists as steeply dipping semi-concordant (to stratigraphy) and discordant pod-like zones hosted in stockwork vein systems within the volcanic and volcaniclastic sequence. High-grade mineralization zones appear to be spatially associated, at least in part, with intensely silicified zones resulting from local silica flooding and over-pressure caprock formation. High-grade mineralization occurs both in the main east-west trending vein stockwork system, as well as in the rarer north-south trending part of the system. Snowden notes that Pretivm has taken these various observations into consideration in its interpretation of the mineralization domains for the VOK.

A threshold grade of 0.3 g/t gold was found to generally identify the limits to the broad zones of mineralization as represented in the drill cores at West Zone and the VOK. In the VOK, a 1 g/t gold to 3 g/t gold 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 grade estimation.

All data was composited to the nominal sample length of 1.5 m prior to statistical analysis and estimation. Statistical analysis of the gold and silver data was carried out by lithological domain (at the VOK) and mineralized domain. Review of the statistics indicated that the grade distributions for the mineralization within the different lithologies are very similar and as a result these were combined for analysis. This is in agreement with field observations which indicate that the stockwork mineralization is superimposed on the stratigraphic sequence. The summary statistics of composite samples from all domains exhibit a strong positive skewness with high (greater than 1.6) coefficient of variation (ratio of the standard deviation to the mean) and some extreme grades.

Because of the extreme positive skew in the histograms of the gold and silver grades within the high-grade domains, Snowden elected to use a non-linear approach to estimation, employing the use of indicator and truncated distribution kriging. In this approach the proportion of high grade in a block was modelled, as was 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 population was estimated using ordinary kriging on the truncated (low-grade; less than 5 g/t gold and less than 50 g/t silver) part of the grade distribution.

Specific gravity was estimated into the model blocks using simple kriging of specific gravity measurements determined on sample pulps by ALS. As part of the 2012 surface drilling and 2013 underground drilling programs, Pretivm selected a portion of the samples (207 and 204 samples, respectively) for core density (water immersion method) as well as the pulp specific gravity measurements in order to determine the impact of porosity.

Results of the comparison between the pulp specific gravity and core density measurements indicate that the core density is on average the same as the pulp specific gravity within the siliceous zone and approximately 3% lower, on average, for all other rock types. Consequently, all specific gravity estimates in the Mineral Resource model (which are based on the pulp specific gravity measurements), with the exception of the siliceous zone, were factored down by 3% to yield the corresponding bulk density

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

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 the “CIM Definition Standards” document.

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 AuEq calculation used is: AuEq = Au + Ag/53. These blocks were then used as a guide to develop a set of wireframes defining coherent zones of mineralization which were 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 at West Zone are within the well-informed portion where the resource is informed by 5 m by 5 m or 5 m by 10 m spaced drilling. Measured Resources within VOK are informed by 5 m by 10 m to 10 m by 10 m underground fan drilling and restricted to the vicinity of the underground bulk sample.

Areas classified as Indicated Resources are informed by drilling on a 20 m by 20 m to 20 m by 40 m grid within West Zone and VOK. In addition, some blocks at the edge of the areas with 20 m by 20 m to 20 m by 40 m drilling, were downgraded to Inferred where the high grades appear to have too much influence. The remainder of the Mineral Resource is classified as Inferred Resources where there is some drilling information (and within around 100 m of drilling) and the blocks occur 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.

The Mineral Resource was reported above a 5 g/t AuEq cut-off grade for the VOK and West Zone (Table 1.1 and Table 1.2). The Mineral Resources are depleted for historical mining in West Zone and the recent underground bulk sample mining in VOK.

Table 1.1 VOK Mineral Resource Estimate Based on a Cut-off Grade of 5 g/t AuEq – December 2013(1)(4)(5)

Category	Tonnes (million)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Measured	2.0	19.3	14.4	1.2	0.9
Indicated	13.4	17.4	14.3	7.5	6.1
M + I	15.3	17.6	14.3	8.7	7.0
Inferred(2)	5.9	25.6	20.6	4.9	3.9

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 Technical Report were estimated 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 Mineral Resource estimate stated in Table 14.4 and Table 14.5 is defined using 5 m by 5 by 5 m blocks in the well drilled portion of West Zone (5 m by 10 m drilling or better) and 10 m by 10 m by 10 m blocks in the remainder of West Zone and in VOK.

(5) The AuEq value is defined as AuEq = Au + Ag/53.

Table 1.2 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
M+I	4.9	5.85	267	0.9	41.9
Inferred(2)	4.0	6.44	82	0.8	10.6

Notes: (1), (2), (3), (4) and (5) - refer to footnotes in 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 2014. The samples tested were generated from various drilling programs, including the samples tested by the bulk sample processing programs. The metallurgical test programs conducted on the Brucejack mineralization included head sample characteristics, gravity concentration, gold/silver bulk flotation, cyanidation, table concentrate melting 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. The industrial runs on the 10,000-t bulk sample for the 2013 bulk sample processing program and the 1,200-t high-grade Cleo mineralization conducted in 2014 showed that the gravity/flotation process flowsheet as designed for the Brucejack mineralization suited the treatment of the bulk sample. The results also showed that the gravity/flotation flowsheet adapted well for the varying mineralization and the wide range feed grades that were experienced during processing of the bulk sample.

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 5,600 kg of gold and 1,900 kg of silver as doré per year and 44,000 t of gold-silver bearing flotation concentrate per year from the mill feed, grading 14.1 g/t gold and 57.7 g/t silver. The estimated gold recoveries to the doré and flotation concentrate are 43.3% and 53.4%, respectively, totalling 96.7%. The estimated silver recoveries reporting to the doré and flotation concentrate are 3.5% and 86.5%, respectively, totalling 90.0%. The LOM average gold and silver contents of the flotation concentrate are anticipated to be approximately 157 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:

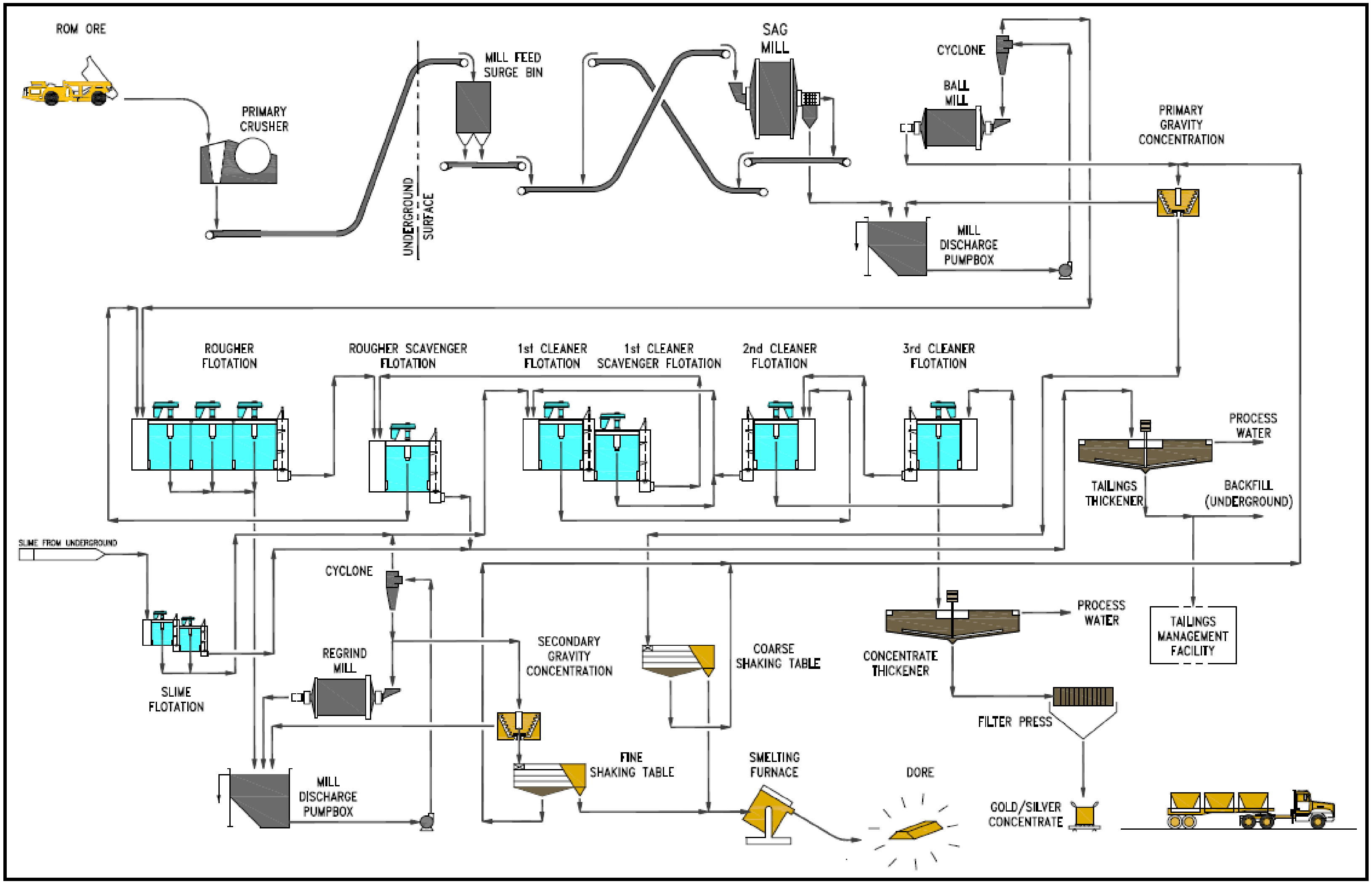
- one stage of crushing (located underground)
- a surge bin with a live capacity of 2,500 t on surface
- a semi-autogeneous grinding (SAG) mill and ball mill primary grinding circuit integrated with gravity concentration
- rougher flotation and rougher/scavenger flotation followed by rougher flotation 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 is expected that the flotation concentrate 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 value of Cdn$180/t of ore was used to define the Mineral Reserves (as used in previous studies). For the feasibility study update, the average site operating costs over the LOM are calculated as Cdn$163.05/t, providing a minimum $16.95/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$1,100/oz gold and US$17/oz silver.

The dilution factors were calculated from standard overbreak assumptions that are based on AMC's experience and benchmarking of similar long hole open stoping (LHOS) operations. The overall LOM ore recovery is estimated at 94% with an overall ore dilution of 12%.

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 1.3). The Mineral Reserves were developed from the resource model, "bjbm_1313_v11_cut", which was provided to AMC by Snowden—on behalf of Pretivm—in February 2014.

Table 1.3 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	2.1	15.6	12	1.1	0.8
	Probable	11.5	15.7	10	5.8	3.9
	Total	**13.6**	**15.7**	**11**	**6.9**	**4.6**
West Zone	Proven	1.4	7.2	383	0.3	17.4
	Probable	1.5	6.5	181	0.3	8.6
	Total	**2.9**	**6.9**	**279**	**0.6**	**26.0**
Total Mine	Proven	3.5	12.2	161	1.4	18.2
	Probable	13.0	14.7	30	6.1	12.5
	Total	**16.5**	**14.1**	**58**	**7.5**	**30.7**

Notes: *Rounding of some figures may lead to minor discrepancies in totals. Based on an NSR cut-off value of Cdn$180/t, US$1,100/oz gold price, US$17/oz silver price, Cdn$/US$ exchange rate = 0.92.

1.7 MINING METHODS

The underground mine design is largely unchanged from the previous feasibility study, supporting the extraction of 2,700 t/d of ore via transverse 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 second decline will be dedicated to conveying crushed ore directly to the concentrator via two conveyors with a combined length of 800 m. There will be a two-year pre-production development period, with steady-state production being reached by the end of Year 2 of an 18-year LOM. The development and production sequence prioritizes high-grade areas while ramping up overall mine tonnage to the steady state, averaging approximately 980,000 t/a through to Year 16.

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.

Underground manpower will consist of technical staff, mining crews, mechanics, electricians, and other support personnel. Pre-production manpower will be supplied by a contractor, except personnel required for maintenance and technical support. Total manpower required for full production is 351, with up to 176 personnel on site at any given time.

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 main access point to the VOK area located on 1,320 Level.

Ore will be trucked from working areas to an underground crusher and then transferred to surface via two, 1.07 m wide conveyors. Waste rock will be disposed in the underground mine whenever possible, with the balance trucked to surface for disposal in Brucejack Lake.

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. Solids captured in the main collection sump will be pumped to the mill for residual gold recovery. 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 initial mining capital during pre-production period, including a 10% contingency, is estimated at Cdn$240 million. Sustaining mining capital of Cdn$280 million has been estimated for the production period. The total underground operating cost over the LOM is estimated to be Cdn$1,512 million, at an average LOM cost of Cdn$91.34/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
- mill building and process plant
- mine site operation camp
- transmission line and substation
- ancillary facilities
- water supply and distribution
- water treatment plant
- waste disposal
- tailings delivery system
- Brucejack Lake 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

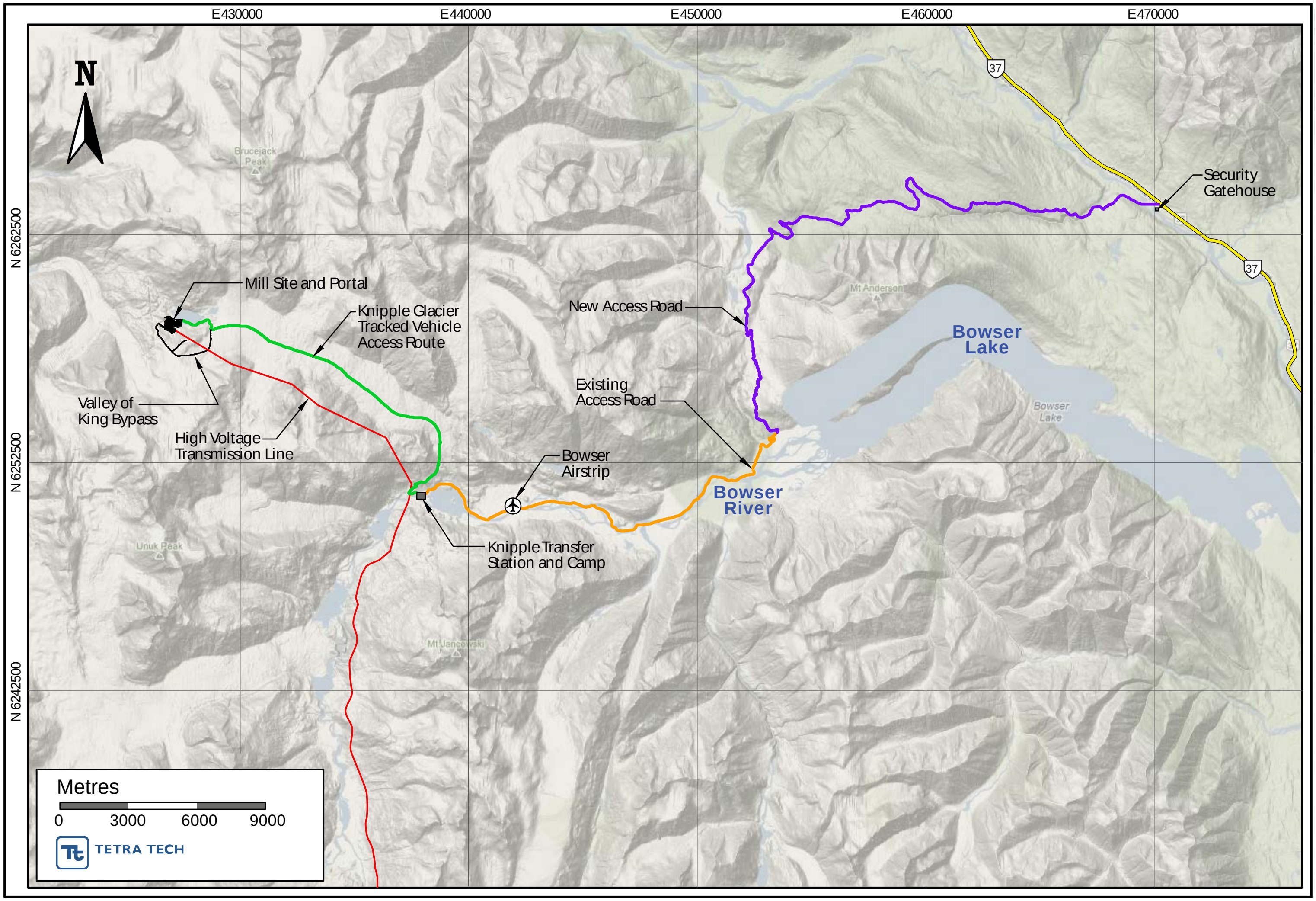


Figure 1.3 Mill Site Layout

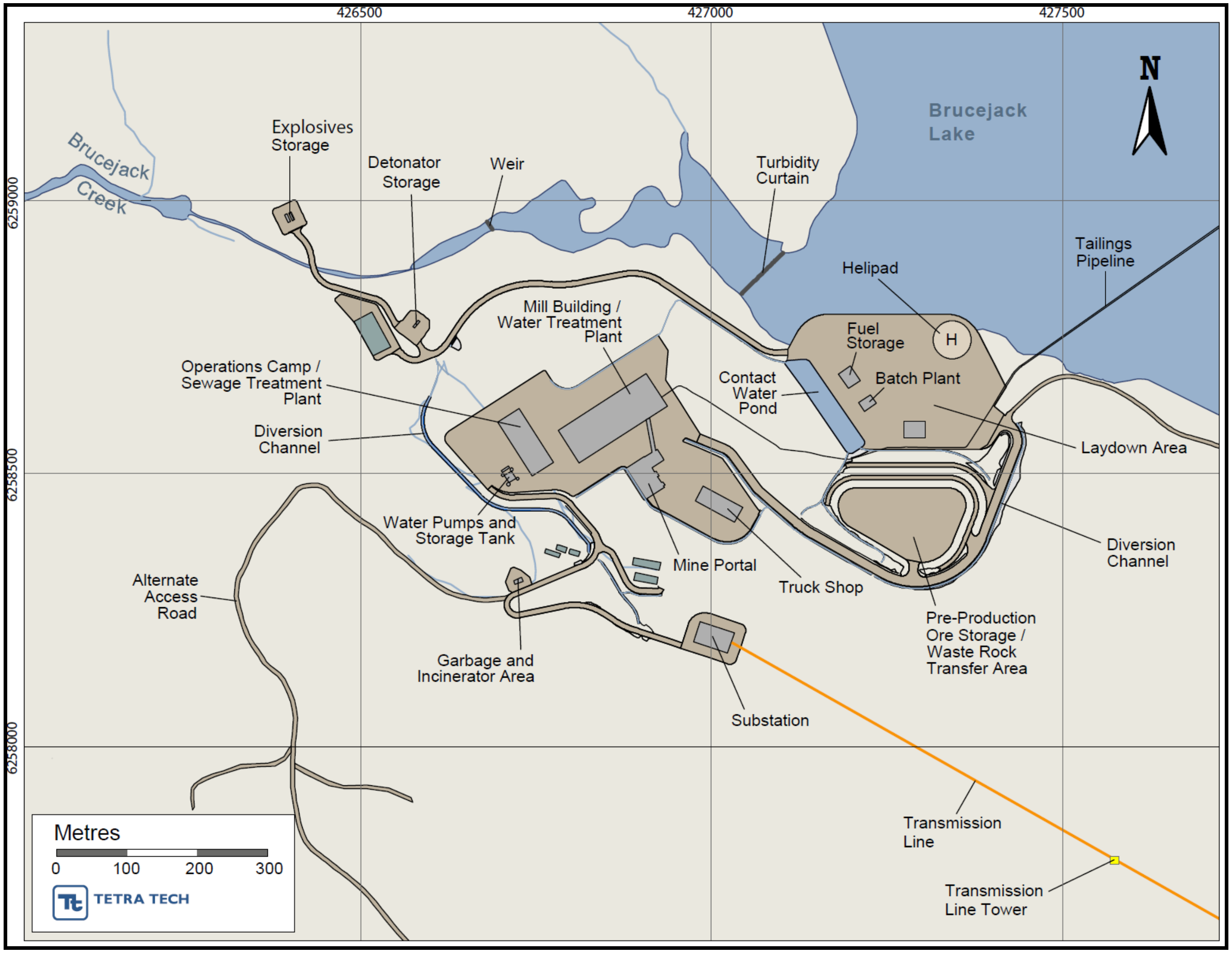
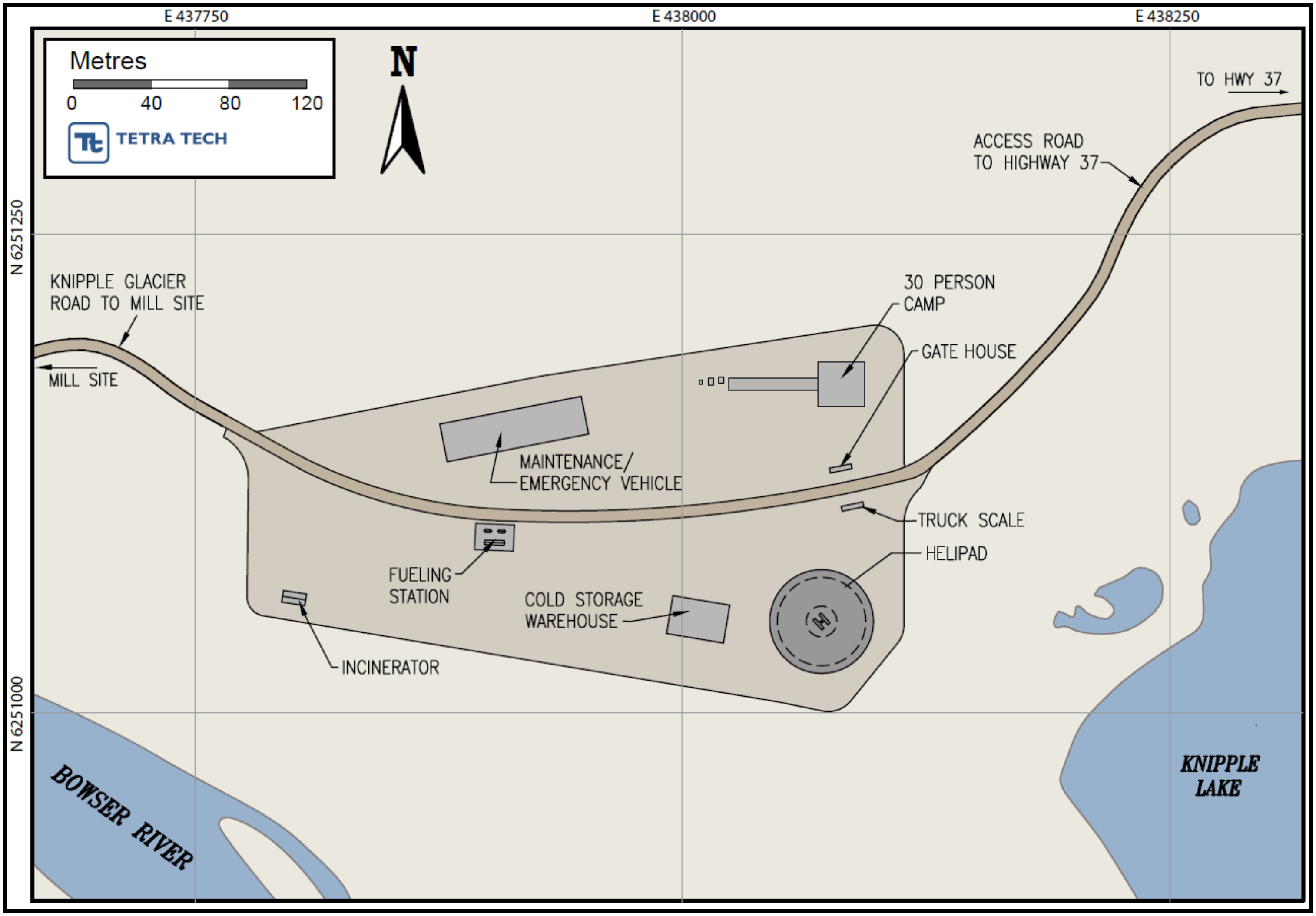


Figure 1.4 Knipple Transfer Station Facility Layout



1.8.1 Avalanche Hazard Assessment

An avalanche hazard assessment has been completed for the Project. Mine site facilities and access routes are exposed to approximately 14 avalanche paths or areas, and the preliminary transmission line alignment crosses several avalanche paths. Avalanche magnitude and frequency varies depending on location. 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 steel 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 lake in a manner which will minimize suspended solids concentrations at the lake outlet. 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 US$746.9 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 1.4 Summary of Initial Capital Cost

Major Area	Area Description	Capital Cost (US$ million)
Direct Costs		
11	Mine Site	21.5
21	Mine Underground	179.5
31	Mine Site Process	53.8
32	Mine Site Utilities	30.5
33	Mine Site Facilities	53.5
34	Mine Site Tailings	3.5
35	Mine Site Temporary Facilities	33.4
36	Mine Site (Surface) Mobile Equipment	14.6
84	Off Site Infrastructure	89.1
Subtotal Direct Costs		**479.4**
91	Indirect Costs	127.5
98	Owner's Costs	71.0
99	Contingency	69.0
Total Initial Capital Cost		**746.9**

Note: Numbers may not add due to rounding.

The purpose of this capital cost estimate is to provide feasibility-level input to the Project financial model.

This s a Class 3 feasibility cost estimate prepared in accordance with the standards of the Association for the Advancement of Cost Engineering International (AACE). The estimated degree of project definition meets or exceeds 30%. The accuracy of this estimate is deemed to be -15%/+20%. 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 2014 and does not include any escalation beyond this date. The quotations used for this feasibility study estimate were obtained in Q2 2014, and have a validity period of 90 days.

The capital cost estimate uses US 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 is divided into four general sections (direct costs, indirect costs, owners' costs and contingency) and was developed based largely on first principles from a design, planning, and cost basis. A list of exclusions is presented in Section 21.0.

1.11 OPERATING COSTS

The total LOM average operating cost for the Project is estimated at Cdn$163.05/t ore milled which includes for:

- mining
- process
- 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, or other government imposed costs, unless otherwise noted.

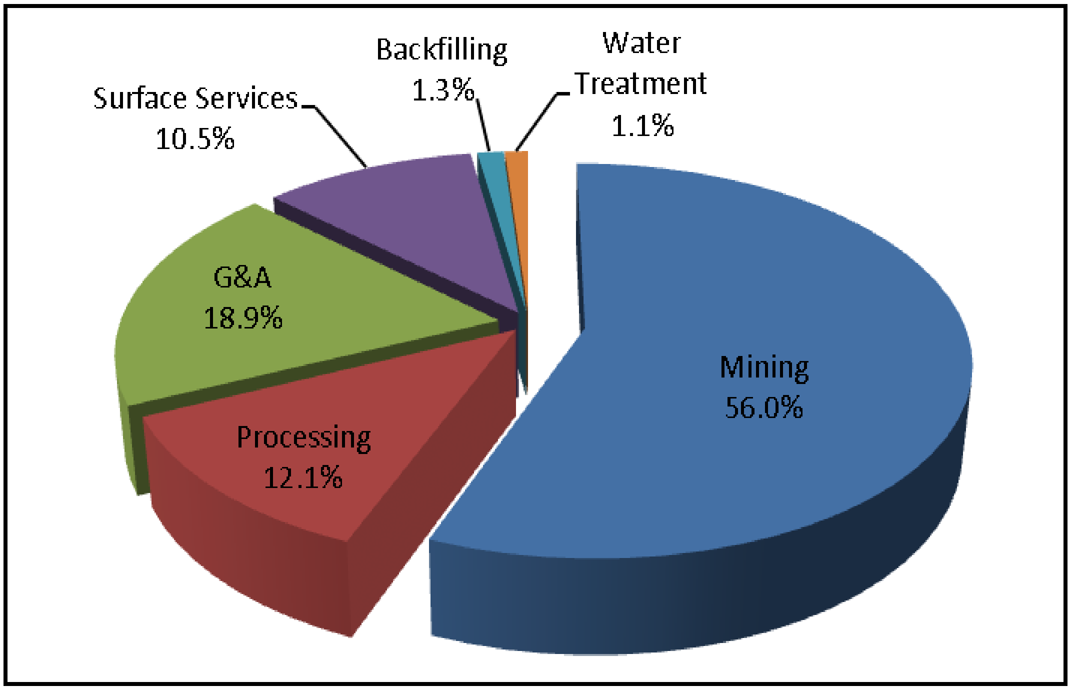
A total of 593 personnel are projected to be required for the Project. The unit cost estimates are based on the LOM ore production and a mine life of 18 years. The currency exchange rate used for the estimate is 1.00:0.92 (Cdn$:US$). The operating cost for the Project has been estimated in Canadian dollars 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 1.5 Overall Operating Cost

Area	Personnel	Unit Operating Cost (Cdn$/t milled)
Mining*	351**	91.34
Processing	100	19.69
G&A	54	30.87
Surface Services	78	17.18
Backfilling	6	2.11
Water Treatment	4	1.86
Total	**593**	**163.05**

Notes: *Average LOM mining cost including crushing cost, cement cost for backfill and back-hauling cost for the preproduction ore stocked on the surface; if excluding the ore mined during preproduction, the estimated unit cost is $91.78/t.
**351 workers during Year 3 to 12 and less mining personnel requirement is estimated for the rest of the operation years.

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 18-year LOM and 16.55 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 34.7% internal rate of return (IRR)
- 2.7-year payback on the US$746.9 million initial capital
- US$2,251 million net present value (NPV) at a 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:

- 28.5% IRR
- 2.8-year payback on the US$746.9 million initial capital
- US$1,445 million NPV at a 5% discount rate.

As indicated in Sections 19.0 and 21.0 of this report, the base case metal prices (Section 19.0) and exchange rate (Section 21.0) used for this study are as follows:

- gold – US$1,100/oz
- silver – US$17.00/oz
- exchange rate – 0.92:1.00 (US$:Cdn$).

1.13 PROJECT EXECUTION PLAN

The Project will take approximately 33 months to complete from the start of basic engineering, through construction, to introduction of first material into the mill. A further two 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 2014 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 1.6 Key Milestone Dates

Year	Quarter	Activity
2014	2	Feasibility Study Update Completion
2014	3	Start of Basic Engineering
2014	3	EPCM Award
2015	2	Start of Stage I Early Works Construction
2016	1	Detailed Engineering Completion
2016	1	Start of Stage II Full Project Execution Construction
2017	2	Surface Mechanical Completion
2017	2	Underground Mechanical Completion
2017	3	Mine Site Commissioning Completion
2017	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 EAC later this year. After obtaining permits, and subject to a production decision, Pretivm anticipates commencing construction of the mine in the first half of 2015. Detailed recommendations for the Project can be found in Section 26.0.

2.0 INTRODUCTION

Pretivm commissioned Tetra Tech to complete a feasibility study update 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
- ERM 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.

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:

- Lynn Olssen, MAusIMM(CP) completed a site visit on August 16, 2013 for five 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. completed 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.
- Trevor Crozier, M.Eng., P.Eng. completed a site visit on September 5, 2013 for one day.
- Sharon Blackmore, M.Sc., P.Geo. completed a site in September 2013 for two weeks.

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	Lynn Olssen, MAusIMM(CP)
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Snowden	Lynn Olssen, MAusIMM(CP)
6.0	History	Snowden	Lynn Olssen, MAusIMM(CP)
7.0	Geological Setting and Mineralization	Snowden	Lynn Olssen, MAusIMM(CP)
8.0	Deposit Types	Snowden	Lynn Olssen, MAusIMM(CP)
9.0	Exploration	Snowden	Lynn Olssen, MAusIMM(CP)
10.0	Drilling	Snowden	Lynn Olssen, MAusIMM(CP)

table continues...

Report Section		Company	QP
11.0	Sample Preparation, Analyses and Security	Snowden	Lynn Olssen, MAusIMM(CP)
12.0	Data Verification	Snowden	Lynn Olssen, MAusIMM(CP)
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	John Huang, Ph.D., P.Eng.
14.0	Mineral Resource Estimate	Snowden	Lynn Olssen, MAusIMM(CP)
15.0	Mineral Reserve Estimate	AMC	Colm Keogh, P.Eng.
16.0	Mining Methods	AMC/BGC	Colm Keogh, P.Eng./ George Zazzi, P.Eng./ Catherine Schmid, M.Sc., P.Eng./ Trevor Crozier, M.Eng., 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	Michael Chin, P.Eng.
	18.5 Grading and Drainage	Tetra Tech	Michael 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	ERM Rescan	Paul Greisman, Ph.D., P.Eng.
	18.13 Brucejack Lake Outflow Monitoring Weir and Suspended Solids Control	BGC/ Tetra Tech	Trevor Crozier, M.Eng., P.Eng./ Dave Ireland, C.Eng., P.Eng.
	18.14 Communications	Tetra Tech	Dave Ireland, C.Eng., P.Eng .
	18.15 Power Supply and Distribution	Tetra Tech	Wayne E. Scott, P.Eng.
	18.16 Fuel Supply and Distribution	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	ERM Rescan/ BGC	Pierre Pelletier, P.Eng./ Hamish Weatherly, M.Sc., P.Geo./ Sharon Blackmore, M.Sc., P.Geo./ Trevor Crozier, 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.

table continues...

Report Section		Company	QP
	21.2.5 Water Treatment Operating Costs	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.2.6 General and Administrative and Surface Services	Tetra Tech	John Huang, Ph.D., P.Eng.
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
23.0	Adjacent Properties	Snowden	Lynn Olssen, MAusIMM(CP)
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 INFORMATION AND DATA SOURCES

A complete list of references is provided in Section 27.0.

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.

3.1 LYNN OLSSEN, B.SC., MAUSIMM (CP)

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.

3.2 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")" dated June 19, 2014.

Sabry Abdel Hafez, Ph.D., P.Eng. also relied on Pretium Resources Inc. for estimating the applicable royalties on the Brucejack Deposit used in the economic analysis. This reliance is based on an agreement document titled "Royalty Agreement" dated August 31, 2001.

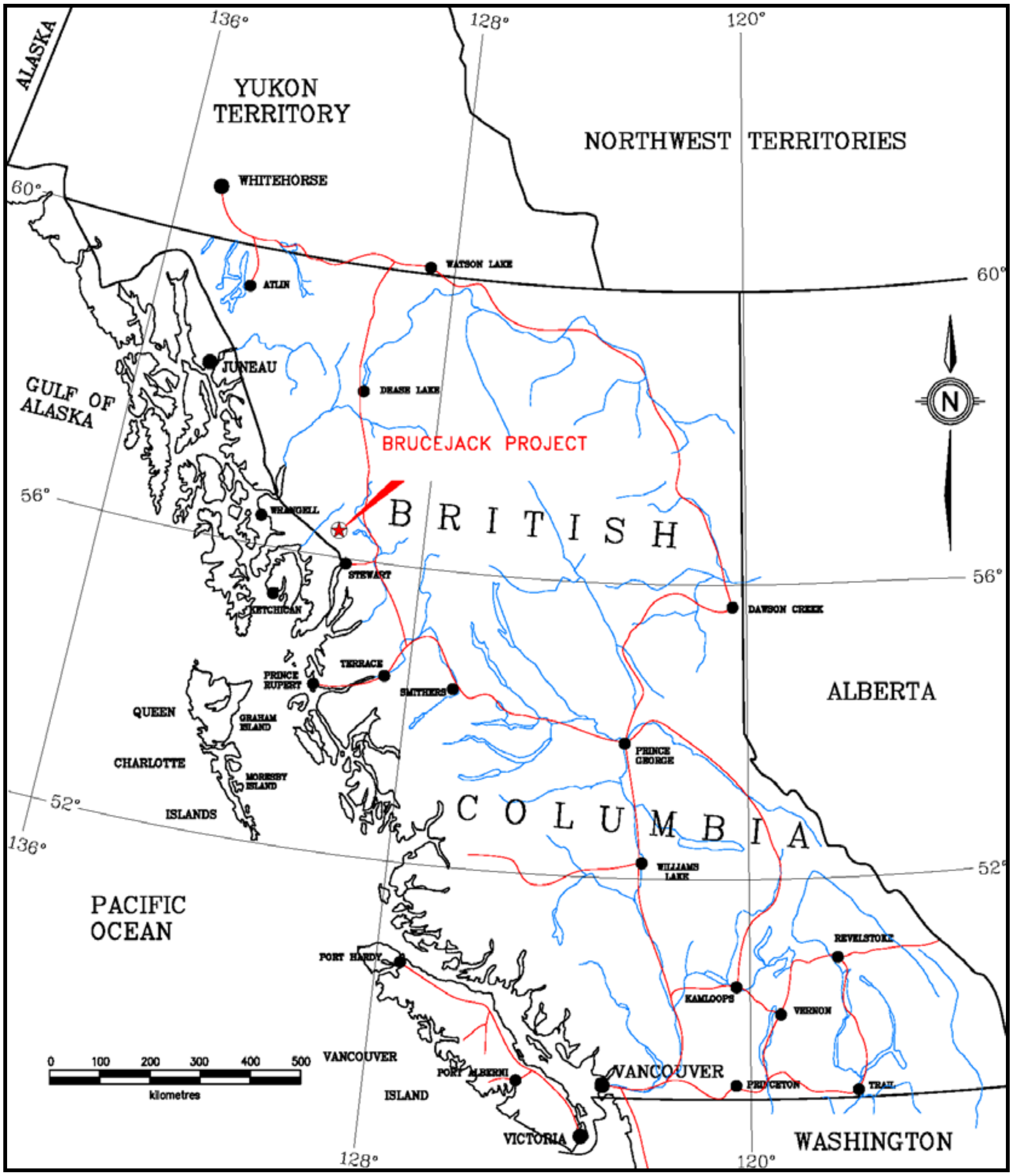
4.0 PROPERTY DESCRIPTION AND LOCATION

Information in this section has been excerpted from Jones (2013).

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



Source: Pretivm

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 is located on provincial Crown land and consists of eleven mineral claims that cover the target Mineral Resource, totaling 3,199.28 ha in area. All claims are in good standing until January 31, 2025 (Table 4.1). These claims are part of a larger block of mineral claims held by Pretivm that includes the Bowser Property (Figure 4.2). The larger block of mineral claims totals 260 mineral claims totalling approximately 104,111 ha in and around the Property (Figure 4.3). The claims extend from the proposed mine site area east to Highway 37, including parts of the Bowser River, Scott Creek, and Wildfire Creek watersheds, and along parts of the transmission line right-of-way. The Project is situated within the Sulphurets District, Skeena Mining District.

Table 4.1 Mineral Claims for the Brucejack Property

Tenure No.	Tenure Type	Map No.	Owner	Pretivm Interest	Status	In Good Standing To	Area (ha)
509223	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	428.62
509397	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	375.15
509400	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	178.63
1027396	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	125.09
1027397	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	53.63
1027398	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	160.92
1027399	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	983.61
1027400	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	500.39
1027429	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	196.61
1027431	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	53.63
1027433	Mineral	104B	Pretium Exploration Inc.	100%	Good	31-Jan-25	143.00
						Total (ha)	**3,199.28**

Snowden relied upon public information, as well as information from Pretivm, regarding the Property claims and has not undertaken an independent verification of title and ownership. However, Snowden verified information relating to tenure, to the extent possible, by means of public information available through the Mineral Titles Branch of the BC MEMNG MTO land tenure database.

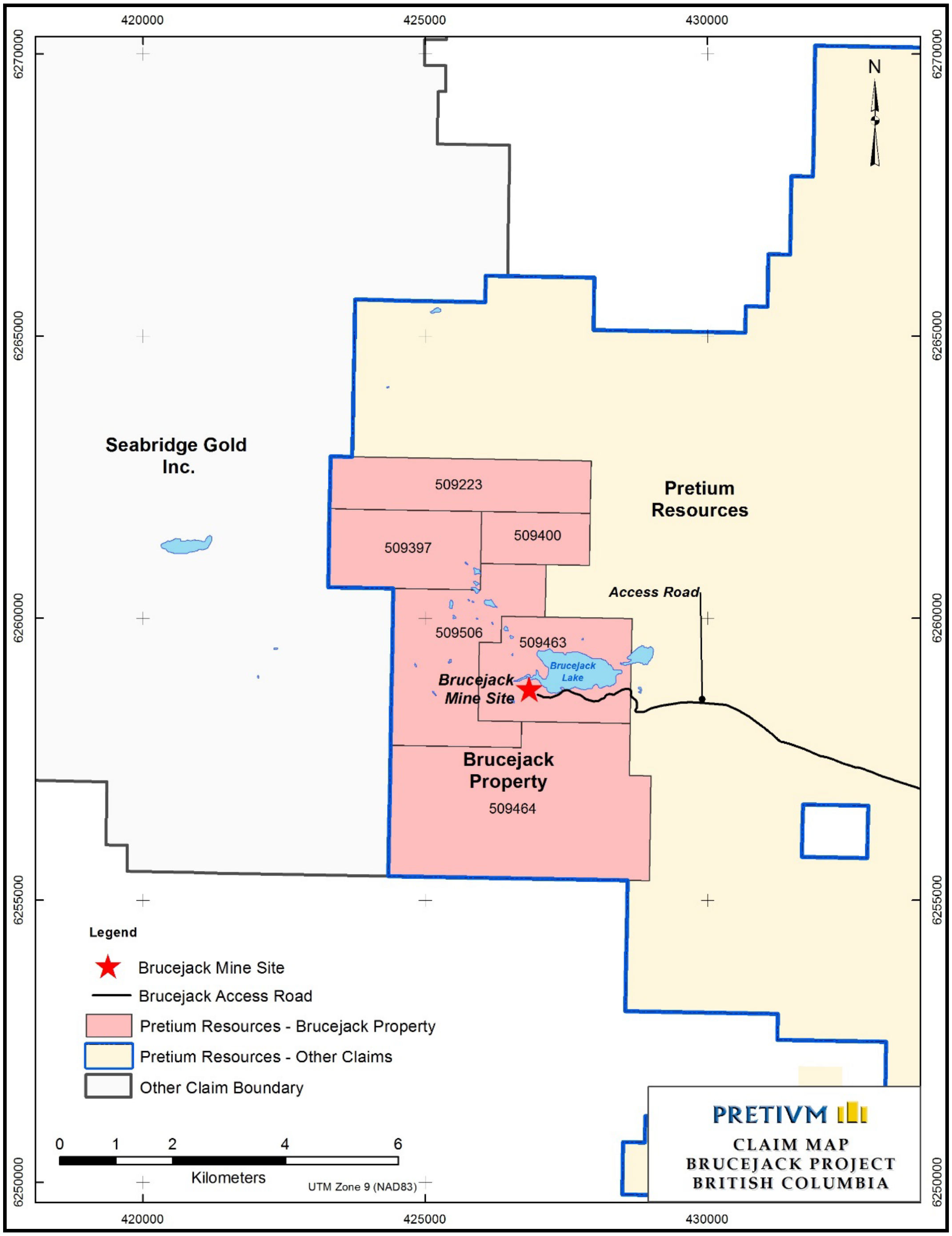
A legal land survey of the claims has not been undertaken.

There are no annual holding costs for any of the 11 mineral claims at this time, as the claims are paid up until January 31, 2025.

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.

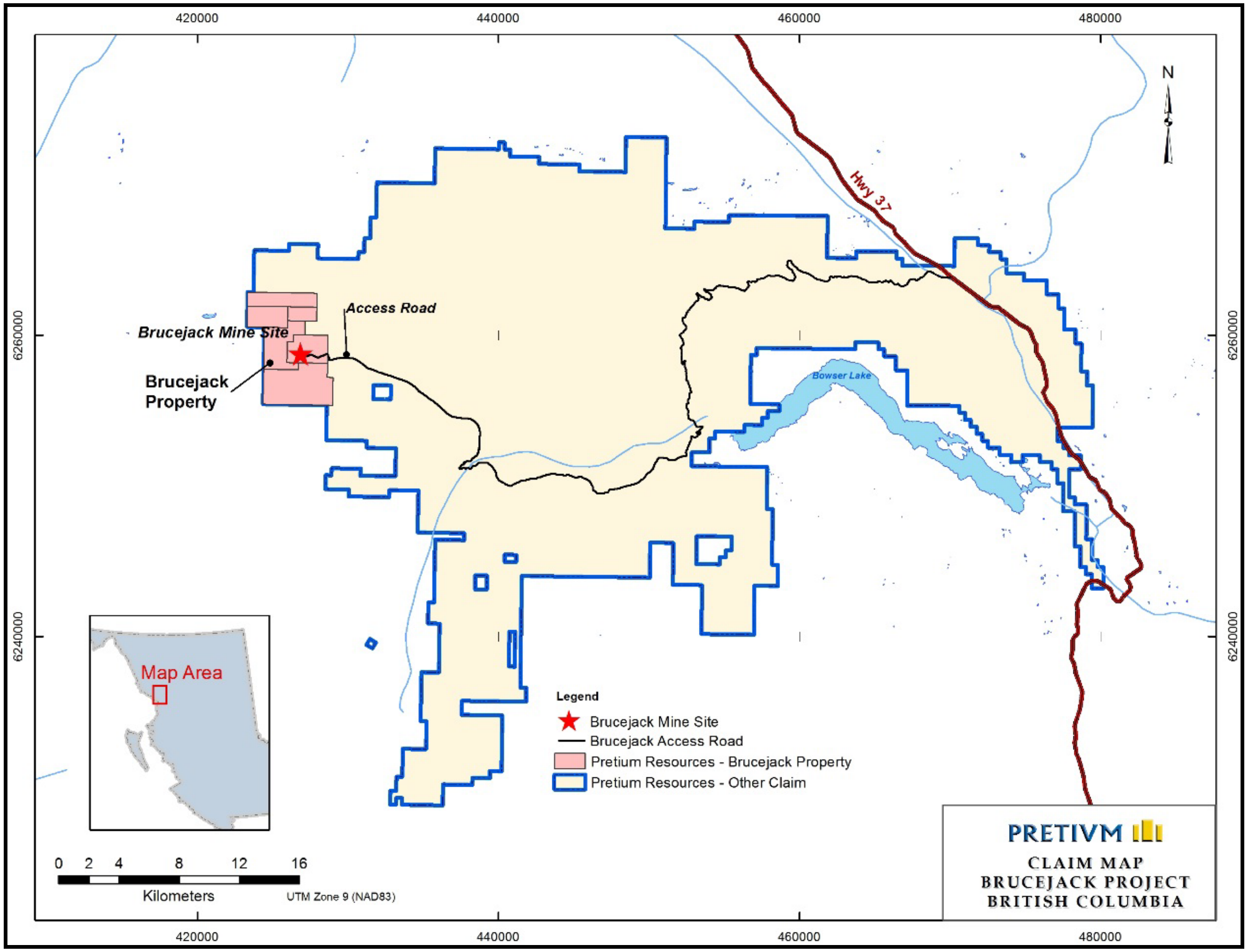
As of the effective date of this report, the land claims in the area are in review and subject to ongoing discussions between various First Nations and the Government of BC.

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 reported properties 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 Jones (2013).

5.1 CLIMATE AND PHYSIOGRAPHY

The climate at the Property is typical of northwestern BC with cool, wet summers, and relatively moderate but wet winters. Annual temperatures range from +20 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.

5.1.1 VEGETATION

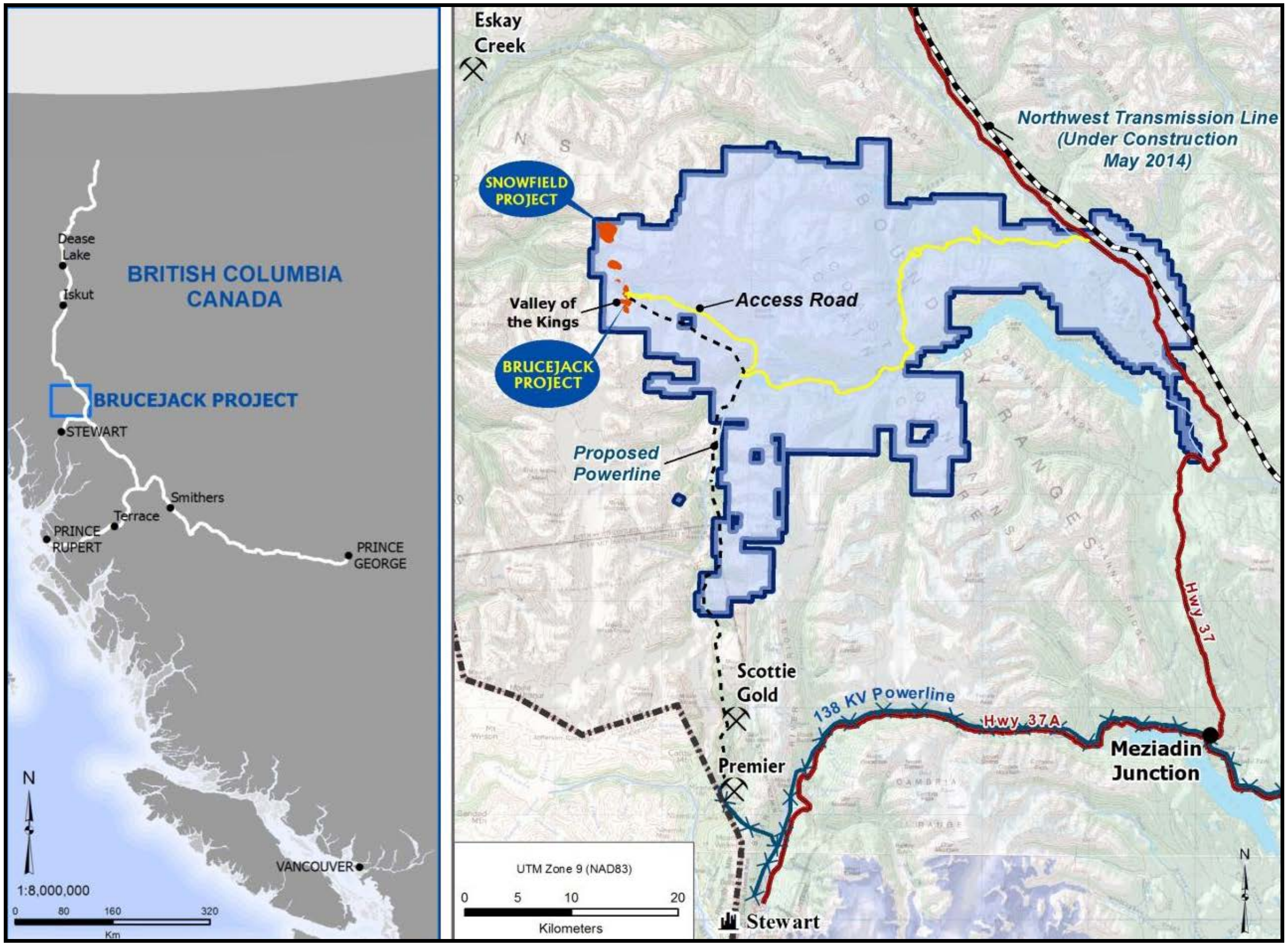
The tree line is at an elevation of approximately 1,200 m. Sparse fir, spruce, and alder grow along the valley bottoms with only scrub alpine spruce, juniper, alpine grass, moss, and heather covering the steep valley walls. The Property, at an elevation above 1,300 m, has only sparse mosses along drainages.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

5.2 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.

Pretivm has completed construction of its 74 km access road that links the Brucejack Camp to Highway 37 via the Knipple Glacier, Bowser Camp and Wildfire Camp (Figure 5.1). Personnel, equipment, fuel and camp provisions are driven to a staging area on the Knipple Glacier (at km 60), before being taken over the glacier to the Brucejack camp. This has significantly reduced transportation costs. The Property area is also 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.

Figure 5.1 Project Access



Source: Pretivm

5.3 INFRASTRUCTURE

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 effective date of this report, the Wolverine and Huckleberry mines were shipping material via this terminal.

A high-voltage, 138 kV transmission line currently services Stewart, BC, and has sufficient capacity to provide power to the Project. BC Hydro is completing a facilities study in respect of the interconnection of a transmission line servicing the Project with the 138 kV transmission line servicing the town of Stewart, BC (Figure 5.1). The study is expected to be completed in mid-2014.

6.0 HISTORY

Information in this section has been excerpted and updated from Jones (2013).

6.1 EARLY EXPLORATION

In 1935, copper-molybdenum mineralization was discovered on the Sulphurets property by prospectors in the vicinity of the Main Copper Zone, approximately 6 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 resulted in the discovery of several small copper and gold-silver occurrences 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. This was the start of what could be termed the era of modern exploration (Table 6.1).

Table 6.1 Exploration History of the Sulphurets Property Between 1960 and 2008

Date	Exploration
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 Sulphurets 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 Sulphurets property and (in 1984) outlined a deposit on the west Brucejack Zone.
1985	Esso dropped the option on the Sulphurets property.
1985	The Sulphurets 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.

table continues…

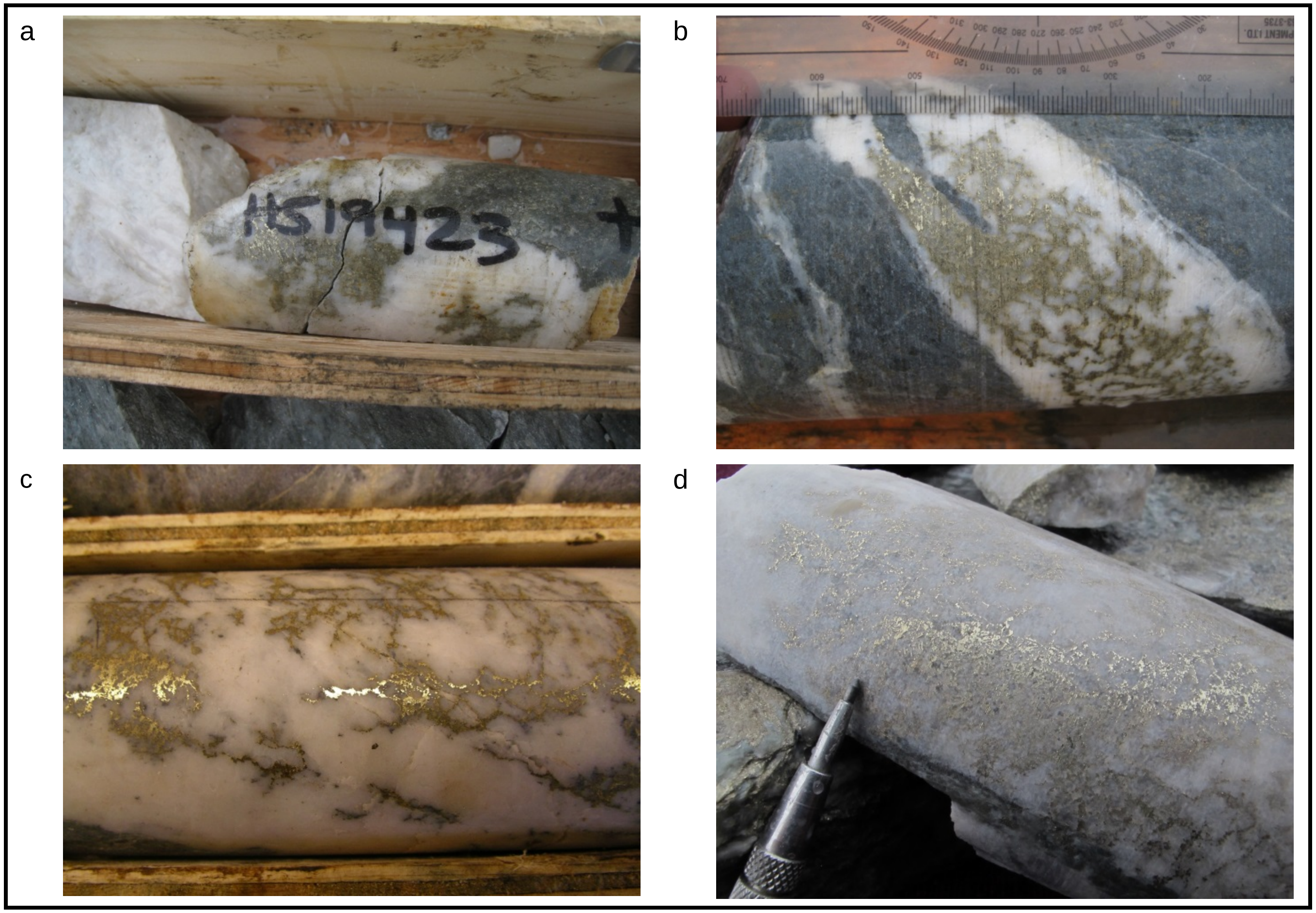
Date	Exploration
1986 to 1991	Between 1986 and 1991, the Newcana JV spent approximately $21 million developing the West Zone and other smaller precious metal veins, on what would later become the Bruceside property. In addition to surface work, a total of 5,276 m of exploratory underground drifting and 33,750 m of underground drilling in 442 drillholes was completed on the West Zone between 1987 and 1990.
1991 and 1992	Newhawk officially subdivided the Sulphurets 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 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 gold over 1.5 m, to 13 g/t gold over 4.9 m (www.infomine.com).
1994	Exploration in the Bruceside property consisted of detailed mapping and sampling in the vicinity of the Gossan Hill Zone, and 7,352 m of diamond drilling (in 20 drillholes) 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. (Black Hawk).
1997 and 1998	No exploration or development work was carried out on the Snowfield and Bruceside properties (Budinski et al. 2001).
1999	Silver Standard Resources Inc. (Silver Standard) acquired Newhawk and with it, Newhawk's 100% interest in the Snowfield property and 60% interest in the Bruceside 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 Bruceside property, giving Silver Standard a 100% interest in the Bruceside property, which it subsequently renamed the Brucejack Project.
1999 to 2008	No exploration or development work was carried out on the Snowfield and Brucejack properties during the period from 1999 to 2008.

6.2 Exploration by Silver Standard Resources Inc. (2001-2010)

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. Based on its successful bulk tonnage drilling on the Snowfield Property, Silver Standard designed the 2009 Brucejack drill program test for additional bulk tonnage resources on the Brucejack Property.

The 2009 program tested five zones with 37 drillholes totalling 18,000 m. A total of 12 drillholes were targeted at what would become the VOK. Drillhole SU-012 (Figure 6.1) is credited as being the discovery drillhole for the VOK intersecting 16,948.5 g/t gold over 1.5 m.

Figure 6.1 Examples of High Grade Gold Intersections in the VOK



Source: Pretivm

Notes: Dendritic latticework electrum in quartz-carbonate vein in: a) discovery drillhole SU-012; b) drillhole SU-084; c) drillhole SU-115; and d) drillhole SU-452; core in photographs is HQ diameter.

The 2010 drill program, which totalled 33,480 m in 73 drillholes, was designed to continue definition of the bulk tonnage mineralization as well as to determine the nature and continuity of the high-grade mineralization observed at VOK. Approximately one third of the 2010 drilling targeted the VOK and included gold intersections of up to 5,840 g/t gold. The bulk tonnage drilling achieved its intended goal through the definition of more than 20 Moz at Brucejack (8 Moz in Measured and Indicated and 12.5 Moz gold in Inferred, at a 0.3 g/t AuEq cut-off; Ghaffari et al. 2011). The relatively dense drilling from the bulk tonnage drilling program, with drill spacings of 100 m by 100 m to 50 m by 50 m, formed the basis upon which the bulk tonnage resource model was built. Numerous high-grade intersections were defined as part of this drilling, allowing for the initial delineation of high-grade mineralization trends.

In 2010, Silver Standard proceeded with the sale of the Snowfield and Brucejack projects to a company formed by the former president specifically to acquire the projects (Pretivm Resources Inc.).

6.3 Previous Feasibility Studies on the Property (1990)

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. Total operating costs of $145/t were estimated based on a 350-t/d mill facility for processing, a capital cost of $42.7 million and a 6.7% pre-tax return at a price of US$400/oz gold and $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 above mentioned 1990 Corona Sulphurets Project feasibility study is no longer relevant, is not NI 43-101 compliant, and should not be relied upon.

6.4 Prior Mineral Production

In the 1980s, 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. Prior to 2013, no ore had been mined or processed from the Property, including the West Zone.

6.5 Preliminary Economic Assessment (2010)

Silver Standard commissioned Wardrop Engineering Inc. (Wardrop; now Tetra Tech) to complete a preliminary economic assessment (PEA) on the combined bulk-tonnage resources of the Brucejack and Snowfield projects 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 Mining Consultants Inc. (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 prefeasibility study, in order to identify opportunities and further assess bulk-tonnage viability of the Property. This report was re-issued for Pretivm in October 2010. The report, however, is no longer current.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

Information in this section has been excerpted and condensed from Jones (2013). The reader should refer to Jones (2013) for detailed information.

7.1 REGIONAL GEOLOGICAL SETTING

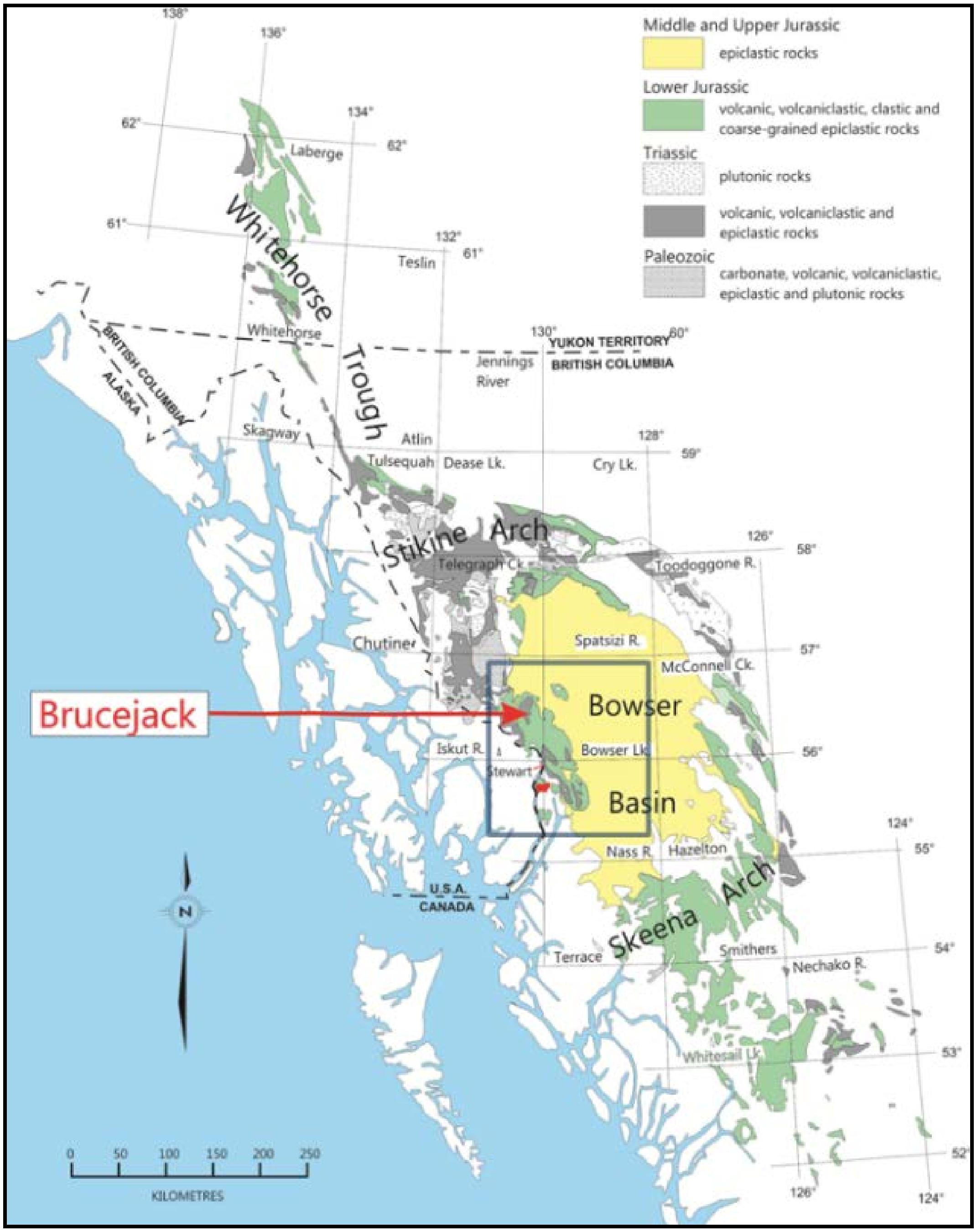
The Property is located in the western Stikine terrane (Stikinia), the largest and westernmost of several exotic terranes in the Intermontane Belt of the Canadian Cordillera (Figure 7.1). Stikinia is interpreted as an intra-oceanic island arc terrane, formed between mid-Palaeozoic to Middle Jurassic time, when it was accreted to the North American continental margin (about 173 Ma; e.g., Nelson and Colpron 2007; Evenchick et al. 2007; Gagnon et al. 2012). Western Stikina was subsequently strongly deformed during the Cretaceous accretion of the outboard Insular Belt terranes (about 110 Ma; Kirkham and Margolis 1995).

Volcano-sedimentary rocks and related Early Jurassic plutons in the north-west part of Stikina represent an exceptionally metals-rich tectonic assemblage in BC (e.g., Nelson et al. 2013). This area includes volcanogenic massive sulphide deposits (e.g., Granduc, Dolly Varden-Torbrit, Anyox, and Eskay Creek), alkaline porphyry copper-gold deposits (e.g., Kerr, Sulphurets, Mitchell, Snowfield), and transitional epithermal intrusion-related precious metal deposits (e.g., Brucejack, Silbak-Premier, Big Missouri, Red Mountain, and Homestake Ridge).

The Property is located on the eastern limb of the McTagg Anticlinorium, the northern closure of the Stewart-Iskut culmination (Figure 7.1). As a result, rocks on the Property are tilted, as well as folded, and generally display a progressive younging towards the east. Volcanic arc-related rocks of the Triassic Stuhini Group form the core of the anticlinorium, and are successively replaced outwards by volcanic arc-related rocks of the Lower Jurassic Hazelton Group and clastic basin-fill sedimentary rocks of the Middle to Upper Jurassic Bowser Lake Group.

The McTagg Anticlinorium is cut by a series of thrusts (e.g., south-east directed Mitchell Thrust) of mid-Cretaceous age, and late-stage brittle faults of probable Tertiary age, including the northerly trending Brucejack Fault.

Figure 7.1 Location of Brucejack and Snowfield Deposits in the Northwest-trending Structural Culmination of Lower Jurassic Rocks of the Stikine Terrane on the Western Side of the Bowser Basin



Source: Pretivm

7.2 BRUCEJACK PROPERTY GEOLOGY

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.

Gold (± silver) mineralization is hosted in predominantly sub-vertical vein, vein stockwork, and subordinate vein breccia systems of variable intensity, throughout the alteration band. The stockwork systems display both parallel and discordant relationships to stratigraphy. The stockwork systems are relatively continuous along strike (several tens of metres to several hundreds of metres).

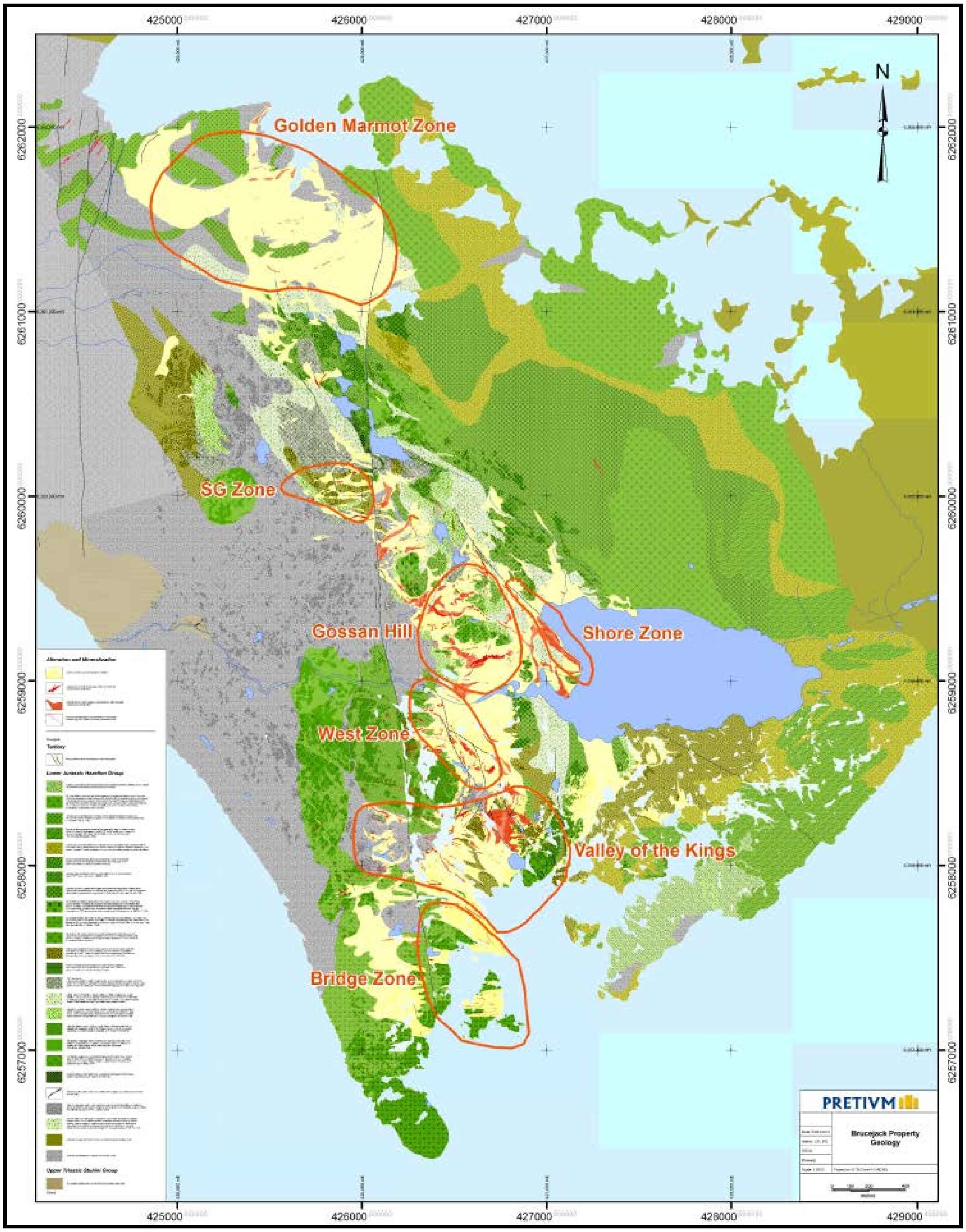
Several mineralization zones have been explored to varying degrees, including (from south to north): Bridge Zone, VOK, West Zone, Gossan Hill, Shore Zone, and SG Zone (Ireland et al. 2013) (Figure 7.2). There are numerous relatively unexplored mineralization showings within the alteration band across the Property that are between the main mineralization zones, highlighting the exceptional exploration potential of the Property.

High grade gold mineralization in the VOK, the current focus of the Project, occurs in a series of west-northwest (and subordinate west-southwest) trending sub-vertical corridors of structurally reoriented vein stockworks and vein breccias. Stockwork mineralization displays both discordant and concordant relationships to the volcanic pile stratigraphy. 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 g/t gold and 27,725 g/t silver over 0.5 m.

Recent underground exploration carried out as a part of the bulk sample program confirmed the location of corridors of stockwork-style mineralization and the lithological contacts in this part of the deposit (within the VOK). In addition the work resulted in the recognition of sub-vertical north-northeasterly trending deformed, curviplanar, and sheared quartz-carbonate veins containing abundant visible electrum. These structures are interpreted as structurally-controlled fluid conduits that were active during development of the porphyry system and associated volcanic pile in the early Jurassic, and which were reactivated during Cretaceous deformation.

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. The West Zone appears to form the northern limb of an anticline that links up with the VOK in the south, and the southern limb of a syncline that extends further to the north. This zone, which is currently defined over 590 m along its northwest strike, 560 m across strike, and down to 650 m in depth, is open to the northwest, southeast, and at depth to the northeast.

Figure 7.2 Geological Map of the Property Showing Location of Defined Mineralized Zones and their Association with the Arcuate Band of Quartz-sericite-pyrite Alteration (shown in yellow)



Note: Enlarged legend provided in Figure 7.3
Source: Pretivm

Figure 7.3 Brucejack Property Geology Legend for Figure 7.2

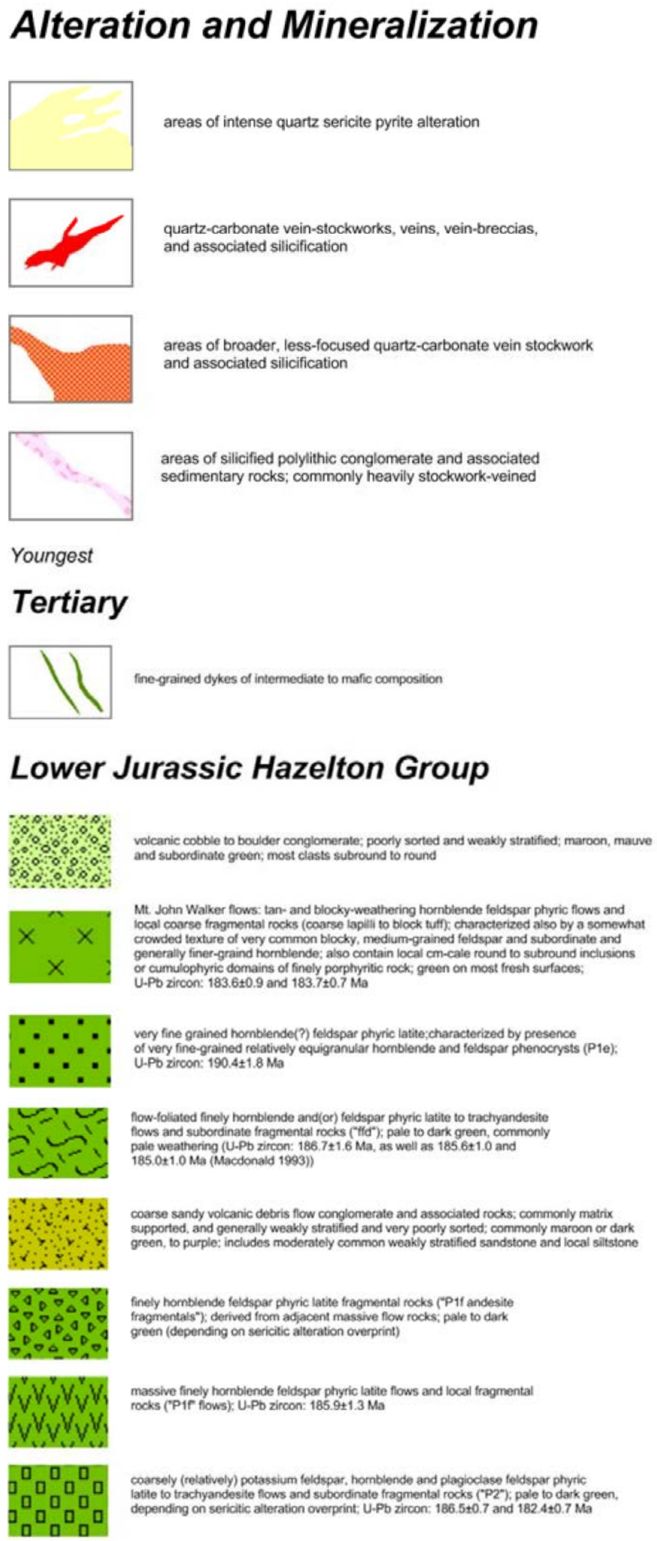


figure continues…

Figure 7.3 *(con't)* Brucejack Property Geology Legend for Figure 7.2

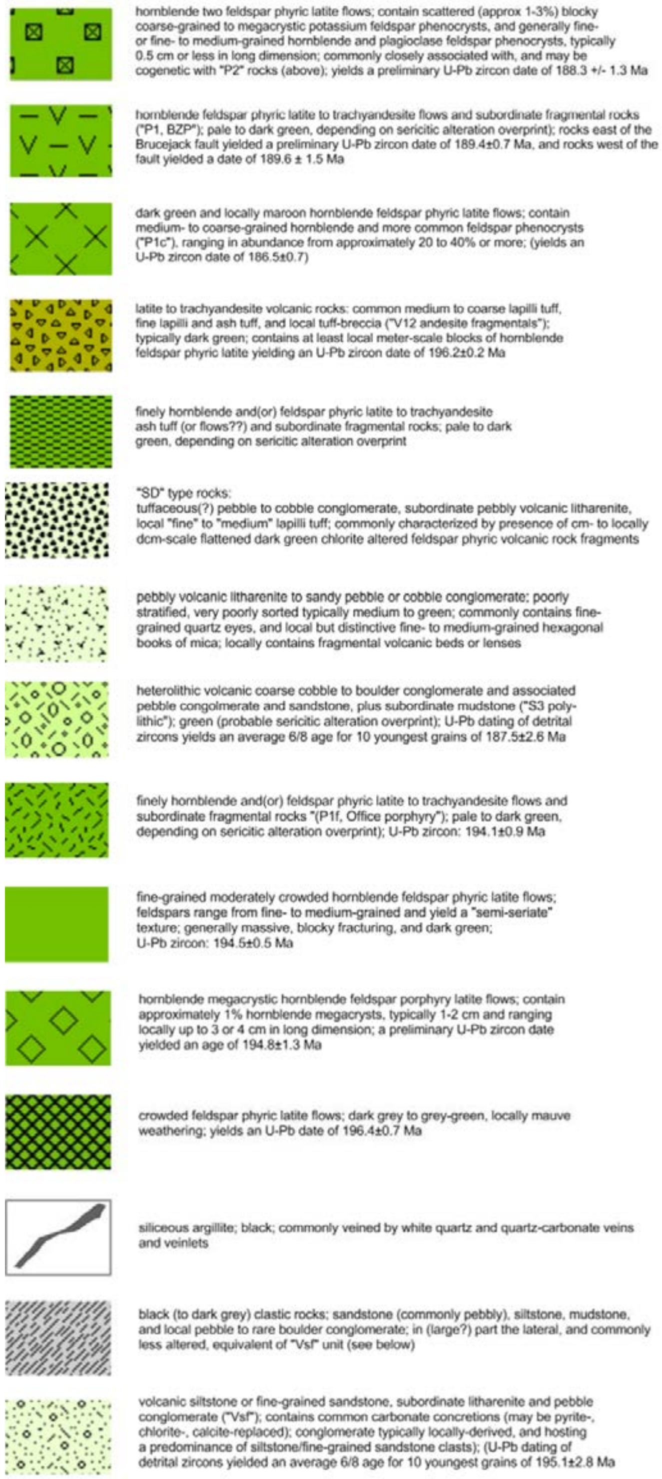


figure continues…

Figure 7.3 *(con't)* **Brucejack Property Geology Legend for Figure 7.2**

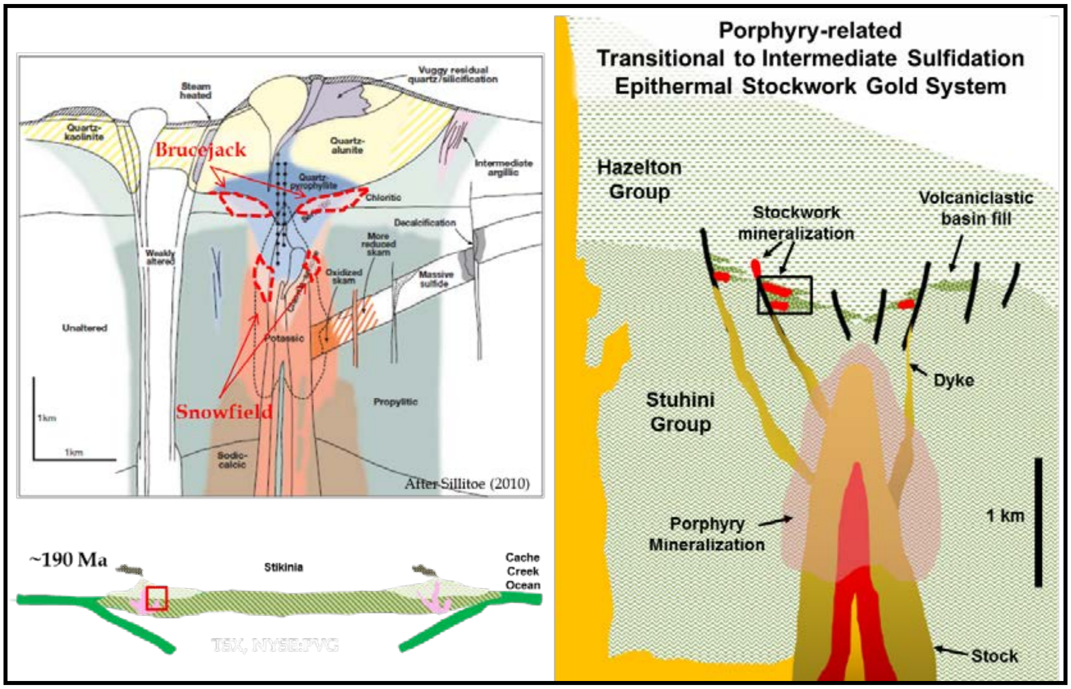
8.0 DEPOSIT TYPES

Information in this section has been excerpted and condensed from Jones (2013). The reader should refer to Jones (2013) for detailed information.

Mineralized zones on the Property are considered to represent a deformed porphyry-related transitional to intermediate sulphidation epithermal high-grade gold-silver vein, vein stockwork, and vein breccia system that formed between approximately 192 to 190 Ma and 184 Ma (Figure 8.1). Initial disseminated mineralization and sulphidation of the host rocks occurred within the evolving intra-arc basin. Progressive development and telescoping of the 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. Epithermal mineralization is considered to have been superimposed on earlier porphyry-associated alteration and mineralization between approximately 185 Ma and 183 Ma, utilizing the structural framework generated in response to syn-arc deformation. Intrusion of post-mineral intermediate dykes at circa 183 Ma reflect the waning of the system.

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 sea-water hydrothermal system developed above the pulsing porphyry system.

Figure 8.1 Schematic Showing Relative Position of the Brucejack Deposit to a Porphyry Copper-gold System



Source: Pretivm

9.0 EXPLORATION

Information in this section has been excerpted and condensed from Jones (2013). The reader should refer to Jones (2013) for detailed information.

In 2011, following the acquisition of the Project in late 2010, Pretivm management decided to shift the exploration focus from the open pit bulk-tonnage approach in favour of a more selective underground high-grade mining approach. Table 9.1 provides a summary of the exploration carried out since the acquisition of the Project.

Table 9.1 Exploration of the Brucejack Deposit

Date	Exploration
2011	A bulk-tonnage resource update was released in February 2011 with a high-grade sensitivity for the VOK.
	Brownfields exploration included detailed surface geological mapping, limited surface sampling, and limited geophysics (Spartan magnetotelluric survey; refer to Ireland et al. 2013).
	A total of 178 diamond drillholes were completed, totalling 72,805 m. The program targeted previously defined high-grade intersections primarily in the VOK (60% of the total), but also in the Gossan Hill, Shore, West, and Bridge zones.
	Dewatering of the historical West Zone underground development was carried out to assess the condition of the workings and determine if the workings could be used as a launching point for a development drive to the VOK.
2012	Detailed Brownfields surface geological mapping and associated supplementary surface geochemical sampling was continued.
	A total of 301 drillholes were completed, totalling 105,500 m of drilling during the 2012 drilling program. Zones within 150 m of surface were drilled at 12.5 m centres, with the deeper parts (down to about 350 m below surface) being drilled at approximately 25 m centres. Drilling at greater depths was generally only able to reliably achieve 50 m centres.
	The results of the 2012 drilling were incorporated into a revised Mineral Resource estimate (Jones 2012b). This resource estimate formed the basis for a feasibility study on the Property, which was completed in June 2013 (Ireland et al. 2013).
2013	A total of 24 surface diamond drillholes (5,200 m) were completed in drillholes SU-590 to SU-626.
	Surface geological mapping and supplementary surface geochemical sampling was continued albeit with a more Greenfields exploration goal (i.e., focussing on the broader area within Pretivm's claims) than in previous years.
	Pretivm elected to extract a bulk sample to further evaluate the geological interpretation and Mineral Resource estimate for the VOK deposit as discussed below.

9.1 BULK SAMPLE

The original bulk sample location and layout was discussed and prepared in agreement with the input of independent consultants. The bulk sample location was selected such that the mineralization, grade, and drilling density in the bulk sample area was representative of the Mineral Resources from the November 2012 Mineral Resource estimate, and covered areas of both low- and high-grade mineralization, so as to test the predictability of the Mineral Resource estimate across the grade spectrum. Figure 9.1 illustrates the location of the planned and actual bulk sample collected.

Owing to the legislative restrictions on the maximum extractable tonnage as part of the bulk sample program (10,000 t), Pretivm decided to test a larger area around the bulk sample workings through underground drilling. A total of 16,500 m of underground drilling at 7.5 m centres was planned to drill off an area measuring 120 m along strike, 60 to 90 m across strike, and 60 m above and below the 1,345 m Level. An additional 16,500 m of underground exploration drilling was designed to test targets outside of the bulk sample area.

Geological mapping (face, back, and ribs), channel, and chip sampling were conducted on a round-by-round basis.

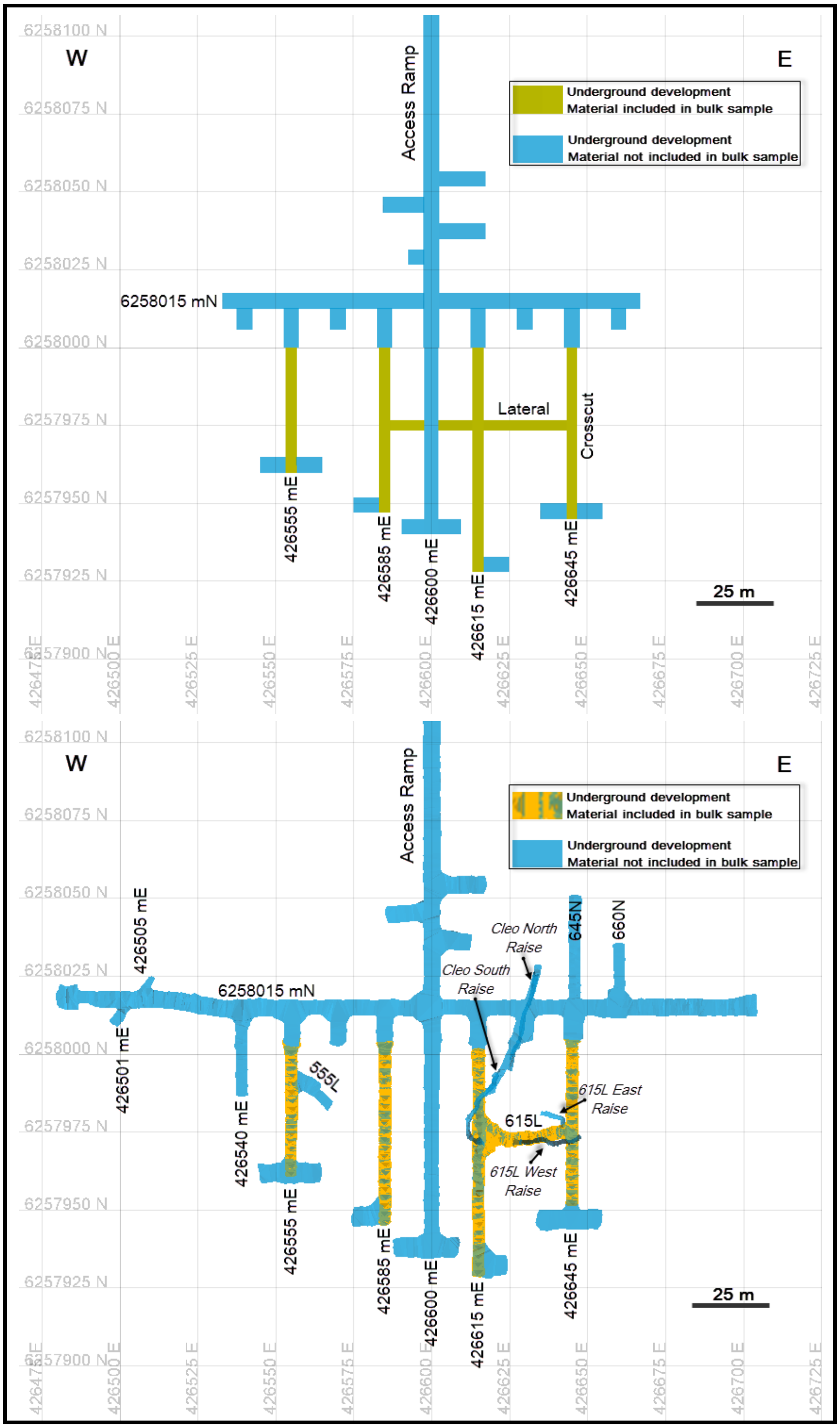
The bulk sample was collected in a series of nominal 100-t rounds in underground development, and processed through a sample tower on site. Each nominal 100-t round was split down to two 30 kg samples after processing through the sample tower, however, the variability in grades from the sample tower on each 100-t round was considered too high to give an accurate representation of the grade of each round.

The bulk sample material from each round (minus the sample splits collected at the sample tower) was sent as defined parcels to the Contact Mill in Philipsburg, Montana, US, for processing. This was done to provide a comprehensive dataset for reconciliation purposes.

Mill results from processing the bulk sample were used to test the validity of the estimation method used for the previous November 2013 Mineral Resource and to potentially refine the method for the updated December 2013 Mineral Resource (Section 13.2).

The results of the processing were compared with the results of the sample tower and this confirmed the poor accuracy of the sample tower results. There was no further use of the sample tower results.

Figure 9.1 Planned (top) Versus Actual Completed (bottom) Bulk Sample Area Layout on the 1,345 m Level, VOK Deposit



10.0 DRILLING

Information in this section has been excerpted from Jones (2013). The reader should refer to Jones (2013) for detailed information.

Table 10.1 summarizes the drilling carried out on the Property.

Table 10.1 Drilling on the Brucejack Deposit

Date	Exploration
Historical 1960-1990	• 452 surface diamond drillholes (60,854 m) in West Zone, Shore Zone, Galena Hill and Gossan Hill. • 442 underground diamond drillholes off the West Zone exploration ramp (33,750 m), providing a drill density of approximately 5 m centres between 5 m and 10 m spaced sections.
Silver Standard 2009	• 17,846 m in 37 drillholes, of which 2,913 m in 6 drillholes were targeted at the VOK. This program successfully discovered several areas with bulk tonnage mineralization within which were locally discreet high grade intersections
Silver Standard 2010	• 73 diamond drillholes were completed which totalled 33,480 m. Of this, 11 drillholes comprising 3,693 m were targeted at the VOK, and two drillholes, totalling 1,119 m at the footwall of West Zone. In the VOK, wide spaced drilling intersected enough high grade mineralization to confirm the exploration potential of the zone.
Pretium 2011	• 178 drillholes were completed totalling 72,805 m, focusing on defining high grade resources. Included in this were 97 drillholes (41,219 m) targeted at the VOK, 16 drillholes (7,471 m) at West Zone, and 21 drillholes (7,220 m) targeting the surrounding areas.
Pretium 2012	• The 2012 diamond drill program was focused on defining the high grade resource in the VOK. 301 drillholes were completed, totalling 105,500 m of drilling during the 2012 drilling program. Zones within 150 m of surface were drilled at 12.5 m centres, with the deeper parts (down to about 350 m below surface) being drilled at approximately 25 m centres. Drilling at greater depths was generally only able to reliably achieve 50 m centres.
Pretium 2012	• As part of the bulk sample program, 409 underground diamond drillholes (38,840 m) were completed with 200 of these drillholes (16,640 m) being in the bulk sample area, and the remainder (209 drillholes totalling 22,200 m) testing targets outside of the bulk sample area. • Drillholes range from 12 m to 450 m in length, with most drillholes being between 50 m and 150 m in length.

For the 2013 surface drilling program, the drilling contractors were Hy-Tech Drilling Limited from Smithers, BC. The drill collars were surveyed by McElhanney Surveying from Terrace, BC. McElhanney Surveying used a total station instrument and permanent ground control stations for reference and completed all the surveying on the Project since 2009. All underground drill collars were surveyed by Procon.

Drillhole paths were surveyed at a nominal 50 m interval using a Reflex EZ single shot instrument. All drillhole paths were checked in a mining software package for deviation errors, which, if present, were corrected on a real-time basis.

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 West Zone underground exploration ramp was AQ (27 mm diameter).

Core sizes for Pretivm's drillholes were PQ (85 mm diameter), HQ (63.5 mm diameter) and NQ (47.6 mm diameter). Approximately 50 to 60% of core was HQ size. All drill core collected from the underground drilling in 2013 was HQ size.

Geotechnical and geological logging was carried out after the entire drill core was photographed. A maximum sample length of 2 m was used with the geologist ensuring samples did not cross geologic contacts. Sample lengths generally averaged 1.5 m. Every drillhole was sampled in its entirety from top to bottom.

It is the author's opinion that the core logging procedures employed were thorough and provided the appropriate level and quality of information required to model the geological and geotechnical aspects of the deposit. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results. The author believes that drilling has been conducted using industry standard practices and that the drillhole sample data is appropriate for use in grade estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Information in this section has been excerpted and condensed from Jones (2013). The reader should refer to Jones (2013) for detailed information.

11.1 SAMPLE PREPARATION AND ANALYSIS

Pretivm's QP for field activities is Mr. Kenneth McNaughton, P.Eng., Chief Exploration Officer.

Drill core samples were either split or sawn in half lengthwise before half core samples were delivered the ALS Minerals sample preparation facility in Terrace, BC. The average sample size was 6.5 kg.

Chip and channel samples taken as part of the underground bulk sample program during 2013 were also delivered to the ALS Minerals sample preparation facility as per the drill core samples. All sample preparation and analysis was the same for drill core, channel, and chip samples.

ALS Global (ALS) was the primary laboratory, with the SGS Canada (SGS) laboratory used as an umpire (check laboratory), both of which are located in Vancouver, BC. The ALS analytical laboratory in Vancouver has been accredited to International Organization for Standardization (ISO) 17025 standards for general testing laboratory procedures by the Standards Council of Canada (SCC).

Analysis included:

- Crush to 70% passing 2 mm, (-10 mesh), riffle split, and 500 g pulverized to 85% passing 75 µm (-200 mesh).
- Gold grade determined using fire assay on a 30 g aliquot with an atomic absorption (AA) finish. A 33 element package which included silver was also assessed using a four acid digest and an ICP-AES analysis.
- The upper limit of acceptable accuracy for gold and silver was 10 ppm gold and 100 ppm silver, respectively, using these methods. For sample results above these levels, the gold and silver content was determined by gravimetric analysis.

Density determinations were also carried out including:

- A total of 2,621 pulp specific gravity determinations carried out by ALS.

- As part of the 2012 surface drilling and 2013 underground drilling programs, Pretivm selected a portion of the samples (207 and 204 samples) for core density (water immersion method) as well as the pulp specific gravity measurements in order to determine the impact of porosity.

Results of the comparison between the pulp specific gravity and core density measurements indicate that the core density is on average the same as the pulp specific gravity within the siliceous zone and approximately 3% lower, on average, for all other rock types. Consequently all specific gravity estimates in the Mineral Resource model (which are based on the pulp specific gravity measurements), with the exception of the siliceous zone, were factored down by 3% to give the bulk density.

11.2 QUALITY ASSURANCE AND QUALITY CONTROL

Snowden analyzed the QA and QC data and accompanying documentation for the Project. GeoSpark managed the Brucejack drillhole and QA/QC database, as well as the routine analysis of the QA/QC results for Pretivm.

The QA/QC protocols included the use of field duplicates, standards and blanks. The QC samples were included at a nominal rate of one field duplicate, one standard and one blank for every 20 samples. Check assays, in the form of pulp duplicates, were also completed by a different laboratory (SGS) and compared with the primary laboratory.

Results of the QC analyses show that there are acceptable levels of precision (90% of the diamond core duplicate pairs have a difference of less than 30% for gold and 25% for silver based on the half absolute relative difference (HARD) statistic) and accuracy given the nuggetty nature of the mineralization at Brucejack. Overall, contamination during the sample preparation and assaying is considered reasonable and within acceptable tolerance intervals. Pulp check assays for gold show a good comparison between assays at ALS and SGS and that a good level of precision is being achieved for the pulp duplicates.

11.3 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 Pretivm's staff, as far as drill core sampling prior to dispatch. Laboratory sample reduction and analytical procedures have been conducted by independent accredited companies using industry standard methods.

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

It is the author's opinion that the sample preparation, sample security, and analytical procedures were satisfactory and appropriate for resource evaluation of Brucejack.

12.0 DATA VERIFICATION

Information in this section has been excerpted and condensed from Jones (2013). The reader should refer to Jones (2013) for detailed information.

Independent sampling and site verification visits were undertaken by Snowden in 2012 and 2013. Mr. Ivor Jones, FAusIMM, Executive Consultant from Snowden's Brisbane office visited the Property on February 15 and 16, 2012, June 3 to 6, 2013 and August 16 to 21, 2013. In addition, Ms. Lynn Olssen, MAusIMM (CP), Senior Principal Consultant from Snowden's Perth office and Mr. Harald Muller, FAusIMM, Senior Principal Consultant from Snowden's Brisbane office, visited the Property from August 16 to 21 2013.

During the site visits, Snowden verified the sample preparation, handling and security procedures on site and in Stewart. In addition Snowden reviewed the underground bulk sample cross cuts and drill core to confirm the nature of the mineralization and appropriateness of the definition of domains, and the estimation method

From June 8 to 10, 2012, Mr. Adrian Martínez Vargas from Snowden's Vancouver office completed the sample validation under the supervision of Ms. Lynn Olssen (the QP for this report) and Mr. Ivor Jones. Mr. Martinez reviewed the sample preparation, handling and security procedures while on site and took independent samples. The independent samples confirmed the grade of the mineralization. It should be noted that the samples taken are only validating the lower-grade parts of the mineral system. This was intentional as the high-grade samples have been verified by independent laboratories.

Snowden also carried out a basic statistical and visual validation of the data prior to estimation and found no significant issues.

The author has not undertaken a complete data verification study, however sufficient checks have been completed to satisfy the author that the Brucejack drilling and sampling data is suitable for use in Mineral Resource estimation.

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
2014	1208011	Inspectorate		√			√
2014	T1172	Gekko	√	√	√		
2014	P-14066	FLS-DM	√				√
2014	MS1542	Met-Solve		√			√
2013	1208011	Inspectorate	√	√	√	√	√
2012	11489	Hazen	-	-	√	-	-
2012	KRTS20734-A	Knelson	√	-	-	-	-

table continues…

Year	Program ID	Laboratory**	Gravity	Flotation	Grindability	Cyanidation	Others
2012	MS1399	Met-Solve	√	-	-	-	-
2012	Feb2012-01	PMCL					√
2012	-	Pocock	-	-	-	-	√
2012	12012	JZM					√
2012	1106811	Inspectorate	√	√	√	√	√
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 or Metallurgical Division, Inspectorate Exploration and Mining Services Ltd.; Met-Solve = Met-Solve Laboratories Inc.; Knelson = FLSmidth Knelson; Pocock = Pocock Industrial Inc.; JZM = Joe Zhou Mineralogy Ltd; PMCL = Process Mineralogical Consulting Ltd.; FLS-DM = FLSmidth Dawson Metallurgical; Gekko = Gekko Systems Pty 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 (West Zone)

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 2014 TEST WORK

Inspectorate, formerly known as Process Research Associates Ltd., 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.

In 2014, Pretivm contracted Inspectorate, Gekko, FLSmidth and Met-Solve to conduct further laboratory test work to investigate metallurgical responses of VOK mineral samples to gravity and flotation concentrations. Gekko tested the amenability of the VOK samples to vertical shaft impact (VSI) crushing. FLS-DM conducted smelting tests on a gravity concentrate upgraded by tabling.

Between September 2013 and February 2014, Pretivm contracted Strategic Minerals LLC to process two batches of bulk mineral samples generated from the VOK deposit of the Property using the Contact Mill facility located in Philipsburg, Montana. The samples processed were approximately 10,300 t for the first campaign and approximately 1,200 t for the second run. Figure 9.1 illustrates the collection locations for the bulk samples.

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 2014 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.

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. The 2014 tests conducted by Inspectorate used the 2013 composite samples.

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

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	-	-	-	-	-	-

The samples used by Gekko, FLSmidth and Met-Solve for the 2014 testing mainly were from the VOK bulk sampling program conducted in 2013.

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

table continues…

Sample ID	Zone	Hole ID
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	-	-
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 Metallics (g/t)	Ag Metallics (g/t)	Ag 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	Head Grade (g/t) 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.

In 2012, PMCL conducted mineralogical examination on six mineral samples generated from the composite samples for the 2012 test work. The samples were identified as Samples VOK1, VOK2, VOK3, VOK4, WZ1, and WZ2. The purpose of the examination was to determine the deportment of gold and silver bearing minerals within each sample, including mineral size, distribution and association. The study showed:

- The samples are primarily composed of non-opaque gangue minerals, which include quartz, K-feldspar and muscovite with minor amounts of pyrite, carbonates, pyroxene and Mg-silicates (clinochlore).
- The primary gold bearing mineral present in all samples is electrum. The Scanning Electron Microscopes-Energy Dispersive Spectrometer (SEM-EDS) analysis of the electrum grains indicates that the gold content ranges between 50 to 75%.
- Distribution of gold between the heavy liquid separation products indicates that approximately 80% and greater of the gold was recovered into the heavy liquid sink products for most samples, excluding Sample WZ-2 which has approximately 70% of the gold recovered into the sink products.
- Gold-bearing minerals occur primarily as fine grains ranging in size from 2 to 10 µm. Samples VOK-1 and WZ-2 illustrate a fairly tight size range between 2 and 16 µm. Sample VOK-2 contains gold with a broader range of gold from 2 to

32 µm. Gold present in Samples VOK-3, VOK-4, and WZ-1 illustrates a finer distribution of gold, mainly less than 8 µm in size.

- Textural determinations made by Backscatter Electron Imaging indicate that gold-bearing minerals occur as fracture fillings in pyrite and as disseminated grains and inclusions interstitial to grain boundaries of non-opaque gangue minerals and pyrite.

Optical examination of the samples yielded no evidence of organic carbon or graphitic carbon. Carbonate minerals are present in each sample in minor amounts (approximately 2 to 3%).

The program also studied silver bearing mineral deportment on Samples VOK-3, VOK-4, and WZ-2 . Silver speciation of the three samples indicated differences in silver occurrence between each sample. The findings are summarized as follows:

- Sample VOK-3 showed that electrum was the primary Ag-bearing mineral with only minor amounts occurring as polybasite, tetrahedrite, acanthite and native silver. Trace amounts of selenopolybasite, stephanite, hessite, petzite, andorite, aguillarite and argentotennantite were also observed. Trace amounts of silver are also present in galena as shown by SEM-EDS analysis. Electrum is mainly present as liberated grains and to a lesser amount as exposed grains on pyrite and non-opaque gangue minerals.
- Sample VOK-4 showed that silver is mainly present in polybasite and acanthite. Moderate amounts of silver are present as tetrahedrite and native silver, while minor amounts of silver are present in selenopolybasite, argentotennantite, electrum, and stephanite. Trace amounts of silver were also observed by SEM-EDS analysis within galena present in the sample. Significant amounts of silver-bearing minerals present in the sample occur as liberated grains with subordinate amounts associated with pyrite and non-opaque gangue minerals.
- Sample WZ-2 is mainly polybasite, selenopolybasite with lesser amounts present as tetrahedrite. Minor amounts of silver are present as argentotennantite, acanthite and andorite. Trace amounts are present as electrum and as silver in galena as well as stephanite. The primary host minerals of silver occur as liberated grains with lesser amounts occurring as grains associated with non-opaque gangue minerals and reduced amounts associated with pyrite.

Mineral abundance of the as-received material for each sample determined by Tescan Integrated Mineral Analyzer is showed in Table 13.12.

Table 13.12 Mineral Abundance of the Samples

Mineral	WZ-1	WZ-2	VOK 1	VOK 2	VOK 3	VOK 4
Quartz	33.9	46.8	43.3	37.8	37.8	41.3
Feldspar	22.4	18.6	27.5	31.1	31.1	29.3
Muscovite	32.3	20.5	11.0	10.8	10.8	10.7
Pyrite/Pyrrhotite	5.60	8.35	5.91	7.42	7.52	6.51
Carbonates	2.69	2.18	3.99	3.41	3.43	3.46
Mg-Silicates	0.21	0.10	2.96	2.93	2.96	1.86
Pyroxene	1.74	2.16	2.12	2.10	2.17	3.20
Plagioclase	0.70	0.44	2.11	3.06	3.01	2.79
Phosphates	0.31	0.33	0.41	0.52	0.40	0.36
Ti-Bearing Minerals	0.14	0.12	0.35	0.48	0.46	0.21
Garnet	0.00	0.00	0.08	0.05	0.05	0.03
Arsenopyrite	0.01	0.00	0.04	0.09	0.08	0.06
Sphalerite	0.04	0.17	0.02	0.04	0.03	0.02
Tungsten Minerals	0.02	0.04	0.02	0.02	0.03	0.10
Chalcopyrite	0.00	0.00	0.02	0.00	0.02	0.00
Galena	0.00	0.03	0.00	0.00	0.01	0.04
Other	0.05	0.18	0.10	0.09	0.10	0.14
Total	**100**	**100**	**100**	**100**	**100**	**100**

In 2014, Inspectorate conducted mineralogical determination on Composite SEU & MU containing approximately 9.9 g/t gold and 10 g/t silver. The mineral compositions of the composite, measured by Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN) Particle Mineral Analysis (PMA) protocols, are summarized in Table 13.13. The findings are summarized as follows:

- Composite SEU & MU contains in total approximately 7.5% w/w sulphide minerals. Pyrite accounts for more than 99% w/w of the total sulphide minerals. Other present sulphide minerals in trace amounts are freibergite, chalcopyrite, bornite, sphalerite, galena, molybdenite, tetrahedrite, tennantite and arsenopyrite.
- The sulphide minerals are embraced in silicon rich non-sulphide gangue host, which dominantly incurs as quartz and muscovite/biotite. The remaining non-sulphide gangue is comprised of k-feldspar, chlorite, calcite, apatite and rutile/sphene.
- At a primary grind size of 80% passing 87 µm, 90% of the pyrite has been liberated. The unliberated pyrite is mostly interlocked with non-sulphide gangue.
- Six pieces of gold was spotted by the determination. All the gold grains are finer than 5 µm in circular diameter. The observed gold mostly occurs as liberated grains. The unliberated gold is associated with pyrite in binary form.

Table 13.13 Chemical and Mineral Composition of Composite SEU & MU

Chemical Assays			Mineral Content (%)			
Element	Unit	Content	Sulphide Minerals	Mass	Non-Sulphide Minerals	Mass
Gold	g/t	9.88	Freibergite	0.01	Iron Oxides	0.2
Silver	g/t	10.0	Chalcopyrite	0.02	Quartz	49.7
Copper	%	0.01	Galena	0.00	Muscovite/Biotite	28.3
Lead	%	0.01	Sphalerite	0.06	K-Feldspars	5.6
Zinc	%	0.03	Other Sulphides	0.00	Calcite	5.2
Iron	%	4.29	Pyrite	7.33	Chlorite	2.1
Sulphur	%	3.95	Arsenopyrite	0.04	Apatite	0.7
-	-	-	-	-	Others	0.7
-	-	-	**Total**	**7.46**	**Total**	**92.5**

Notes: Others include rutil/sphene, barite and corundum.

13.3.6 ORE HARDNESS TEST WORK

Table 13.14 and Table 13.15 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 13.14.

Table 13.14 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

table continues…

Sample ID	BWi (kWh/t)	Cut Particle Size (Screen Aperture) (µm)	RWi (kWh/t)	CWi (kWh/t)	UCS (psi)	Ai (g)
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 13.15 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).

In 2014, Gekko conducted crushability tests to determine the amenability of three different grade samples to VSI crushing. Gekko indicated that the samples are amenable to VSI crushing. The recirculating loads at a product particle size of 100% passing 1.18 mm were a range between approximately 340% and 460%. At a product particle size of 100% passing 2.36 mm, the recirculating loads reduced to approximately 200% for the low grade material and 225% for the high grade material.

13.3.7 SAMPLE SPECIFIC GRAVITY

The specific gravity of the Brucejack mineral samples are shown in Table 13.16 and Table 13.17. The specific gravity data varied narrowly from 2.71 to 2.87.

Table 13.16 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 13.17 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)

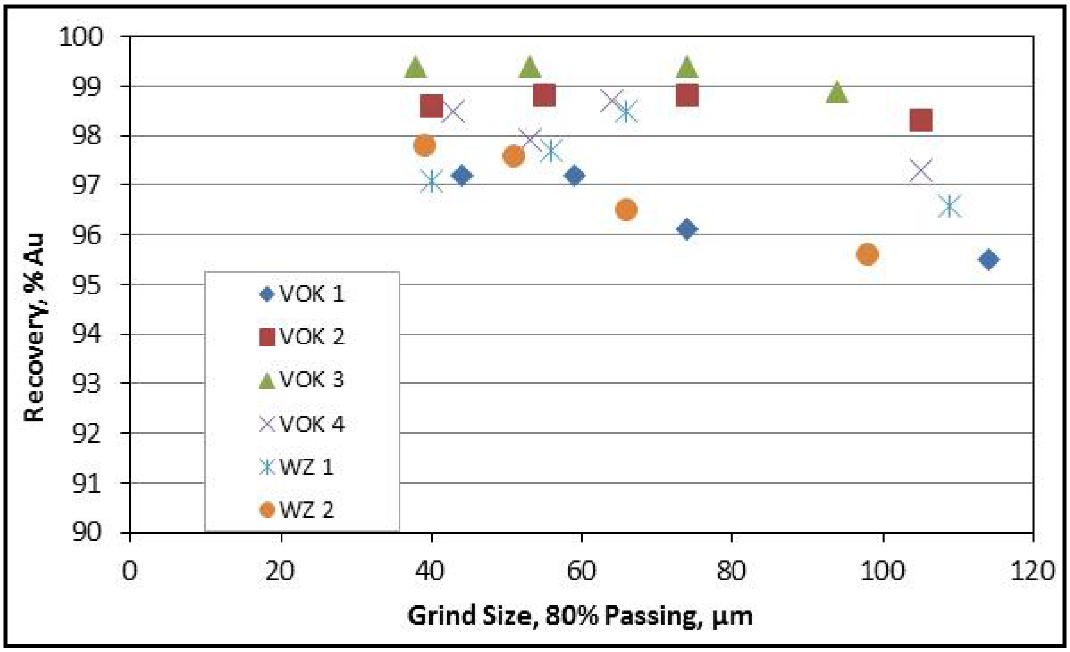
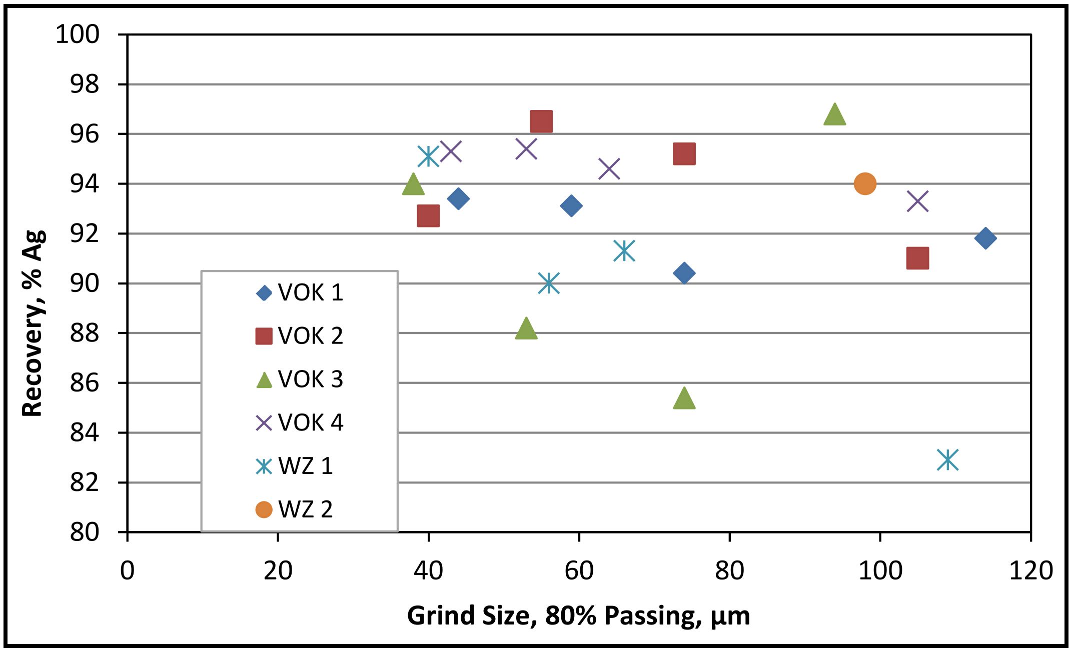


Figure 13.2 Silver Recovery versus Primary Grind Size (2012)



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)

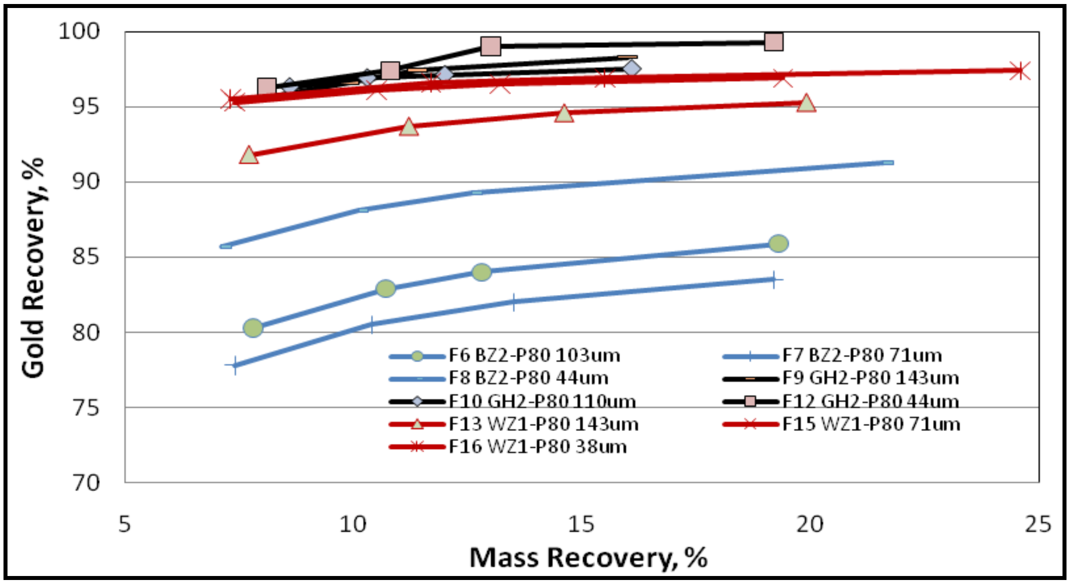
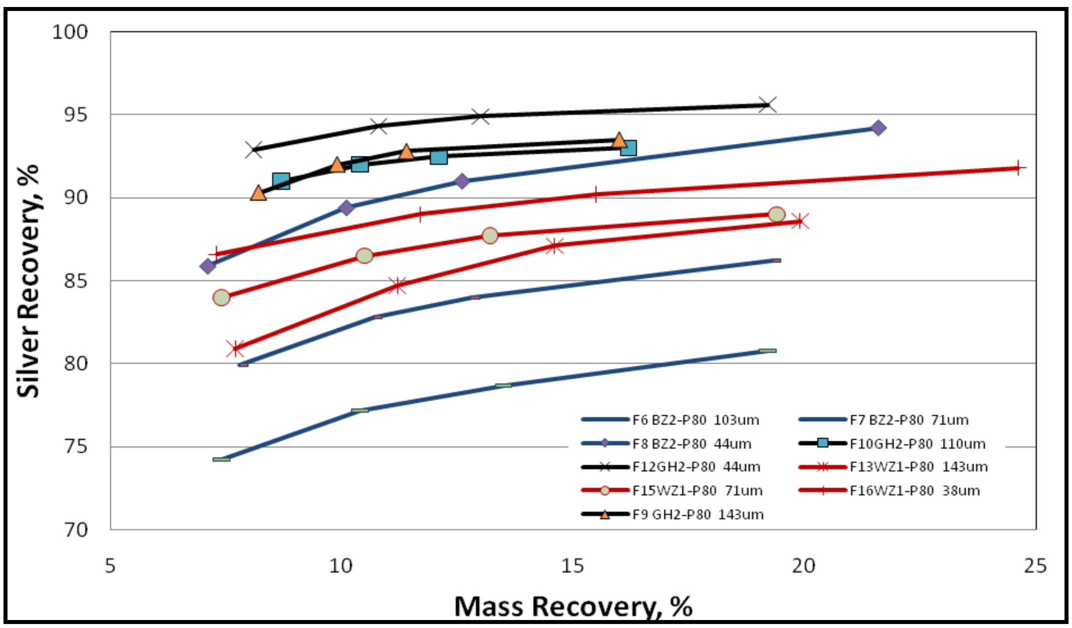


Figure 13.4 Effect of Primary Grind Size on Silver Recovery (2010 to 2011)

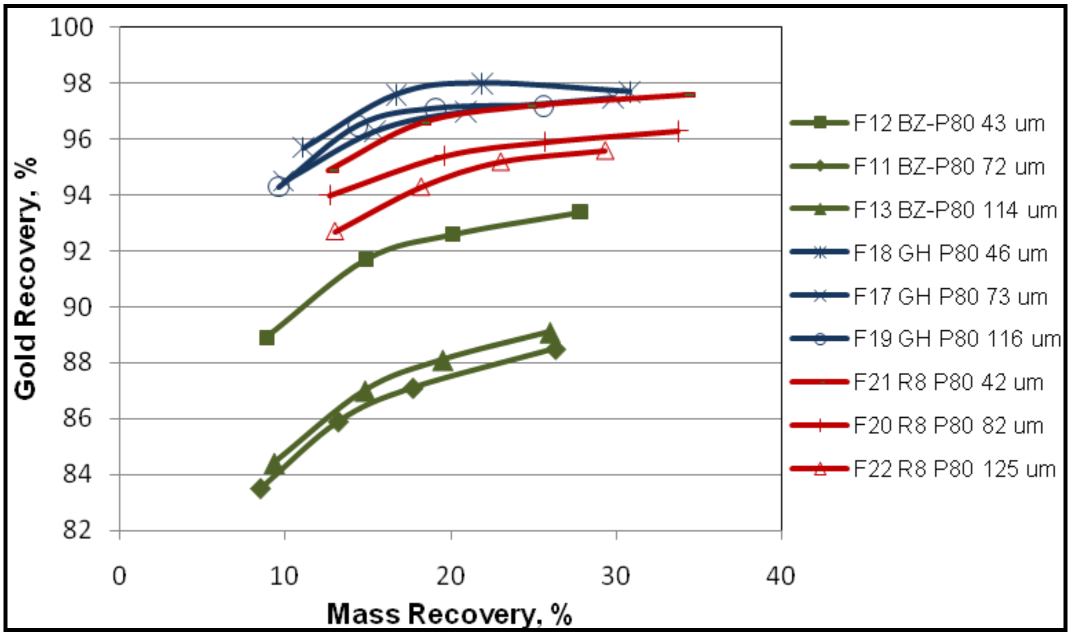


Gold and silver recoveries from the Galena Hill Zone sample and the West Zone sample were higher than the Bridge Zone sample. At a grind size of 80% passing 143 µm, the gold recoveries of the Galena Hill Zone sample and the West Zone sample were approximately 97% and 95%, respectively.

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)

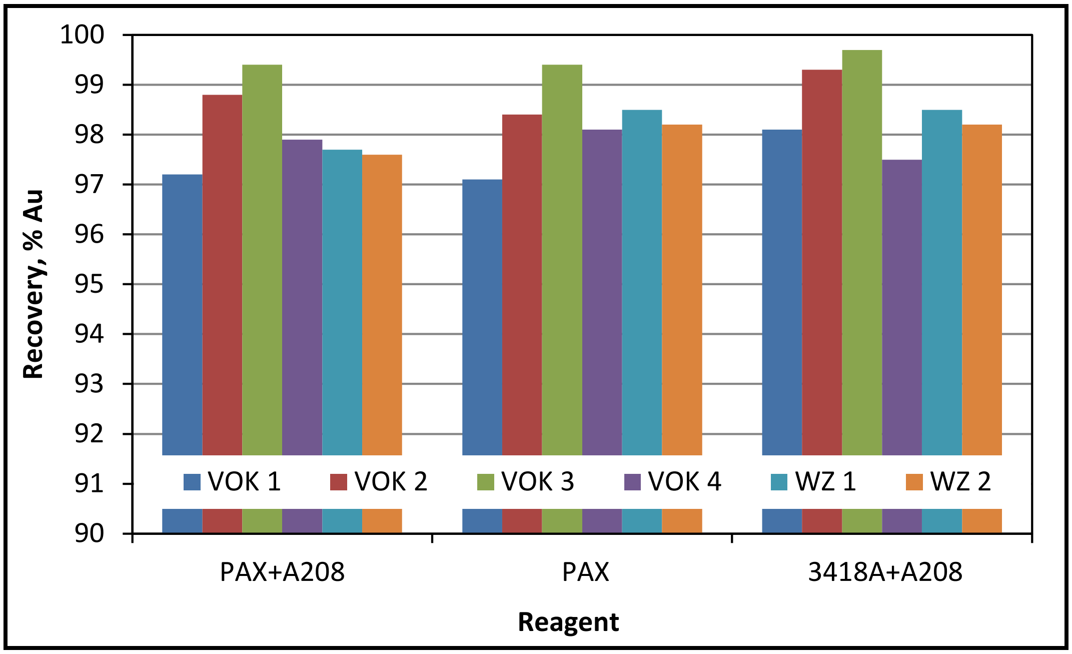
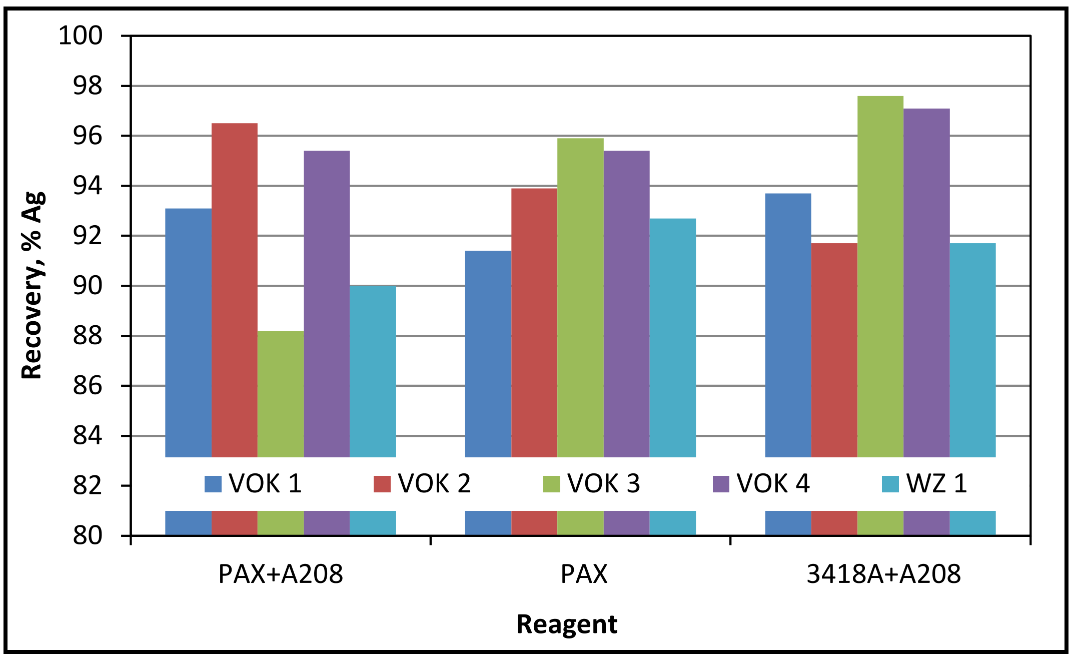


Figure 13.7 Collector Screening Tests – Silver Recovery (2012)

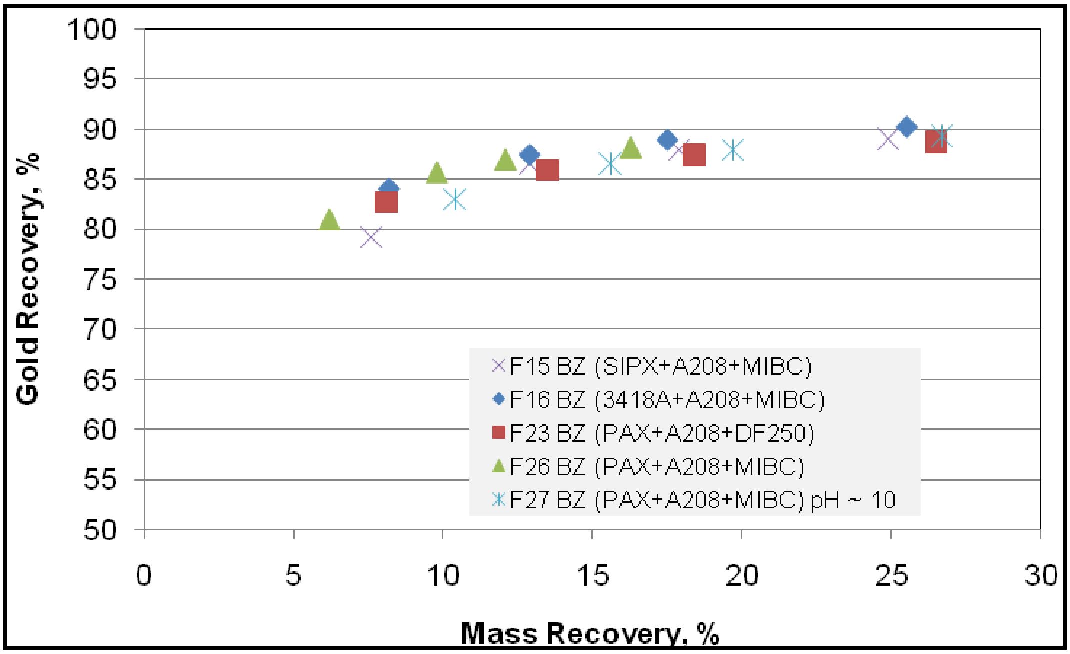


Copper sulphate has been used in the test work since 2009; however, the 2012 and 2013 test results indicate that the addition of copper sulphate does not improve metal recovery.

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)

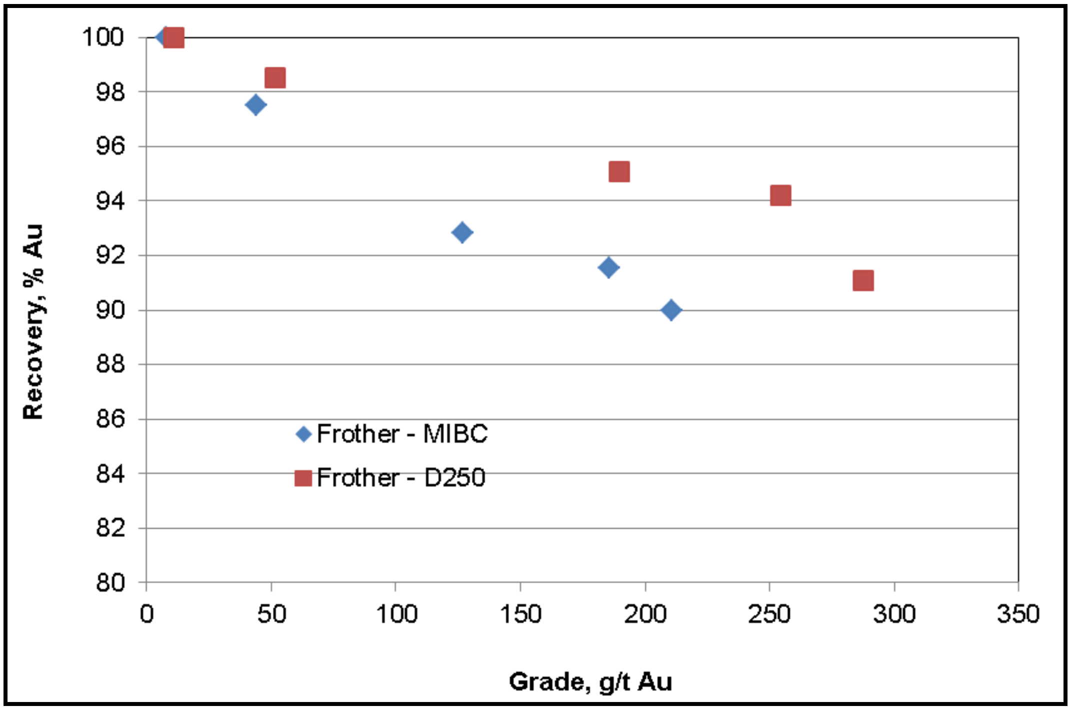
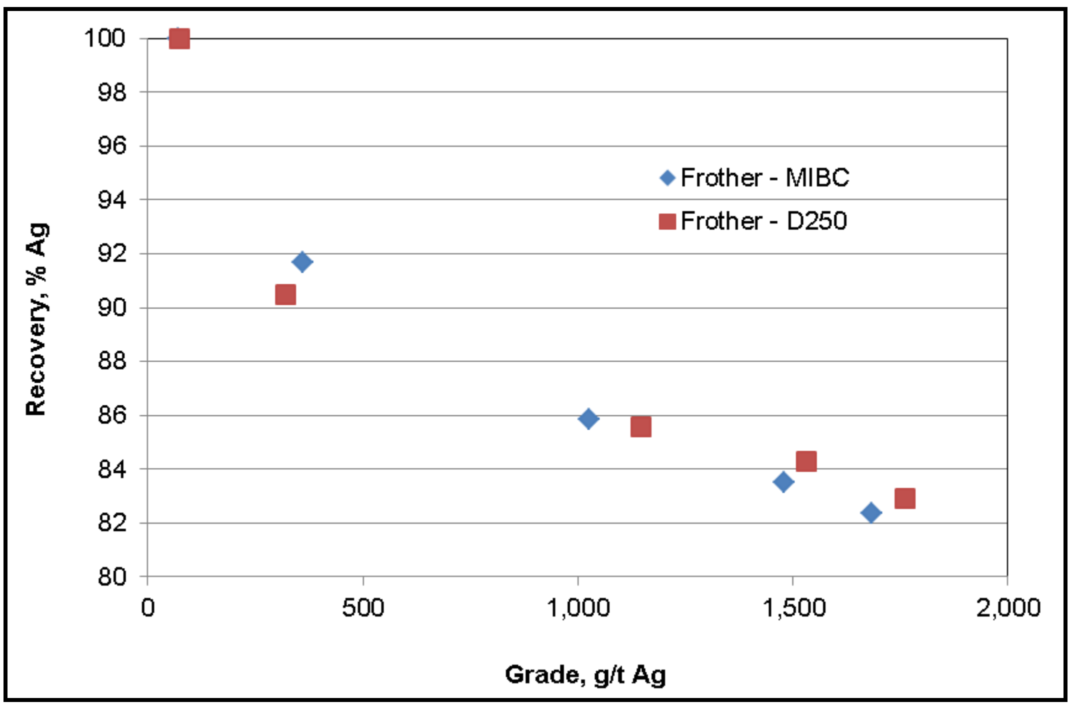


Figure 13.10 Effect of Cleaner Flotation on Silver Recovery (Composite, 2012)



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)

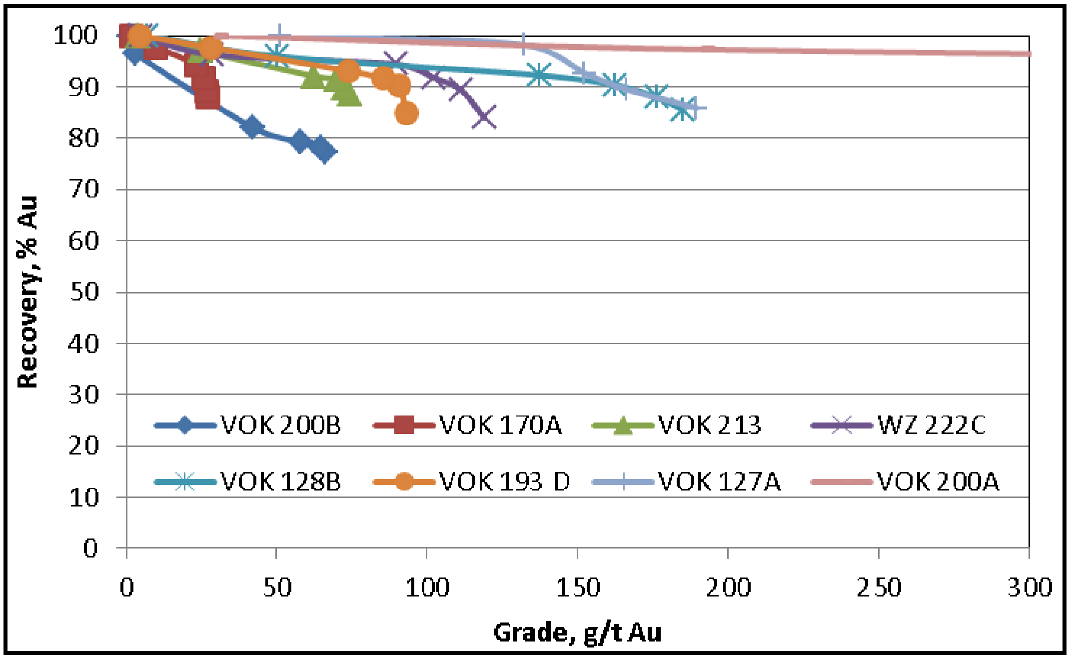


Figure 13.12 Effect of Cleaner Flotation on Gold Recovery (Composites, 2013)

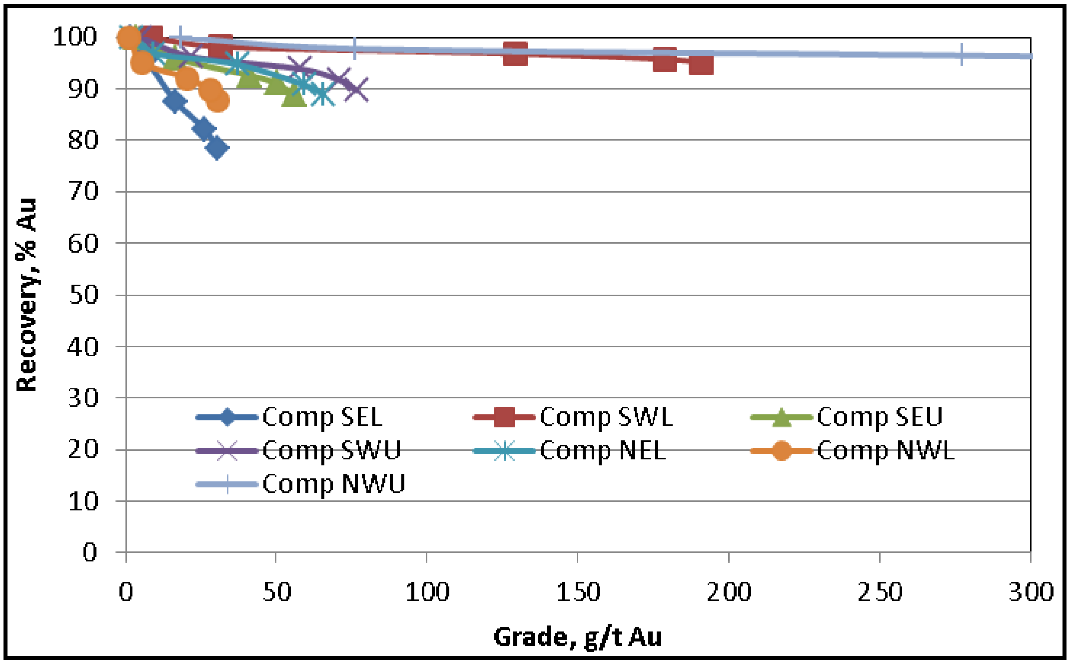


Figure 13.13 Effect of Cleaner Flotation on Silver Recovery (Interval Samples, 2012)

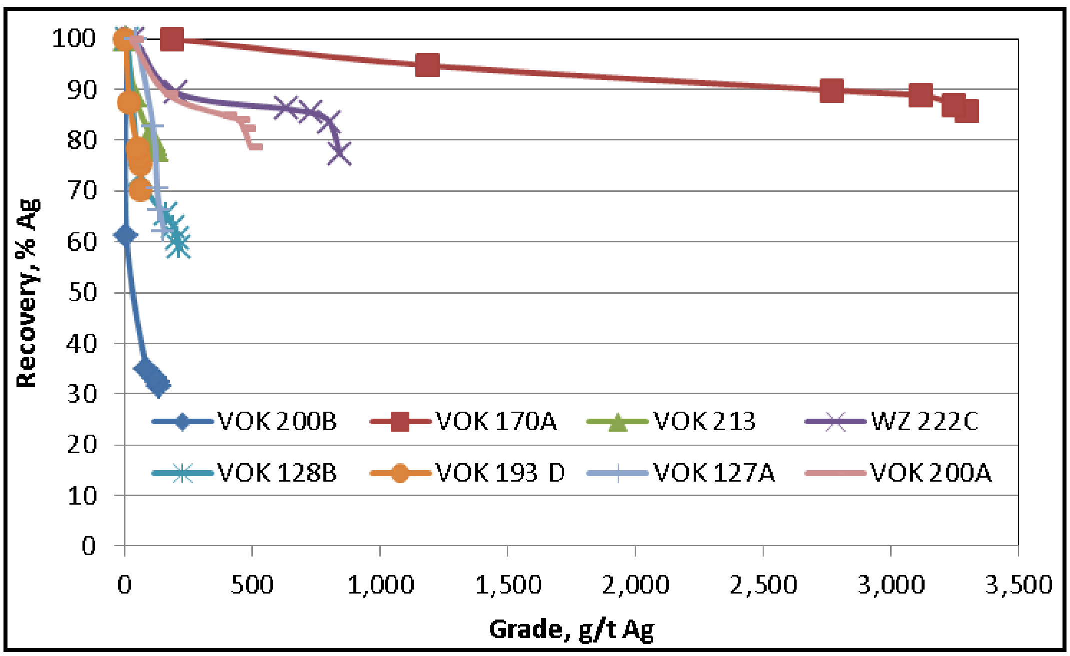
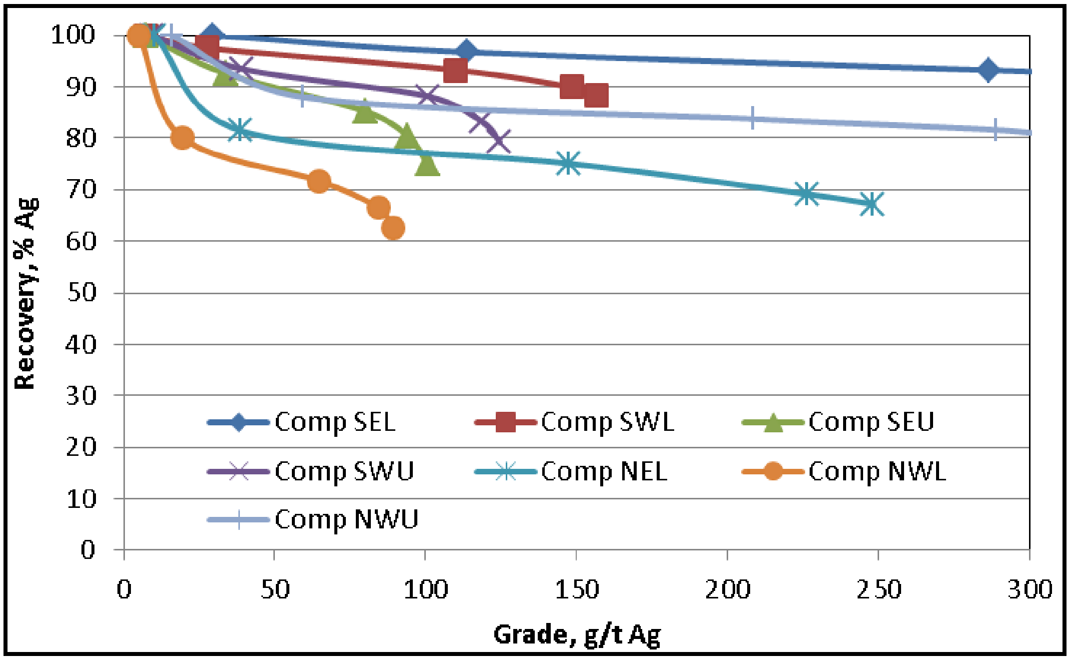
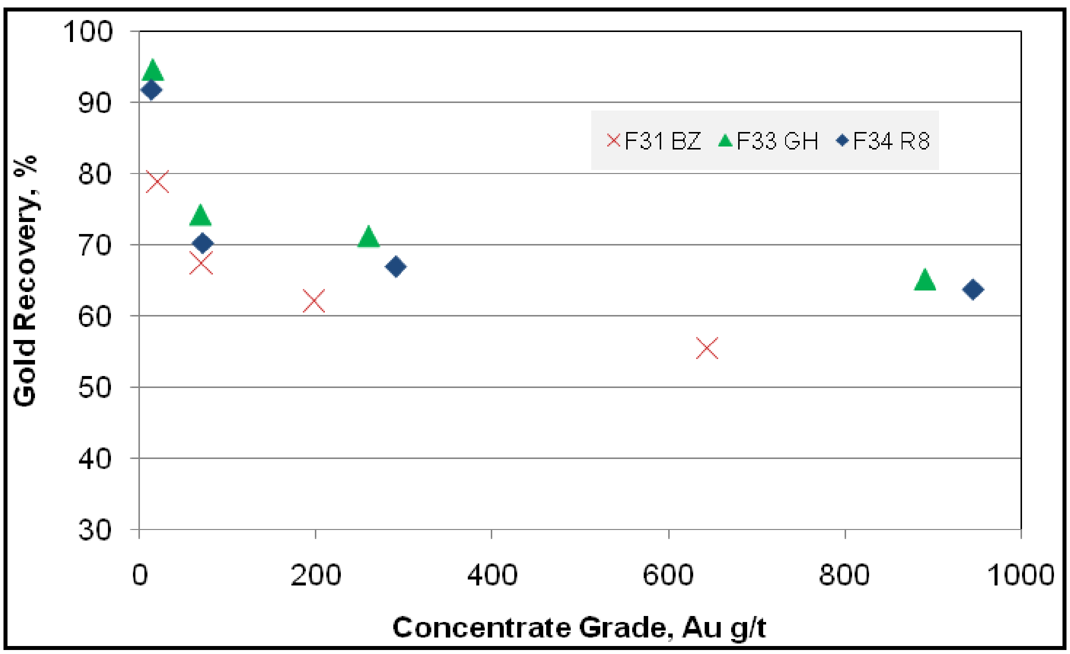


Figure 13.14 Effect of Cleaner Flotation on Silver Recovery (Composites, 2013)



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 to 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.

In 2014, Inspectorate conducted further test work in an effort to improve flotation concentrate grade for low grade materials using the previously developed gravity and flotation processing flowsheet. The test work investigated regrinding and other potential procedures to improve flotation concentrate grades, including reduced float retention time, increasing slurry pH using lime, altering collector type, and the use of a synthetic sulfide depressant. The test results indicated that regrinding of the bulk flotation concentrate prior to cleaning offered only minor benefits to increase product concentrate grade. Mineralogical examination on the concentrates showed that the reground concentrate contained approximately 98% of sulphide minerals, compared to 96% of sulphides in the concentrate produced without regrinding. The concentrates are mostly dominated by liberated pyrite. Other sulphide minerals, including silver bearing minerals, copper sulphides, galena and sphalerite, are all in relatively low amounts, and mostly presented as either liberated particles or binary particles interlocking with pyrite. The observed silver bearing minerals using SEM scanning are hessite, acanthite/argentite, freibergite, tetrahedrite and tennantite.

The 2014 test results by Inspectorate appeared to show an increased flotation retention time required in order to maintain gold recovery on the reground rougher concentrate. The retention time increase may be due to regrinding. The test work also showed that flotation concentrate grade could be improved by using aggressive flotation conditions. However, these procedures would significantly reduce gold recovery. Further test work using locked cycle testing was suggested to investigate the effect of the middlings streams on concentrate grade and gold recovery.

Met-Solve conducted a preliminary flotation test to recover gold from the fine fraction (slime, approximately 96% passing 37 µm) generated from a de-sliming classifier. The test result showed that approximately 83% of the gold could be recovered to a flotation concentrate at a mass recovery of 36%, in compared to a mass pull of 20% by a Falcon ultra-fine centrifugal concentrator. The test using a combined gravity and flotation flowsheet showed that a total gold recovery of 93 to 94% was achieved, including gravity separation recovering 84% of the gold. Flotation of this fine material required a significant amount of sodium silicate (Na_2SiO_3) due to a high viscosity slurry even at a low pulp density.

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 13.18 to Table 13.20.

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 13.18 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**
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 13.19 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**

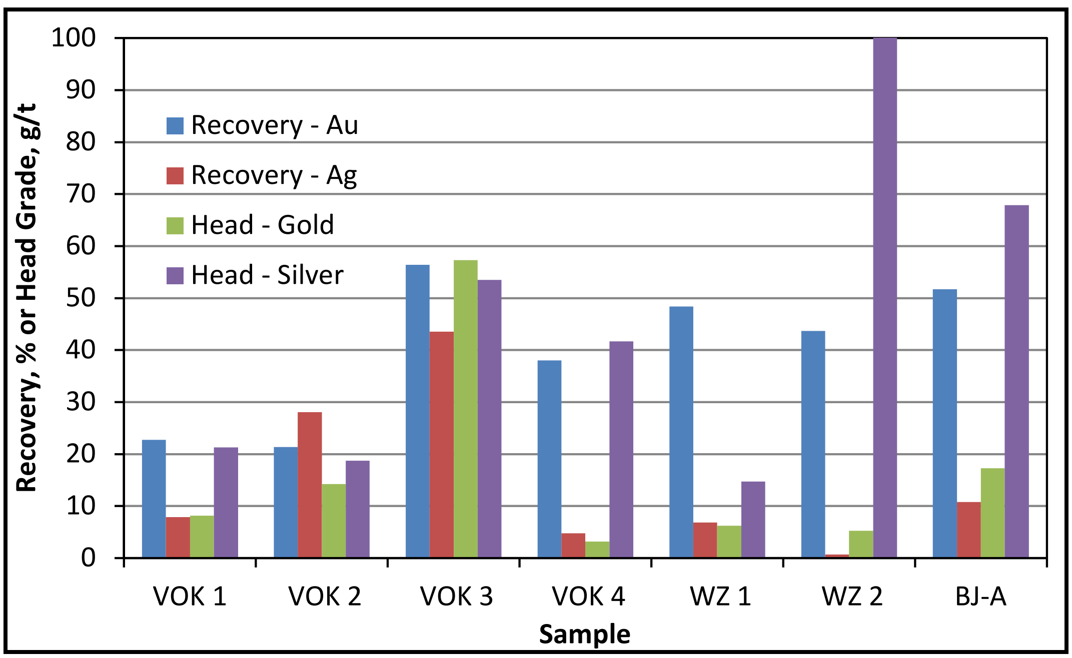
Table 13.20 Metallic Gold Test Results – Individual Samples (2009 to 2010)

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 13.21). 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 13.21 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

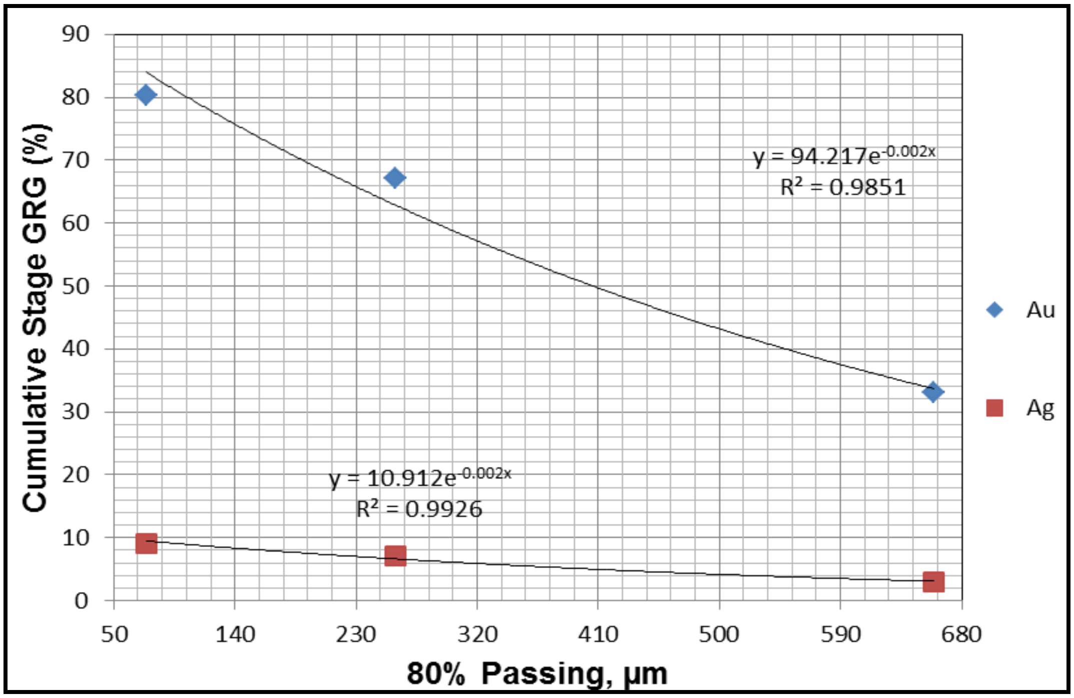
In 2012, Met-Solve and Knelson each performed gravity recoverable gold (GRG) and gravity recoverable silver (GRS) tests to investigate the gold and silver recoveries by gravity concentration and cyanide leaching.

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 13.22.

Table 13.22 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

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)
Centrifugal gravity concentrate upgrading by tabling, assuming gold is present as native gold						
XD20	40	11	15	38	1,920	10,168
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 13.23 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 13.23 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 13.24 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. As predicted by Met-Solve, an additional centrifugal concentrator installed for the cyclone overflow would improve overall gold recovery.

Table 13.24 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

In 2014, Gekko conducted gravity tests on the VOK samples containing three different gold grades. The tests used a continuous tabling gravity concentration procedure to simulate the performance of an inline pressure jig. The results showed that the maximum gold recovery by the jigging process could be in a range of 59 to 74% at a concentrate mass pull between 19% and 21%. The gold recovery reduced to between 43% and 67% when the concentrate mass recovery was decreased to approximately 5%. It appeared that the low grade materials responded better than the high grade samples.

In 2014, FLS-DM conducted tabling tests to upgrade the centrifugal concentrates grading 5,672 g/t gold and 3,309 g/t silver (based on back calculation). The upgrading includes three stages of sequential tabling. The precious metal material balance for the gravity upgrading is given in Table 13.25.

Table 13.25 Precious Metal Material Balance

Product	Weight (g)	Assay (g/t)		Distribution (%)	
		Au	Ag	Au	Ag
Table Concentrate	60.4	199,935	107,297	23.0	21.1
Table Middlings 1	250.7	40,571	24,715	19.3	20.2
Table Middlings 2	2655.2	9,058	5,290	45.7	45.8
Table Tailings	6308.6	1,000	629	12.0	12.9
Calculated Head	9274.9	5,672	3,309	100.0	100.0

The results showed that the tabling increased the precious metal contents of the gravity concentrate from 5,672 to 199,935 g/t for gold and from 3,309 to 107,297 g/t for silver. The gold and silver recoveries to the final table concentrates by the open circuit upgrading were approximately 23% and 21%, respectively.

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)

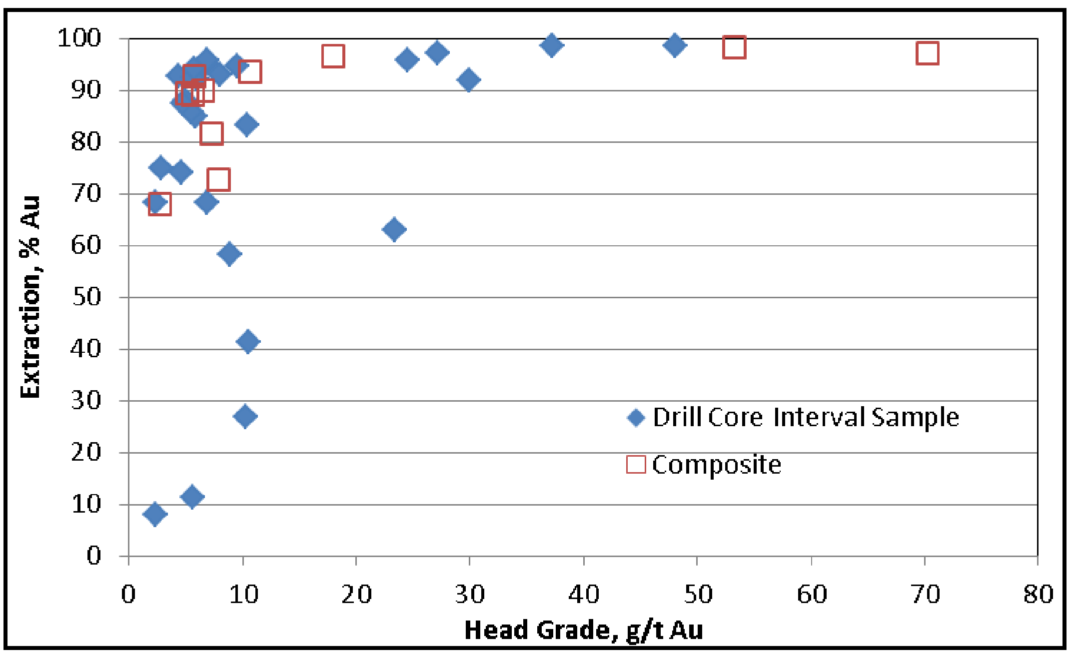
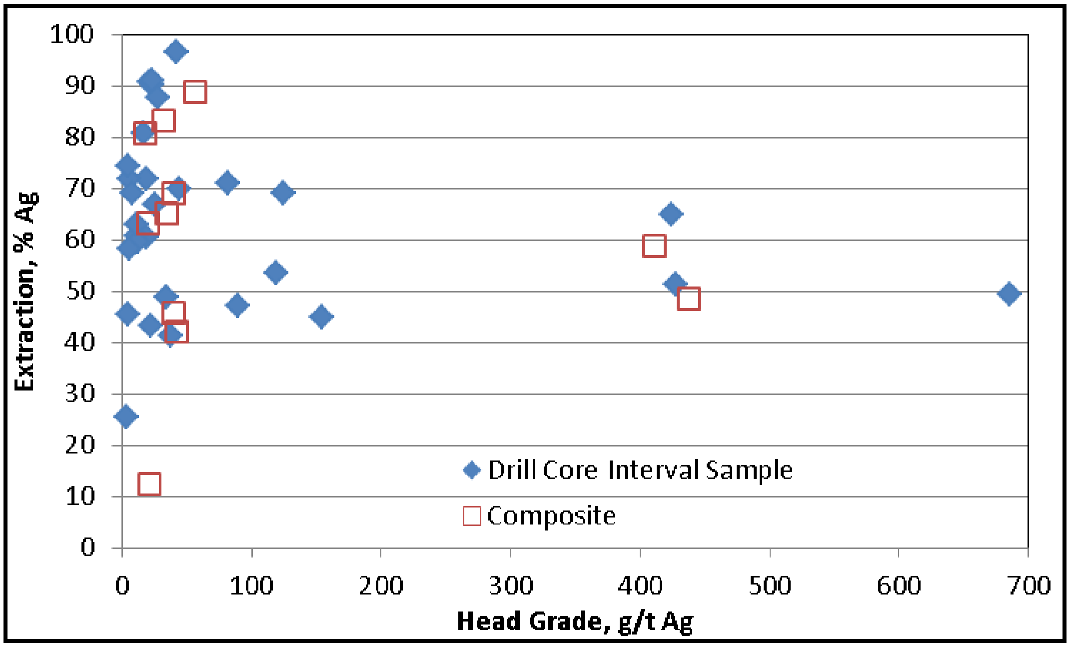


Figure 13.19 Silver Cyanide Extraction – Whole Ore Leach (2012)



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)

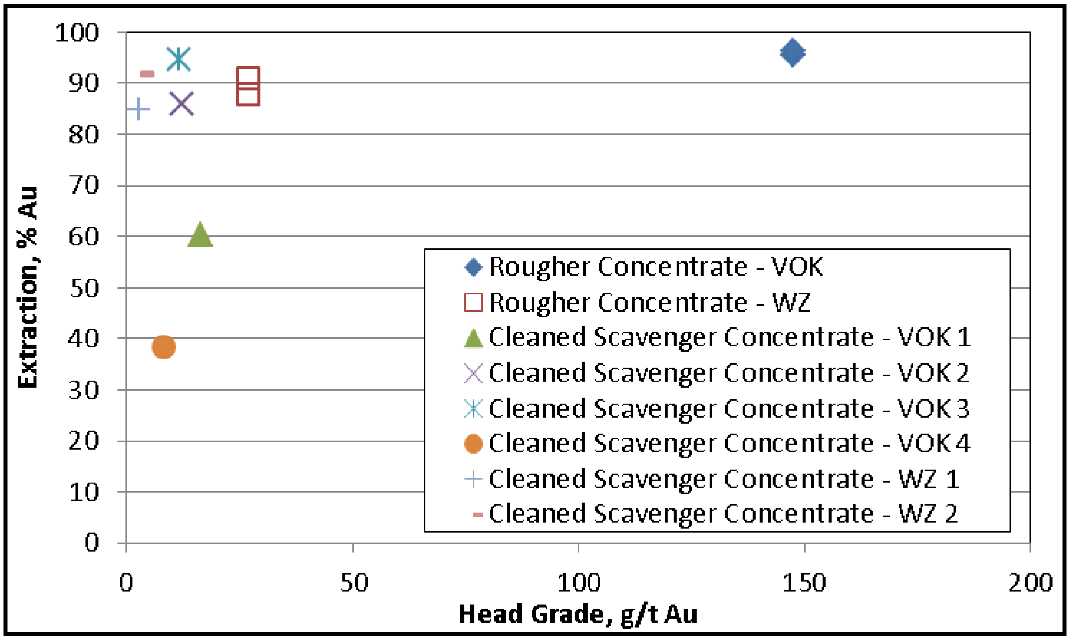
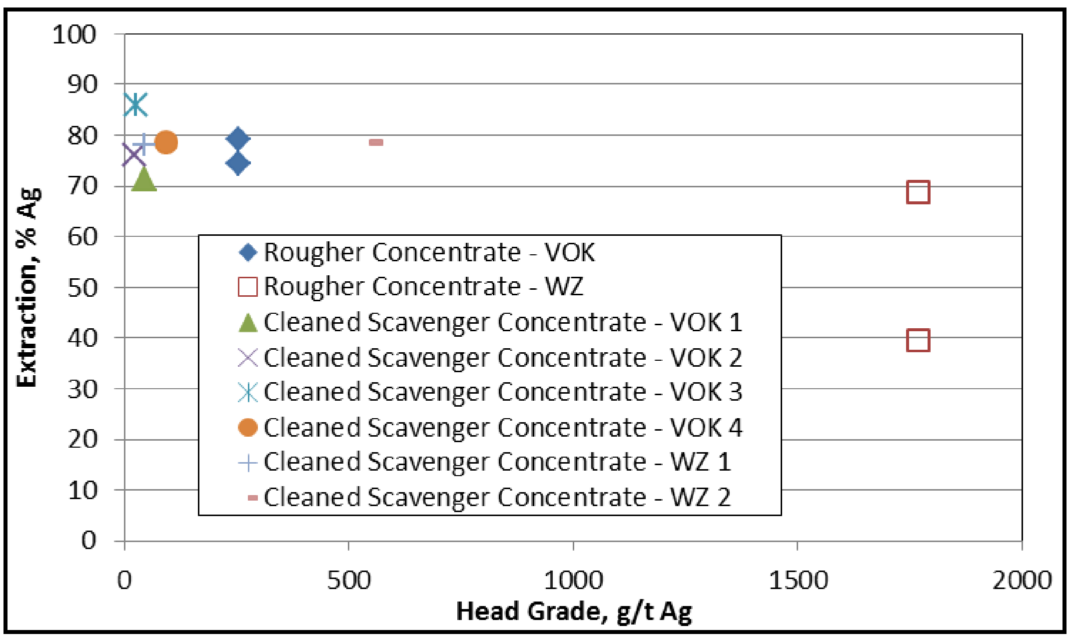


Figure 13.21 Silver Cyanide Extraction – Concentrate Leach (2012)



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.

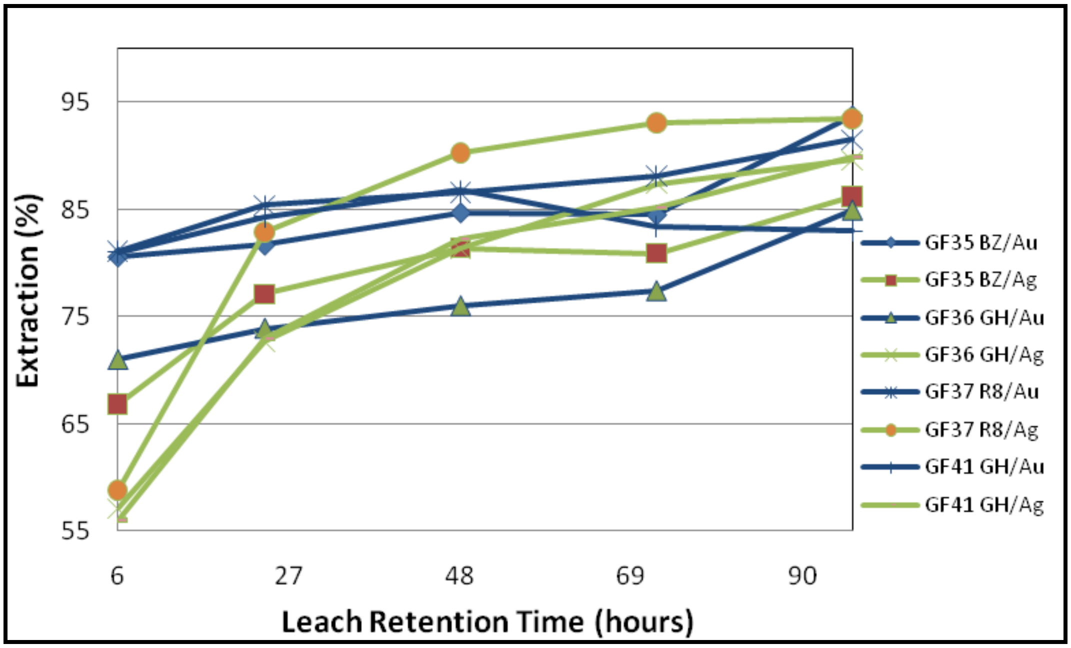
The study indicated that the gold in both leach residue samples occurred mainly as submicroscopic gold, with a modest amount of microscopic gold. The submicroscopic/microscopic gold occurrences are the main cause of gold losses, since conventional leaching cannot recover the gold without finer grinding 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 13.26 summarizes the gold occurrences in various forms in the residue.

Table 13.26 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 13.27. 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 13.27 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 13.28.

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 13.28 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 13.29 to Table 13.31.

As shown in Table 13.29, 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 13.29 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 13.30, 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 13.30 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 13.31, 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 13.32, 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 13.31 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 13.32 show that additional regrinding and re-leaching extracted 13% more gold and 51% more silver from the leaching residue.

Table 13.32 Gravity/Leaching Test Results on Re-ground Flotation Concentrate (2010 to 2011)

	Regrind Size	Concentrate Grade (g/t)		Recovery/Extraction (%)	
		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)

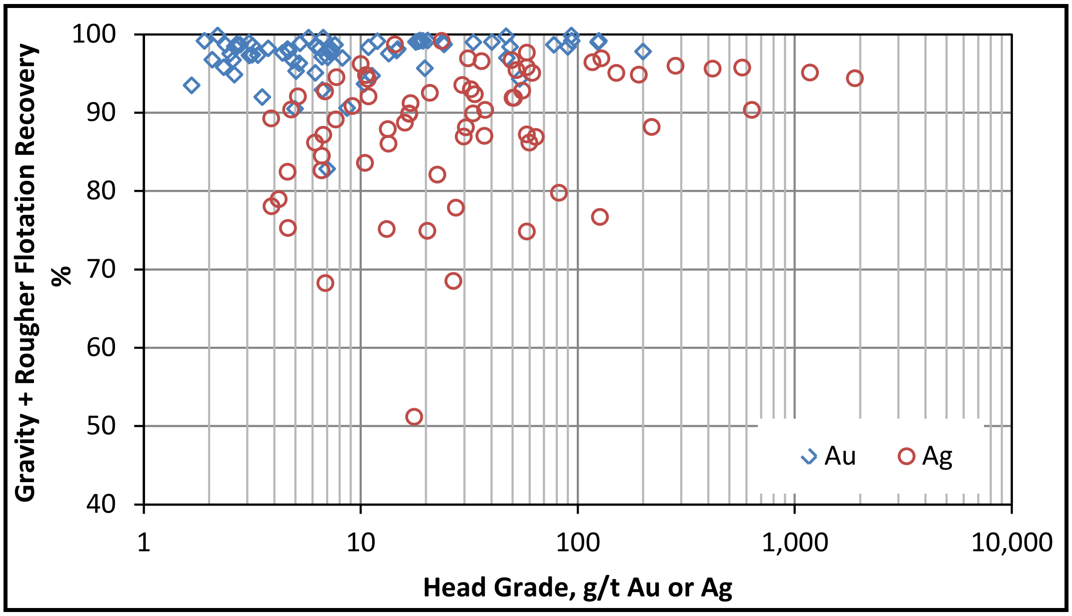
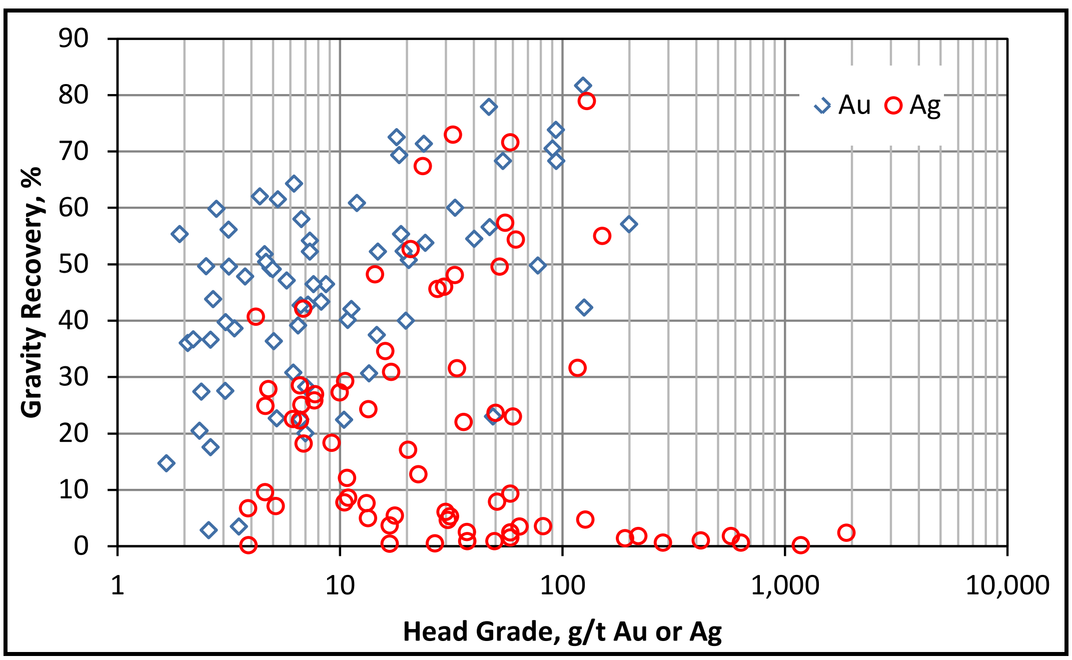


Figure 13.24 Metal Recovery - Gravity Concentration (2012)



In general, the test results were consistent with 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)

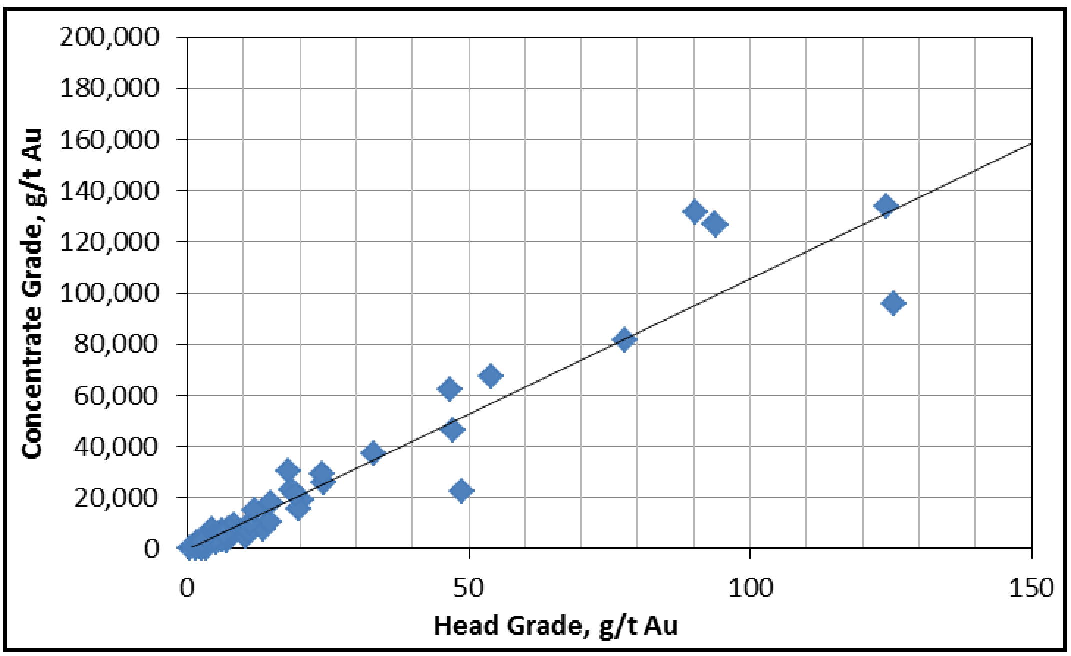
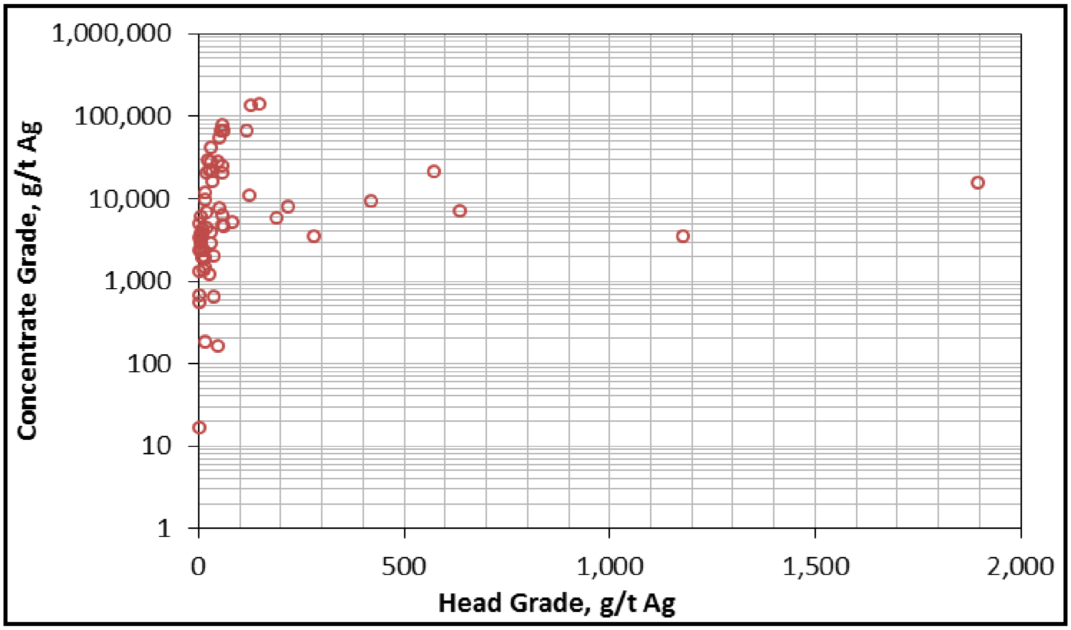


Figure 13.26 Gravity Concentrate Grade versus Head Grade – Silver (2012)



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 13.33.

Table 13.33 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 (%)		
	Au	Ag	Au	Ag	Grind Size
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)

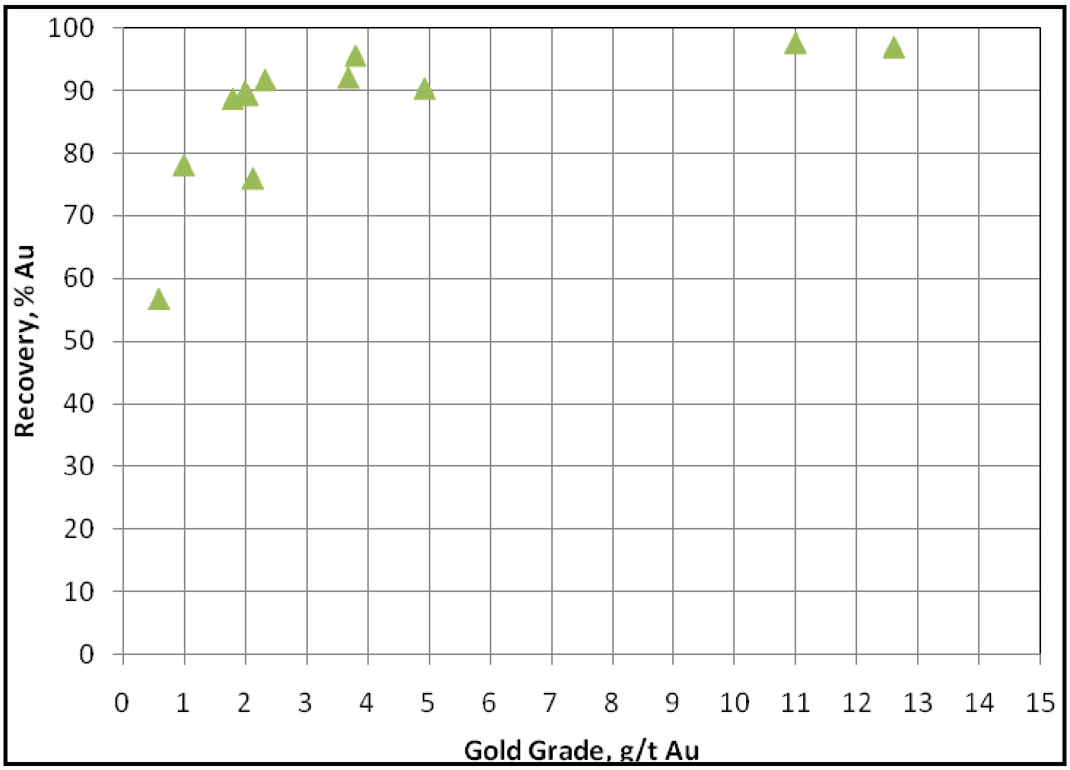
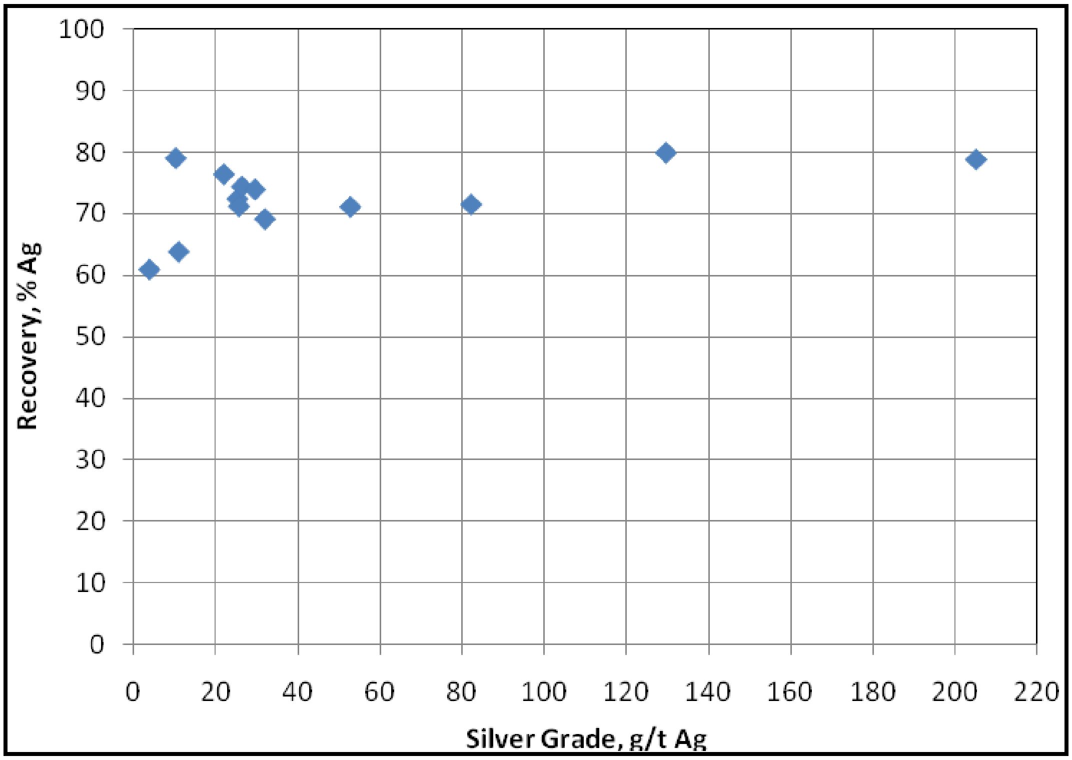


Figure 13.28 Variability Test Results – Silver Metallurgical Performance (2010-2011)



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 13.34 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 13.34 Locked Cycle Tests Results

					Gravity Concentration				Flotation					
		Head Grade Calculated			Recovery		Concentrate Grade		Concentrate Grade				Recovery	
Composite	Test No.	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 MELTING TEST WORK

In 2014, FLS-DM conducted a preliminary melting test on the tabling concentrate. The concentrate used for the test was generated from two Knelson concentrates produced from the VOK 10,300-t bulk sample. The centrifugal concentrates were upgraded by tabling. The upgraded table concentrate contained approximately 20% of gold and 11% of silver. The smelting flux used was a mixture of 62.5% borax, 20% sodium nitrate, 8.75% soda ash and 8.75% silica. FLS-DM indicated that although the table concentrate grade was less than typical for smelting, there were no problems in achieving a good smelt. Smelted doré metal grades were 64% gold, 34% silver and 2% lead.

13.3.11 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 13.35, Table 13.36, 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 13.35 Conventional Thickening Test Results for Flotation Concentrate

Feed Solids (%)	CCD Stage Simulated	Flocculant Dose (g/t)	Flocculant Concentration (g/L)	Rise Rate (m³/m²/h)	Maximum Test Density (%)	Maximum Test Time (min)	Solids Density (%)			Unit Area (m²/t/d)		
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 13.36 Conventional Thickening Test Results for Flotation Tailings

Feed Solids (%)	Flocculant Dose (g/t)	Flocculant Concentration (g/L)	Rise Rate (m³/m²/h)	Maximum Test Density (%)	Maximum Test Time (min)	Solids Density (%)			Unit Area (m²/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

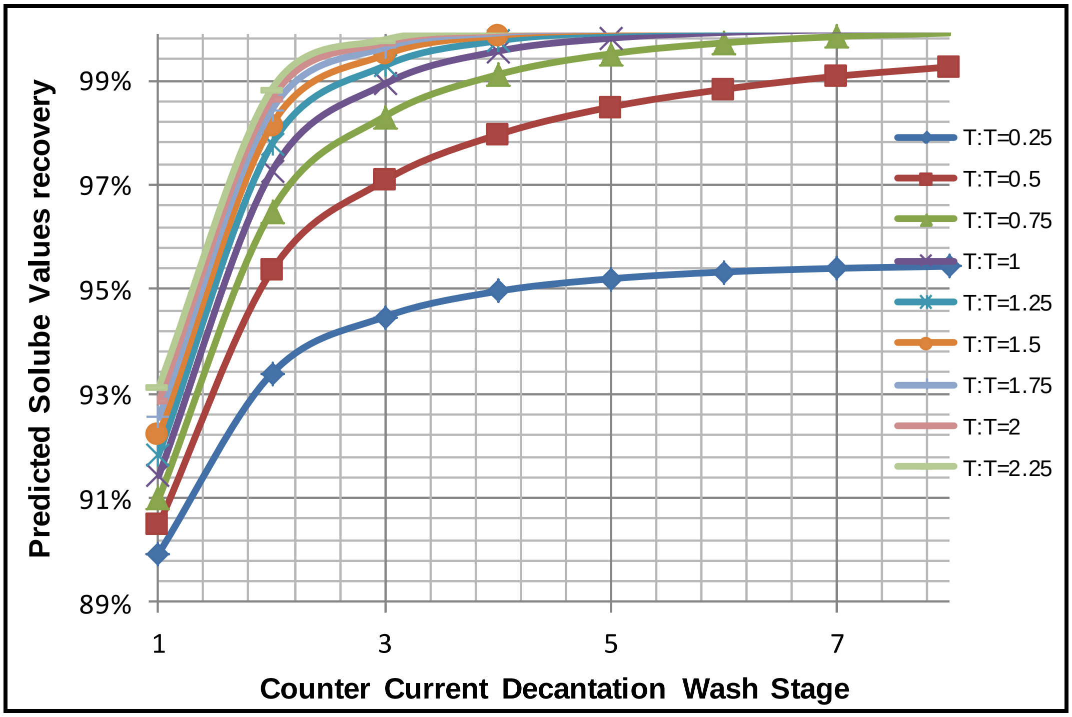


Table 13.37 Recommended Thickening Design Parameters

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 13.38) may be decreased by choosing a membrane-type filter press to get a high-pressure cake.

Table 13.38 Filtration Test Results and Sizing Summary

Filter Type	Material	Filter Feed Solids (%)	Bulk Cake Density (kg/m³)	Filter Sizing Basis (m³/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/m³)**	**Filter Production Rate (kg/m²/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.

13.4 BULK SAMPLE PROCESSING

Between September 27 and December 12, 2013, Pretivm contracted Strategic Minerals LLC to process approximately 10,300 t of the mineralization from the VOK deposit of the Property using the Contact Mill facility located in Philipsburg, Montana. From February 4 to 12, 2014, approximately 1,200 t of the high grade materials from the Cleopatra structure zone of the VOK deposit were also processed at the Contact Mill facility. The primary purpose of the bulk sample runs was to substantiate the resource estimate and evaluate the proposed flowsheet for the Brucejack mineralization. Independent metallurgists were contracted by Pretivm to observe the process.

13.4.1 SAMPLE DESCRIPTION AND HEAD ASSAY ANALYSIS

The materials from the bulk VOK exploration were crushed at the Brucejack site using a jaw crusher and cone crusher in closed circuit with a screen. The 95% finer than 10 mm crushed product was conveyed to a sampling tower where 30 kg samples were recovered for assay purposes. The sample tower rejects were bagged in tote super sacks for shipment to the Contact Mill for processing. For the 2013 runs, five different cross-cuts, identified as 585, 556, 645, 616 and 616L, were processed through the mill in five distinctive rounds. For the 2014 processing, the bulk sample used was from the Cleopatra structure (Cleo) vein.

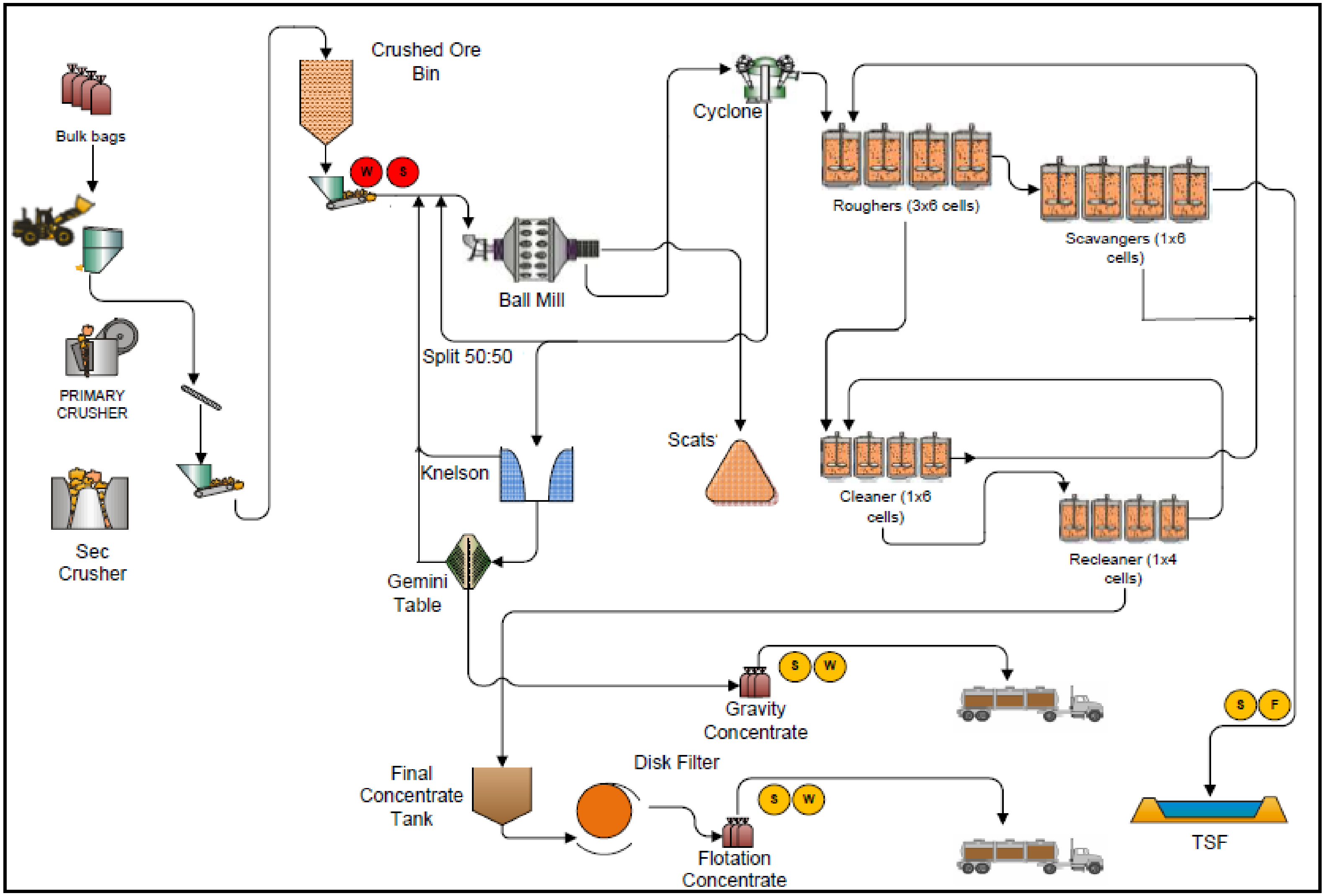
The daily feed grades to the mill ranged widely from less than 1 g/t to more than 130 g/t gold for the samples processed by the 2013 processing campaign and from approximately 40 to 300 g/t gold for the Cleo sample processed in 2014.

13.4.2 Process Description

The process flowsheet used for the bulk sample process campaign was based on the existing process flowsheet at the Contact Mill by incorporating a Knelson 12'' centrifugal concentrator to recover liberated gold/silver by gravity. Approximately 50% of the cyclone underflow in the grinding circuit was directed to the Knelson concentrator while the rest of the cyclone underflow was directed to the ball mill feed chute. The flowsheet used for the campaign is shown in Figure 13.30. During processing of the Cleo sample, in addition to the centrifugal gravity concentration, a jigging and tabling gravity concentration circuit was incorporated in the grinding circuit to recover coarse gold-silver grains.

At the end of the treatment of each cross-cut, the circuit was cleaned using a grind-out and clean-up procedure. At the completion of the program, the grinding, gravity concentration, flotation and filtration circuits were thoroughly cleaned, including the removal and cleaning of the mill charge and liners, the draining of all sumps and the concentrate filter boot.

Figure 13.30 Bulk Sample Process Flowsheet



The process flow through the mill differed slightly from the flowsheet proposed for the project due to space and equipment limitations in the mill. The differences between the design and the operation are outlined in Table 13.39.

Table 13.39 Flowsheet Difference between Bulk Sample Process Program and Proposed Process for the Project

Area	Bulk Sample Process Flowsheet	Proposed Process Flowsheet
Comminution	Multi-stages of crushing + ball mill grinding. On average, primary grind sizes were in the range of between 65% and 75% passing 200 mesh (75 µm).	One stage of crushing + SAG mill/ ball mill grinding. The designed primary grind size is 80% passing 200 mesh (75 µm).
Gravity Circuit	One stage of centrifugal concentration in the primary grinding circuit (for the Cleo materials, jigging+tabling process was incorporated prior to the centrifugal concentration). Gravity concentrate was upgraded by a Gemini table.	Two stages of centrifugal concentration, one in the primary grinding circuit and one in the regrind circuit. Gravity concentrates are upgraded by two separate tables with different surface patterns.
Rougher Flotation Concentrate Regrinding	No regrinding.	One stage of regrinding to 80% passing 35 to 40 µm, with a centrifugal concentrator.
Flotation - Circuit	Three stages of rougher flotation, one stage of scavenger flotation, two stages of cleaner flotation.	Three stages of rougher flotation, one stage of scavenger flotation, three stages of cleaner flotation, one stage of 1st cleaner scavenger flotation.
Flotation - Reagents	Collector: PAX. Frother: D250.	Collector: PAX. Frother: MIBC.
Flotation – Feed Slurry Solids Density	15 to 25% solids.	Approximately 33% solids.
Flotation – Retention Time	Rougher Flotation: 20 to 30 min. Scavenger Flotation: 6 to 10 min.	Rougher Flotation: ~80 min. Scavenger Flotation: ~25 min.
Concentrate Filtration	Vacuum Disc Filter.	Filter Press.

The bulk sample runs showed that the differences from the design flowsheet appeared to have little impact on the overall processing of the bulk samples and metal recovery.

13.4.3 Results and Discussion

In general, the results from the bulk sample processing campaigns demonstrated that the flowsheet used for the program can effectively recover gold and silver into gravity and flotation concentrate products. The results confirm that the process flowsheet as designed for the Brucejack mineralization suited the treatment of the bulk sample. The results also showed that the gravity/flotation flowsheet adapted well for the varying mineralization and the wide range feed grades that were experienced during processing of the bulk sample. The bulk sample represents specific locations within the VOK deposit.

The daily samples collected were analyzed using fire assay procedure by the on-site assay laboratory. Each of flotation concentrate bags were sampled and sent to Inspectorate or ALS for assay, including using metallic gold assay procedure. The randomly selected head, concentrate and tailings samples were sent to the off-site laboratories for assay. A daily mass balance was generated by Contact Mill using the Contact Mill assay data.

A summary of the processing results is listed in Table 13.40. The assay data for flotation concentrate and flotation tailings by the Contact Mill assay laboratory have been reconciled according to the assay data from Inspectorate and ALS. For the 2013 campaign, approximately 97.5% of the gold and 86.9% of the silver was recovered into the gravity and flotation concentrates, including approximately 42% of the gold reporting to the table concentrate. For the Cleo material, the gold and silver reporting to the gravity and flotation concentrates were approximately 98% for gold and 96% for silver. The gold reporting to the table concentrate was higher, averaging approximately 48%. A total of gold and silver content in the table concentrates is approximately 20% to 39%. The data in Table 13.40 is preliminary since the final settlements for the total recovered gold and silver from the gravity and flotation concentrates sent to further treatments/sold to smelter are still in progress. The final metallurgical balances with reconciling the final settlements may be different from the preliminarily estimated data.

Table 13.40 Bulk Sample Processing Metallurgical Performances

Materials	Feed			Recovery, %						Grade, g/t			
	Tonnage	Calculated Grade, g/t		Table Concentrate		Table Concentrate + Table Middlings		Gravity+ Flotation Concentrate		Table Concentrate		Flotation Concentrate	
	t	Au**	Ag**	Au*	Ag*	Au*	Ag*	Au	Ag	Au*	Ag*	Au***	Ag***
585	*2,169*	*3.9*	*9.6*	*25.7*	*4.9*	*34.2*	*6.7*	*93.6*	*71.0*	*193,975*	*90,210*	*36*	*96*
555	*1,596*	*4.1*	*11.9*	*44.2*	*8.9*	*51.8*	*10.5*	*95.2*	*83.4*	*164,241*	*95,559*	*25*	*122*
645	*1,694*	*2.2*	*5.1*	*40.4*	*8.3*	*46.5*	*9.6*	*92.1*	*71.4*	*138,473*	*65,898*	*14*	*43*
615	*3,050*	*36.5*	*28.1*	*40.4*	*20.9*	*46.1*	*24.2*	*98.0*	*91.8*	*277,293*	*110,189*	*143*	*170*
615L	*1,792*	*26.9*	*22.4*	*49.5*	*26.7*	*55.0*	*30.2*	*97.5*	*89.2*	*268,885*	*120,923*	*107*	*169*
2013 Material	10,302	17.5	17.1	41.8	18.2	47.6	21.0	97.5	86.9	259,487	110,146	79	129
Cleo Material	1,203	82.6	59.7	47.9	36.6	56.2	44.0	98.0	96.3	247,999	136,877	398	402

Notes: *Based on assay data from Contact Mill laboratory.
**Including cleanout.
***Flotation concentrate only.

13.5 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/gravity concentration on the rougher and scavenger concentrates to improve gravity gold recovery
  - 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.
- The gravity concentrates responded very well to intensive leach by cyanide.

13.5.1 Recommendations

Further test work is recommended to further investigate:

- effect of flotation retention time to gold and silver recoveries
- tailings thickening condition optimization
- effect of regrinding of rougher and scavenger concentrates on metal recovery and concentrate quality
- flotation performance for the slime that may be produced from the underground dewatering system.

13.6 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 and the bulk sample processing programs conducted during 2013 and 2014.

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, the results from the bulk sample processing programs conducted during 2013 and 2014 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 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 13.41 for the VOK Zone mineralization and Table 13.42 for the West Zone mineralization.

Table 13.41 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	= (18.5*(Head Grade, g/t) $^{-0.091}$)
130 to 400	= 10.0
> 400	= 5.0
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

table continues…

Head Grade (g/t)	Gold and Silver Recovery (%)
Flotation Concentration - Silver	
< 3.0	= 35
3.0 to 130	= 2.793*ln(Head Grade, g/t) - (18.5*((Head Grade, g/t) $^{-0.091}$)) + 78.07
130 to 400	= 82.0
> 400	= 87.5

Table 13.42 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) + 2.0
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

Information in this section has been excerpted and condensed from Jones (2013). The reader should refer to Jones (2013) for detailed information.

In December 2013, Snowden completed a Mineral Resource estimate for the VOK Zone of the Project. The West Zone estimate remains unchanged from the April 2012 Mineral Resource estimate (Jones 2012a).

The December 2013 estimate has been updated based on over 40,000 m of additional drilling including 24 surface drillholes (5,200 m) and 409 underground drillholes (38,840 m) drilled in support of the underground bulk sample. In addition to the drilling, a 10,000-t bulk sample has been processed through a mill and detailed test work has been carried out to both validate the previous Mineral Resource and refine the estimation process for the updated Mineral Resource.

The result of the test work is an improved confidence in both the geological model and the grade estimate, with the definition of Measured Resources as part of the December 2013 Mineral Resource.

Mineral Resources were prepared by Ms. Lynn Olssen, Snowden's QP and the author of this report, and Mr. Ivor Jones. The author, Ms. Lynn Olssen, is an employee of Snowden Mining Industry Consultants and, by way of experience and qualifications, is a QP as defined by NI 43-101 and is independent of Pretivm.

All Mineral Resource estimation was carried out using CAE Studio software (formerly Datamine). Snowden Supervisor software was used for statistical and geostatistical analysis.

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

14.1 INPUT DATA

The input data for the VOK Mineral Resource estimate comprised 932 drillholes totalling 218,238 m. These included:

- 9 historic drillholes (579 m)
- 490 surface drillholes drilled between 2009 and 2012 (173,619 m)
- 24 surface drillholes drilled in 2013 (5,200 m)
- 409 underground drillholes drilled in 2013 (38,840 m).

The input data for the West Zone estimate comprised 756 drillholes (63,208 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.

14.2 ESTIMATE TEST WORK IN THE BULK SAMPLE AREA

Prior to the December 2013 mineral resource estimate, the underground bulk sample results were used to investigate the local accuracy of the November 2012 Mineral Resource estimate within the VOK, and to determine whether the estimation methodology could be improved for the December 2013 Mineral Resource estimate.

A series of statistical tests were run to determine whether any bias exists between the surface diamond drilling, underground diamond drilling, underground channel samples, and chip samples. No statistical bias, based on these statistics, was evident between the different sample types. However, as the chip samples and channel samples are mostly co-located, the chip samples were excluded from all remaining test work and estimation are the sampling method used for the chip sampling is inherently less precise than that used for the channel sampling.

Additional test work in the estimation did, however, display some bias caused by directional drilling in the area of the bulk sample. The underground drilling had been aligned in a north-south orientation which is consistent with the orientation of some high-grade mineralization identified in the bulk sample..

Pretivm also completed a substantial amount of underground drilling associated with the bulk sample. This drilling was closely spaced, but based mostly on a north-south grid and appears to have created a directional bias in the drilling information because it is consistent with the orientation of some high-grade mineralization identified in the bulk sample. Removal of the underground drillholes resulted in an increase in the grade of the local estimate. This was particularly evident in those cross-cuts dominated by north-south mineralization (e.g. 426615E) and resulted in a significantly better correlation with the results from processing in the mill. However, this drilling, along with the results of processing the bulk sample, was used to assist in the improvement of grade estimation parameters. It was noted as a part of this test work, however, that the result of including the new drilling information in the resource estimate further under-estimated the grade in the bulk sample because of this directional bias.

While there is no bias evident between the channel samples and the drilling, the location of numerous channel samples in the centre of some of the higher grade mineralization does result in a local overestimation around the bulk sample cross-cuts. Consequently the decision was made not to use the channel samples for the final Mineral Resource estimate.

The final metal and tonnes from the mill accounting were compared to those predicted by the November 2012 Mineral Resource estimate for each drive to assess the appropriateness of the modelling process. This test work has in part relied on comparisons between the test estimates and results from the bulk sample processing.

However, the reader should be warned that there is a significant difference in the sample support for the resource estimate (each block in the resource estimate represents 2,700 t whereas the bulk sample packages are around 100 t), and the grade is not homogenous throughout any block. In other words, the grade can vary from a high-grade side of the block to a low-grade side of the block, whereas the block grade represents an average of the whole block. If the bulk sample happens to take a high-grade part of the block, then the comparison will look like the resource estimate under-estimated the grade, and conversely if the bulk sample takes a low grade part of the block, then the comparison will look like the resource estimate over-estimated the grade in the block. Whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate locally, it does provide the best opportunity to fine-tune the estimate to some hard data. The reader should be warned that the results are only used to give some local perspective to the grade estimates.

The results indicated that the November 2012 Mineral Resource underestimated the total metal content in the bulk sample by about 10%. In more detail, the November 2012 Mineral Resource estimated high-grade into lower-grade areas, and low-grade into the high-grade areas, a result of extrapolating the high-grade values around the high-grade core. This extrapolation of high-grade values was based on the nature of the mineralization and the interpreted continuity of the high-grades.

Based on the bulk sample comparisons, Snowden concludes that the November 2012 Mineral Resource was a good indicator of the contained metal within the VOK deposit and satisfactory for bulk underground mining, but that it was not locally accurate at the 10 m block scale. As a result further test work was undertaken to adjust the estimation methodology for the December 2013 Mineral Resource, to produce an estimate that is more responsive to the local high-grades.

In order to produce an estimate that is more responsive to the local high grades, a series of test estimates were created inside the local test area surrounding the bulk sample crosscuts. The estimates were compared to the bulk sample mill results on a round-by-round basis, as well as on a more global basis within the local test area.

Several options were assessed as part of the test work including:

- Looking at the use of channel samples to assist in defining the local grade more accurately around the bulk sample cross-cut.
  - These tests indicated that the use of channel samples resulted in a local overestimation of the grade in the bulk sample cross-cuts. As a result channel samples were not included in the December 2013 Mineral Resource estimate.
- Assessing the impact of constraining the north-south mineralization and estimating it separately to the dominantly east-west mineralized corridors.
  - The test work indicated that the estimate without any north-south constraint is a better local predictor of metal, with underestimation of the north-south mineralization. Based on the outcomes of the test work and bulk sample analyses, Snowden and Pretivm agreed that the more conservative

approach, not using the north-south constraints, should be applied for the December 2013 Mineral Resource estimate. As a result, the December 2013 Mineral Resource estimate is considered to be a conservative estimate of the contained metal in the VOK deposit.

- Adjusting the estimation parameters and methodology to reduce smoothing, including the method for re-blocking the high-grade MIK estimates, parent cell size, and search neighbourhood parameters.
  - Review of the estimation parameters resulted in slight adjustments to the search neighbourhood to produce a more selective estimate, and changing the re-blocking of the high-grade estimates to use the parent cell size. A parent cell size of 10 mE by 10 mN by 10 mRL was retained for the estimation of the VOK and resulted in less smoothing of the estimate.
- Comparing ordinary kriging and inverse distance weighted estimation methods.
  - Test work using ordinary kriging and inverse distance weighted estimation methods showed that these methods, when run using a "typical" top cut of the 99.9th percentile or lower, significantly underestimate the metal in the bulk sample cross-cuts. Given the style of mineralization, the level of selectivity required and the estimation of unrealistic high-grade areas defined by these methods, the methods were not considered appropriate for grade estimation at Brucejack.

14.3 ESTIMATION

The mineralization in the VOK exists as steeply dipping semi-concordant (to stratigraphy) and discordant pod-like zones hosted in stockwork vein systems within the volcanic and volcaniclastic sequence. High-grade mineralization zones appear to be spatially associated, at least in part, with intensely silicified zones resulting from local silica flooding and over-pressure caprock formation. High-grade mineralization occurs both in the main east-west trending vein stockwork system, as well as in the rarer north-south trending part of the system. Snowden notes that Pretivm has taken these various observations into consideration in its interpretation of the mineralization domains for the VOK.

A threshold grade of 0.3 g/t gold was found to generally identify the limits to the broad zones of mineralization as represented in the drill cores at West Zone and the VOK. In the VOK, a 1 g/t gold to 3 g/t gold 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. West Zone was interpreted for the April 2012 estimate using a nominal 0.3 g/t gold cut-off grade (Jones 2012a). There are no high-grade corridors defined at West Zone.

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 the VOK) and mineralized domain. Review of the statistics indicated that the grade distributions for the mineralization within the different lithologies are very similar and as a result these were combined for analysis. This is in agreement with field observations which indicate that the stockwork mineralization is superimposed on the stratigraphic sequence. The summary statistics of composite samples from all domains exhibit a strong positive skewness with high coefficient of variation and some extreme grades.

Both the West Zone and the 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 gold. These grades have been shown from the mining to be a normal part of the mineralization, in some instances continuous, and definitely not anomalous. In addition, a review of the upper tail of the cumulative distribution function (CDF) shows that the extreme grade population is continuous and does not break down, supporting this observation. As a result of this population distribution, standard estimation techniques have been found to significantly over-smooth the grades.

Because of the extreme positive skew in the histograms of the gold and silver grades within the high-grade domains, Snowden elected to use a non-linear approach to estimation, employing the use of indicator and truncated distribution kriging. In this approach the proportion of high-grade in a block was modelled as was 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 population was estimated using ordinary kriging on the truncated (low-grade; less than 5 g/t gold and less than 50 g/t silver) part of the grade distribution.

A block model with cell dimensions of 10 mE by 10 mN by 10 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 of the VOK and the majority of the West Zone.

Within the well-informed portion of the 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 MIK estimates so that these point estimates could be subsequently re-blocked to take into account the correct degree of smoothing for 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 the VOK and the majority of the West Zone and 1.25 mE and 1.25 mN by 1.25 mRL for the well-informed portions of the West Zone.

A summary of the estimation parameters used for the high-grade domains in VOK is provided in Table 14.1.

Density was estimated using simple kriging of specific gravity measurements determined on sample pulps by ALS Chemex. Bulk density estimates in the final model were determined by simply factoring down pulp specific gravity estimates by 3% in all lithologies except in the intensely silicified conglomerate (Section 11.1).

Table 14.1 December 2013 Estimation and Search Parameters Within High-grade Mineralized Domains for the VOK

Estimate	Parameter	December 2013
High-grade domains - low-grade population	Estimation method	OK
	Parent cell size	10 mE by 10 mN by 10 mRL
	Re-blocking cell size	10 mE by 10 mN by 10 mRL
	Search ellipse – pass 1	60 m by 100 m by 20 m
	Minimum samples – pass 1	12
	Maximum samples – pass 1	26
	Search ellipse – pass 2	120 m by 200 m by 40 m
	Minimum samples – pass 2	8
	Maximum samples – pass 2	26
	Maximum composites per drillhole	8
High-grade domains - high-grade population	Estimation method	MIK
	Parent cell size	2.5 mE by 2.5 mN by 2.5 mRL
	Re-blocking cell size	10 mE by 10 mN by 10 mRL
	Search ellipse – pass 1	35 m by 35 m by 20 m
	Minimum samples – pass 1	12
	Maximum samples – pass 1	16
	Search ellipse – pass 2	105 m by 105 m by 60 m
	Minimum samples – pass 2	2
	Maximum samples – pass 2	6
	Maximum composites per drillhole	8
High-grade domains - probability	Estimation method	MIK
	Parent cell size	2.5 mE by 2.5 mN by 2.5 mRL
	Re-blocking cell size	10 mE by 10 mN by 10 mRL
	Search ellipse – pass 1	35 m by 35 m by 15 m
	Minimum samples – pass 1	12
	Maximum samples – pass 1	16
	Search ellipse – pass 2	70 m by 70 m by 30 m
	Minimum samples – pass 2	2
	Maximum samples – pass 2	6
	Maximum composites per drillhole	8

14.4 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; grade trend plots; and comparing the results of the model to the bulk sample cross cuts.

Comparison of the updated Mineral Resource to the previous November 2012 Mineral Resource and the bulk sample results shows that the updated estimate is more locally accurate than the previous Mineral Resource (Table 14.2). The updated Mineral Resource underestimates the north-south mineralization which may result in additional ounces if more of these features are discovered during mining. Snowden considers that the underestimation is a function of an orientation bias in the underground drilling which is aligned with the highest grade mineralization. Whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate locally, it does provide the best opportunity to fine-tune the estimate to some hard data. The reader should be warned that the results are only used to give some local perspective to the grade estimates.

Table 14.2 December 2013 Mineral Resource versus November 2012 Mineral Resource and Mill Results by Bulk Sample Cross-cut

Cross-cut	Mill			November 2012 Mineral Resource			December 2013 Mineral Resource		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555E	1,416	4.41	201	1,491	4.89	234	1,481	3.97	189
426645E	1,875	2.28	137	1,959	5.64	355	1,987	3.26	208
426585E	2,169	4.02	280	2,231	12.12	869	2,269	2.35	171
426615E	2,878	39.35	3,642	3,089	18.88	1,876	3,127	15.74	1,582
615L	1,964	25.52	1,611	1,535	38.78	1,914	1,574	38.42	1,945
Final clean out	-	-	52.0	-	-	-	-	-	-
Total	**10,302**	**17.88**	**5,923**	**10,305**	**15.84**	**5,248**	**10,438**	**12.20**	**4,096**

A more global comparison of the November 2012 Mineral Resource estimate and the December 2013 Mineral Resource estimate within the local test area highlights the additional selectivity in the updated 2013 estimate. There are less tonnes at a higher grade in the December 2013 estimate above a 5 g/t AuEq cut-off (Table 14.3) than in the November 2012 estimate.

This change in the local responsiveness to the grades is in line with expectation as the test work carried out (Section 13.2) indicated that the November 2012 estimate was not locally accurate and the method was adjusted for the December 2013 estimate to produce an estimate that is more responsive to the local high-grades.

Table 14.3 November 2012 versus December 2013 Mineral Resource Estimates Within Local Test Area above a 5 g/t AuEq Cut-off

Model	Tonnes	Au (g/t)	Au (koz)
November 2012 Mineral Resource	1,745	16.54	928
December 2013 Mineral Resource	1,326	19.54	833

14.5 CLASSIFICATION

The resource classification definitions (Measured, Indicated, Inferred) used for this estimate are those published by the CIM in the "CIM Definition Standards" document.

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 AuEq calculation used is: AuEq = Au + Ag/53. These blocks were then used as a guide to develop a set of wireframes defining coherent zones of mineralization which were classified as Measured, Indicated or Inferred and reported (Table 14.4 and Table 14.5).

Classification was applied based on geological confidence, data quality and grade variability. Areas classified as Measured Resources at West Zone are within the well-informed portion where the resource is informed by 5 m by 5 m or 5 m by 10 m spaced drilling. Measured Resources within VOK are informed by 5 m by 10 m to 10 m by 10 m underground fan drilling and restricted to the vicinity of the underground bulk sample.

Areas classified as Indicated Resources are informed by drilling on a 20 m by 20 m to 20 m by 40 m grid within West Zone and VOK. In addition, some blocks at the edge of the areas with 20 m by 20 m to 20 m by 40 m drilling, were downgraded to Inferred where the high-grades appear to have too much influence. The remainder of the Mineral Resource is classified as Inferred Resources where there is some drilling information (and within around 100 m of drilling) and the blocks occur 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.

14.6 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 consistent with the November 2012 Mineral Resource. In that evaluation, the AuEq value was calculated according to the formula (AuEq = Au + Ag/53) based upon prices of US$1,590/oz and US$30/oz for gold and silver respectively. Recoveries for gold and silver are assumed to be similar.

High-grade Mineral Resources for the VOK and the West Zone are summarized in Table 14.4 and Table 14.5, respectively. The Mineral Resources are depleted for historical mining in West Zone and the recent underground bulk sample mining in VOK.

Table 14.4 VOK Mineral Resource Estimate Based on a Cut-off Grade of 5 g/t AuEq – December 2013(1)(4)(5)

Category	Tonnes (million)	Gold (g/t)	Silver (g/t)	Contained(3) Gold (Moz)	Contained(3) Silver (Moz)
Measured	2.0	19.3	14.4	1.2	0.9
Indicated	13.4	17.4	14.3	7.5	6.1
M + I	15.3	17.6	14.3	8.7	7.0
Inferred(2)	5.9	25.6	20.6	4.9	3.9

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 Technical Report were estimated 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 Mineral Resource estimate stated in Table 14.4 and Table 14.5 is defined using 5 m by 5 by 5 m blocks in the well drilled portion of West Zone (5 m by 10 m drilling or better) and 10 m by 10 m by 10 m blocks in the remainder of West Zone and in VOK.

(5) The AuEq value is defined as AuEq = Au + Ag/53.

Table 14.5 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)	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

Notes: (1), (2), (3), (4) and (5) - refer to footnotes in Table 14.4.

15.0 MINERAL RESERVE ESTIMATES

15.1 GENERAL

The mine design and Mineral Reserve estimate have been completed to a level 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_1313_v11_cut", which was provided to AMC by Snowden–on behalf of Pretivm–in February 2014.

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 previous studies, including the June 2011 PEA (Ghaffari et al. 2011), the February 2012 Updated PEA (Ghaffari et al. 2012) and the June 2013 Feasibility Study (Ireland et al. 2013). In the 2013 Feasibility Study, site operating costs were ultimately estimated at approximately $156/t based on a 2,700 t/d operation, of which $93/t was attributable to mining. The cut-off grade thus provided a minimum margin of $24/t of ore mined.

This feasibility study update provides a platform for increasing the accuracy of cost estimations relative to previous studies. The design production rate has not changed for the current study. Estimates of average site operating costs over the LOM have been updated as follows:

- Mining: $91.34/t
- Processing: $19.69/t
- Surface Services and Others: $21.15/t
- G&A: $30.87/t
- Total: $163.05/t

The use of $180/t NSR cut-off grade provides a minimum $16.95/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 which are 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 refining, concentrate treatment, transportation, and insurance costs.

The metal price assumptions are US$1,100/oz gold and US$17/oz silver. Costs assume a Cdn$ to US$ exchange rate of 0.92.

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 and other related knowledge acquired during the study. AMC considers the magnitude of any parameter variation to have no material impact on the study findings.

Table 15.1 Net Smelter Return Parameters

Items		Units		Parameters
Currency	Conversion Rate	Cdn$/US$	-	0.92
Metal Prices	Gold	US$/oz	-	1,1100.00
	Silver	US$/oz	-	17.00
Doré				
VOK Process Recoveries	Gold	%	For Au <0.5 g/t	0
		%	For Au ≥0.5 g/t and Au <9.99 g/t	-0.147 x Au^2 + 5.68 x Au - 1.214
		%	For Au ≥9.99 g/t and Au <40.00 g/t	7.32 x ln(Au) + 24.012
		%	For Au ≥40.00 g/t	52
	Silver	%	For Ag <3.00 g/t	23
		%	For Ag ≥3.00 g/t and Ag <130.00 g/t	18.5 x Ag 0.091
		%	For Ag ≥130.00 g/t and Ag <400.00 g/t	10
			For Ag ≥400.00 g/t	5
West Zone Process Recoveries	Gold	%	For Au <0.5 g/t	0
		%	For Au ≥0.5 g/t and Au <9.99 g/t	-0.147 x Au^2 + 5.68 x Au - 1.214
		%	For Au ≥9.99 g/t and Au <40.00 g/t	7.32 x ln(Au) + 24.012
		%	For Au ≥40.00 g/t	52
	Silver	%	For Ag <3.00 g/t	23
		%	For Ag ≥3.00 g/t and Ag <130.00 g/t	34.643 x Ag 0.48
		%	For Ag ≥130.00 g/t and Ag <400.00 g/t	1.5
		%	For Ag ≥400.00 g/t	0
Selling Costs	Metal Payable – Gold	%	-	99.8
	Metal Payable – Silver	%	-	99.8
	Refining Charge – Gold	Cdn$/oz	-	1
	Refining Charge – Silver	Cdn$/oz	-	1
	Transport/Port Handling Costs	Cdn$/oz-gold	-	1
	Insurance (Net Invoice Value)	% NIV	-	0.5

table continues…

Items		Units		Parameters
Flotation Concentrate				
VOK Process Recoveries	Gold	%	For Au <0.5 g/t	30
		%	For Au ≥0.5 g/t and Au <5.64 g/t	85.44 x $Au^{0.056}$ - (-0.147 x Au^2 + 5.68 x Au - 0.714) + 2.6
		%	For Au ≥5.640 g/t and Au <9.99 g/t	1.18 x ln(Au) + 0.147 x Au^2 – 5.68 x Au + 94.694
		%	For Au ≥9.99 g/t and Au <40.00 g/t	-6.14 x ln(Au) + 69.558
		%	For Au ≥40.00 g/t	46.5
	Silver	%	For Ag <3.00 g/t	35
		%	For Ag ≥3.00 g/t and Ag <130.00 g/t	2.793 x ln(Ag) - (18.05 x Ag-0.091) + 78.07
		%	For Ag ≥130.00 g/t and Ag <400.00 g/t	82
		%	For Ag ≥400.00 g/t	87.5
West Zone Process Recoveries	Gold	%	For Au <0.5 g/t	30
		%	For Au ≥0.5 g/t and Au <5.64 g/t	85.44 x $Au^{0.056}$ - (-0.147 x Au^2 + 5.68 x Au - 0.714) + 2.0
		%	For Au ≥5.640 g/t and Au <9.99 g/t	1.18 x ln(Au) + 0.147 x Au^2 – 5.68 x Au + 94.094
		%	For Au ≥9.99 g/t and Au <40.00 g/t	-6.14 x ln(Au) + 69.958
		%	For Au ≥40.00 g/t	46.9
	Silver	%	For Ag <3.00 g/t	35
		%	For Ag ≥3.00 g/t and Ag <130.00 g/t	2.9741 x ln(Ag) - (34.643 x Ag-0.48) + 73.956
		%	For Ag ≥130.00 g/t and Ag <400.00 g/t	2.9741 x ln(Ag) + 72.456
		%	For Ag ≥400.00 g/t	91.5
Selling Costs	Metal Payable – Gold	%	For Au <1.0 g/t	0
		%	For Au ≥1.0 g/t and Au ≤3.00 g/t	3.968 x Au + 85.61
		%	For Au >3.00 g/t	90
	Metal Payable – Silver	%	For Ag ≤8.00 g/t	0
		%	For Ag >8.00 g/t	90

table continues…

Items		Units		Parameters
Selling Costs	Penalty Charge – Arsenic	Cdn$/t-concentrate	For As (ppm) / S(%) >25.4	0.35479 x (As (ppm) / S (%)) + 1.0061
		Cdn$/t-concentrate	For As (ppm) / S(%) ≤25.4	0
	Treatment Charge	Cdn$/t-concentrate	-	200
	Refining Charge – Gold	Cdn$/oz	-	N/A
	Refining Charge – Silver	Cdn$/oz	-	N/A
	Transport/Port Handling Costs	Cdn$/t-concentrate	-	193.24
	Concentrate Production	% of mill feed	-	2.29 x S(%) -0.5072
	Concentrate Moisture	%	-	10
	Insurance (Net Invoice Value)	% NIV	-	0.5

Note: NIV = net invoice value

15.4 MINING SHAPES

AMC used the Mineable Shape Optimizer (MSO) module from the CAE Studio 3 (formerly Datamine) 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 where necessary to ensure stope geometry viability and 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 the 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.

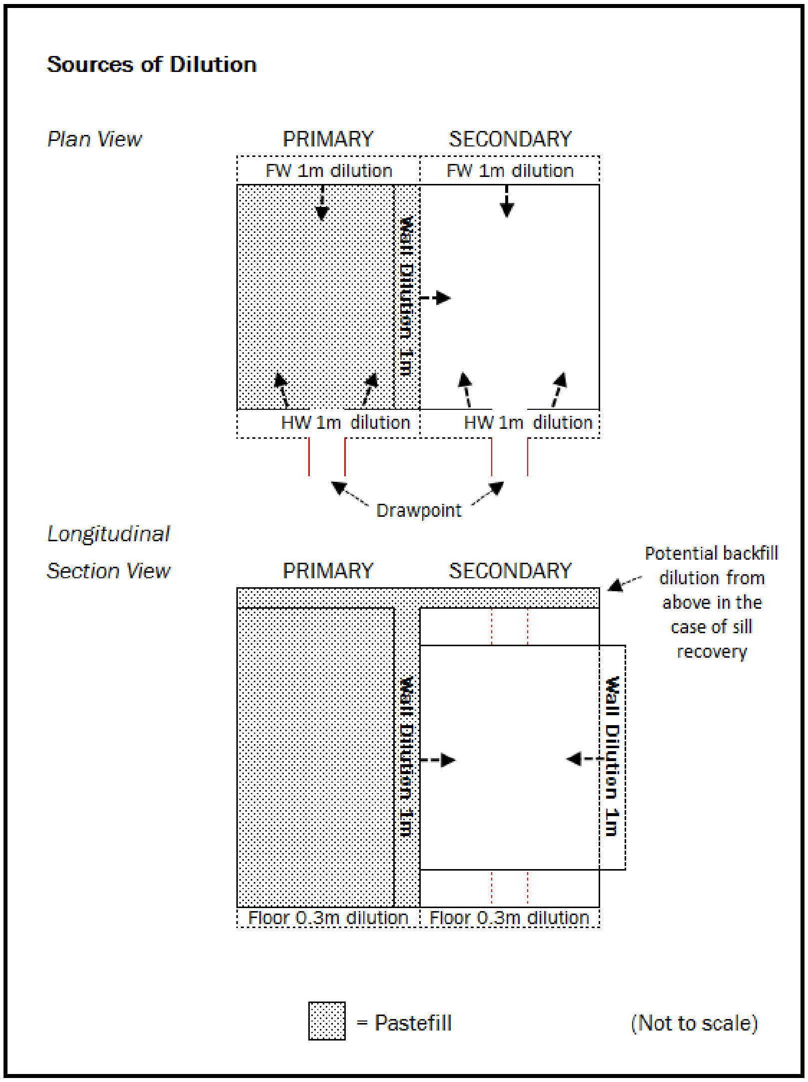
The dilution factors were calculated from standard overbreak assumptions that are based on AMCs experience and benchmarking of similar long-hole open stope operations.

- Longhole stopes (primary, secondary, tertiary) carry 1.0 m of dilution from paste or country rock overbreak into the design hanging wall and design footwall, and 0.3 m of backfill dilution from the floor.
- Secondary or tertiary 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.

Figure 15.1 shows the typical sources of stope dilution

Figure 15.1 Sources of Stope Dilution



The anticipated recovery of ore in proximity to the Brucejack Fault has been capped at 75%. This percentage was applied to all excavations within the fault, which is currently estimated to host approximately 0.72 Mt (4%) of Mineral Reserves.

The application of the above parameters yields an overall LOM ore recovery of 94% and an overall ore dilution of 12%. 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 570 m vertical distance, from the 990 m elevation to surface (approximately 1,560 m elevation).

15.6.1 VOK ZONE

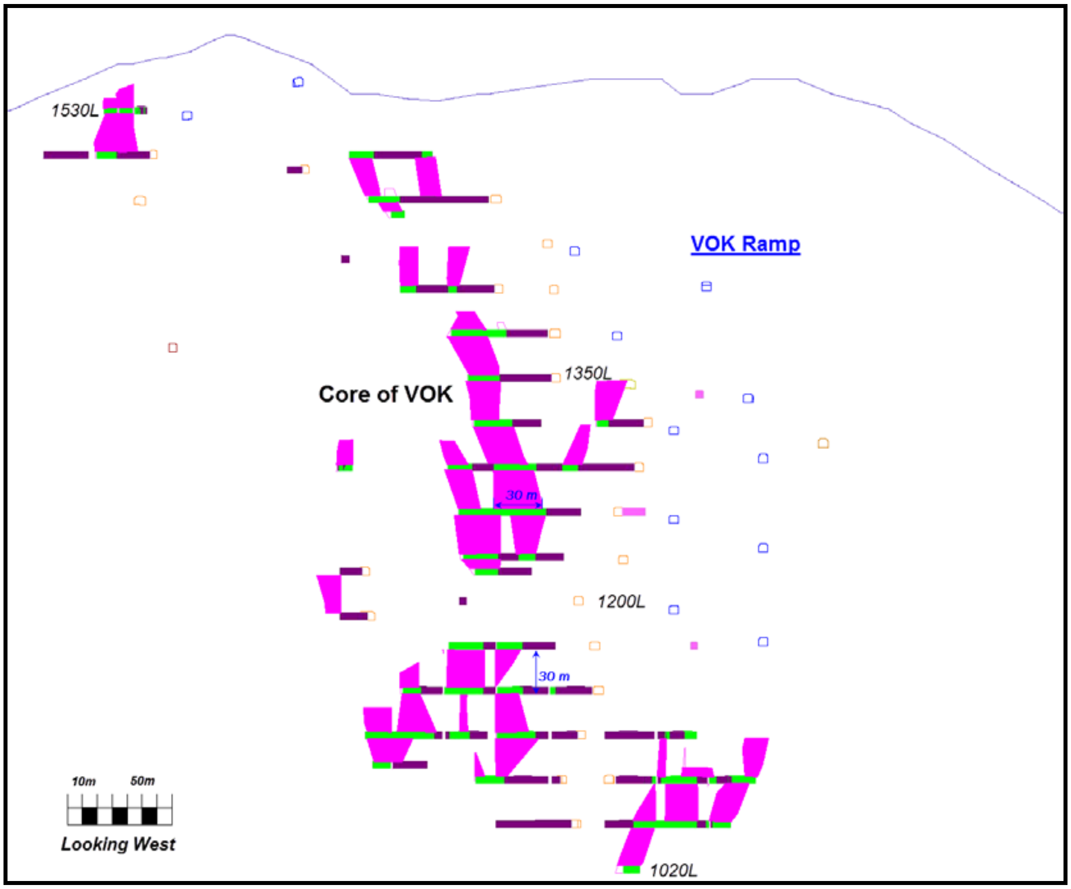
The VOK Zone hosts two main lenses that include 13.6 Mt of Mineral Reserves.

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 the core of the VOK Zone has a strike length of approximately 300 m. The other main lens in the VOK fault zone has a strike of approximately 350 m.

Orebody thickness varies considerably with elevation but, on average, the core of the VOK Zone is 20 m thick in the lower elevations (below 1,200 Level), 50 m thick in the mid-elevations (1,200 to 1,350 Level), and 25 m thick in the higher elevations (above 1,350 Level). Narrow Mineral Reserves have been delineated down to a minimum 3 m mining thickness. The VOK Zone has a slight plunge towards the east. Figure 15.2 illustrates typical widths found in the VOK Zone.

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

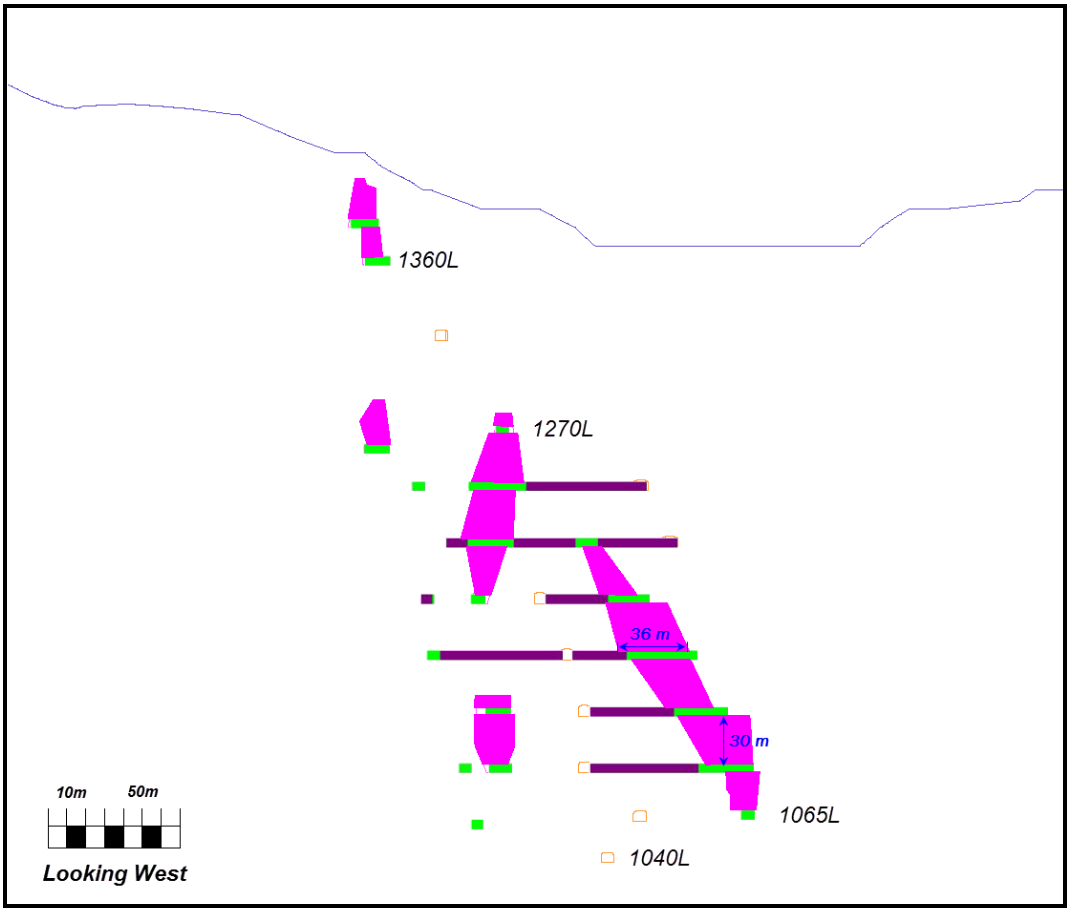


15.6.2 WEST ZONE

Mineral Reserves within the West Zone are contained within four lenses, three of which host 90% of West Zone reserves. Strike lengths vary considerably with elevation, averaging approximately 100 m in the larger lenses, while the smaller lenses are no more than 35 m along strike.

The average thickness is approximately 25 m, with the smaller lenses averaging only 15 m. Figure 15.3 illustrates typical widths found in the West Zone.

Figure 15.3 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 15.2. 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 also discussed in Section 16.0.

Table 15.2 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	2.1	15.6	12	1.1	0.8
	Probable	11.5	15.7	10	5.8	3.9
	Total	**13.6**	**15.7**	**11**	**6.9**	**4.6**
West Zone	Proven	1.4	7.2	383	0.3	17.4
	Probable	1.5	6.5	181	0.3	8.6
	Total	**2.9**	**6.9**	**279**	**0.6**	**26.0**
Total Mine	Proven	3.5	12.2	161	1..4	18.2
	Probable	13.0	14.7	30	6.1	12.5
	Total	**16.5**	**14.1**	**58**	**7.5**	**30.7**

Note: *Rounding of some figures may lead to minor discrepancies in totals. Based on Cdn$180/t cut-off grade, US$1,100/oz gold price, US$17/oz silver price, Cdn$/US$ exchange rate = 0.92.

Table 15.3 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	4.3	578	16.9	12	2.3	1.6
VOK Middle	5.7	503	14.9	10	2.7	1.9
VOK Lower	3.7	530	15.5	9	1.8	1.1
VOK	**13.6**	**534**	**15.7**	**11**	**6.9**	**4.6**
WZ Upper	0.6	304	4.2	407	0.1	8.0
WZ Lower	2.3	350	7.6	245	0.6	18.1
WZ	**2.9**	**340**	**6.9**	**279**	**0.6**	**26.0**
Mining Block Total	**16.5**	**500**	**14.1**	**58**	**7.5**	**30.7**

Note: *Rounding of some figures may lead to minor discrepancies in totals. Based on Cdn$180/t cut-off grade, US$1,110/oz gold price, US$17/oz silver price, Cdn$/US$ exchange rate = 0.92.

Figure 15.4 Reserve Shapes and Mining Blocks in the VOK Zone

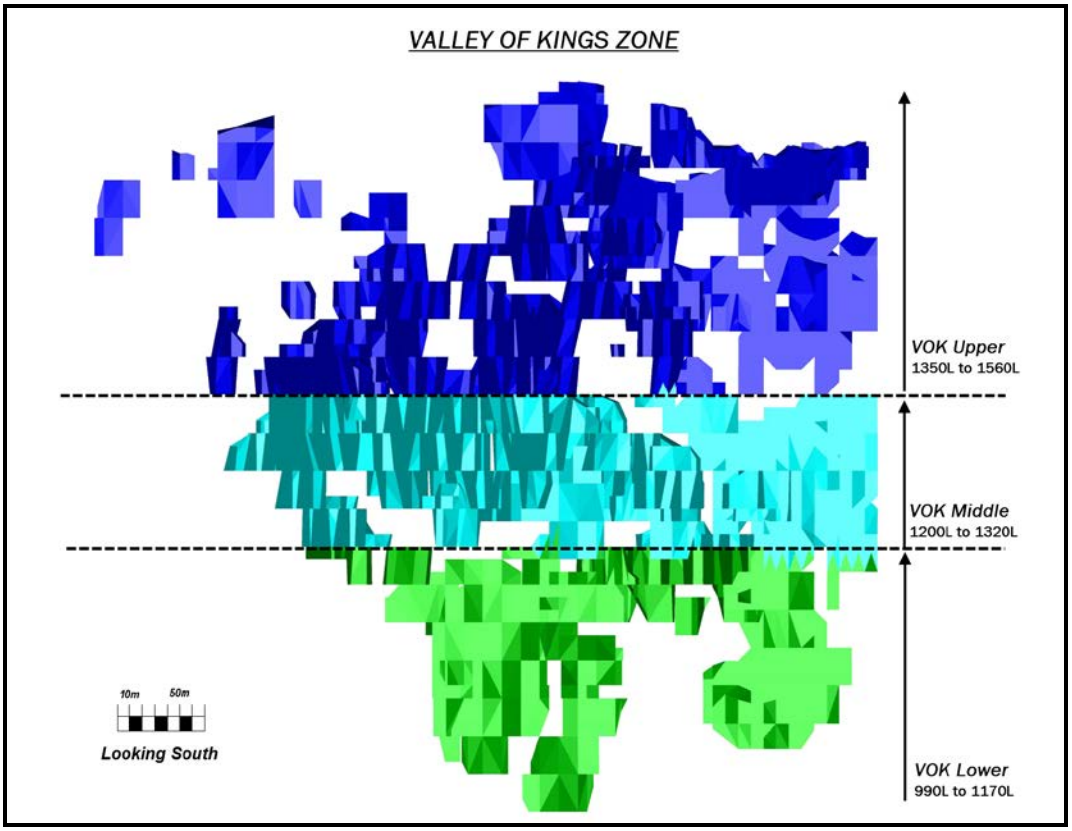
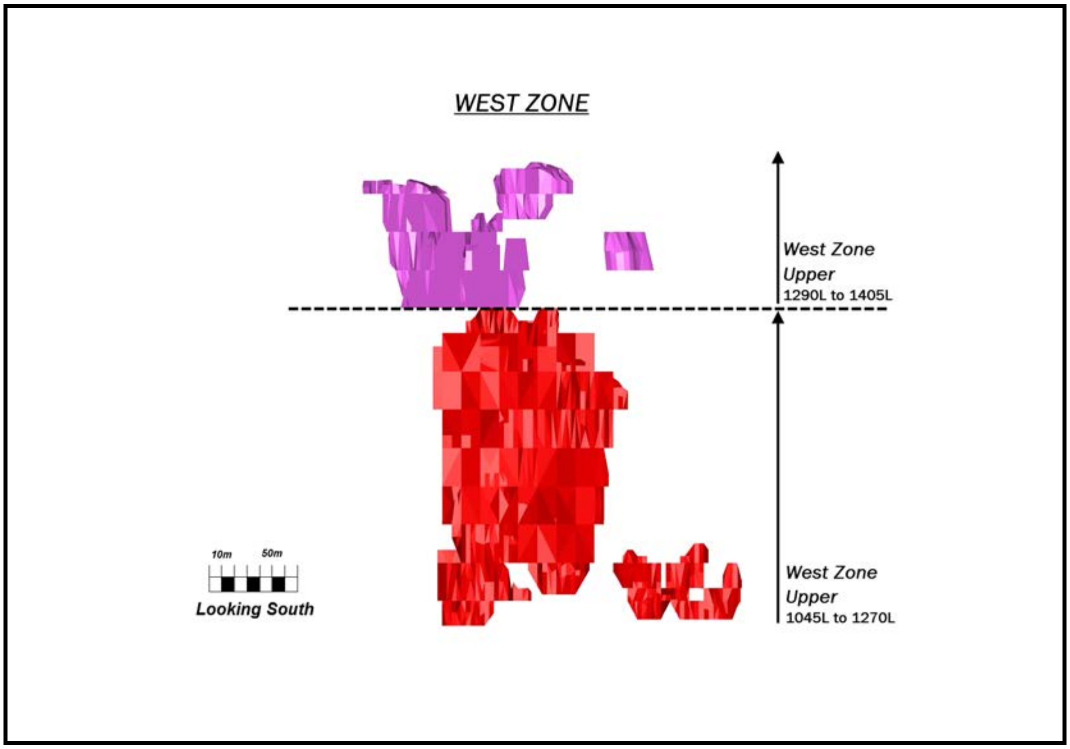


Figure 15.5 Reserve Shapes and Mining Blocks in the West Zone



16.0 MINING METHODS

16.1 GENERAL

The mine design is largely unchanged from the 2013 feasibility study (Ireland et al. 2013); although the crusher location has been moved closer to the VOK orebody in order to minimize truck haulage. This involved the addition of a second conveyor to suit the new geometry. Locations for other infrastructure items were likewise changed to suit the updated crusher location and to realize early development opportunities from the existing bulk sample access drive on the 1,345 Level.

Furthermore, the underground workshop has been scaled down from the previous design and the bulk of maintenance activities will now be performed at the surface facility. The underground workshop will be dedicated to light repairs and servicing.

The final material difference with respect to the previous design concerns the introduction of vertical clarifiers at the main sumps. The water ingress is estimated to be approximately 100 L/s.

As with the previous study (Ireland et al. 2013), the updated 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 two conveyors with a combined length of approximately 800 m. The existing West Zone portal will continue to provide access (and egress) to the mine, although this route will not generally be required for day to day operations once the main declines have been established.

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. The option to introduce ore passes into the design to reduce haulage requirements from the upper levels of the VOK was examined in detail during the feasibility study update. The study concluded that truck haulage without ore passes would yield similar costs over the long term, but with lower up-front capital, such that ore passes have not been brought into the updated design.

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. 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 600 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 420 m/mo during the first 12 years of production, and will decrease considerably in the latter years of mine life following completion of West Zone infrastructure development.

Major underground infrastructure will include the crusher, conveyors, ventilation raises, fans, heating system, pumping stations, a maintenance facility, electrical substations, a fuelling facility, explosives magazines, pastefill booster pump station, 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.

Two independent internal ramps, one for the VOK Zone and one for the West Zone, 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 main access decline will join the main surface portal to the VOK ramp at the 1,335 Level in proximity to the updated crusher location, on the same elevation. The workshop has been placed on the existing bulk sample access drive for early development and central location. The West Zone will be likewise be accessed from the existing bulk sample access drive in the latter half of the mine life. Figure 16.1 illustrates the general development arrangement.

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.

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

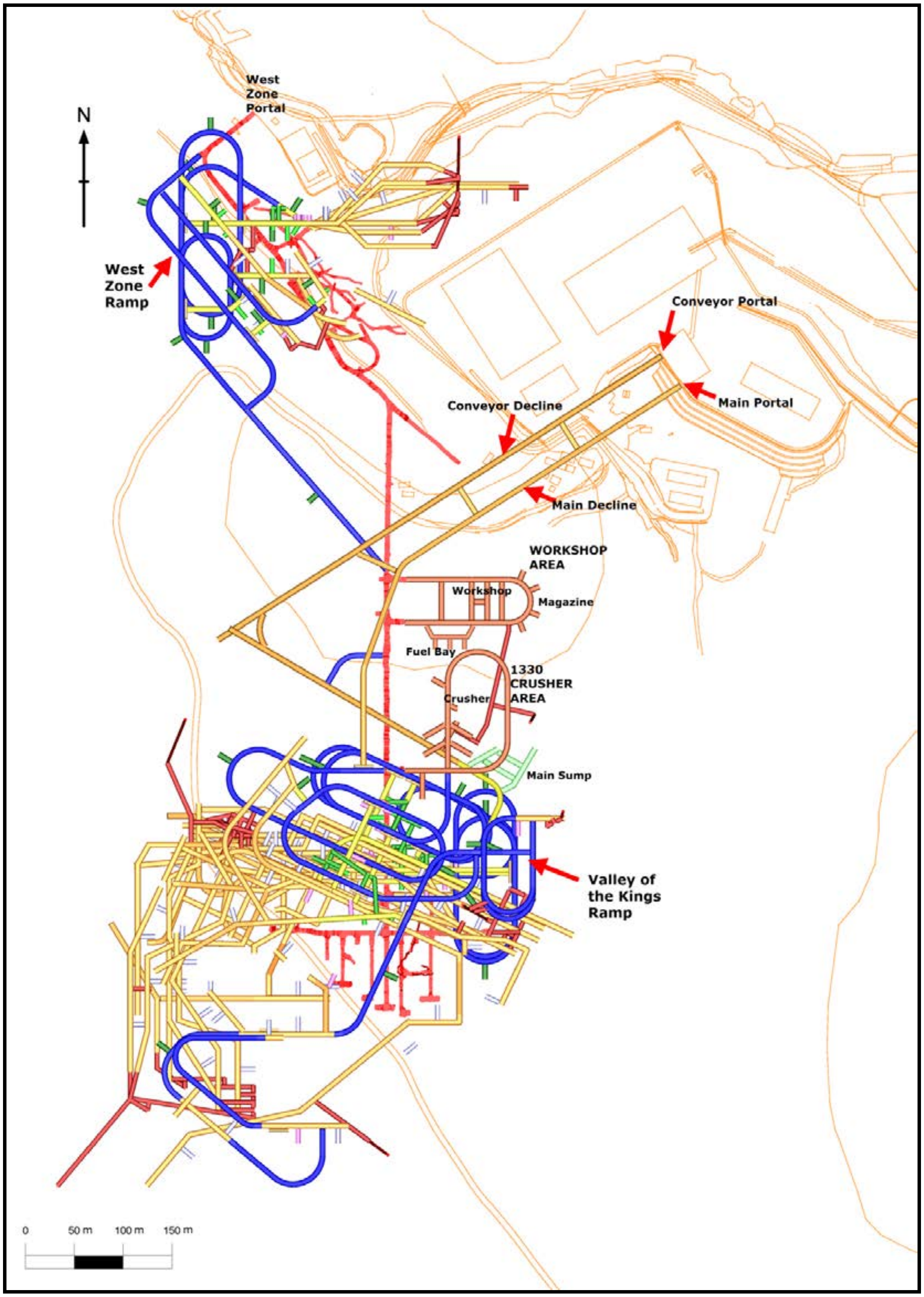
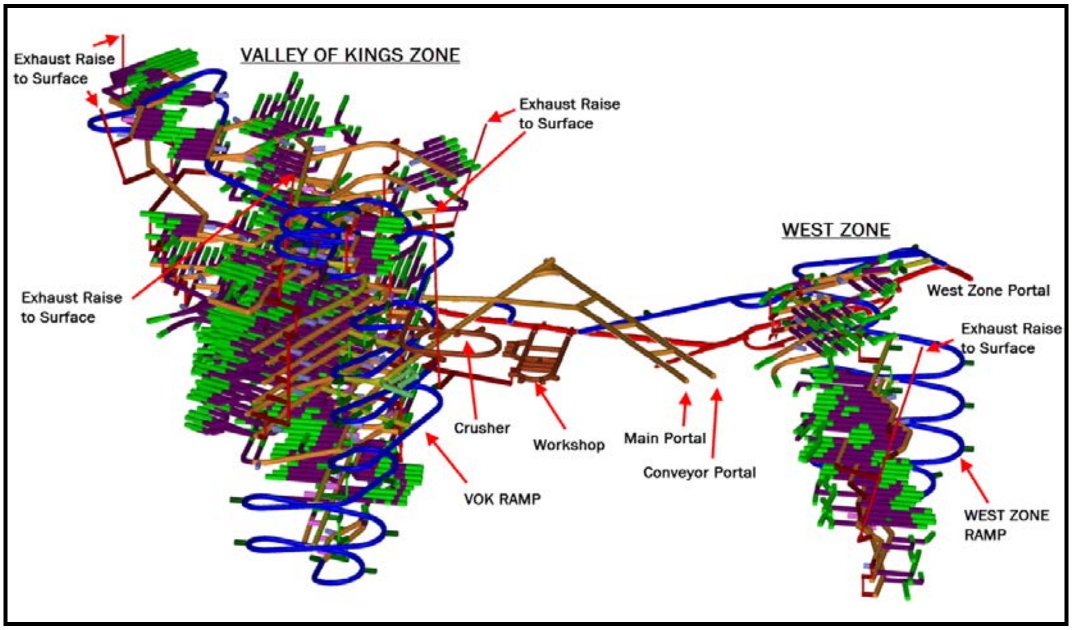


Figure 16.2 Brucejack Twin Declines and Ramp System

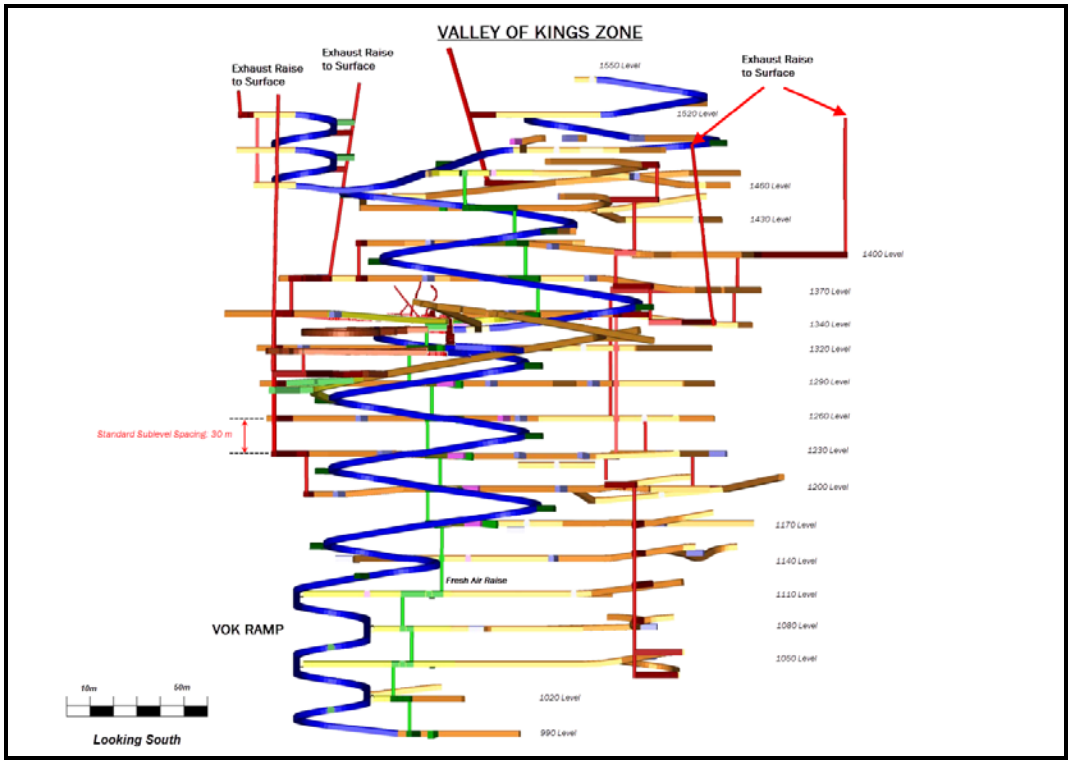


16.2.1 Level Development

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



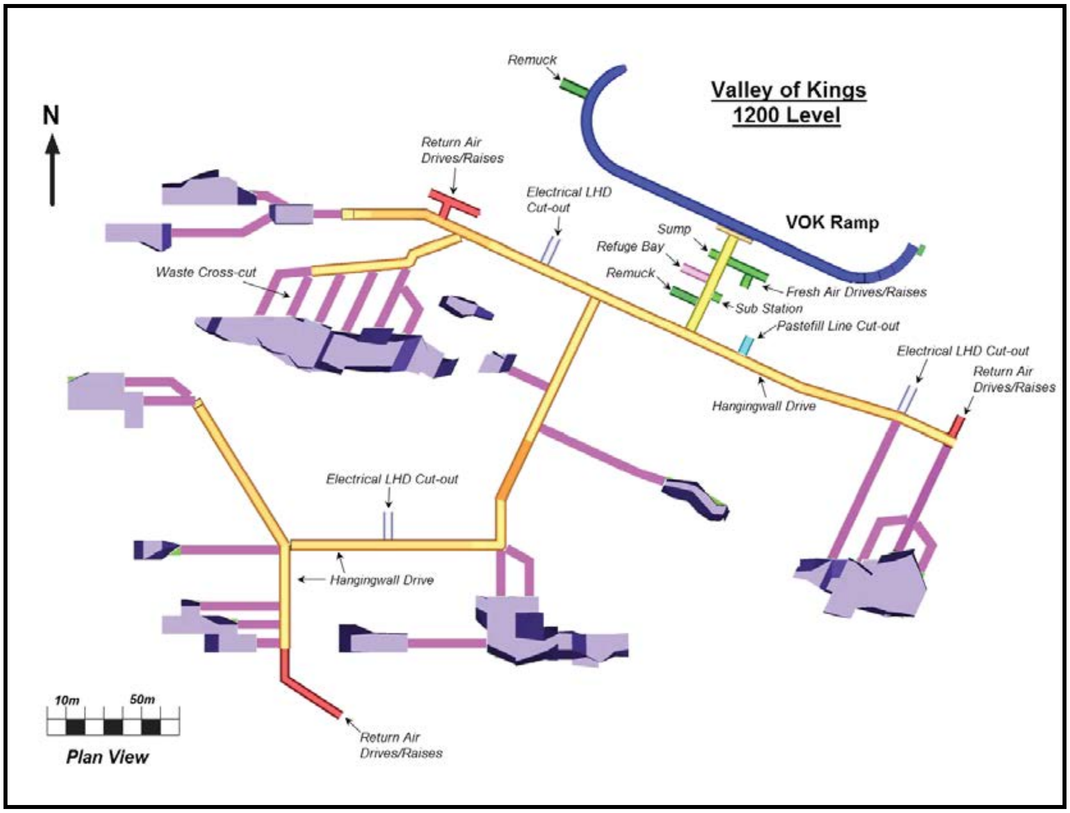
Level development will follow the general strike of the various lenses, providing access to the mineralized zones in a manner that promotes transverse mining wherever possible. Level development will generally be in the hanging wall, with hanging wall drives typically including excavations for sumps,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

Stope access crosscuts outside of the fault zone 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 stopes that are designed on 10 m spacings. Likewise, all fault zone ore will be on 10 m spacings to accommodate poorer ground conditions.

Following the recommendations of the previous feasibility study (Ireland et al. 2013), the updated design shows extraction of fault zone ore in stopes that are retreated perpendicular to the axis of the fault.

Figure 16.4 illustrates typical level development requirements.

Figure 16.4 Typical Level Plan – 1,200 Level in the VOK Zone



Lateral development design considered equipment size, services, and required activity. The 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 Designs – Hanging Wall Drive

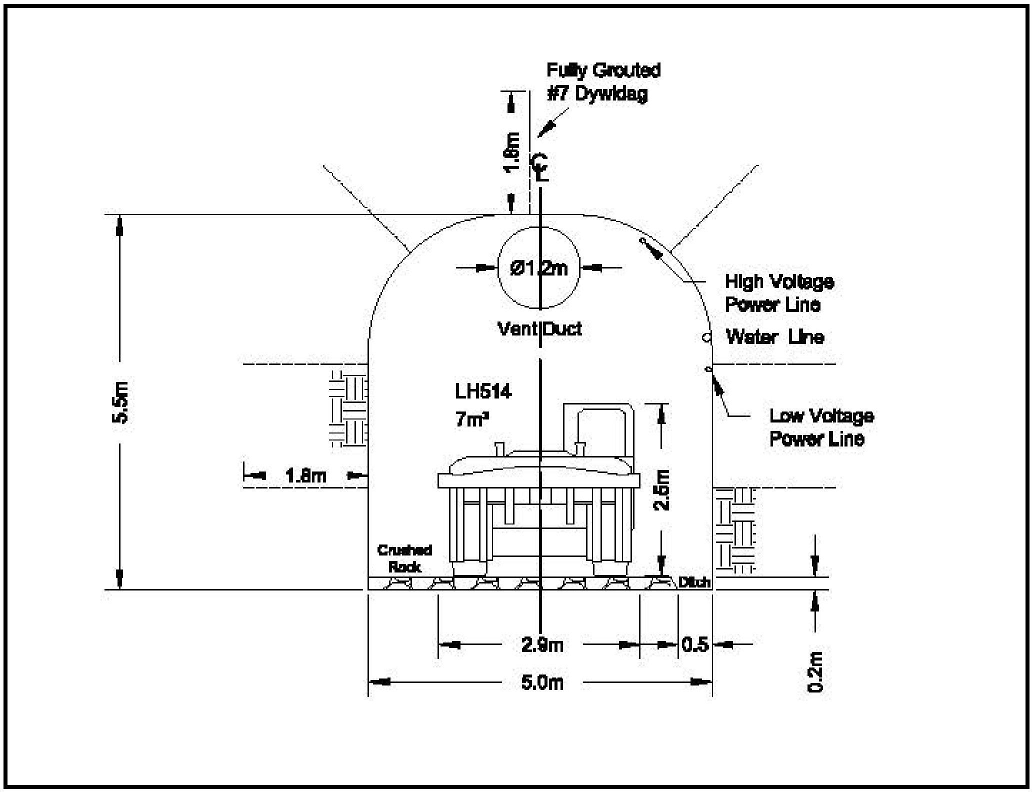


Figure 16.6 Standard Design – Main Decline

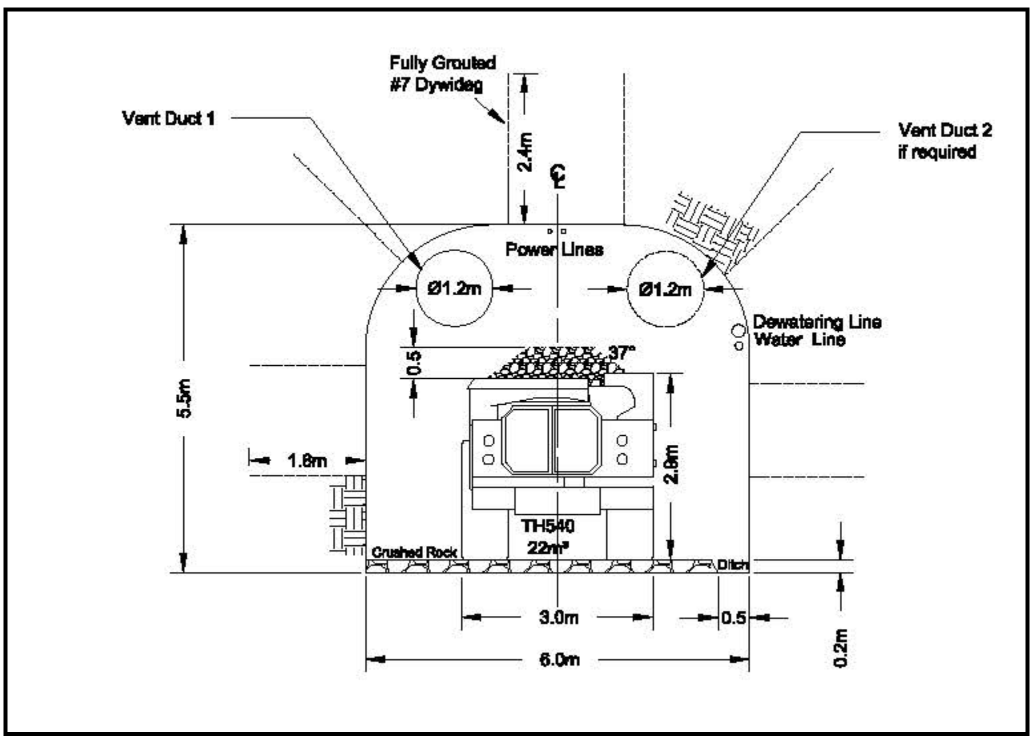


Table 16.1 Development Design Parameters

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

16.2.2 Stope Design

AMC used the MSO module from the CAE Studio 3 (formerly Datamine) 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 mining plan. The LOM plan includes 1058 stopes in the VOK Zone and 138 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 MSO Stope Shapes – VOK Zone

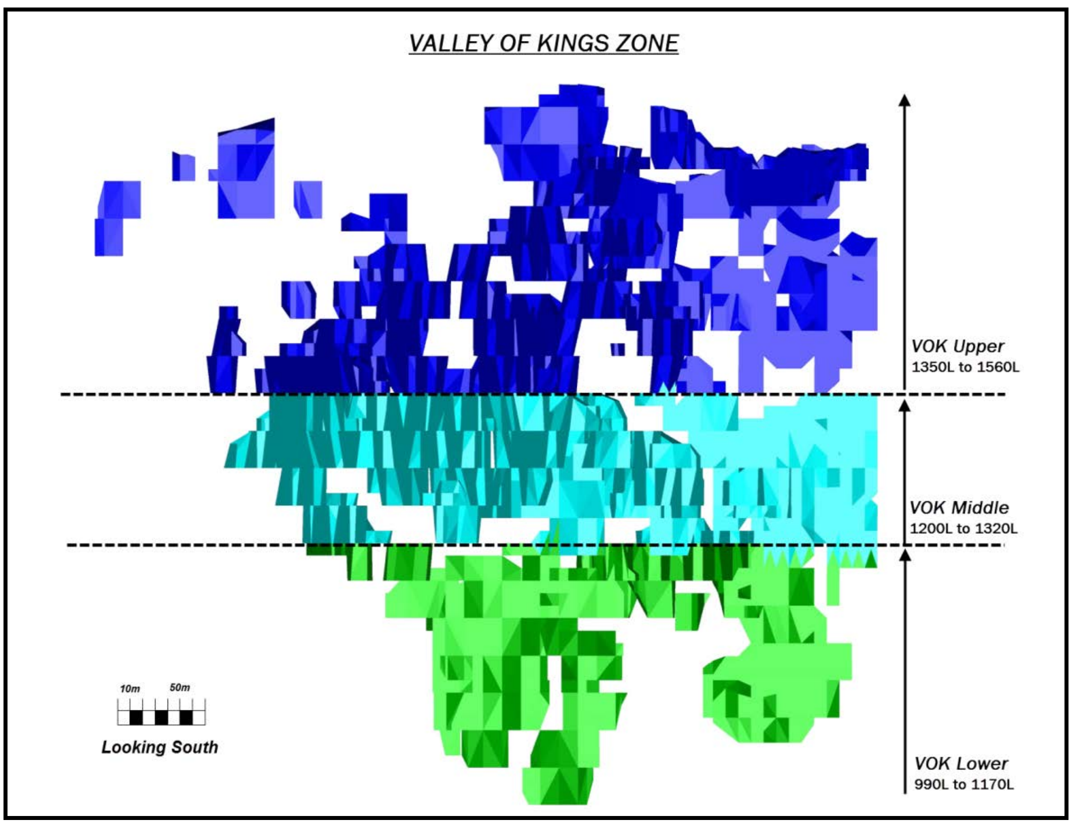
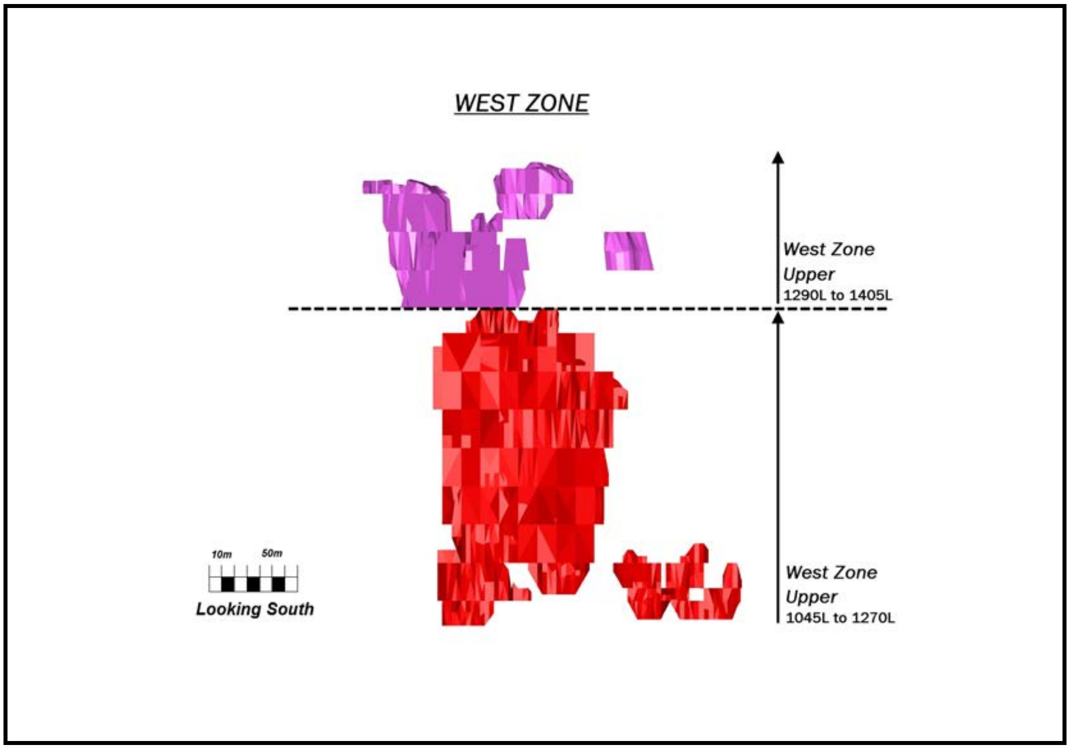


Figure 16.8 MSO Stope Shapes – West Zone



16.3 Mining Method and Sequence

16.3.1 Block Definition

The orebody was divided into five 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 each 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. Natural breaks in economic mineralization were favoured.
- Pre-production development requirements: The VOK Upper block and the VOK Middle block are close to existing workings, and will support the ramp-up of production in a reasonable timeframe.

- 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 already have 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.

However, in contrast to the previous feasibility study, full width slashing was not retained for poor or difficult ground, including sill recovery and stopes within the fault zone.

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 draw point brow is open to the extent where the operator may be exposed to uncontrolled sloughing from the stope cavity.

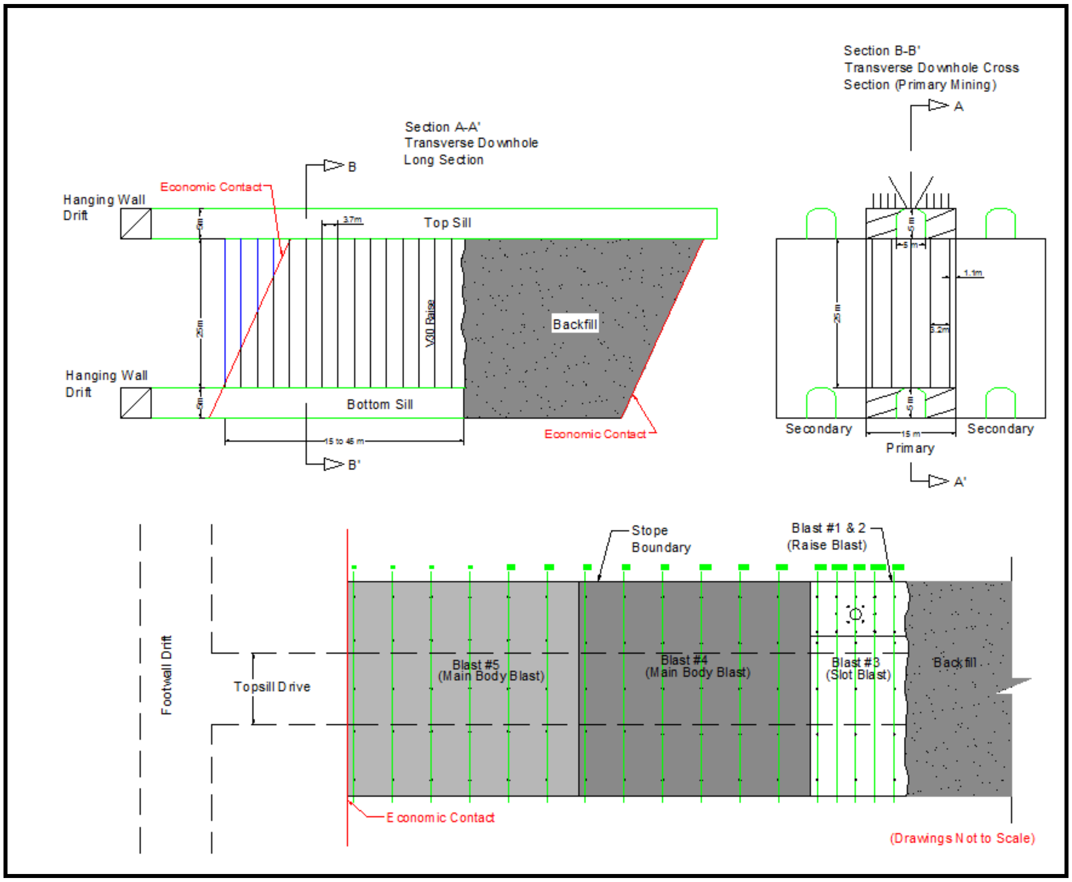
The empty stope will be remotely surveyed with cavity monitoring equipment. A barricade will be constructed in the draw point and the stope backfilled to just below the floor elevation of the top level. Crushed aggregate or ROM waste may be spread over the fill surface to reduce backfill dilution and 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 mine, 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.

Where applicable, the overcut and undercut will be slashed to the footwall and hanging wall contacts, although in numerous longitudinal stopes, no overcut is required and ore will be extracted via uppers drilling. In all other respects, the stope cycle will be similar to transverse LHOS.

Figure 16.9 Typical LHOS Design

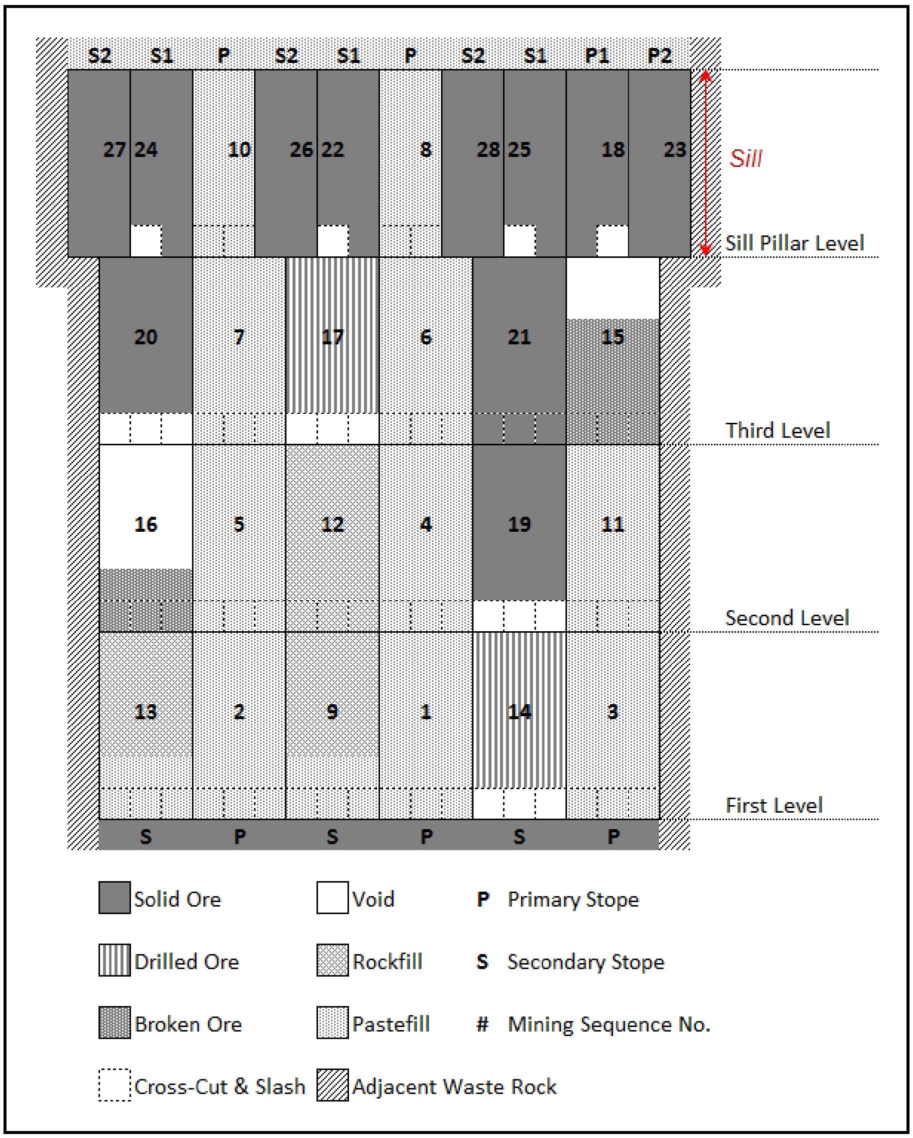


16.3.3 STOPE SEQUENCE

The mining sequence in any area 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 mine.

Figure 16.10 Example of Primary/Secondary LHOS at Brucejack Mine



16.3.4 BACKFILLING

The primary means of backfilling 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	122,432	800	5.5	1.40	171,405	9,427
Regular Paste	3,701,036	300	3.9	1.40	5,181,450	202,077
Low-strength Paste	1,192,444	100	2.8	1.40	1,669,421	46,743
Total	**5,015,912**	-	-	-	**7,022,276**	**258,247**

16.3.5 PASTE BACKFILL TEST WORK

Pretium 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 determined paste fill strengths. The density of the paste was low and resulting strengths required higher-than-expected cement content to achieve the target strengths.

Pretium 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 strength (UCS) 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 Stage 2 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	90	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

Stage 3 strength and rheology test work on bulk sample material is being completed to update paste recipes and binder dosages for the key strength targets. For this study, AMC adopted industry standard dosages to achieve the required 28-day strengths, as outlined in Table 16.4.

Waste Management and Stope Filling

Considerable quantities of 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 projected volumes of waste to be generated from milled ore and development headings, and the destination of these volumes over time. Over the LOM, 62% of development waste and 45% 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 ('000 m^3)	Paste Fill Volume ('000 m^3)	Tailings Underground ('000 t)	Waste to Surface ('000 t)
-2	0	0	324	0	0	0	324
-1	81	0	343	0	0	0	343
1	839	876	303	0	332	471	303
2	929	884	274	42	281	399	155
3	979	932	315	59	276	392	150
4	984	936	282	91	257	365	27
5	988	941	316	90	241	342	63
6	999	951	286	91	247	350	32
7	986	939	213	73	258	366	8
8	996	948	230	75	252	358	19
9	994	946	321	89	259	369	71
10	987	940	315	99	277	393	36
11	985	938	292	88	231	327	45
12	993	945	179	62	261	371	4
13	986	939	32	11	346	491	0
14	981	934	40	14	386	548	0
15	991	943	52	18	372	529	1
16	908	864	46	16	337	478	1
17	663	632	31	11	274	389	0
18	281	267	3	1	129	184	0
Total	**16,550**	**15,755**	**4,198**	**934**	**5,016**	**7,123**	**1,583**

16.4 Development and Production Schedule

16.4.1 Production Rate

Given the decrease in Measured and Indicated Mineral Resource tonnage in the resource model for the updated feasibility study, the ability to achieve and maintain the target 2,700 t/d production rate was necessarily reviewed. A high level mine model comprising the 2012 mine design against the backdrop of the updated resource model was used for this purpose.

Assumptions inherent in the estimation of stope cycle times were revisited with the conclusion that they remained valid for the review of overall mine production capability. The key cycle time parameters are:

- mining of any given stope proceeds at 450 t/d, inclusive all unit operations
- only two stopes per level are active at any given time.

In the interest of determining the maximum achievable production rate, the schedule was not constrained by the availability of resources such that no restrictions were placed on the following:

- ventilation volume
- pieces of equipment operating at one time
- total workforce requirements or availability
- backfill requirements
- number of stopes working at any given time other than two stopes per level
- number of levels or areas in production.

The analysis concluded that the unconstrained production rate would be 3,500 t/d over the LOM. Given the inevitable impact of resource constraints on the optimal production rate, it was reasoned that detailed scheduling should proceeded at 2,700 t/d, unchanged from the previous feasibility study (Ireland et al. 2013).

A detailed mine design was subsequently completed for the new resource model and scheduled to 2,700 t/d steady state ore production, although the final schedule is actually closer to 2,730 t/d. As with the 2012 feasibility study, the resources required to ramp up and maintain this rate were estimated and shown to be reasonable and achievable.

The final schedule was constrained to reflect realistic mining practices and availability of equipment. The model limited the number of active stopes at any one time to; six blasting and mucking (with a maximum of two per level and three per zone), one backfilling, three drilling and up to seven undercutting. The average number of active stopes at any one time is 12, with variations from 10 to 17. The number of available stopes could be higher.

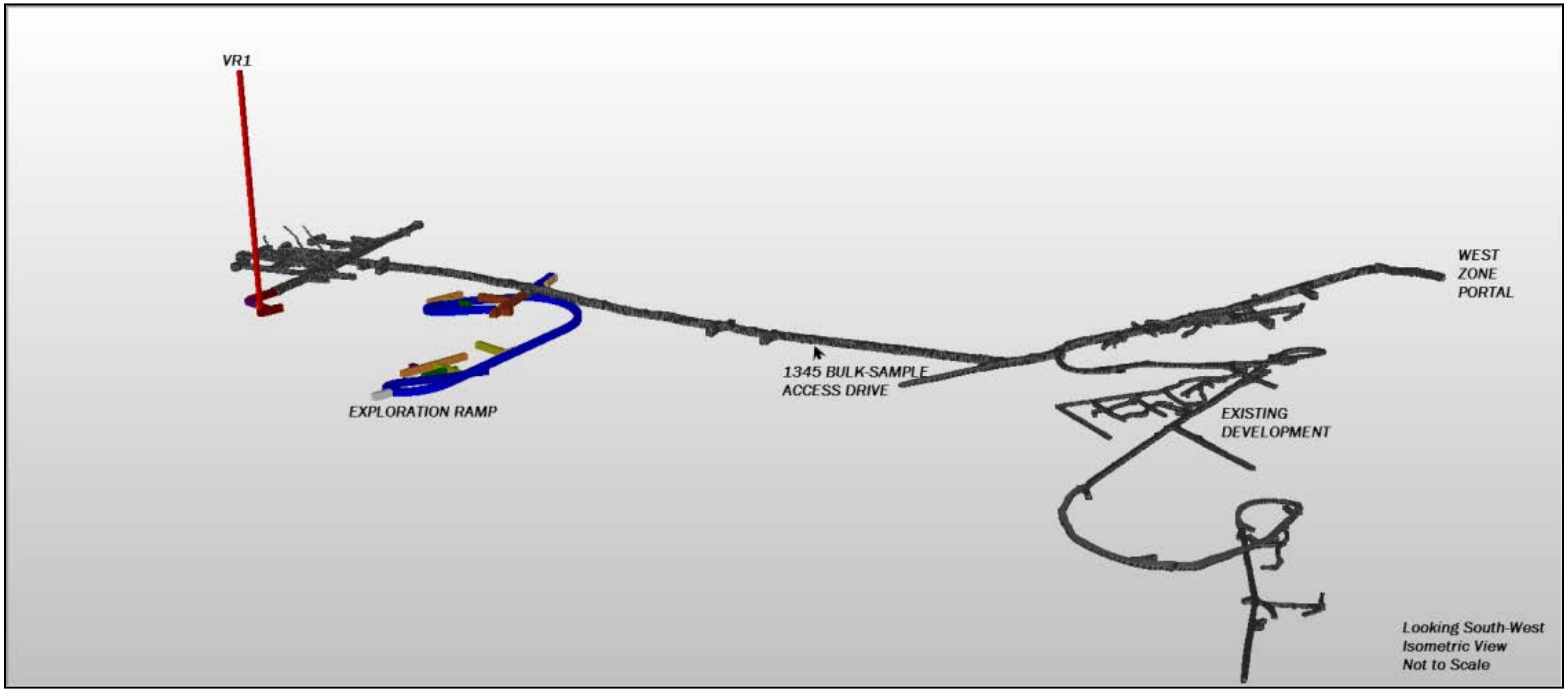
Ireland et al. (2013) concluded that a steady state production rate between 2,700 t/d and 3,000 t/d would be achievable, and a conservative view was adopted at that time. AMC still considers that the optimum production rate from the mine may be slightly higher than 2,700 t/d, but, in the absence of operating experience, the more conservative approach was maintained in this feasibility study update.

Again, similar to Ireland et al. (2013), detailed scheduling has shown the 2,700 t/d rate to be achievable with a high level of confidence.

16.4.2 PRE-PRODUCTION DEVELOPMENT

Exploration development completed to date includes the West Zone portal, cross cut at approximately 1,315 m elevation to the VOK Zone and bulk sample development at the 1,345 m elevation in the VOK. Further exploration development is planned from the existing 1,315 cross cut down to the 1,260 m elevation, as well as 1,350 Level raise access and ventilation raise VR1 from the south end of the 1,345 bulk sample development. Figure 16.11 shows the extent of development that has been or will be completed at the start of the pre-production development phase.

Figure 16.11 Extent of Development Prior to Project Start



Pre-production underground development will occur over a 24-month timeframe before the first stope is extracted. Mobilization is expected to occur two months prior to the start of development.

Development drive ore produced during this period will be hauled directly to a surface stockpile pending commissioning of the material handling system. Once the mine is in production, stockpiled ore will be backhauled underground to the crushing plant. All waste rock and ore will be hauled to temporary stockpiles near the portals where surface haul trucks will re-handle the material to the final waste disposal location or ore stockpile.

The mining strategy established five mining blocks. The VOK will have two sill elevations, one at 1,200 and one at 1,350, plus the mine bottom at the 990 Level. These are labeled the VOK Lower, Middle and Upper blocks respectively. There will also be two mining blocks in the West Zone, one at the 1,030 Level and a sill at the 1,270 Level. These are labelled the WZ Lower and WZ Upper blocks respectively.

The development strategy targets the VOK Middle and Upper blocks as the first priority, followed by the more distant lower block to sustain production. Excavation for required 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 from the 1,200 Level and 1,350 Level of the VOK Zone area. Critical path pre-production activities include:

- access development to the top and bottom of the crusher chamber, excavation and support of the crusher chamber, and installation of the crusher
- decline development to the 1,200 Level, development of the 1,200 Level, and continuation of ventilation raise VR1 from 1,200 to 1,350 to establish a ventilation circuit in the lower part of the mine
- twin decline development from surface and underground to allow the installation of the portal structure, main ventilation fans and underground conveyor system
- excavation and construction of the sump, maintenance workshops, magazine, fuel bay, and other ancillary installations
- development of 1,200, 1,230, 1,350 and 1,380 Levels including ventilation raise VR3 to allow commencement of stoping

Figure 16.12 illustrates the extent of development required for the main onset of stope production between the 1,200 and 1,380 Levels. A total development requirement of 12,728 lateral meters and 947 vertical meters is planned in the first 24 months. Up to 627 m/mo of development advance will be required at the peak activity level with an average rate of 460 m/mo and 605 m/mo in years -2 and -1 respectively.

Figure 16.12 Extent of Mine Development at the Main Onset of VOK Stoping

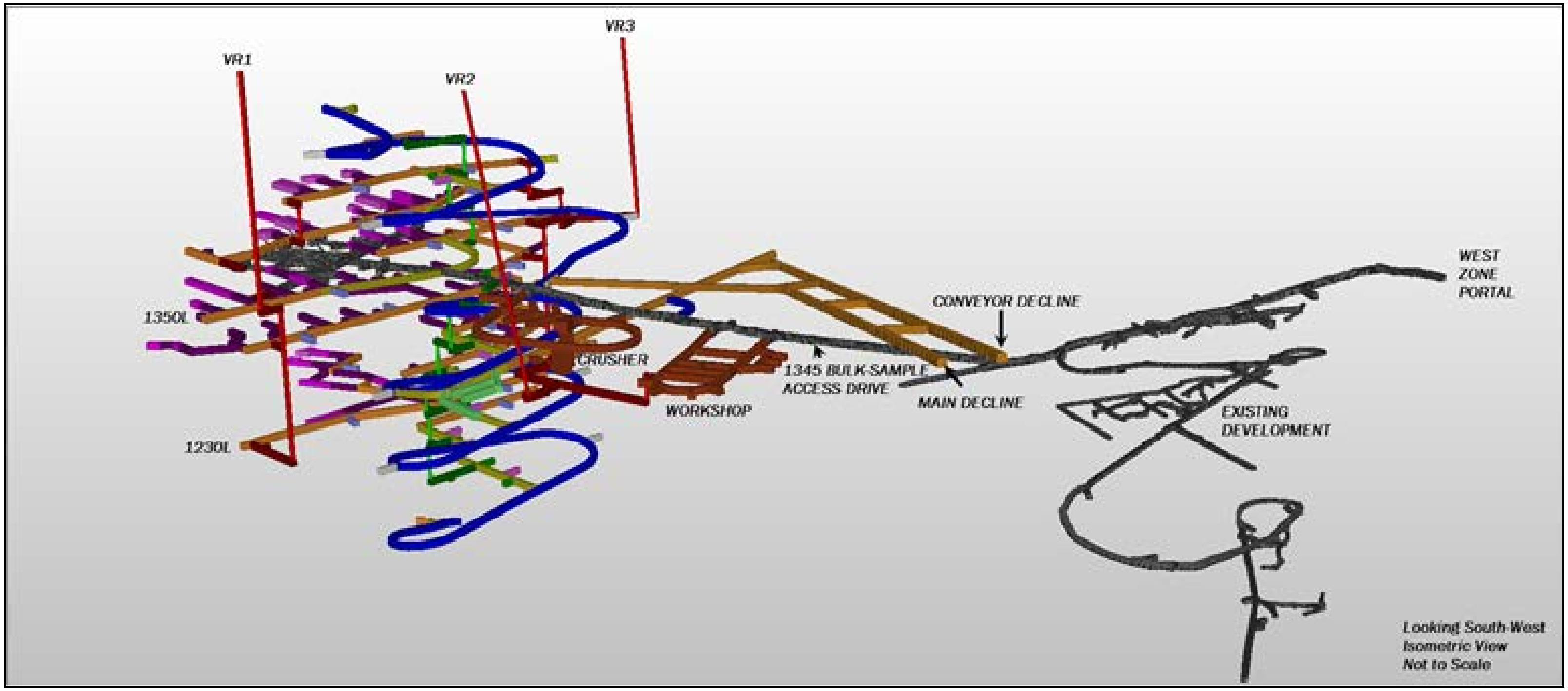
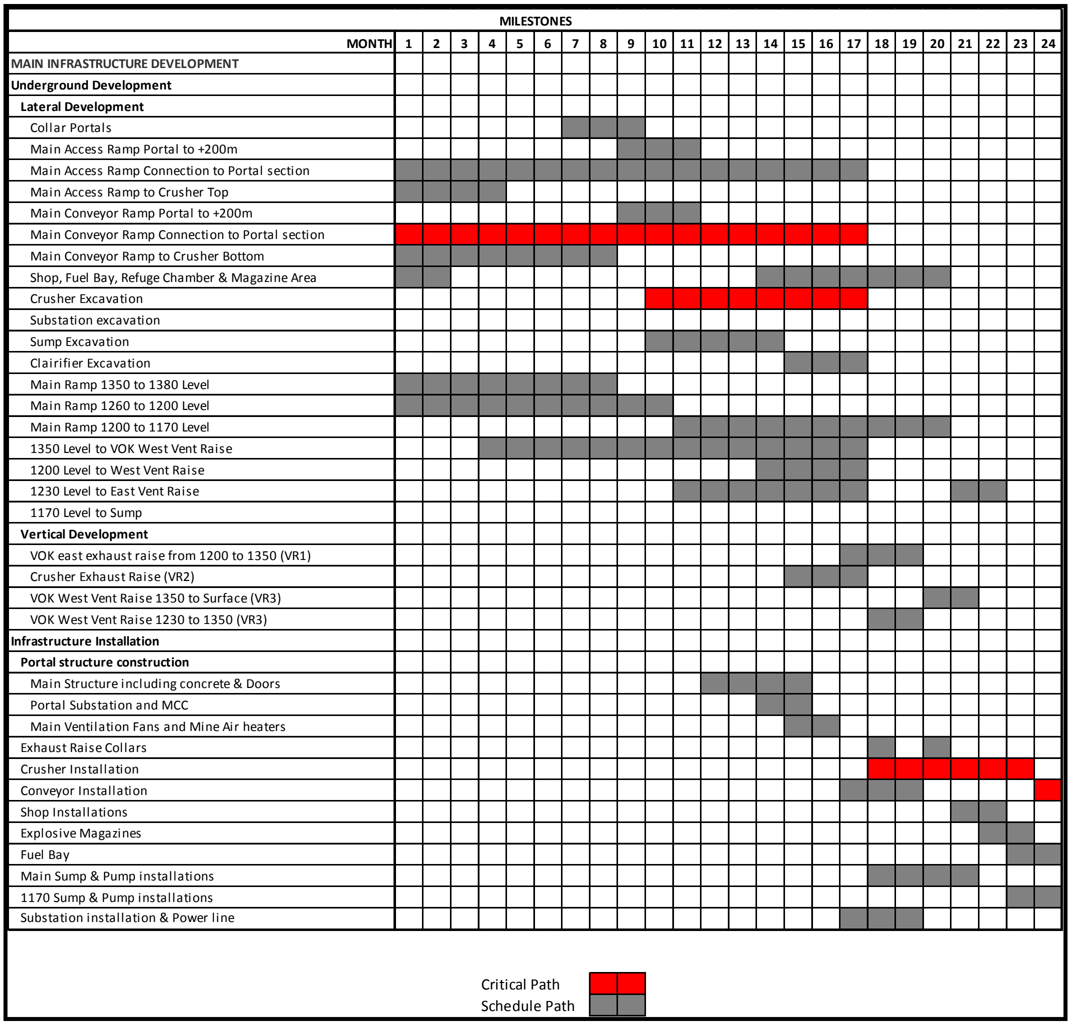


Figure 16.13 presents the critical path activities leading to the commissioning of the material handling system and initial stoping in the VOK Zone.

Figure 16.13 Critical Path Construction and Development Activities



16.4.3 Execution and Transition Plan

Pre-production development will be completed by the Contractor with a transition period to owner personnel and equipment starting six months prior to production. The pre-production period is broken down into four phases.

- Mobilization period – first six months. The Contractor is building up development crews and begins underground development.
- Peak Development period – months 7 to 12. The Contractor employs the full complement of development crews to meet scheduled development.
- Initial Transition phase – months 12 to 24. The Owner's crews begin to supplement Contractor crews. Longhole drilling will begin with Owner crews and

one of the Contractor's development crews will be replaced by an Owner development crew.

- Final Transition phase - months 25 to 30, Owner now employs three development and all production crews. Contractor continues to operate one development crew.

For the purposes of this study, a development crew was considered to contain the following primary equipment: one, 2-boom jumbo; one, 14-t loader (LHD); one, 40-t truck, one bolter shared between two crews, one cable bolter shared between four crews.

Eleven miners will be required per shift to man the equipment as well as do support work such as services installation and logistics. An additional six persons will be required for maintenance support (mechanics and electricians), together with personnel for two Alimak crews and technical support staff. The total manpower requirement (excluding infrastructure installation crews) will be 43 per crew or 170 in total, with 85 persons on site at any time.

Mobilization of mining equipment will begin two months prior to the anticipated mining start date. Two weeks have been allocated to complete site setup for development, with only two crews active during the initial two months of development. Four excavation crews will be active from months three through to month six. Development rates for the first six months were reduced with respect to steady state expectations to account for the anticipated level of efficiency during the start-up period.

During the peak development period, four development crews will operate with multiple headings available for each crew. The development rate was limited to 150 m/mo per crew. The main access ramps will be excavated during this phase to take advantage of the hiatus in surface construction during the first winter. Personnel on site will increase to 88, excluding installation crews, from months seven to 11.

During the initial transition phase, owner mining crews will be phased in and site manpower will gradually increase from 88 to 165 during this period. The onset of longhole drilling and slot raise excavation will also take place during this time. The Contractor's four development crews will be reduced to one crew, losing one crew every four months. The Owner will begin phasing in development crews at the same time, such that four development crews are continuously active on site.

During the development phase, equipment will be provided to the Contractor by the Owner from the Owner's permanent fleet. In the case of insufficient equipment being available, the Contractor will supply additional equipment required for the development phase.

In addition to development work, installation of the conveyor, crusher, sumps, workshops and other underground infrastructure will take place during this time. To complete this work, 40 persons will be required, with 20 persons on site at any time. These personnel will first be required 12 months after the initiation of development, when the portal will be available for installation. Installations will continue through the remaining 12 months of the development phase to production start-up.

Table 16.6 shows the manpower build-up through the development phases.

Table 16.6 Crew Size During Development and Transition Phases

Personnel on Site	Month						
	1 to 3	4 to 6	7 to 12	13 to 18	19 to 24	25 to 30	30+
Contractor							
Staff	4	4	4	4	4	4	0
Mining	29	54	54	54	42	13	3
Maintenance/Support	1	2	2	2	2	0	0
Construction	0	0	3	20	20	6	0
Owner							
Staff	7	8	8	25	28	26	22
Mining	0	0	0	2	30	99	99
Maintenance/Support	9	17	17	33	40	58	58
Total On Site	50	85	88	140	165	205	181
Total Personnel	100	170	176	279	330	410	362

16.4.4 Sustaining Development

Development of the VOK Upper and Middle blocks alone is insufficient to sustain 2,700 t/d of ore production. The VOK Lower 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 990 Level
- development of the 990, 1,020 and 1,050 levels
- continuation of VR1 from the 1,050 Level to the 1,200 level
- excavation of the fresh air raise system from the 1,050 Level to the 990 Level

The VOK ramp development will not be interrupted at the 1,200 Level, but will advance continuously to the bottom of the mine (990 Level). Levels will continue to be developed and stoping will continue in all three VOK blocks. Development to the WZ will begin later in the mine life to allow production from the Lower and Upper WZ blocks. This development is timed such that the 2,700 t/d mining rate can continue as long as possible without interruption. LOM development rates are shown in Table 16.7.

Table 16.7 LOM Development Requirements

Year	Capital		Operational		Total	
	Lateral (meq)	Vertical (m)	Ore (meq)	Waste (meq)	Lateral (meq)	Vertical (m)
-2	4,660	117	0	809	5,469	117
-1	5,086	831	930	1,243	7,259	831
1	1,810	0	1,630	2,027	5,467	0
2	1,467	110	2,081	1,744	5,292	110
3	2,013	261	1,636	1,575	5,224	262
4	1,783	144	1,906	1,489	5,179	144
5	2,160	75	1,542	1,436	5,139	75
6	1,595	170	1,719	1,716	5,030	170
7	742	34	2,406	1,853	5,001	34
8	1,124	21	1,965	1,595	4,684	21
9	2,199	65	1,521	1,479	5,199	65
10	2,360	97	918	1,194	4,472	97
11	1,719	371	1,731	1,613	5,063	371
12	819	90	2,459	1,332	4,610	90
13	0	0	1,507	419	1,926	0
14	0	0	835	525	1,360	0
15	0	0	783	674	1,457	0
16	0	0	1,020	592	1,612	0
17	0	0	535	409	944	0
18	0	0	89	43	132	0
Total	**29,537**	**2,386**	**27,215**	**23,767**	**80,518**	**2,386**

16.4.5 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 and the phasing of the various blocks.

Figure 16.15 shows the LOM split of production by development and stoping.

Table 16.8 is a summary of projected LOM production tonnes and grade.

Figure 16.14 Life of Mine Production Schedule by Mining Block

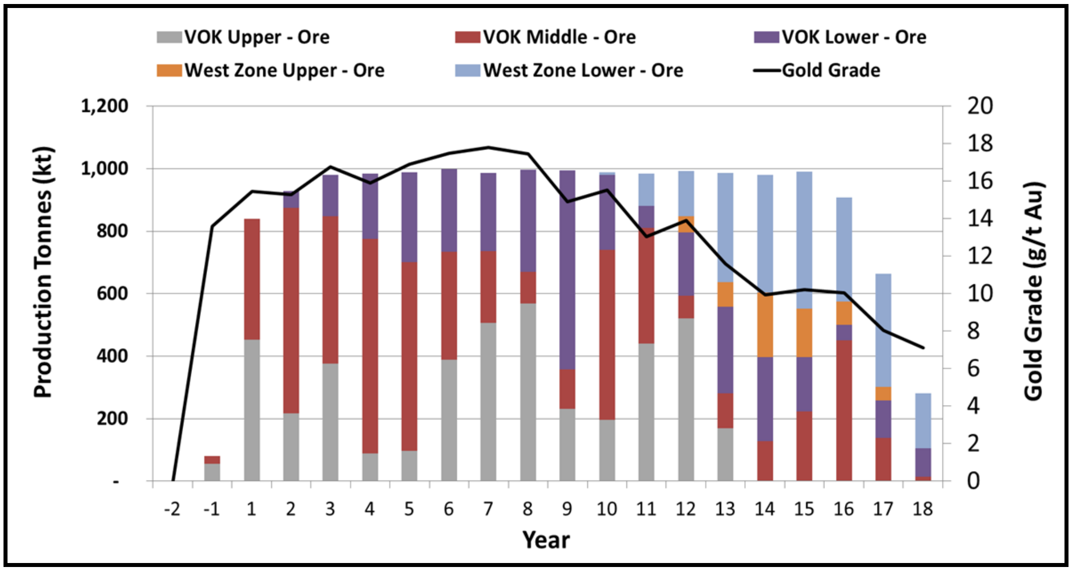


Figure 16.15 LOM Production Schedule by Activity

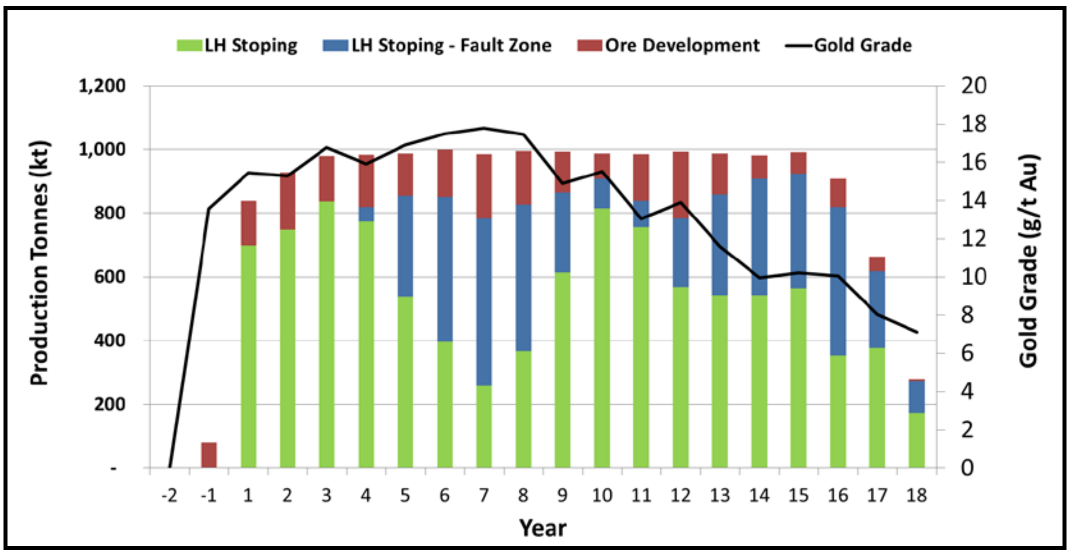


Table 16.8 LOM Tonnes and Grades

Year	Ore (kt)	Au (g/t)	Ag (g/t)	NSR ($/t)
-2	0	0	0	0
-1	81	13.6	11.5	451
1	839	15.4	11.7	521
2	929	15.3	11.7	516
3	979	16.8	12.8	574
4	984	15.9	9.9	539
5	988	16.9	11.0	579
6	999	17.5	10.6	601
7	986	17.8	11.8	612
8	996	17.5	11.7	600
9	994	14.9	10.2	507
10	987	15.5	11.2	525
11	984	13.0	29.3	444
12	993	13.9	69.2	497
13	986	11.6	102.8	430
14	981	9.9	151.9	393
15	991	10.2	158.7	407
16	908	10.4	104.1	376
17	993	8.0	254.7	375
18	281	7.1	271.9	350
Total	**16,550**	**14.1**	**57.7**	**500**

16.5 GEOTECHNICAL

Geotechnical designs and recommendations contained in the 2013 Feasibility Study (Ireland et al 2013) are based on the results of site investigations and geotechnical assessments completed by BGC on behalf of Pretivm. The assessments included rock mass characterization tasks, structural geology interpretations, excavation and pillar stability analyses, and ground support design.

Geotechnical site investigations completed to support the 2013 Feasibility Study (Ireland et al. 2013) 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.

For the 2014 Feasibility Study Update, the proposed twin portal has moved approximately 60 m to the west, the elevation of the surface decline intersection with main mine development has risen by approximately 30 m, the infrastructure excavations (crusher, etc.) have moved approximately 250 m to the east, and the mining plan through the Brucejack Fault Zone has been modified. No new rock mechanics site investigations or analysis work was completed for the feasibility study update. The effect of the above noted changes on the 2013 Feasibility Study rock mechanics assessments are noted in the appropriate sections below.

16.5.1 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 16.9.

Table 16.9 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.

16.5.2 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).

Pretivm has not yet developed an authoritative structural geology model for the Project; therefore, 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.

16.5.3 Underground Rock Mechanics

Stope Design Criteria

For the 2013 Feasibility Study (Ireland et al. 2013), 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. 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 16.10.

Table 16.10 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 10 m, contingent on encountered ground conditions, 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	-	

table continues…

Opening Type	Cross Section (w by h, m)		Ground Support Type	Length (m)	Spacing (m)	Shotcrete Estimate (%)	Additional Notes
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.
Shotcrete estimate (%) is based on the percentage of total development length estimated to require shotcrete support.
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 16.11. Primary stopes should be tight-filled as best as possible.

Mining through the Brucejack Fault

The Feasibility Study mine plan had development (stope access drifts) within and sub-parallel to the Brucejack Fault Zone. At the current level of study, the interpreted Brucejack Fault Surface has been used to plan developments near or within the fault zone to 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 will 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 will 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.

For the FS, 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. Because the crusher location has changed for the current study, the proposed location now encompasses rock within the VOK D3 geotechnical domain. In general, the VOK D3 domain has higher rock mass strength than the VOK D2 geotechnical domain; for planning purposes, the FS designs are considered appropriate for this revised location.

Recommendations for additional site investigation at the next level of study are provided in Section 26.0.

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 16.11.

Table 16.11 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 proposed portal site has moved approximately 60 m to the west since the 2013 Feasibility Study (Ireland et al. 2013) site investigations were completed. Therefore, there is no site-specific geotechnical data available for the proposed location shown in the feasibility study update mine plan. The following recommendations assume that the rock mass at the proposed locations is of similar quality as the rock mass encountered in the two drill holes completed at the original feasibility study portal site. Overburden thickness estimates based on those holes should not be used for material takeoff estimates for the new location.

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. Benches serve as catch-benches for small rockfall and reduce the overall slope; however a double-bench can be excavated at the immediate portal face to allow congruency with portal infrastructure. 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 time periods and general conditions covered by this work extend back to 2010, when initial investigations were conducted by BGC at the site. Data available through late-2013 and early-2014 were used in model development, calibration, and benchmarking. 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 Environmental Assessment - Numerical Hydrogeologic Model" and dated June 6, 2014 (BGC 2014).

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 Brucejack Lake catchment is approximately 27% glaciated; estimates of glacier contributions to streamflow or to groundwater recharge were not available at the time of this analysis.

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. Based on available data, there is no apparent relationship between hydraulic conductivity and the major structure in the immediate vicinity of the Project area (the Brucejack Fault). However, the structure referred to as the Bruce Fault, a westward trending feature occupying Brucejack Creek at the outlet of Brucejack Lake, appears to act as a control on groundwater flow in that area.

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. The numerical model was initially developed in support of the feasibility study submission (BGC 2013), and was subsequently refined in support of the Environmental Assessment (EA) submission. The model was 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, 32 hydraulic head targets, streamflow data and winter low-flow estimates for the period 2008 to 2012, and volumetric discharge data available from mine dewatering activities that occurred from late-2011 to early-2013.

An iterative approach was adopted to adjust parameter values and compare results for the average annual, or steady state simulations, and transient simulations for both seasonal and dewatering conditions, until a suitable calibration was achieved. The groundwater model was considered calibrated when the best match to steady-state hydraulic head targets in standpipe piezometers and groundwater monitoring wells, and low-flow stream flows were achieved, while maintaining a good match to seasonal variations for the head targets in the transient seasonal simulations and drawdown due to adit dewatering.

Prior to predictive simulation runs, an additional run was completed to represent ongoing dewatering at the site, and benchmark the model with observed dewatering data. Model benchmarking suggests that the model may slightly over-predict groundwater inflow to the underground workings, discussed further in Section 16.6.4.

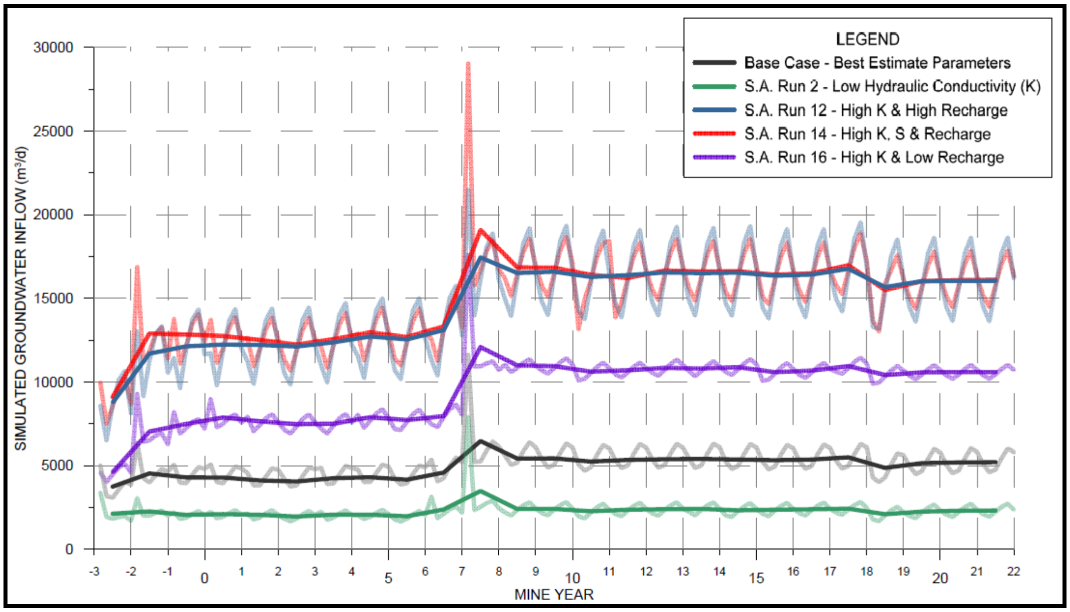
16.6.4 PREDICTIVE SIMULATIONS AND INFLOW ESTIMATES

Predictive simulations are based on the 22-year underground mine plan received from AMC on July 3, 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 annual schedule in the mine 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 years 1 to 7 of mine life, ranging between 4,100 m^3/d and 4,600 m^3/d. The rate of inflow to the underground workings is predicted to increase to an annual average peak of approximately 6,500 m^3/d in year 8, with the initiation of development of the WZ resource. During years 9 to 18 of mine life, predicted annual average inflows range between 5,200 and 5,500 m^3/d, before decreasing slightly and ranging between 4,900 and 5,200 m^3/d for the final four years of mine life. The overall average flow for the entire simulated mining period is 4,900 m^3/d.

A series of 16 sensitivity scenarios were completed using the transient predictive mining operations model, to evaluate changes to predicted groundwater elevations and flow rates for a range of input parameters. Estimated mine inflow to the underground workings is illustrated in Figure 16.16, for the base case modeling scenario in addition to five sensitivity scenarios.

Figure 16.16 Estimated Inflow to Underground Workings for Base Case Predictive Simulation and Selected Sensitivity Scenarios



Note: Faded lines represent model stress period flows (3 x 2-month stress periods per year), while solid-colour lines represent average annual predicted flows.

Large changes in mine inflows were observed for sensitivity scenarios where K of the bedrock fabric was increased (S.A. Run 1) or decreased (S.A. Run 2) relative to the base case. S.A. Run 1, with hydraulic conductivity increased by a factor of 5, resulted in increased flows by a factor of approximately 2.4, on average. S.A. Run 2, with hydraulic conductivity decreased by a factor of 5, resulted in corresponding decreases in average inflow and peak annual inflow by a factor of 0.5.

The high K sensitivity simulations (S.A. Runs 1, 12, 14, and 16) yielded the highest inflow estimates, and are described in more detail below. Predicted inflows to the underground mine averaged approximately 11,700 m^3/d, 14,600 m^3/d, and 14,700 m^3/d, respectively, for S.A. Runs 1, 12, and 14. As with the base case modeling scenario, peak inflows for all sensitivity scenarios are predicted to occur in year 8 of mining operations, with the development of the WZ resource. The annual average peak flows associated with the high K sensitivity scenarios are 14,400 m^3/d (S.A. Run 1; factor of 2.2), 17,400 m^3/d (S.A. Run 12; factor of 2.7), and 19,100 m^3/d (S.A. Run 14; factor of 2.9).

- *S.A. Run 12 (increased K and recharge)* – Increasing K alone (S.A. Run 1) resulted in a factor of 2.4 increase in groundwater flows to the underground workings, relative to the base case, while increasing recharge alone (S.A. Run 4) resulted in a factor of 1.4 increase in flows. Increasing both hydraulic conductivity and recharge (S.A. Run 12) resulted in a factor of 3.0 increase to average mine inflows, and a factor of 2.7 increase to peak annual inflows.

- *S.A. Run 14 (increased K, recharge, and storage)* – As with S.A. Run 12, increasing the hydraulic conductivity, recharge, and storage properties resulted in an increase to average mine inflows by a factor of 3.0. A greater increase in peak annual inflows was observed relative to S.A. Run 12 (factor of 2.9 vs. factor of 2.7) due to the increased storage in S.A. Run 14.

- *S.A. Run 16 (increased K and decreased recharge)* – Increasing K alone (S.A. Run 1) resulted in a factor of 2.4 increase in groundwater flows to the underground workings, relative to the base case. Simultaneously decreasing recharge (S.A. Run 16) tempers this response, resulting in a factor of 1.9 increase in both average annual and peak annual inflows, (9,400 m^3/d and 12,100 m^3/d, respectively).

It is worth noting that while these runs (and S.A. Run 1) are considered conservative from a feasibility perspective (i.e., they result in the highest rates of groundwater inflow to the underground workings), none are supported by the calibration results.

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 S.A. Runs 12 and 14 (Figure 16.16), while the base case modeling scenario (Figure 16.16) provides the best estimate of inflows using the numerical model calibrated to the existing dataset.

16.7 MOBILE EQUIPMENT REQUIREMENTS

16.7.1 PRE-PRODUCTION PHASE

During the preproduction phase, 12,728 m of lateral development must be completed to meet the ramp-up production schedule. Development during this period will peak at 627 m/mo, but with the average over the pre-production timeframe at about 530 m/mo. The Contractor will 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.

During the development phase, equipment will be provided to the Contractor by the Owner from the Owner’s permanent fleet. In the case of insufficient equipment being available, the Contractor will supply additional equipment required for the development phase. Table 16.12 shows the breakdown of Pretium and Contractor-supplied equipment for the pre-production development phase. The ancillary equipment during this period will be supplied by Pretium.

Table 16.12 Contractor and Pretium Equipment during Pre-production Development

Description	Contractor	Pretium	Total
Two boom mining jumbo	1	3	4
LHD, 14 t, Diesal (Development)	1	3	4
LHD, 14 t, Electric (Production)	0	0	0
Haulage truck, 40 t	0	4	4
Bolter	0	2	2
Cable bolter	1	0	1
Shotcrete sprayer	0	2	2
ITH long hole drill	0	1	1
TH long hole drill	0	0	0
Transmixer	0	2	2
Scissor Lift	0	2	2
Explosives loader, diesel, emulsion (face charger)	0	2	2

Note: ITH = in-the-hole, TH = top hammer

JUMBOS

Each jumbo is scheduled to achieve 150 m/mo of development. Four jumbos will be required to meet the peak scheduled advance.

LHDS

Four diesel LHDs will be required to move the blasted material from the various development headings and from the mass excavations during the pre-production period. The electric LHDs planned for the production phase will not have the flexibility of movement required during the pre-production phase. The Contractor will provide one diesel unit for this period, and Pretium will provide three. The three diesel LHDs supplied by Pretium will be required for ongoing development headings during the production phase.

TRUCKS

Four 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.

The Contractor will supply the cable bolting drilling capacity

SHOTCRETE SPRAYERS

It is anticipated that 5 to 10% of all development over the LOM will require shotcrete. The 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 and to ensure no shotcreting delays.

Long Hole Drills

A long hole drill will be required to pre-drill in the VOK Upper and Middle blocks as well as excavate slot raises. A single cubex long hole drill equipped with a V30 borehead will be sufficient to handle this.

Explosive Loaders

Two face chargers will be required for development loading during the pre-production period.

16.7.2 Production Phase

Pretium will supply all equipment during production with the exception of Alimak raise climbers, which will be included in the raising contracts. Table 16.13 lists the required equipment for development, stoping, and support activities.

Table 16.13 Underground Development and Production Equipment List

Description	Availability (%)	Utilization (%)		Quantity
		Peak	Average	
Two boom mining jumbo	86	64	53	3
LHD, 14 t (diesel) (development)	80	76	59	3
LHD, 14 t (electric) (production)	80	85	73	5
Haulage truck, 40 t	85	90	72	6
Bolter	76	82	58	3
Cable bolter	71	90	68	1
Top hammer long hole drill	66	66	58	3
ITH long hole drill	66	88	59	2
Explosives loader, diesel, emulsion (face charger, development)	93	73	54	2
Explosives loader, diesel, emulsion (production)	93	20	15	2
Shotcrete sprayer	87	53	27	2
Transmixer	87	57	30	2

Jumbos

Development advance (in ore and waste) will average approximately 450 m/mo during the first 12 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 seven, 14 t loaders will be required for the scheduled stope and development volumes. An additional unit will also be dedicated to feeding the crusher. A total of eight 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 three 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 the desired 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 costs, 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. Bolters should be equipped for the installation of either bolt type. A maximum of three units will be required by Year 5 of production, when increased bolter capacity is required for the scheduled stope volumes in the VOK fault zone.

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 cable bolter will be sufficient.

Long Hole Drills

Production drilling will be done with appropriately sized top hammer drills. Slot raises will be excavated using a small diameter boring machine.

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. However, a second unit is planned in the interest of ensuring capability for this critical task.

Shotcrete Sprayers

It is anticipated that 5 to 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 16.14 presents the complete list of support equipment.

Table 16.14 Support Equipment List

Item	Description	Availability (%)	Utilization (%)	Quantity
1	Personnel carrier, diesel, underground	93	22	3
2	Scissor lift truck, diesel	87	39	2
3	Lubrication service truck, diesel	93	36	1
4	Boom truck, diesel	93	18	1
5	Explosives truck, diesel, emulsion (transport)	93	15	1
6	Tractor	93	22	11
7	Utility vehicle	93	22	19
8	Telehandler, diesel	87	34	1
9	Wheel loader with tire handler	93	11	1
10	Motor grader (tracks and wheels)	88	16	1
11	1500 cfm, 300hp electric, portable compressor	93	59	2

Personnel Carriers

Though many personnel will be transported in tractors, carriers will be needed for most personnel transport. The transporter selected is capable of carrying 22 people. With three transporters, 66 people can be brought to work each shift. 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, ventilation duct, and an average of 12 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 three hours, enabling up to four headings to be serviced 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 for daily transport of materials from surface to underground 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. One unit will provide adequate capacity.

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 six 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 7 t tank to the empty 24,000 L ISO tank in the emulsion bay. Consumption will average three 7 t tanks per week. All other explosives will be transported to the cap and powder magazines by the explosives handling truck. Approximately 250 caps and 250 primers will be required daily.

Tractors and Utility Vehicles

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.

Utility vehicles will be used by personnel for quick transport between headings, and will be the preferred mode of transport for supervision and technical support staff.

The following crews will be issued tractors and/or utility vehicles for use during their shifts:

- development blasters

- backfill crew
- mechanics
- electricians
- production blasters
- diamond drillers
- warehouse
- managers/shifters & technical support staff

16.8 VENTILATION

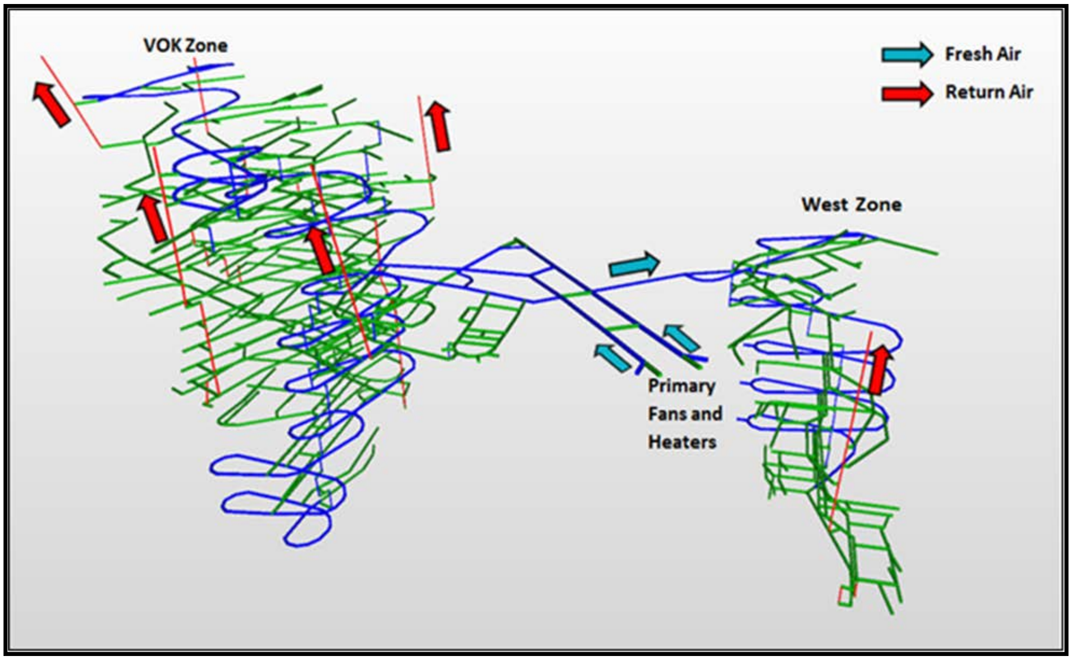
The ventilation system was designed to meet the requirement of the Health, Safety and Reclamation Code for Mines in British Columbia – 2008 (The Code), which requires a minimum of 0.06 m^3/s of ventilating air for each kilowatt of power of diesel powered equipment operating. The design is based on a "push" configuration, with permanent surface fans located at the portal of the twin declines.

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 16.17 shows an isometric view of the Brucejack ventilation system.

Figure 16.17 Brucejack Ventilation System – Looking West



16.8.1 Design Criteria

This study has drawn information from The Code as basis for the design of the ventilation system. As stated in Part 4, Section 4.6.1 "minimum of 0.06 m^3/s of ventilating air for each kilowatt of power of the diesel powered equipment operating shall be circulated by mechanical means through every workplace where diesel-powered equipment is operating".

A diesel engine exhaust emissions (DEE) dilution rate of 0.06$m^3/s/kW$ has therefore been used for this study.

16.8.2 Total Airflow Requirements

Airflow requirements were determined based on the 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 mine:

- three development headings advancing
- three 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 16.15.

Table 16.15 Total Airflow Requirements

Areas		Flow (m^3/s)	Number	Total (m^3/s)
Development		38	3	114
Production	Stope Mucking and Loading	23	3	68
	Drilling/Charging/Services	10	2	20
Infrastructure	Crusher Chamber/Truck Loop	45	1	45
	Workshop/Magazine/Fuel Bay	25	1	25
	Lower Conveyor Leg	12	1	12
Leakage and Balancing	20%	-	-	57
Total (rounded)				**340**

16.8.3 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 supply 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.

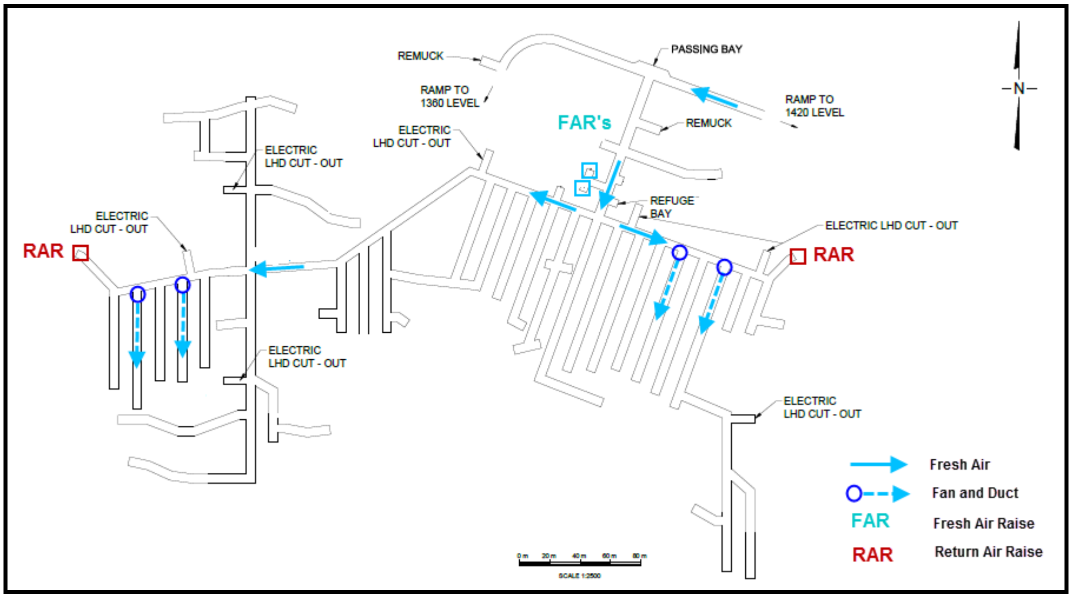
Modelling indicates that two ducts will be required. Each duct will have two 55 kW fans bolted together in series. The duct size is 1,200 mm in diameter. This will supply 38 m^3/s up to a distance of 850 m. This arrangement will allow for adequate overhead clearance for a fully-loaded 40 t truck.

Drawpoint Ventilation

An allowance of 10 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 400 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 16.18 shows a ventilation configuration for a typical production level.

Figure 16.18 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 fan 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.

16.8.4 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 16.16.

Table 16.16 Primary Fan Specifications

Description	Specification
Duty	Two each @ 170 m^3/s @ 1,200 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	266 kW to 710 rpm, variable frequency drive capability

16.8.5 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.
- Discussion on the use of electric and propane mine air heating can be found in Section 16.9.13.

16.8.6 Conveyor Decline

The conveyor decline will be a main mine intake with planned dimensions of 6.0 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 6.0 m/s, the air velocity in the conveyor decline should not exceed approximately 5.0 m/s.

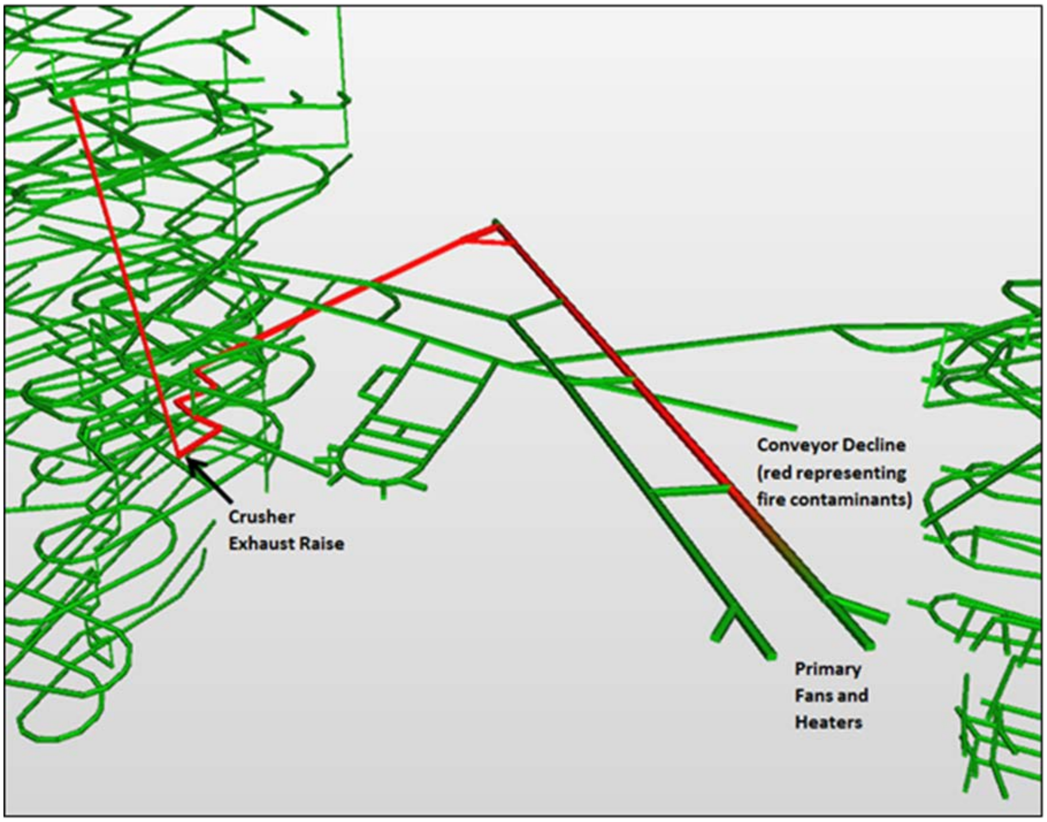
Given that the conveyor will be located in a primary air intake, the risk of the conveyor catching fire must be managed. The design includes allowance for the following:

- 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. Figure 16.19 shows the isolation of conveyor fire contaminants from the ventilation circuit.

Figure 16.19 Conveyor Fire Isolation



16.8.7 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.
- The escape ladderway will be located in the fresh air raise installed as part of the development of the ramps.
- 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 within the shop complex and will service both the West Zone and the VOK Zone.
- 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.

The emergency response procedures will incorporate trained on site mine rescue teams made up of a cross section of the workforce and staff. These teams will be trained in administration of first aid and firefighting procedures. As this site is considered remote, a first aid facility run by a trained person, sufficient supplies and provisions for air ambulance and landing pad have been taken into consideration.

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 within the shop complex and will service both the West Zone and the VOK Zones. It is required to provide refuge for 40 persons during an emergency. This refuge station will be independent of compressed air and 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 88; a typical underground workforce will be 65. It is estimated that 16 people will be working predominantly in the workshop/crusher 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.

As the mine does not reticulate compressed air from the surface, the refuges including the permanent refuge station do not have an external back-up supply of air. Whilst the planned MineARC system provides a supply of air as well as robust back-up systems, an opportunity exists to introduce a further air supply back-up system through placement of a compressor on the surface and reticulate compressed air to the permanent refuge station. Therefore, there is an opportunity to investigate the potential costs/benefits of installing a surface compressor and reticulating compressed air down the decline direct to the permanent refuge station. Its purpose would be strictly for a back-up to the existing air supply systems in the refuge station.

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
- crusher tipple.

Portal doors will also be designed to meet fire door criteria.

16.9 Underground Infrastructure

16.9.1 Mine Dewatering

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 (including service water); 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 for the Brucejack Project is handled by a combination of submersible and horizontal centrifugal pumps located throughout the West Zone (WZ) and Valley of Kings (VOK) working levels. The pumps will handle ground inflow and spent drill water via multiple 90m lifts throughout the mine.

Dewatering during mine development will begin with the establishment of the main sump at the VOK 1,290 Level. As VOK development continues upwards from the 1,290 Level, water will be drained down to the main sump through the use of boreholes. Development progressing downward will require the establishment of temporary cylindrical sumps at each level, where submersible pumps (such as an Eliminator submersible pump) will cascade waste water up to the main sump. When the locations for permanent intermediate lift stations (VOK levels 1,170, 1,080, and 990) are reached, the stations will use high head centrifugal pumps and will lift dirty water up to the main sump, with boreholes being used to drain water down to the lift stations. The cylindrical sumps can then be decommissioned and the submersible pumps can be used to develop levels below the permanent lift station until the next lift station is established. This process will continue until the VOK is completely developed. The same development process will be used for the WZ dewatering, with lift stations being established at 1,290, 1,210, 1,120, and 1040 Levels.

The main sump will consist of three settling columns and one clear sump. The flow of water reporting to the main sump will be divided equally between the three settling columns. The settling columns will consist of 2.4 m diameter boreholes between the 1,290 and 1,260 Levels with cone bottoms. The cone bottoms will integrate flush water ports and manholes to allow for cleaning of caked solids. In the settling columns solids and slimes will settle to the bottom where they will drain to the suction line for two positive displacement pumps (one running, one backup) that will pump slimes to the process plant on surface via a dedicated line.

Clear water will overflow from the settling columns into the clear sump. The clear sump will report water to the water treatment plant on surface via four centrifugal pumps (two running, two backup).

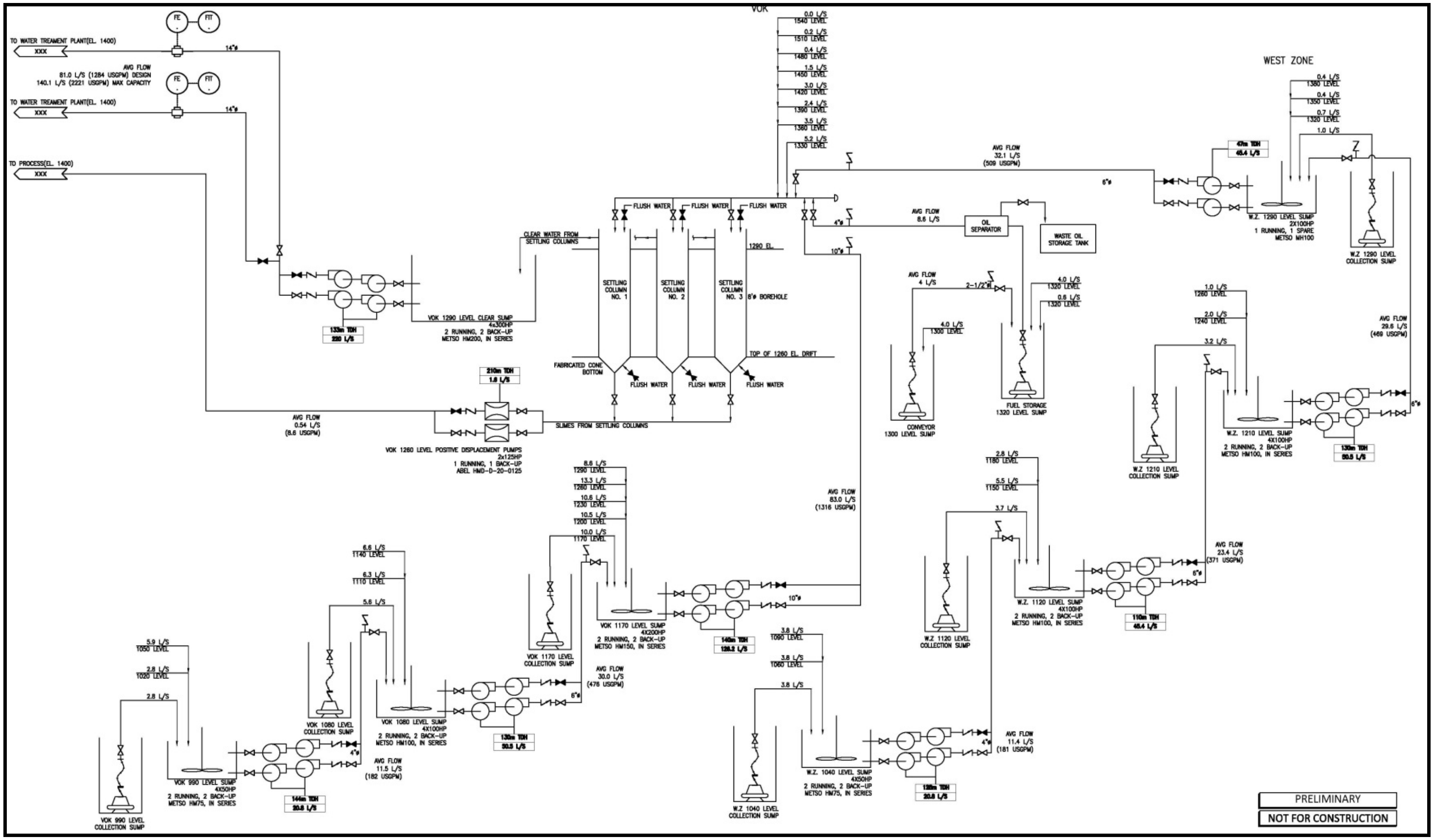
Intermediate lift stations for the VOK and WZ will consist of a collection sump with a submersible pump and an intermediate sump with two centrifugal pumps (one running and one backup). Screens will be installed on pump suctions to limit the maximum particle size to 7.6 mm. A collection sump will collect the water from a level and pump it to the intermediate sump, along with water from any above levels drained through boreholes or received from lower intermediate sumps.

To minimize up-front capital, the pump procurement will be staged such that pumps only arrive as their assigned sumps are excavated. Table 16.17 shows the pump installation schedule. Figure 16.20 is a line diagram of the dewatering system.

Table 16.17 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
Service Water Pump																				
VOK Intermediate Sump 1,170																				
VOK Intermediate Sump 1,080																				
VOK Intermediate Sump 990																				
West Zone Level Sump 1,290																				
West Zone Level Sump 1,210																				
West Zone Level Sump 1,120																				
West Zone Level Sump 1,040																				
Main Sump 1,285																				
Shop Sump 1,320																				
Fuel Storage Pump																				

Figure 16.20 Dewatering Plan

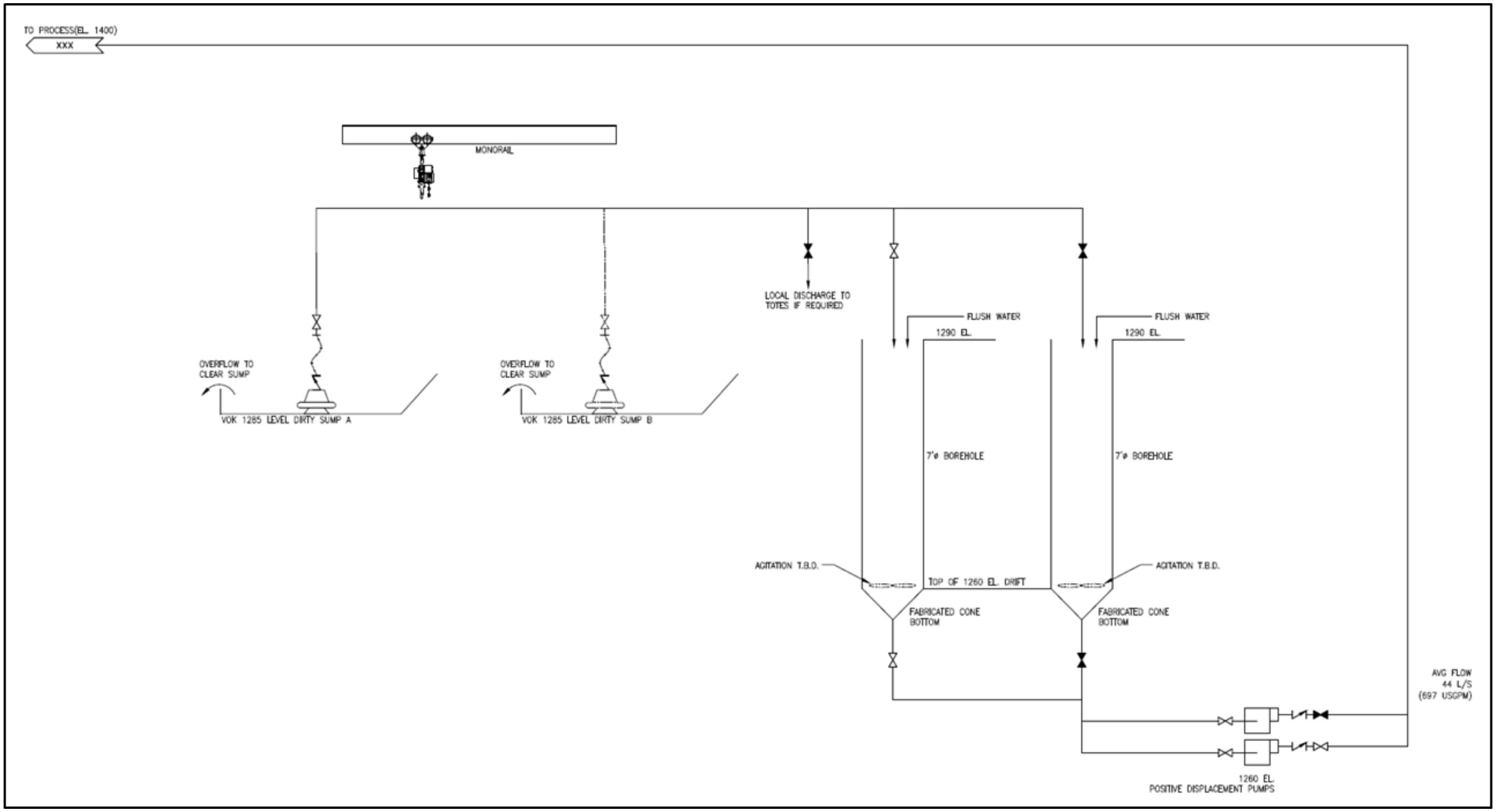


16.9.2 Solids and Slimes Handling

Solids and slimes entrained in water pumped through the dewatering system will be allowed to settle at the main sump located at the 1,285 elevation. This main sump consists of three dirty water settling columns and one clear water sump. All dewatering water first enters one of the dirty columns where clear water is allowed to overflow with solids and slimes left behind. Once the level of solids in a dirty column reaches the maximum allowable level, dewatering water is diverted to the alternate dirty column. The bottom of the columns have agitators and a fabricated cone with a flanged outlet that leads to two positive displacement pumps (one running and one backup). When the level in the column reaches a maximum these pumps lift the slimes and solids out of the underground workings and to the process stream to recover any contained gold.

Figure 16.21 shows the underground solids and slimes handling plan.

Figure 16.21 Underground Solids and Slimes Handling

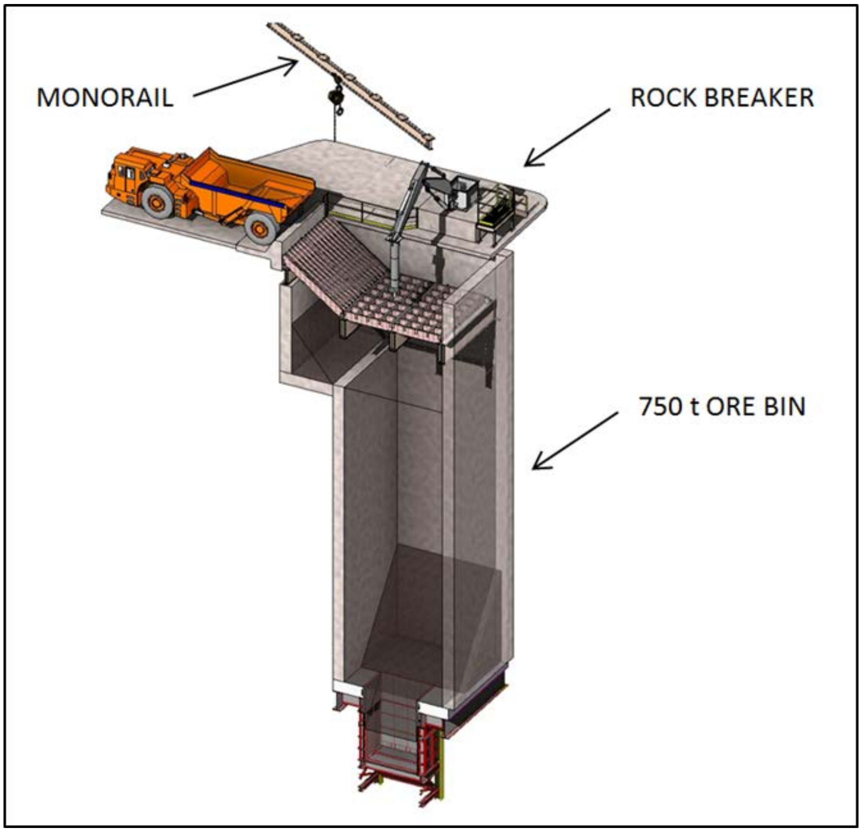


16.9.3 MATERIALS HANDLING

As part of the FS update, a trade-off study was completed to review options for ore handling underground. The study concluded that the crusher should be moved closer to the VOK deposit. Other ore handling systems remain unchanged from the previous FS. The new crusher location is at the 1,298 m elevation directly off the VOK ramp, with the tipple located at the 1,330 m elevation.

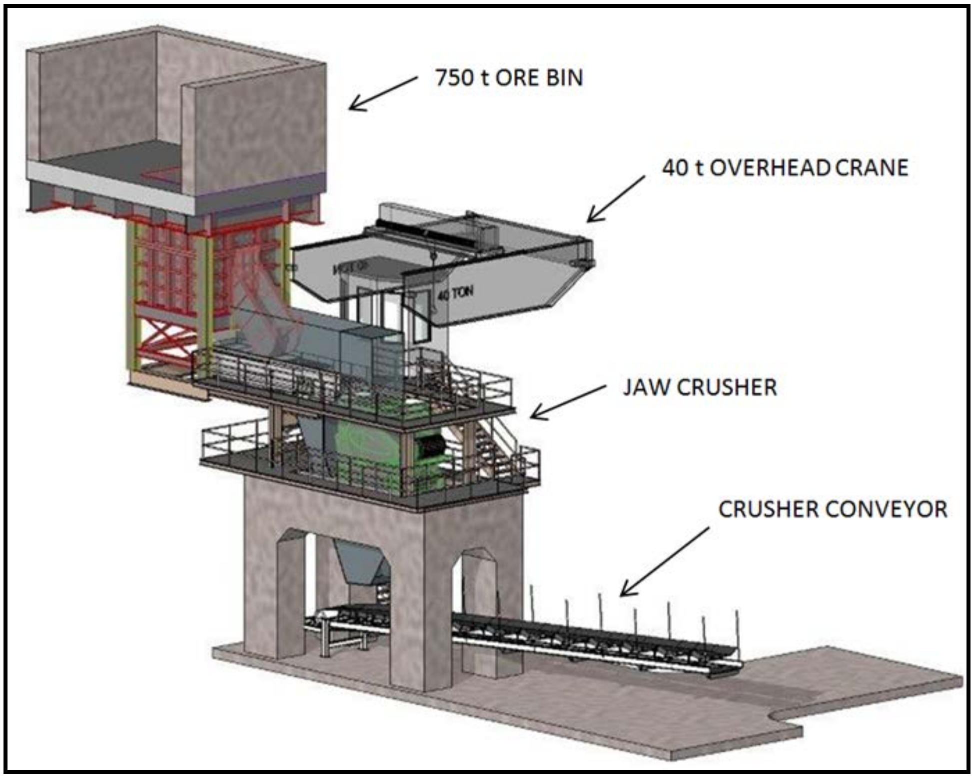
ROM material will be transported underground by truck from the West Zone and VOK Zone and deposited into the ore storage bays or directly onto the scalping grizzly. Material stockpiled in the storage bays will be re-handled and deposited onto the scalping grizzly by an electric LHD. At the scalping grizzly, 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 16.22 shows a sectional projection through the Coarse Ore Bin with the rock breaker and scalping grizzly.

Figure 16.22 Tipple and Ore Bin Sectional Projection



The 750 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 allows fines (less than 64 mm) to fall through and down the fines chute to the crusher conveyor. The jaw crusher will reduce the larger material down to 65-75 mm in size and drop this product down the fines chute to the crusher belt conveyor. Figure 16.23 shows an isometric view of the crusher feed and crusher.

Figure 16.23 Crusher Feed and Crusher



The 1 m wide belting on the crusher conveyor will carry material at a rate of approximately 225 t/h from the crushing area to the intermediate conveyor at a speed of 1 m/s. The ore on the crusher conveyor will move past a magnet that will remove any tramp iron, depositing this iron into a waiting bin.

The intermediate conveyor will also move at a rate of 1 m/s, transporting the ore up the intermediate decline tunnel to the main conveyor. The main conveyor will exit the decline tunnel into the portal structure where the 335 horsepower drive unit is located. The ore will be dropped onto the mill feed conveyor, which will exit the portal structure and carry the ore to the mill through an enclosed, heated, rectangular gallery. The gallery structure will be elevated, continuing at a grade of 12° from the portal structure to the mill, allowing for traffic underneath.

The main conveyor belt speed of 1 m/s, along with the expected air speed of 5 m/s down the decline will provide a combined speed of 6 m/s. This speed was assessed as reasonable in terms of keeping dust lifted from the belt to a minimum. Table 16.18 summarizes the conveyor parameters.

Table 16.18 Conveyor Parameters

Conveyor	Width	Length	Speed	Incline Angle
Crusher	1 m (42")	38.2 m (125'4")	1 m/s (197 fpm)	4°
Intermediate	1 m (42")	244.9 m (803'4")	1 m/s (197 fpm)	8.53°
Main	1 m (42")	553.2 m (1815'6")	1 m/s (197 fpm)	8.53°
Mill Feed	1 m (42")	83.5 m (273'11")	1 m/s (197 fpm)	12°

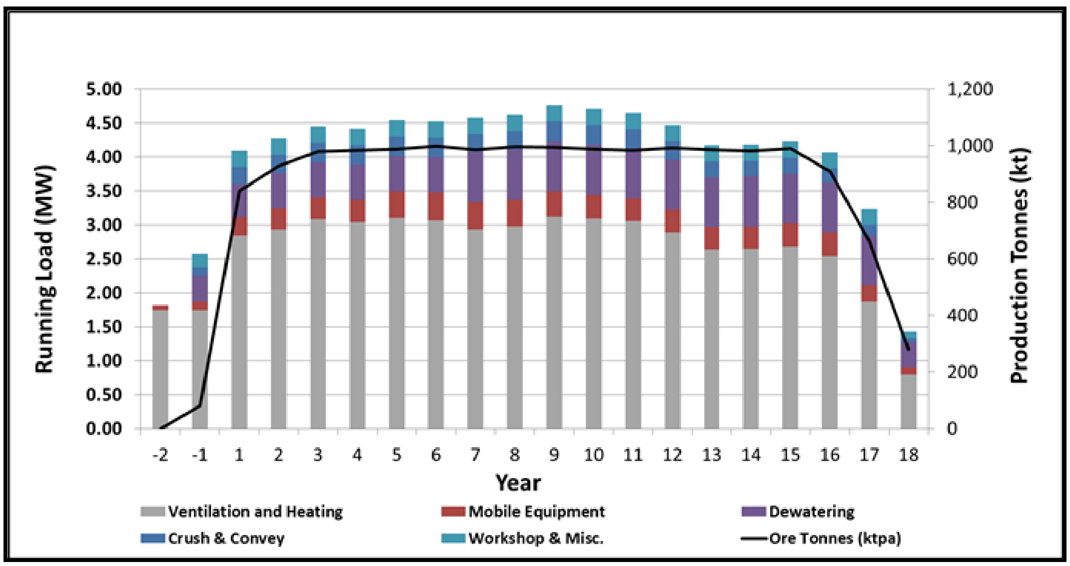
16.9.4 POWER REQUIREMENTS AND ELECTRICAL DISTRIBUTION

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 16.24 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 4.8 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 16.24 Underground Power Requirement Profile



Electrical power will be supplied to the portal building by four separate 4160 Volt feeder circuits from the site main substation. Two feeders (2 x 3 conductor 500 MCM each) will supply power to the 4160 volt distribution equipment located at the portal building and the other two feeders (2 x 3 conductor 350 MCM each) will continue on through the portals to the underground access declines, one feeder within the main access decline and one within the conveyor decline.

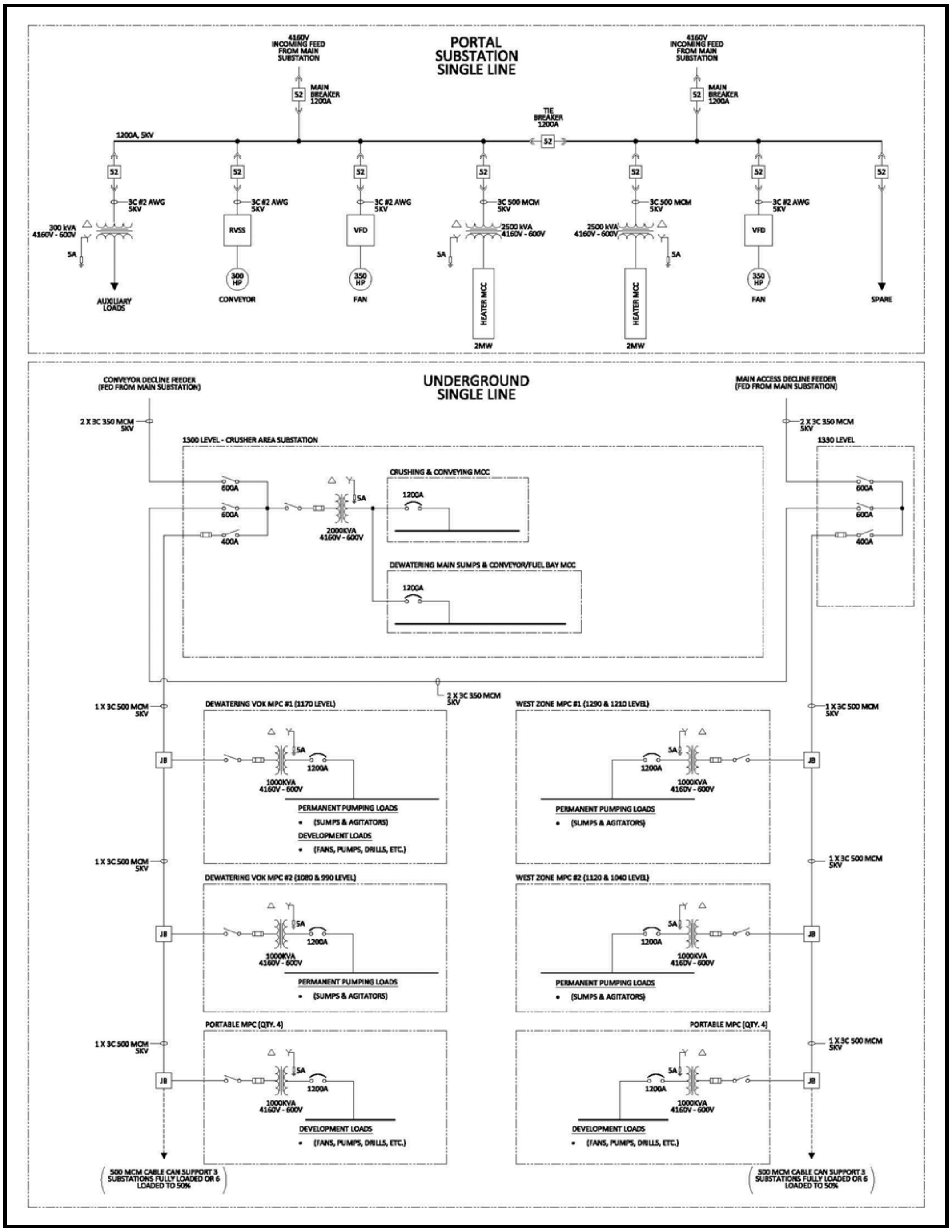
The portal building 4160 Volt switchgear will incorporate two main incoming circuit breakers, a tie circuit breaker and suitable feeder circuit breakers to supply power to the 2500 kVA electric heater unit supply transformers (2), 350 HP fan drives (2), 300 HP conveyor drive, and other auxiliary equipment via a 300 kVA delta-wye step down transformer c/w a 5 Amp continuous rated neutral grounding resistor. A 600 Volt, 600 Amp motor control centre will be supplied from the step down transformer to distribute power to various small motors, lighting and utility loads located in the portal building.

Each 4160 Volt decline feeder will terminate at fused disconnect assemblies located at the 1,298 (crusher) level and 1,330 (truck dump) level respectively. A tie circuit will connect the two underground fused switch assemblies to allow for a redundant power feed system from either underground feeder. A step down transformer located at the 1298 level substation will provide 600 Volt supply to various electrical loads including the crusher, conveyors, main sump, lighting, etc.

Additional 4160 Volt feeders (1 X 3 conductor 500 MCM) supplied from fused disconnects at the 1,290 and 1,330 level to both the Valley of the Kings and the West Zone working levels will provide power to portable substations to be used for development, pumping, ventilation, lighting, etc.

Figure 16.25 shows single line electrical diagrams for the underground mine.

Figure 16.25 Portal Substation and Underground Single Line Diagrams



16.9.5 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.

16.9.6 Service Water Supply

Service water for drilling and dust control will be supplied via a 100 mm (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 689 kPa (100 psig).

Water requirements for Drills, Bolters and other equipment will increase from an initial 228 m^3/day (42 USGPM) during mine development, up to an anticipated 471 m^3/day (86 USGPM) in Year 6 of the mine life.

To supply service water to the higher-working levels in the VOK area, 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 16.26 is a schematic of the main water distribution system. Table 16.19 shows estimated water consumption by year.

Figure 16.26 Mine Water Distribution Schematic

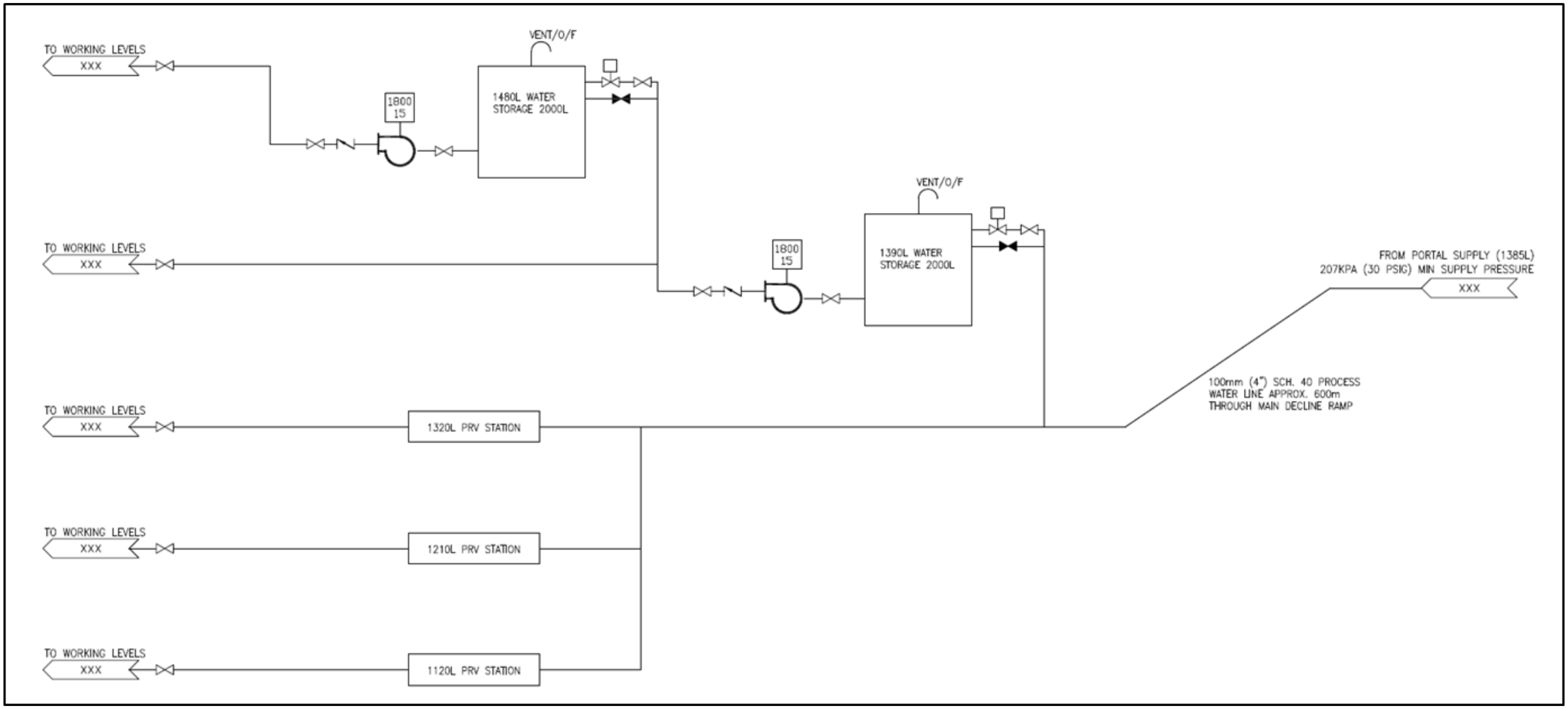


Table 16.19 Total Water Consumption

Equipment	Flow time	50%	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**Total**
	Flow (l/s)																						
Drill Jumbo, 2-Boom Sandvik DD421	2.50	m3	38,533	51,618	39,303	38,942	37,895	37,577	37,036	35,921	36,257	33,802	37,172	31,857	36,833	34,381	14,512	9,925	10,587	12,021	6,948	976	**582,097**
Drill Longhole, Sandvik DL421-15C, T	2.60	m3	-	-	16,311	17,450	19,534	20,298	27,833	31,196	31,363	30,748	26,430	23,481	21,702	23,621	27,954	30,311	30,497	30,737	20,386	8,880	**438,732**
Drill Longhole, Cubex Aries, DU411,	2.60	m3	-	-	6,554	7,012	7,850	9,540	20,557	26,005	28,150	25,975	18,041	12,160	11,217	15,857	20,626	23,005	22,894	26,164	15,316	6,549	**303,469**
Bolter, Sandvik DS411	1.30	m3	10,821	15,024	14,288	15,092	14,511	14,659	16,658	17,141	18,071	16,764	15,833	13,076	15,061	15,902	10,416	9,297	9,526	10,796	6,423	2,122	**261,481**
Cable Bolter, Sandvik DS421	1.30	m3	870	2,059	4,538	5,360	5,067	5,511	7,431	8,425	9,430	8,609	6,637	4,873	5,242	6,991	6,723	5,989	6,111	7,176	4,277	1,791	**113,111**
Water truck		m3	3,403	5,753	5,273	5,127	5,446	5,616	5,855	6,050	6,025	6,147	6,951	7,329	8,022	7,638	6,323	6,147	6,204	6,201	5,909	2,752	**118,173**
Water for Development Mucking	3%	m3	9,115	10,305	9,682	8,214	9,451	8,473	9,470	8,578	6,404	6,896	9,633	9,440	8,774	5,363	968	1,214	1,558	1,367	944	100	**125,947**
Water for Production Mucking	3%	m3	-	2,432	25,185	27,859	29,379	29,508	29,648	29,965	29,586	29,872	29,812	29,617	29,544	29,795	29,590	29,417	29,722	27,234	19,901	8,426	**496,490**
Total Required Water per year		m3	62,743	87,192	121,135	125,056	129,132	131,183	154,489	163,282	165,286	158,812	150,508	131,834	136,394	139,547	117,112	115,305	117,098	121,696	80,103	31,595	**2,439,501**
Water Recycled from drilling	0%	m3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Total New Water Req'd per year		m3	62,743	87,192	121,135	125,056	129,132	131,183	154,489	163,282	165,286	158,812	150,508	131,834	136,394	139,547	117,112	115,305	117,098	121,696	80,103	31,595	**2,439,501**
Average Daily Supply		**m3**	**174**	**242**	**336**	**347**	**359**	**364**	**429**	**454**	**459**	**441**	**418**	**366**	**379**	**388**	**325**	**320**	**325**	**338**	**223**	**88**	**339**

16.9.7 FUELING AND LUBRICATION

Daily fuel consumption is estimated to be about 5,500 L. A fuel bay area will be located on the 1,320 infrastructure level between two automatic fire doors, as shown in Figure 16.27. 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.

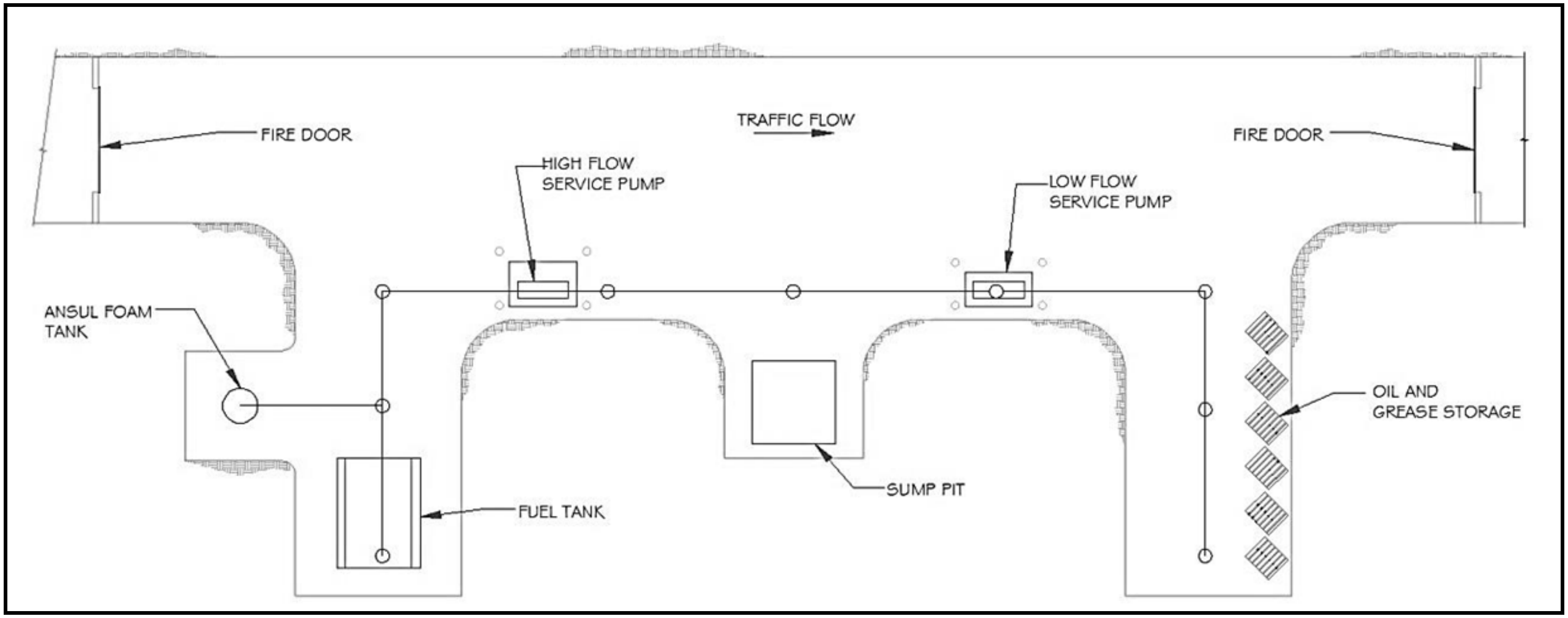
The fuel bay area will include a main drift and three bays. A 20,000 L fuel storage tank and a fuel delivery pumping system will be located in one of the bays located inside the fuel bay area.

The fuel storage tank will be filled from a fuel delivery truck coming from surface. 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 main 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 16.27 Fuel Bay Layout



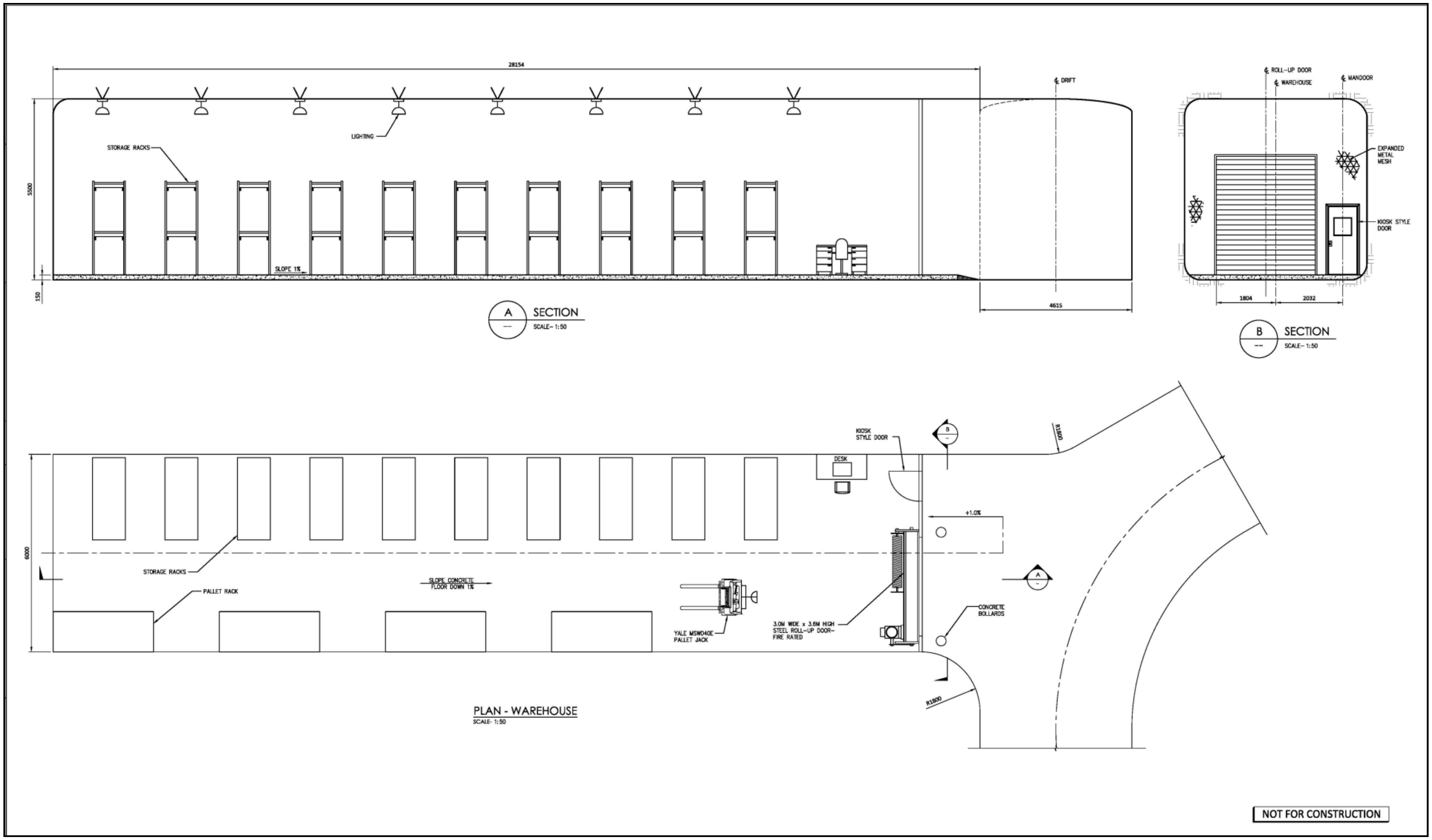
16.9.8 Workshop and Stores

The main maintenance area will be located on surface (covered in Section 18 of this report). All major scheduled PM and rebuilds will take place in the surface shop. Two small service bays will be located underground to complete low level maintenance such as lubrication and small repairs. The service bays will have finished concrete floors, monorail hoists, tire storage, lube storage and the capacity to make hydraulic hoses.

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.

A warehouse and small office will also be located near the underground service bay area. Figure 16.28 shows the warehouse plan and sections.

Figure 16.28 Warehouse Plan and Sections



16.9.9 Explosives Magazines

Two bays will be provided for the storage of bulk emulsions, each containing one 24,000 L storage tank 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 powder bay will be designated for the storage of all other explosive products (other than the bulk emulsion and the detonators) on wooden shelves. A concrete wall with a steel door will separate this bay from the rest of the mine works.

A fourth bay will be designated for the storage of detonators on wooden shelves. A concrete wall with a steel door will separate this bay from the rest of the mine works.

The bays are shown in Figure 16.29 and Figure 16.30.

Figure 16.29 Bulk Emulsion Storage - Plan and Sections

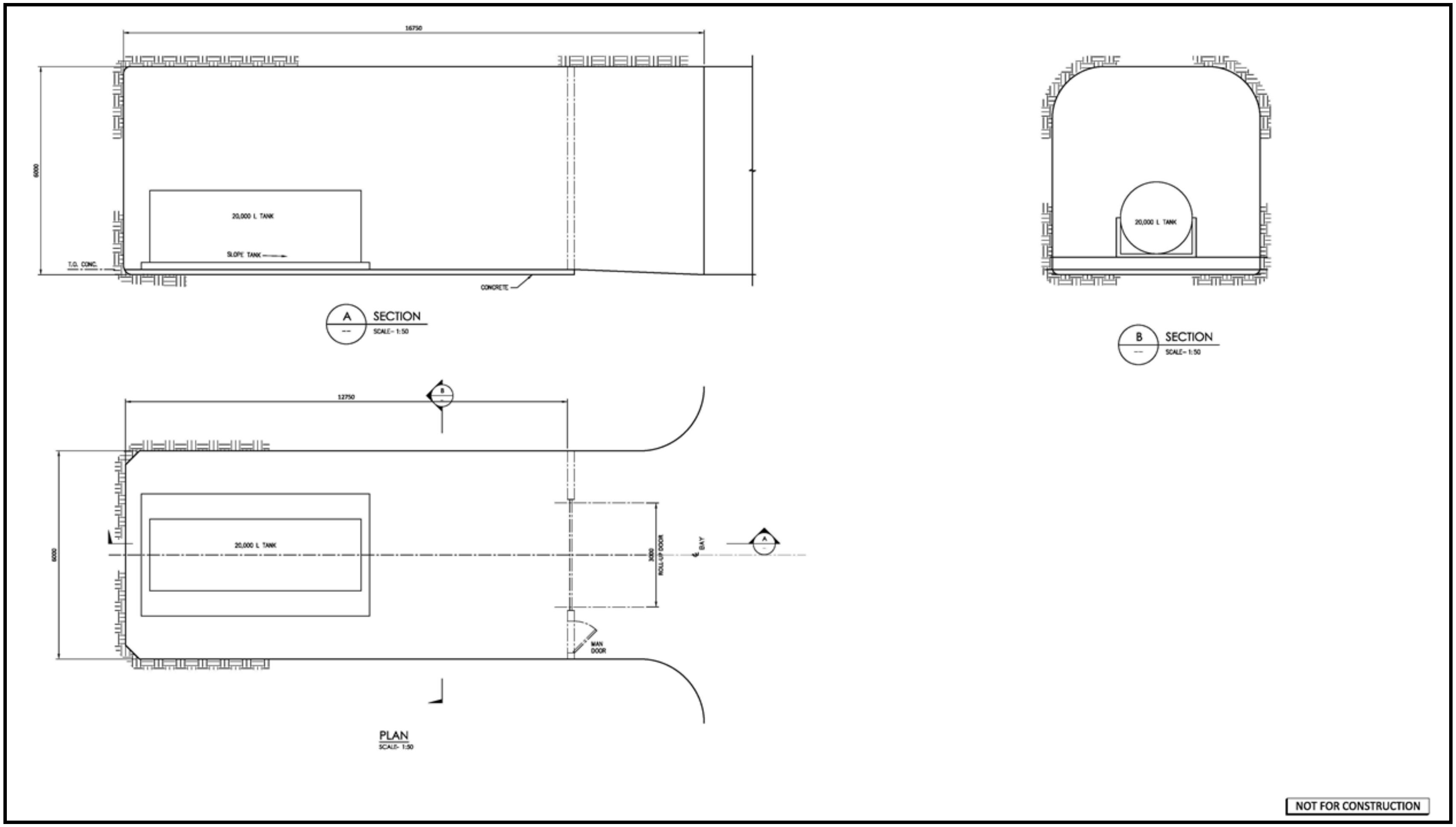
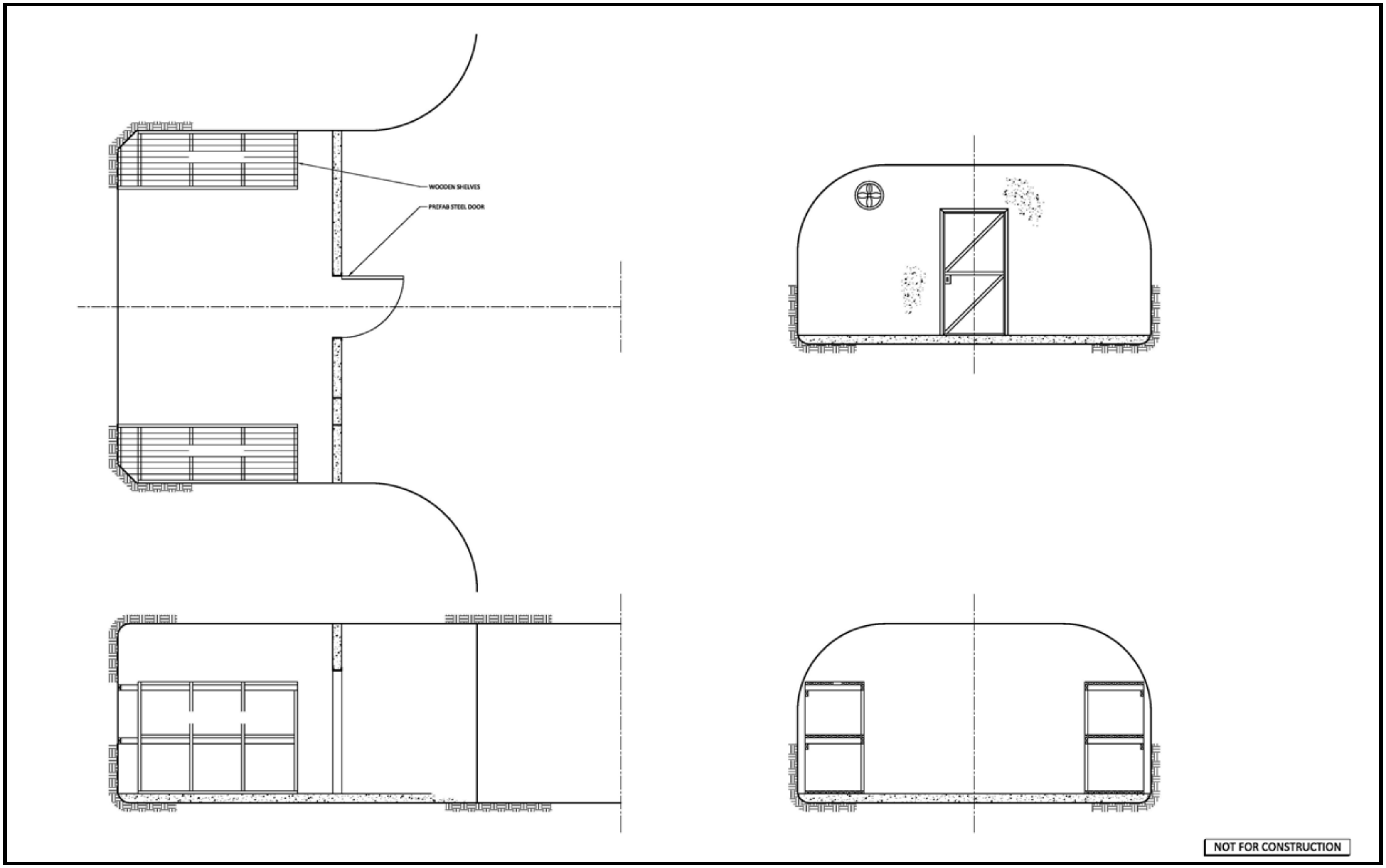


Figure 16.30 Cap and Powder Storage - Plan and Section



Bulk emulsion will be transported by the explosives supplier directly from the manufacturing plant to the Knipple Transfer Station. Each shipment will be delivered via transport trailer in three custom made 7 t (6,000 L) ISO tanks per load. The three 7 t ISO tanks will be transferred immediately upon arrival, at the Knipple Camp, via a fork-lift, onto the explosives transport vehicle for delivery along the Knipple glacier to the Brucejack mill site. Once the three full 7 t ISO tanks have been offloaded from the transport trailer, three other empty 7 t ISO tanks will be loaded onto the trailer to be returned to the manufacturing plant for refill with emulsion.

When the explosives vehicle arrives at the Brucejack mill site, a fork-lift will immediately transfer one of the 7 t ISO tanks onto the underground explosives boom-truck for delivery to the underground emulsion bays. The emulsion is then transferred with a pneumatic emulsion pump from the 7 t ISO tank into the 24,000 L storage tank in the emulsion bay. The boom-truck then returns the empty 7 t ISO tank to the mill site and the fork-lift transfers the empty 7 t ISO tank back onto the surface explosives vehicle. The process is then repeated for the two other full 7 t ISO tanks. The explosives transport vehicle delivers the three empty 7 t ISO tanks to the storage area at the Knipple Camp.

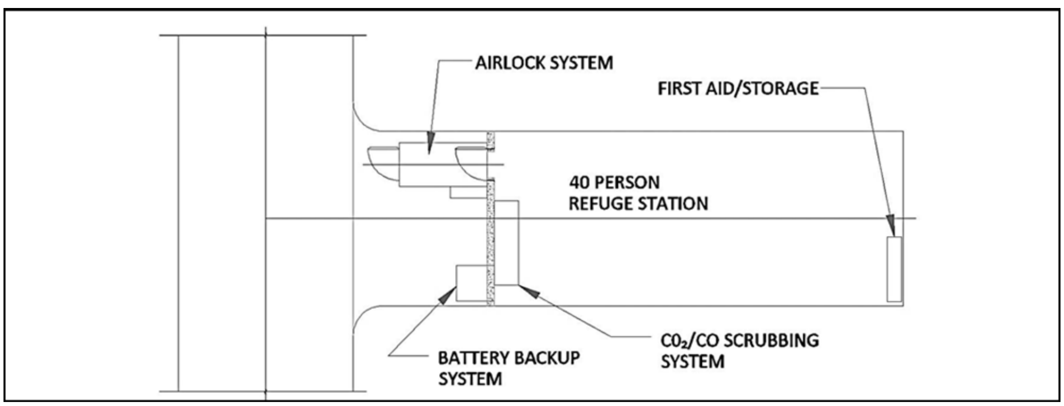
16.9.10 Refuge Stations

A refuge station (see Figure 16.31) will be located between the two decline tunnels at the mine. 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 will also be used as a lunchroom.

Figure 16.31 Permanent Refuge Station



16.9.11 COMMUNICATIONS

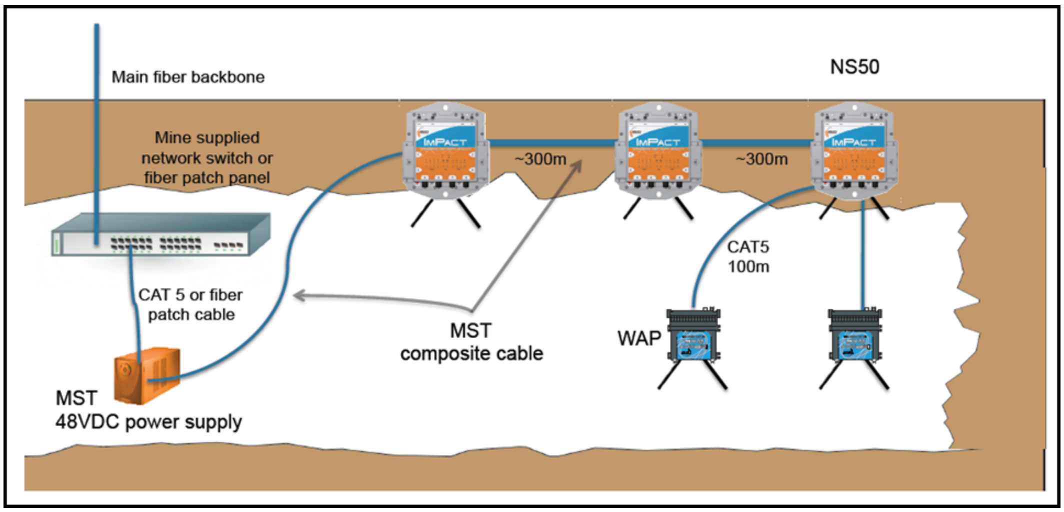
FIBER OPTICS AND PHONE AND RADIO COMMUNICATIONS

The underground wireless network infrastructure will consist of:

- Voice over Internet Protocol (VoIP) MinePhones
- cap lamps
- asset and personnel tracking

Radio communications are to be established underground by virtue of a wireless digital, Local Area Network (LAN) protocol WiFi compatible system. 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 16.32 shows a schematic of the proposed typical underground communications system.

Figure 16.32 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 AND EQUIPMENT TRACKING

Personnel tracking will be accomplished by virtue of a RFID tag system. An integrated communications cap lamp (ICCL) will contain the RFID Tag. The ICCLs will also contain a

UHF radio for an 'all-in-one underground miner solution'. 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. 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

The 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 crushing, conveying, pumping, ventilation, air quality and quantity monitoring and control will be by virtue of the PLC system. The PLC system processor (main rack) will reside in the portal indoor substation. Remote PLC I/O racks will be placed near equipment as necessary.

As indicated, the PLC system will be tied to the mill/control room network. Camera systems can also use this network bridge to provide camera data to the control room. Camera systems can be controlled by virtue of a PC with IP cameras positioned at critical areas underground. This can be replicated on a large monitor or television in the control room as necessary.

16.9.12 Portal Structure

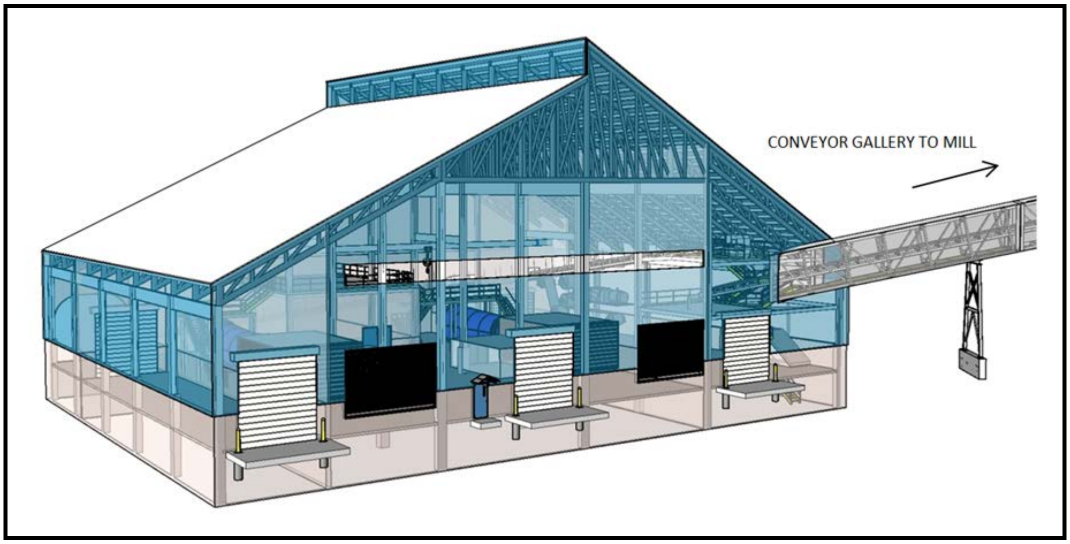
A portal structure will be constructed at the access to the underground decline tunnels. The structure will span the area between both decline tunnels and will house the mine air heaters and ventilation fans, the conveyor drive motor and structure, and an electrical sub-station. The main decline conveyor will exit up from the tunnel and transfer ore to the mill feed conveyor. This transfer will be located inside the portal structure. 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 16.33 shows the portal structure.

Figure 16.33 Portal Structure



16.9.13 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 option for the Project.

For the feasibility study update, the estimated amount of electric power available for mine air heating was consistent with the feasibility study projection. This, in combination with updated costs for propane and electric power further supports the use of a hybrid electrical/propane powered mine air heating system.

Climatic Data

Climactic data from site was analyzed to quantify the amount of annual electric 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 (4.5 years, October 2009 to March 2014) 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.

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 and proximity to glaciers accounts for this differential value. AMC concluded 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.

16.9.14 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 16.20 shows the projected monthly and annual propane requirement during steady state operations.

Table 16.20 Propane Consumption

Month	Propane Consumption (L)
January	96,740
February	93,690
March	70,530
April	6,980
May	30
June	30
July	30
August	40
September	40
October	11,400
November	56,850
December	108,220
Total	**444,580**

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 Knipple Transfer Station. At the Knipple 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 two 114,000 L tanks.

The propane delivery strategy to site is as follows:

- An 18,000 L propane delivery truck drives from Terrace, BC to the Knipple Transfer Station.
- The truck will transfer its load into a 24,000 L ISO 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 site 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 (December) at the maximum anticipated airflow volume, the mine air heaters will consume approximately 3,500 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.

16.10 PASTE FILL DISTRIBUTION

Paterson & Cooke (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 ramp to VOK. This booster pump will pump paste up to the Upper VOK Zone and Galena Hill (1,330 Level and above).
- The paste pumps will both be positive displacement piston pumps of 100 m^3/h peak capacity with a pressure rating of 120 bar. The nominal flow rate for the system will be 80 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 250 Pa with a range of 100 to 375 Pa. This equates to an cemented paste percent solids of 66.1% solids by weight (ranging from 62 to 69% 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 Zone

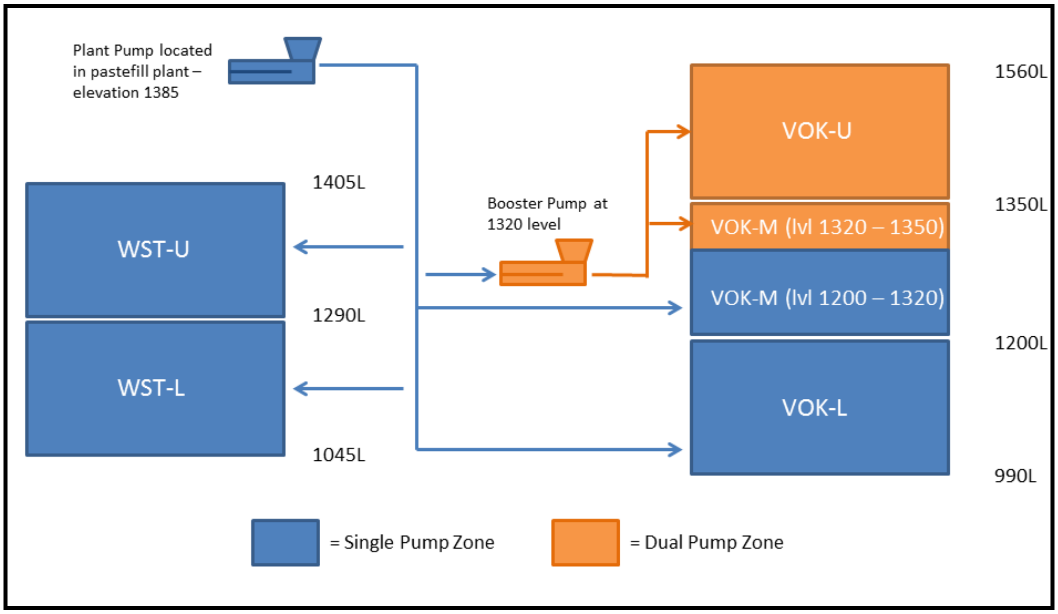
The mining schedule breaks down the Brucejack orebody into six areas: VOK-U, VOK-M, VOK-L, GAL, WST-U, and WST-L, as shown with their respective elevations in Figure 16.7 and Figure 16.8. The first areas to be mined are the VOK-U, GAL and VOK-M; coming on-line in Years 1, 2 and 3, respectively. Production in VOK-L will start in Year 2, while the WST Zones will only come on-line after Year 12. The VOK zones have continuous production scheduled until Year 20. The paste fill distribution system was designed with the schedule shown in Figure 16.13 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 to the main entrance to the VOK Zone on the 1,330 Level. This booster pump will pump paste up to the Upper VOK Zone and Galena Zone (1,330 Level and above). Figure 16.34 shows the breakdown of the Brucejack ore zones by the paste pumps feeding them: single pump zone and dual pump zone.

Figure 16.34 Paste Fill Distribution System Schematic Showing Paste Pumping Zones



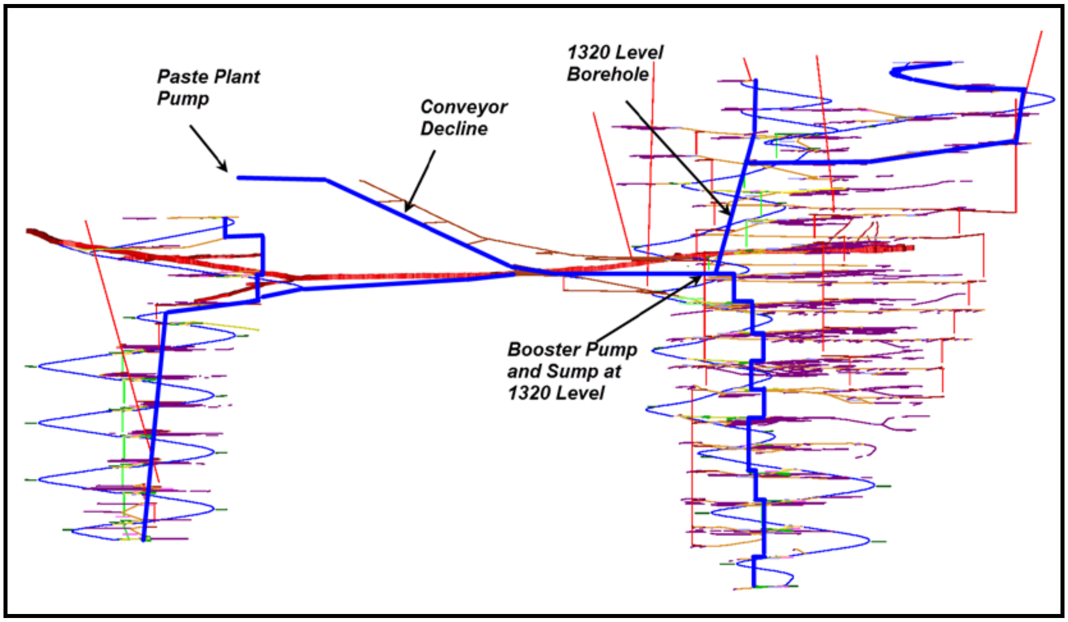
16.10.3 Distribution System Layout

The underground perspective view of the paste fill distribution system is provided in Figure 16.35.

Key points of the piping strategy are:

- one pump plus installed spare at the pastefill plant
- one booster pump plus installed spare near the ramp to VOK, 1,330 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.

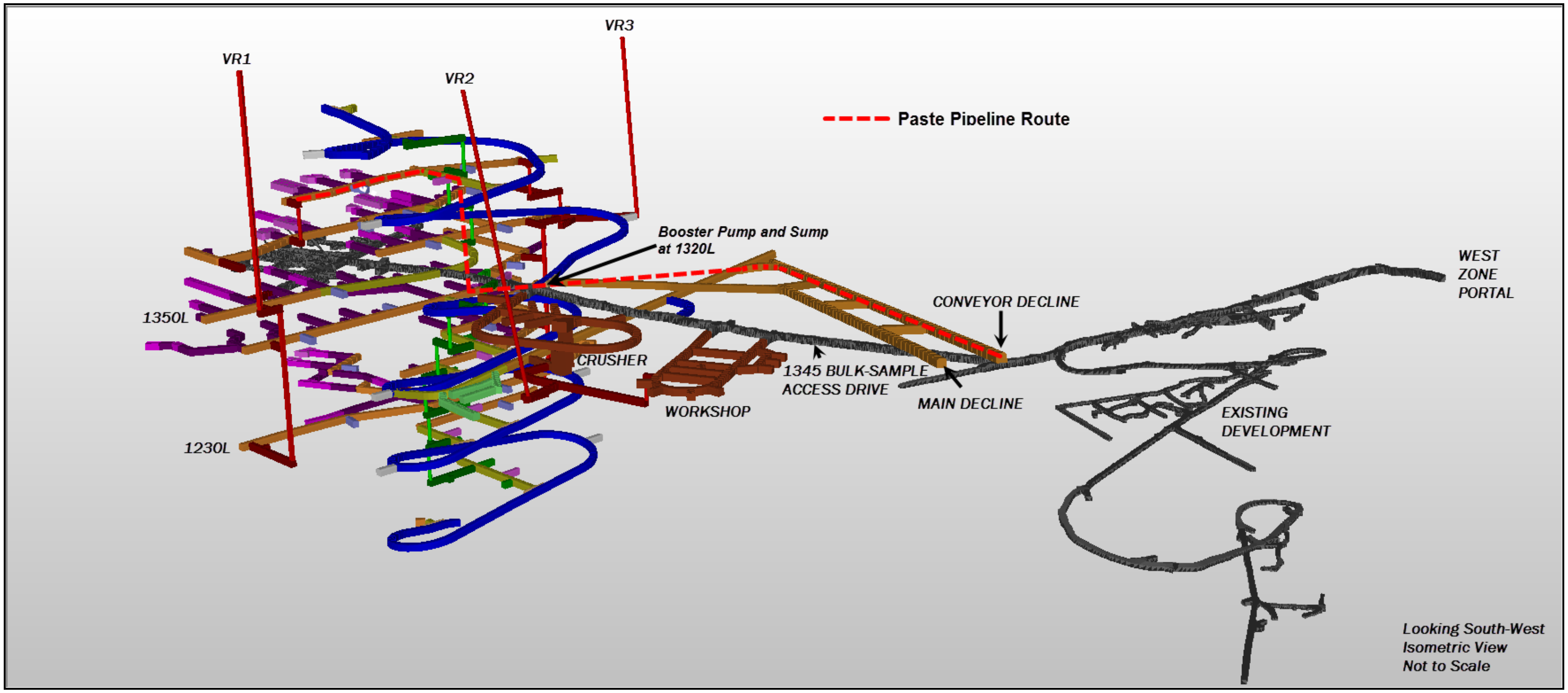
Figure 16.35 Paste Fill Distribution System Schematic



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 16.36 illustrates the placement of the paste fill line during this phase.

Figure 16.36 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. Details for the pre-production manpower requirements and transition can be found in section 16.4.2 and 16.4.3.

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 9.0 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. There are a number of recent examples of BC mines that operate under similar shift variances.

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 16.21 shows the total personnel by operational group when the mine has reached full steady state production.

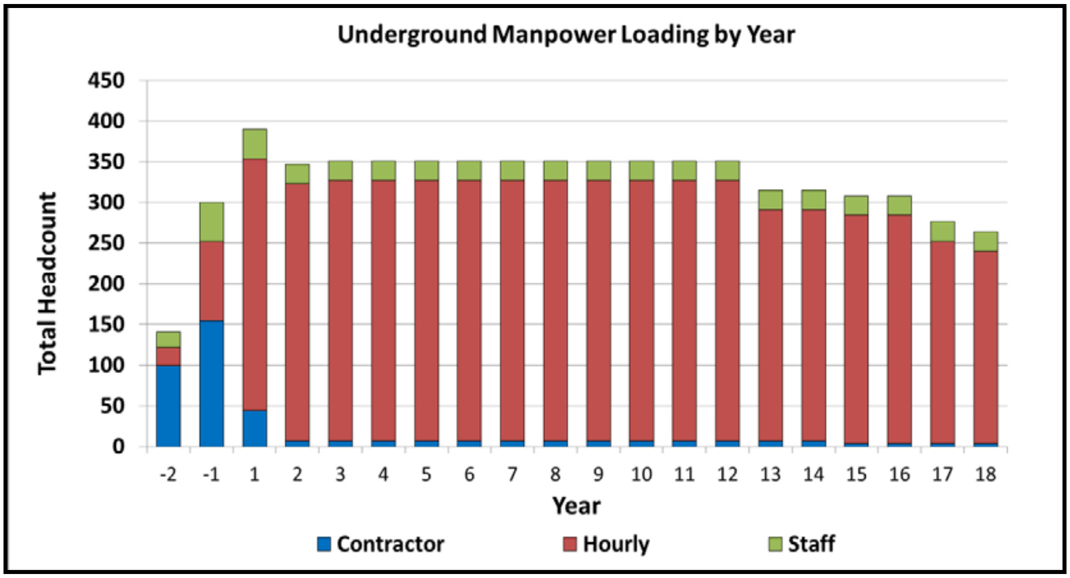
Table 16.21 Manpower by Operational Group

Role	Head Count	Role	Head Count
Mining Supervision (7)			
Underground Superintendent	1	Mine Captain	2
Safety/Training/First Aid	4		
Development Crew (68)			
Development Shift Boss	4	LHD Operators	12
Jumbo Operators	12	Truck Operators	8
Bolter Operators	8	Blasters	8
Cable Bolter Operators	4	Service Installers	12
Production Crew (106)			
Production Shift Boss	4	Crusher Operator	6
Long Hole Drillers	20	Crusher Labourer	4
Blasters	12	Backfill Leader	4
LHD (Electric) Operators	20	Timber Men	8
Truck Operators	16	Backfill Operator	8
General Labourers	4		
Raising (3)			
Raise Leader (Contract)	1	Raise Mechanic (Contract)	1
Raise Miner (Contract)	1		
Logistics (21)			
Underground Chief of Logistics	1	Boom Truck/Grader Operators	8
Underground Warehouse Manager	4	Clerk/Labourer/Forklift Operator	8
Maintenance (74)			
Maintenance Superintendent	1	Welders	8
Master Mechanic	2	Chief Electrician	1
Mechanics	20	Lead Electrician	1
Mill Wrights	8	Electricians	16
Apprentices/Labourers	16	Maintenance Planner	1
Technical Services (36)			
Technical Services Manager	1	Senior Engineer	1
Clerk	2	Engineer	3
Senior Geologist	1	Mine Planning & Scheduling	2
Production Geologist	4	Surveyors	8
Geological Technologist	8	Geotechnical Engineer	1
Production Diamond Drillers	4	Ground Control Technician	1
Additional Hires (36)			
Total Personnel	**351**		

Figure 16.37 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 and maintenance provided by the Owner. During Months 18 through 24, mine forces will transition from Contractor to Owner. Additional hires are personnel employed to compensate for shortages due to vacations, absenteeism and turnover.

Figure 16.37 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 as described in Section 16.0.

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 5,600 kg of gold and 1,900 kg of silver as doré and 44,000 t of gold-silver bearing flotation concentrate from the mill feed, grading 14.1 g/t gold and 57.7 g/t silver. The estimated gold recoveries to the doré and flotation concentrate are 43.3% and 53.4%, respectively, totalling 96.7%. The estimated silver recoveries reporting to the doré and flotation concentrate are 3.5% and 86.5%, respectively, totalling 90.0%. The LOM average gold and silver contents of the flotation concentrate are anticipated to be approximately 157 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 gravity concentration

- rougher flotation and rougher/scavenger 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 2014, the 2013 and 2014 bulk sample processing programs, 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.

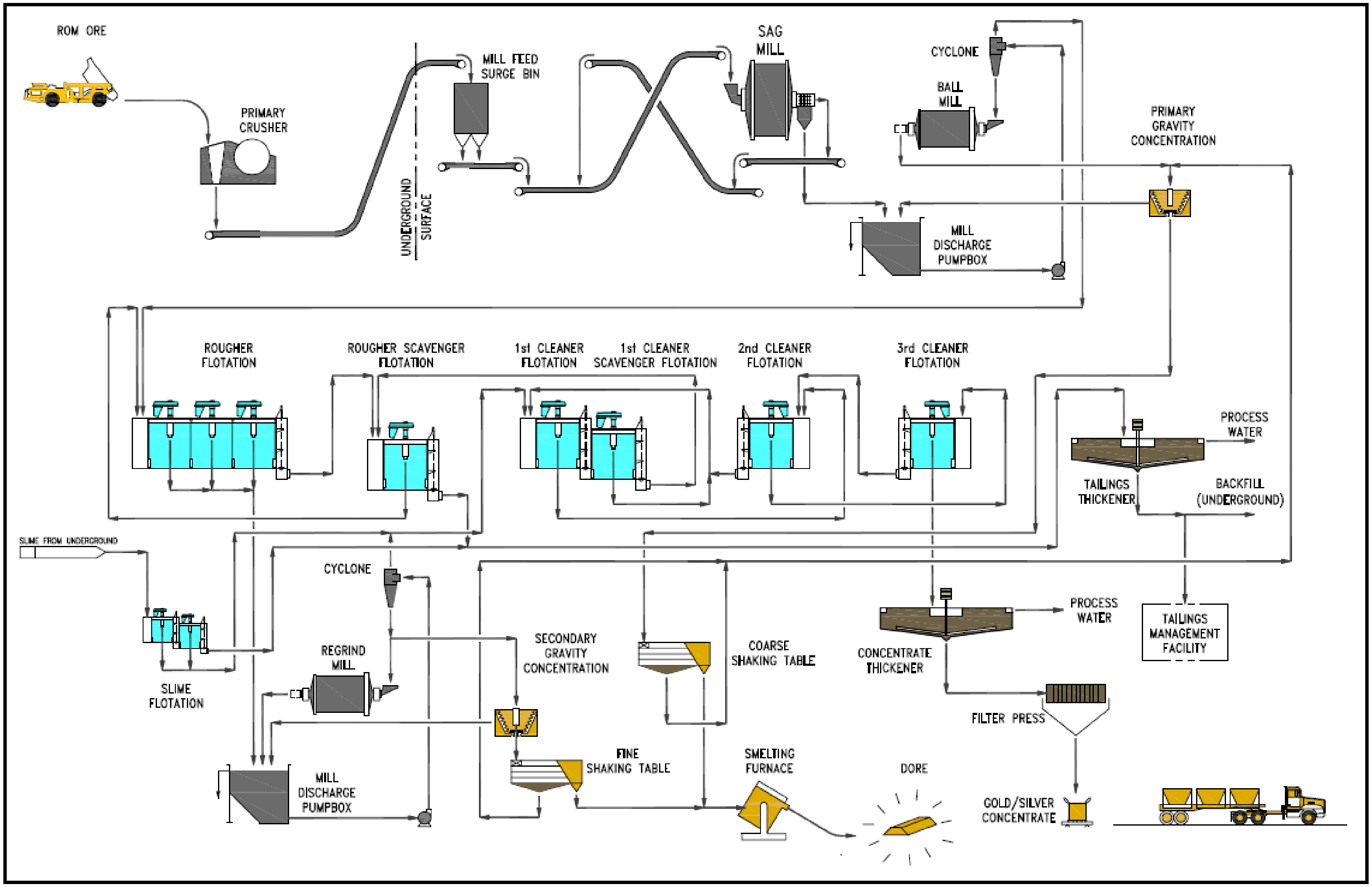
Various test programs evaluated the cyanidation process to extract the gold and silver from the test head samples and flotation concentrates. Although the mineralization responds reasonably well to the process, a few of the samples showed poor cyanidation responses, which may be a result of the presence 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 and economic viability 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 60
	% Ag	1 to 30
Metal Recovery – Flotation Concentrate	% Au	30 to 70
	% Ag	60 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	90
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 Regrind 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 (150 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 portal building, 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 three 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 1,066 mm wide jaw crusher discharge belt conveyor
- a transfer tower located at the surface

- a belt conveyor, 914 mm wide by 85.0 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, 914 mm wide by 6.0 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 2,590 mm long (19 ft by 8.5 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
- two centrifugal gravity concentrators and ancillary screens
- 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. Provisions have been made to provide sufficient space in the grinding area to accommodate a pebble crushing circuit if required at a later date.

The ball mill will be operated in closed circuit with hydrocyclones and two centrifugal gravity concentrators. The product from the ball mill will be discharged into the gravity concentrator feed pump box. The entire ball mill discharge will report to the gravity concentration circuit. The stream will be split in two and each stream will feed a safety screen with the undersize reporting to the centrifugal gravity concentrator. The gravity concentrator tailings will report to the gravity concentrator pump boxes where it will be pumped to the hydrocyclone feed pump box. The gravity concentrator tailings will be combined with the SAG mill trommel screen undersize slurry. The blended slurry in the pump box will be pumped to the hydrocyclones for classification. The hydrocyclone underflow will return by gravity to the ball mill. 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 90 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 ball mill discharge. Tailings from gravity concentration will return to the hydrocyclone feed pump box by pumping. The gravity concentrate will be pumped to the gold room for further upgrading by tabling. Tailings from the tabling will be further processed by a centrifugal concentrator in the gold room. The concentrate produced from the centrifugal concentrator will be recycled back to the tabling circuit while the tailings will be pumped to the feed well of the centrifugal gravity concentrators located in the grinding circuit. The concentrate from the tabling circuit 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 operating 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 the collector and MIBC as the 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 head of rougher flotation for re-processing. 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.

Slimes Flotation

An additional flotation circuit consisting of two 3 m^3 flotation cells will be included in the circuit to handle slime material that is generated in the underground. This material will enter the circuit at approximately 30% solids. The reagents PAX and MIBC will be added to enhance flotation recovery. The concentrate recovered in this circuit will be pumped to the head of the first cleaner flotation circuit. Tailings from the circuit will report to the rougher scavenger tailings pump box for disposal.

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 2,710mm diameter by 4,120 mm long (EGL) ball mill, driven by a 375 kW motor
- one centrifugal gravity concentrator and ancillary screen
- 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 mill 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 together with the concentrate from the slime flotation circuit 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.

Gravity Concentrate Upgrading

The gravity concentrates produced from the primary grinding circuit and the regrind circuit will be upgraded by conventional tabling followed by smelting to produce gold-silver doré. Upgrading will be conducted in a secure facility within the mill building, 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 centrifugal gravity concentrator with a 12’’ concentration bowl

- one table concentrate dryer
- flux reagent storage
- one flux mixer
- one 175 kW induction melting furnace
- one vault for storing doré and table concentrate
- one electrostatic dust collector
- one off-gas and dust scrubbing system
- ancilliary equipment, including slag treatment devices.

The centrifugal gravity concentrate from the primary grinding circuit will be pumped to a sizing screen located in the gold room where it will be split into a coarse and fine fraction prior to tabling. The coarse fraction will enter the coarse table stock feed tank via gravity. The fine screen fraction will be directed to the fine table feed stock tank. The regrind circuit gravity concentrate will be pumped to the fine feed stock tank located in the gold room. The two concentrates will be then upgraded by dedicated tables on a batch basis. The coarse fraction will be treated over two shaking tables while the fine will be concentrated over one table. 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 a scavenger centrifugal gravity concentrator located in the gold room. The concentrate will report to the gravity sizing screen located in front of the table feed stock tanks. Tailings from the gravity circuit will be pumped to the safety screen located at the feed 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 together with an electrostatic precipitator 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 hoods. 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 (5,000 mm diameter by 6,000 mm high)
- one tower-type up to 40 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 60% solids. The concentrate stock tank will be an agitated tank, which serves as the feed tank for the concentrate filter. A tower-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 2.500 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 backfilled to the excavated underground stopes, the water from the water treatment plant or from Brucejack Lake will be sent to the mixing tank. 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 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.

Mine site potable water supply systems will be located in the mill building. 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 within the mill, camp and truck shop.

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 dry air will be provided by dedicated compressors and desiccators 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.

Inline Sample Analysis

The plant will rely on the on-stream or in-stream particle size analyzer and various flow rate 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 for the expected mine life of 18 years. 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 5,600 kg of gold and 1,900 kg of silver contained in doré, and 44,000 t of gold-silver bearing flotation concentrate. On average, the flotation concentrate will contain approximately 157 g/t gold and 1,000 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	839,490	15.4	11.7	314.2	2.8	5,715	1,450	44.1	14.8	47,061	52.7	70.2	145	146	96.8	84.9
2	994,511	15.2	11.7	343	2.6	6,656	1,718	44.1	14.8	52,423	52.7	70.2	152	156	96.8	85.0
3	994,512	16.7	12.8	295	2.3	7,517	1,866	45.2	14.7	45,114	51.8	70.5	191	199	97.0	85.2
4	983,608	15.9	9.9	344	2.5	6,997	1,456	44.7	15.0	49,464	52.2	69.5	165	136	96.9	84.5
5	988,266	16.9	11.0	296	2.1	7,473	1,614	44.7	14.9	41,595	52.2	69.9	210	182	96.9	84.8
6	998,838	17.5	10.6	285	2.1	7,866	1,574	45.0	14.9	40,193	51.9	69.7	226	183	97.0	84.7
7	986,207	17.8	11.8	307	2.2	7,925	1,725	45.1	14.8	42,377	51.8	70.2	215	193	97.0	85.0
8	995,722	17.5	11.7	273	2.1	7,850	1,725	45.2	14.8	41,882	51.8	70.2	215	195	97.0	84.9
9	993,721	14.9	10.2	259	2.2	6,480	1,522	43.8	15.0	42,325	53.0	69.6	186	167	96.8	84.6
10	987,218	15.5	11.2	323	2.6	6,748	1,576	44.1	14.2	50,788	52.7	70.6	159	154	96.8	84.9
11	984,791	13.0	29.3	319	2.7	5,450	1,680	42.5	5.8	52,835	54.1	82.1	132	448	96.6	88.0
12	993,151	13.9	69.2	270	2.3	5,933	2,187	43.0	3.2	46,286	53.6	87.1	160	1,294	96.6	90.3
13	986,322	11.6	102.8	212	2.5	4,678	2,316	40.9	2.3	49,665	55.6	87.7	128	1,789	96.4	90.0
14	980,578	9.9	151.9	209	2.6	3,882	2,675	39.8	1.8	51,310	56.1	88.8	107	2,577	96.0	90.6
15	990,726	10.2	158.7	223	2.5	4,086	2,731	40.4	1.7	50,578	55.6	88.9	111	2,762	96.0	90.6
16	907,805	10.0	104.1	255	2.3	3,524	1,945	38.7	2.1	42,010	57.6	87.7	125	1,973	96.2	89.8
17	663,357	8.0	254.7	225	2.7	1,853	2,739	34.8	1.6	36,457	60.9	90.2	89	4,179	95.7	91.8
18	280,857	7.1	271.9	289	2.8	633	1,214	31.7	1.6	15,698	63.9	90.3	81	4,393	95.5	91.9
LOM	16,549,680	14.1	57.7	281	2.4	101,268	33,711	43.3	3.5	798,062	53.4	86.5	157	1,034	96.7	90.0

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 73.5 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, utilidor and pads
- a mill building containing process equipment, water treatment plant, paste backfill plant, potable water treatment plant and laboratory
- water management infrastructure, including diversion ditches for both contact and non-contact water, interceptor ditches, and a contact water drainage collection pond and pump(s) to direct water to a water treatment plant
- a water treatment infrastructure, to treat underground infill water and surface contact water with a treatment plant which will discharge to process and fresh/fire water tanks
- sewage treatment infrastructure
- an incinerator
- solid waste management systems, including domestic waste disposal
- power distribution from the mine site substation to all the facilities
- process control and instrumentation
- communication systems
- ancillary facilities including:
  - on site fuel storage

- on site explosive storage
- detonator magazine storage
- camp accommodation with recreation area, commissary, laundry facilities, mine dry and medical clinic.
- truck shop with first aid/emergency response
- a heli-pad
- a laydown area
- a covered storage building
- an enclosed concrete batch plant and storage for cement, sand and aggregate.

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 maintenance and emergency vehicle building
- on-site fuel storage and dispensing
- potable water treatment plant
- a sewage treatment plant
- an incinerator
- a laydown area
- a communications system
- a 30-person camp
- a temporary covered storage.
- a gatehouse
- a truck scale
- a heli-pad.

Figure 18.1 Brucejack Overall Site Layout

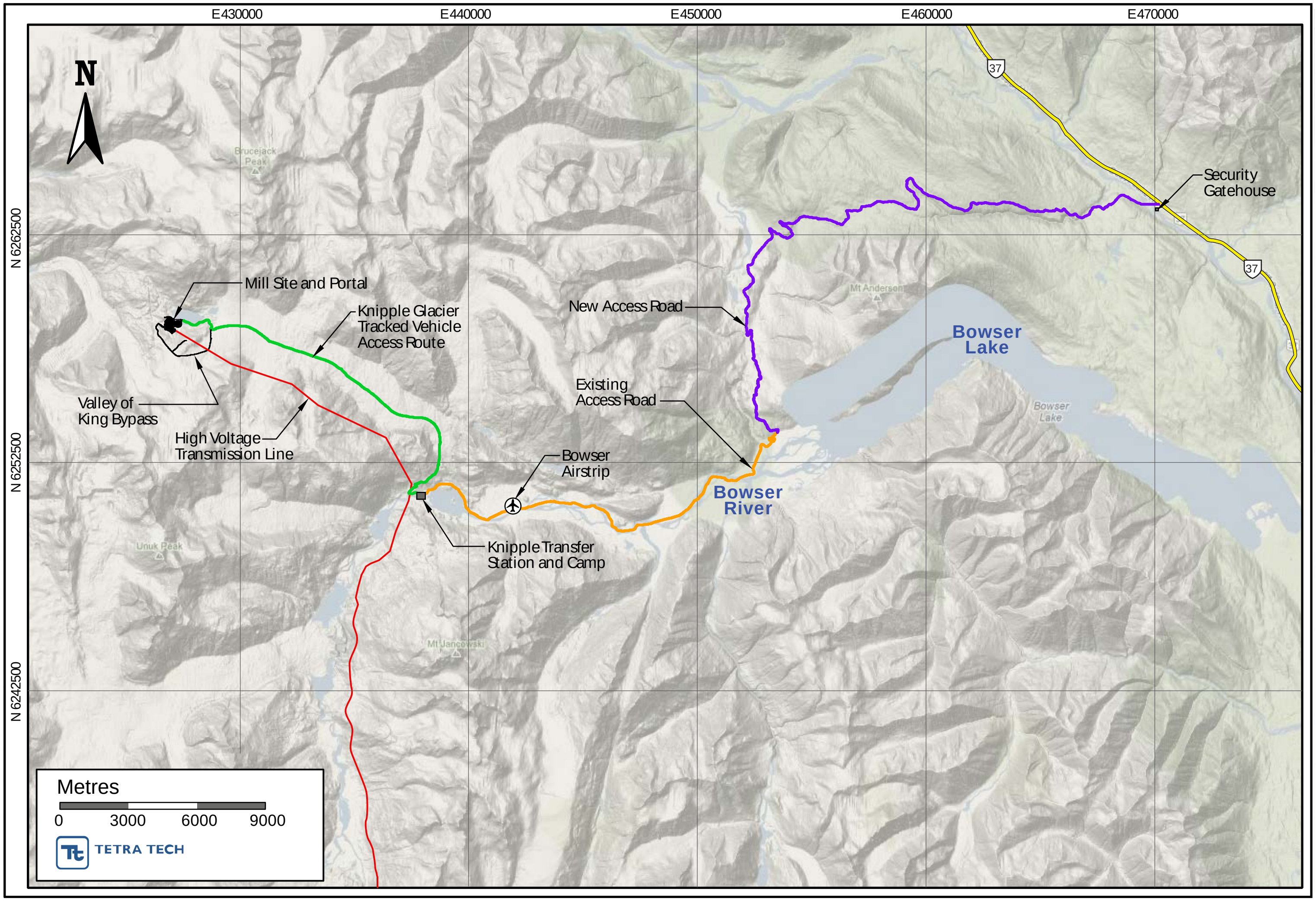
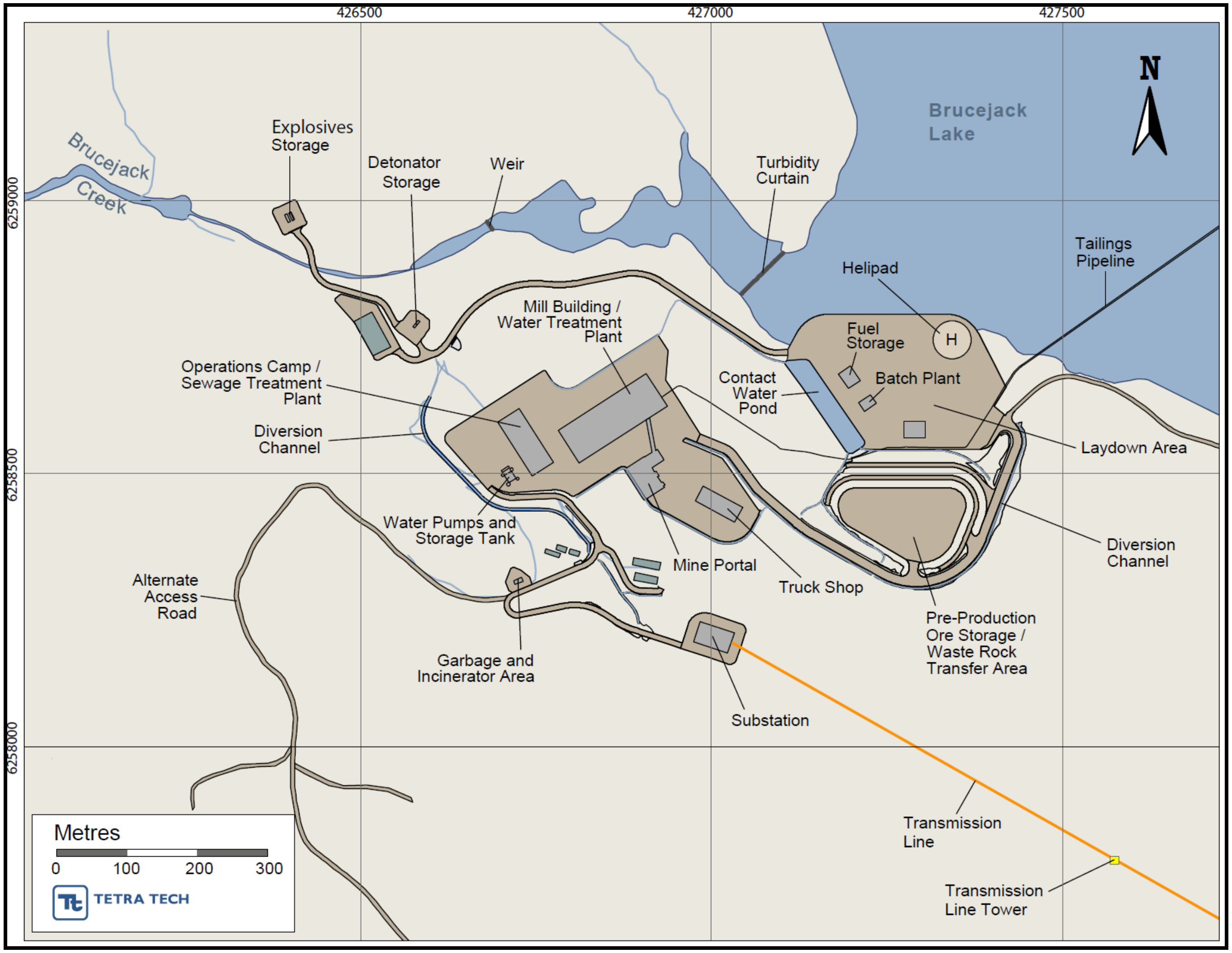


Figure 18.2 Brucejack Mill Site Layout



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 2013 to evaluate the subsurface conditions in the area of the proposed mill building. The investigations completed in 2013 consisted of geotechnical drilling only. Investigations consisting of geotechnical drilling, test pit excavations, and geophysics were also undertaken in 2011 and 2012 to evaluate previously proposed site layouts. Details regarding the investigations are provided in BGC's 2014 design report, "Feasibility Level Geotechnical Analysis and Design of the Plant Site Foundations". Based on the investigations completed, the overburden within the footprint of the mill building is generally less than 1 m thick and overlies good quality bedrock. In other locations across the site, overburden up to 8 m thick has been encountered overlying fair to good quality bedrock.

18.2.2 FOUNDATIONS

For all buildings and facilities located within the mill building, foundations will consist of conventional spread footings, strip footings, or other mass concrete foundation elements founded on good quality bedrock. Recommendations were provided for allowable bearing pressures of the mill building based on the results of the site investigations (BGC 2014). 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 20 mm and 8 mm, respectively. Table 18.1 provides a summary of the recommended allowable bearing pressures.

Table 18.1 Recommended Allowable Bearing Pressures and Maximum Foundation Widths

Foundation Stratum	Foundation		Allowable Bearing Pressure (kPa)
	Type	Maximum Width[2] (m)	
Bedrock	Mat	30	1,000
	Spread Footings	10	1,000
	Strip Footings	5	1,000
Structural Fill[1]	Spread Footings	3	200

Notes: [1]Foundations on structural fill assume a minimum embedment of 1 m below the surrounding grade.
[2]A minimum foundation width of 1 m has been assumed.

For the design cold year, the depth of frost penetration is estimated to be up to 3 to 4 m in overburden and up to 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 (2014). 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 2014). 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 and scaling may be required based on field engineer's review
Cut Slope	Rock - Benched	30 (refer to comments)	Refer to comments	6 m bench height, 5 m bench width, 75° bench face angle; spot bolting and scaling 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 (2014).

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 an increase in design speed to 40 km/h where possible, minor re-alignments of the sharper curves, reductions of the steeper grades, and additional surfacing of some sections. The work will include:

- Variable depth surfacing (pit run gravel or 3" minus shot rock, which has been compacted, of 200 to 300 mm depth) and shaping from 3 to 34 km and from 54.5 to 58 km.
- Upgrading to favourable grades of 12% (14% for pitches less than 150 m) and adverse grades of 8% (10% for pitches less than 100m).
- Re-work of horizontal curves to a design radius of 65 m, a MAXIMUM 3% super-elevation (or crossfall) is prescribed (1/3 of an inch per foot or 3 cm/m).
- Safely address two way traffic along the Wildfire Road, in all locations with sight distances below the Minimum Site Stopping Distance of 95 m (blind vertical and horizontal curves) the road width is proposed to be widened to 10 m (double lane) to allow traffic to safely pass. The road width which would allow for safe passage of two rock trucks is 12 m.
- Posted signage in both directions of travel to identify the proposed road speed zones with the exception of the 10 km zones across bridges beyond the Wildfire Bridge.

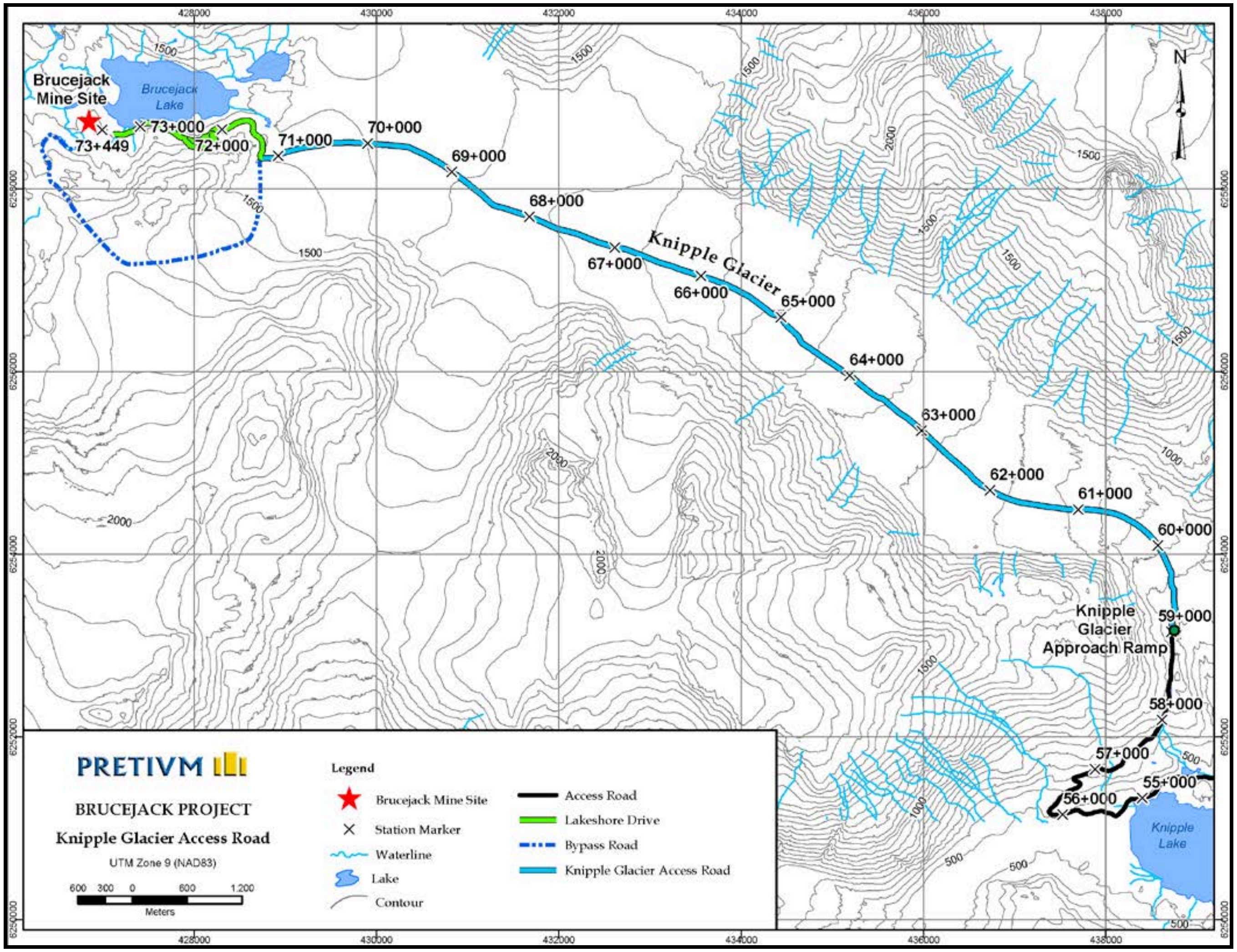
During mine operation, the road will be maintained throughout the year by road grooming equipment and snow plows. Regular patrols will be conducted in potential avalanche areas and avalanche control measures will be utilized.

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 2014 report entitled Feasibility Level 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.

The frequency of the standard axle load and subgrade modulus has 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/operations camp/truck shop
- substation

- laydown
- 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/camp/truck shop 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. It will not be used as rock or general fill for the site, unless tested and proven to be non-PAG. The pads will be finished with a gravel surface, and will drain to ditches. Retaining walls are proposed as required in several locations.

The feasibility-level design incorporates walls in two locations around the mill/camp/truck shop pad. These will be mechanically stabilized earth (MSE) type walls which will vary in height from 0 m to approximately 19 m. The walls will be designed with appropriate geotechnical recommendations and professionally reviewed by geotechnical and structural engineers.

Pretivm provided Light Detection and Ranging (LiDAR) data on October 19, 2012, that was deemed to be within ±0.5 m vertical accuracy. This level of accuracy 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 deficit of material with the current design, which is largely attributed to the lower laydown area. The overall deficit will be supplanted with material from the borrow quarry that is approximately 1.3 km east of the mill site.

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 drilling and 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.

Ditches are designed for the 200-year, 24-hour storm event, plus snow melt. The contact and non-contact water ditches on site are all proposed to have 2H:1V side slopes, a minimum 0.5% longitudinal grade, and be either v-shaped or trapezoidal. All water ditches will be a minimum of 1 m deep and designed to not overtop during the 200-year, 24-hour storm event. Non-contact water ditches will be erosion protected with riprap underlain by geotextile where scouring velocities are exceeded. Contact water ditches will be lined with a high-density polyethylene (HDPE) liner underlain by a cushion layer of 30 mm, minus granular material.

Culverts are designed to discharge a 10-year storm event without static head at the entrance, and to discharge the 200-year, 24-hour storm event utilizing available head at the entrance. Culvert corrugation and wall thicknesses were specified in accordance with structural load-carrying capacity under anticipated live loads 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.

Contact water drainage will include drainage from rock cut surfaces for the mill/camp/substation pad, rock cut surfaces for the truck shop/portal pad, and pre-production ore storage areas. Contact water will be collected and conveyed via lined contact water ditches to a contact water pond on the west side of the lower laydown area. The contact water pond will be 48,700 m^3 to accommodate the 200-year, 24-hour storm plus snow melt event. The dimensions of the contact water pond will be 45 m wide by 150 m long at bottom, with side slopes proposed as 3H:1V, and a 3 m freeboard. It will be lined with a HDPE liner underlain by a cushion layer of 30 mm, minus granular material. Water discharge from the pond will be pumped to the water treatment plant.

During construction and operation, erosion control measures will include riprapped/geotextiled and HDPE lined ditches and will mitigate sediment release. Residual sediments will be handled in accordance with the sediment control plan.

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 are based on Drawing # 100000-10-010 (Progress Print - May 7, 2014), and include:

- explosives storage– preliminary position
- detonator storage – preliminary position
- substation and generators
- temporary water treatment plant

- mill building including administration, warehouse, and water treatment
- mine portals
- tailings pipeline
- helipad
- covered laydown area
- batch plant and fuel storage
- pre-production ore/waste rock transfer storage
- diversion channel
- garbage and incinerator area
- operations camp and mine dry
- sump pump
- site drainage collection pond
- 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 garbage and incinerator area, some sections of the site access roads, and the pre-production ore/waste rock transfer 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.

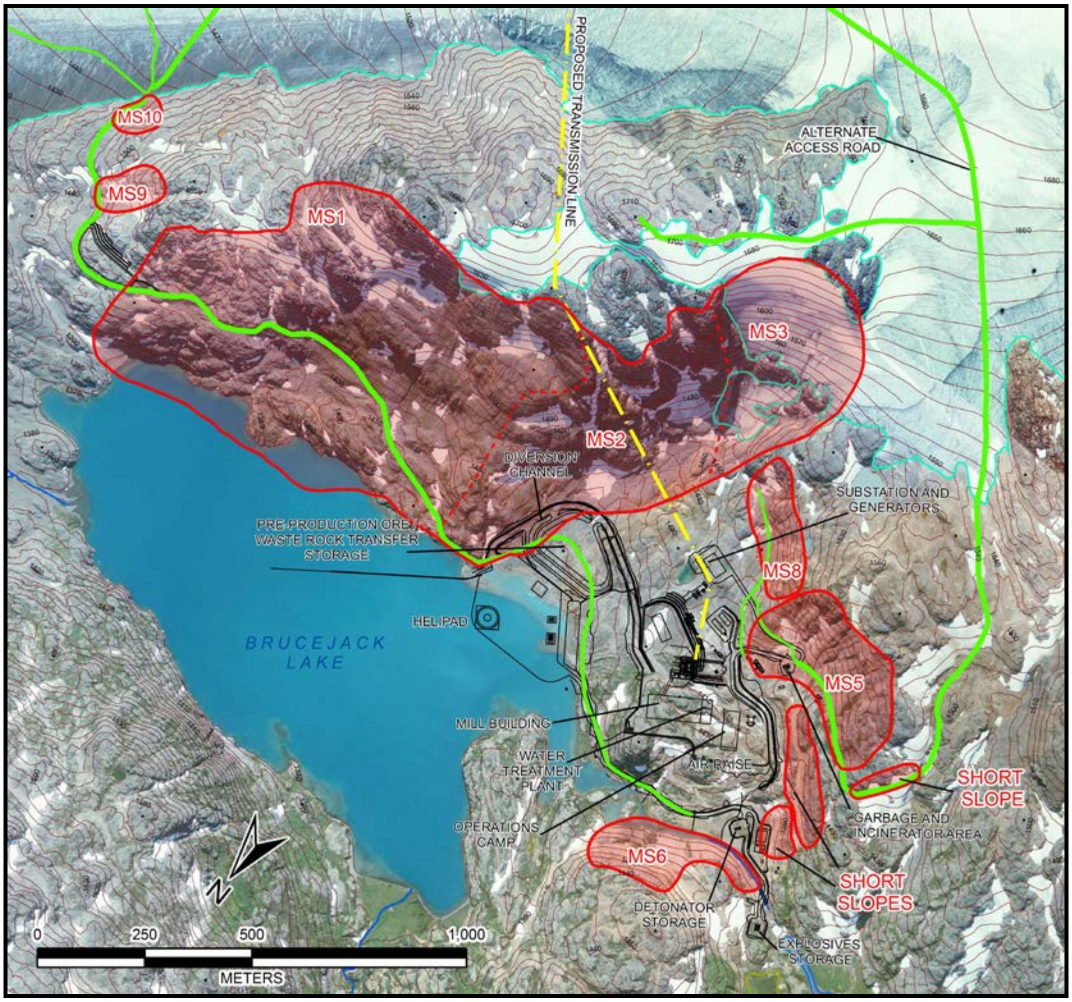
Table 18.5 provides a summary of avalanches reaching the mine site area and Figure 18.4 illustrates the approximate hazard locations.

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/waste rock transfer 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 5	MS5	Garbage and incinerator area	1,420	RZ	20	1:1	1:3	

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.

Garbage and Incinerator area

Size 2 and 3 avalanches from Path MS5 are estimated to reach the garbage and incinerator area with annual return frequency. Potential consequences of avalanches reaching this area include damage to vulnerable infrastructure (e.g. non-structural components of the facility, 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

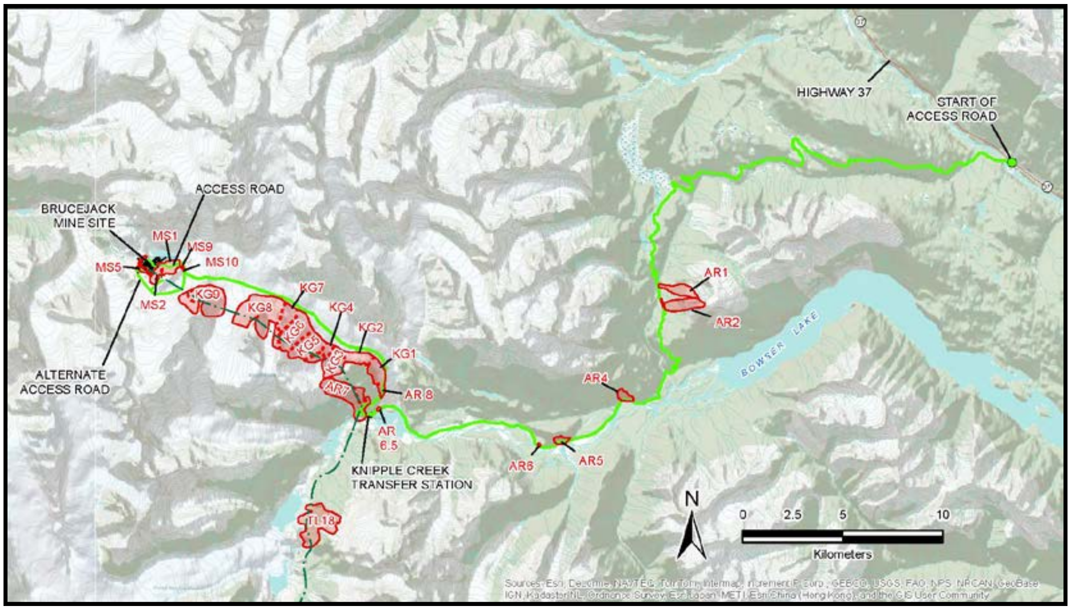
Path or Area ID	Avalanche Atlas Polygon Label	Facility Affected	Approximate Elevation of Facility (m)	Facility Position in Path	Approximate Length of Facility Affected (m)	Estimated Return Frequency (events:years)		
						Size 2	Size 3	Size 4
Access Road 1	AR1	Access Road	580	-	-	-	-	P
Access Road 2	AR2	Access Road	580	-	-	-	-	P

table continues...

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 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. From this facility, 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. At this time the exact alignment has not been finalized; however, several preliminary supporting structure (tower) locations have been proposed. 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 of the preferred alignment indicates that the Salmon Valley segment of the line crosses sporadic small avalanche terrain and a few larger paths. The segment above the Knipple Glacier crosses extensive large avalanche terrain along much of the alignment. Avalanches would only pose a hazard if supporting structures (towers) were built in avalanche paths, or conductors were low enough to the ground to be impacted by the turbulent flow of a large avalanche (up to 50 m impact height). Potential consequences of avalanches impacting the line 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. Preliminary tower locations have been assessed using desktop analysis, and towers that are potentially be affected include Towers 5, 8, 9, 12, 13, 18, 19, and 54. The final alignment of the transmission line (including specific tower 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. Forces on towers from static snow forces can be estimated during final location planning for each tower, and appropriate mitigation can be determined at that time.

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. Portions of the transmission line route will require detailed assessment in the field to determine specific avalanche features in conjunction with selecting the tower structure locations.

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 transmission system 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 18.6). 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

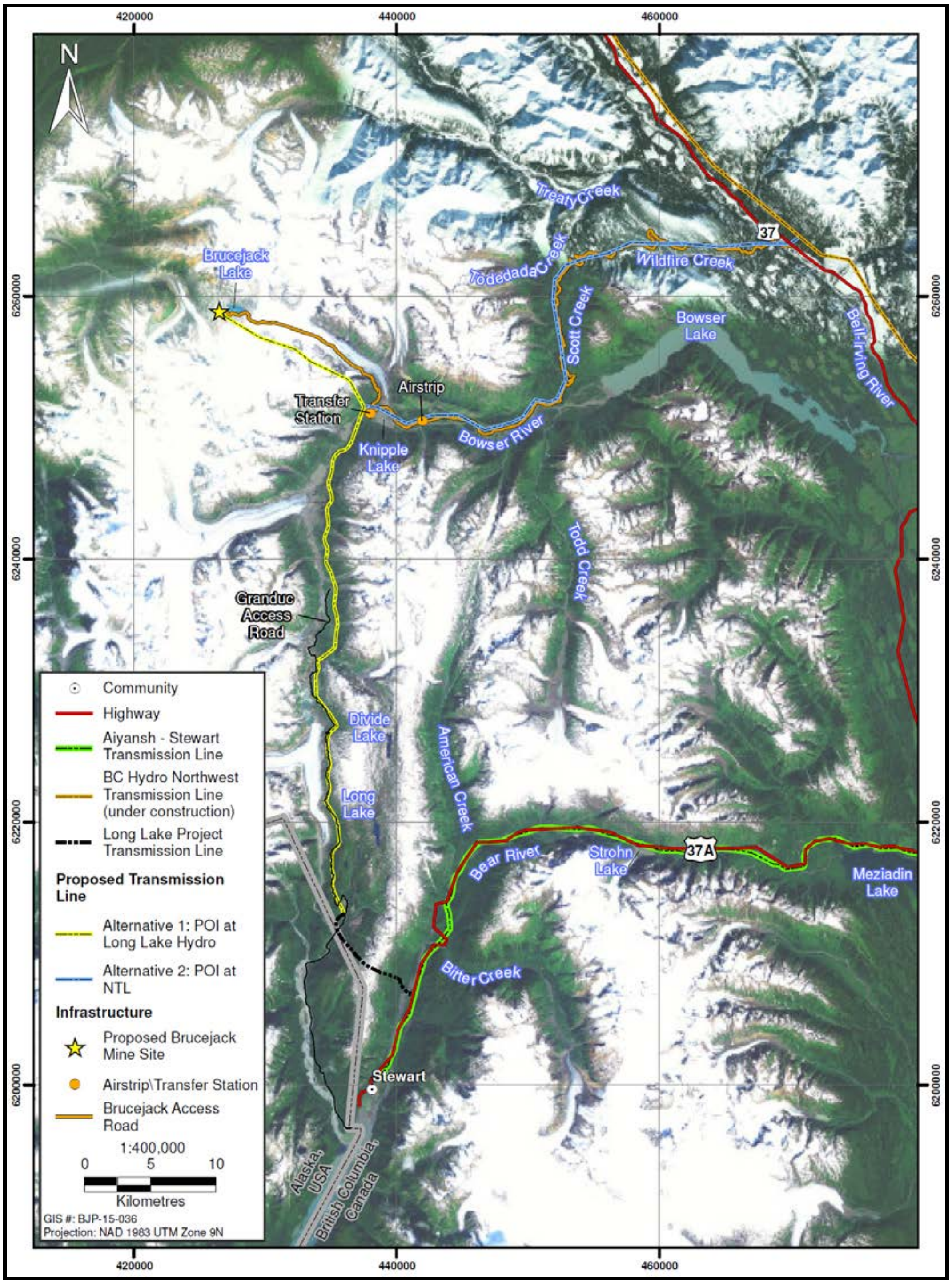


Figure 18.7 Photo of Typical Slopes in the Salmon River Valley

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.

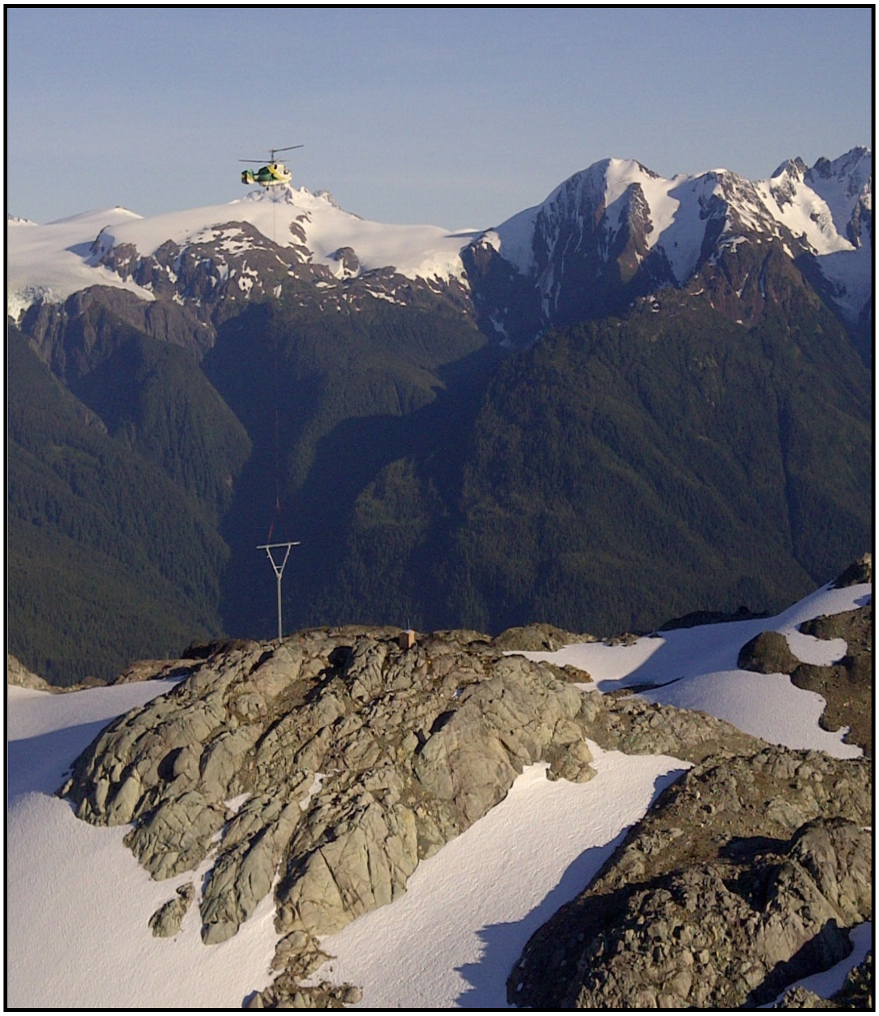
Engineering and construction constraints for the Project include:

- 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 300 MCM 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 Transmission Line contains additional information and a discussion of these constraints. Following initial review of the Project, Valard developed an initial engineering design for the transmission line to determine the likely placement of the transmission towers, based on detailed topography and the necessary ground clearance for the conductors. Concurrently, it was determined that some reroutes of the transmission line were necessary to avoid mineral tenure holders. The net effect was to decrease the span between structures (thus increasing the number of structures). The number of structures will be included in the final engineering design once detailed topography is available for the entire transmission route.

Additional engineering work on the line using updated terrain and weather information resulted in some changes from the previous design. These included changes in the engineering parameters due to extreme wind and ice loading; repositioning of some structures to avoid snow avalanche zones; addition of OPGW; and reroutes based on some local constraints. The result was to increase the number of towers from 148 to 156, although this number is expected to decrease based on detailed engineering of the final alignment using LiDAR and detailed field information.

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, cross arms, 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 updated by Valard on June 10, 2014 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 also includes the indirect costs for the Project including engineering, project management/administration, 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 10% soil, and 90% rock. The foundation estimates also include 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 most of the transmission line route.
- Helicopter stringing support will be carried out along the entire route.
- Note that this estimate assumes site planning/preparation and 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 may be incurred.

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 and these costs are not included in the estimate for the annual costs for the transmission line.

18.8 Surface Facilities

18.8.1 Architectural Design Basis

The architectural design criteria outlines the design parameters of structures and facilities for the Project used to obtain budgetary pricing. 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 of 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 Buildings

The mill building was designed as one large structure to house multiple facilities under one roof. This was done to minimize issues with high snow drifts and snow removal between multiple buildings. The footprint of the building is about 136.5 m by 63.5 m. This structure will include the following infrastructure and equipment:

- SAG mill feed surge bin
- process equipment including grinding mills, flotation cells and tanks, reagent storage, thickeners, and concentrate handling and load out and gold room
- paste backfill plant
- administration offices
- electrical and control rooms
- assay and metallurgical laboratories
- potable water treatment plant
- water treatment plant

The truck shop is a standalone pre-engineered steel building with a footprint of about 80 m by 24 m. Included within the truck shop structure are the following facilities:

- five repair bays to service both surface and underground vehicles

- welding bay
- truck wash bay
- emergency vehicle bays
- first aid and emergency equipment storage
- warehouse
- mechanical room
- electrical room
- washrooms

The 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.

Modular Buildings

The new permanent camps and scale house will be constructed using modular building design. The buildings will include heating, ventilation, and air-conditioning (HVAC) where necessary, 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 outside 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 where practicable.

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.

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, 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.

The laboratories will be constructed as a modular design and installed at a temporary location to operate during the pre-production stage. After the mill building is erected, the modules will be moved into the mill building as a permanent location.

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.

Warehouse Facility

The warehouse facility located within the truck shop structure will house a mill shop, electrical and instrumentation shop, mechanical room, and tool crib.

First Aid

The first aid and emergency response facility located within the truck shop structure 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.

Camp for Construction Phase and Operations

The new permanent camp will accommodate 400 people and will include a kitchen, recreation and exercise facilities, mine dry and a medical clinic. An existing camp kitchen and 120 dormitories will be available to supplement accommodation during peak construction periods.

Mine Dry

The mine dry will be constructed as part of the camp to accommodate 400 people, each with individual lockers and hanging baskets. The wicket and lamp rooms will be located in the camp adjacent to the dry where underground personnel will be picked up by underground vehicles and transported to and from the underground mine.

Batch Plant Facility

A metal roofed enclosed building will house a concrete batch plant, storage for cement, sand and aggregate and parking for two cement trucks. The footprint will be about 75 m by 20 m. This facility will be in operation for the construction period and during operation.

Incinerator

A batch-fed containerized incinerator system will process up to 1,300 kg of mixed solid waste material generate at the mine site per day. Solid waste will include mixed camp waste, non-hazardous solid waste consisting of food-waste, kitchen waste including packaging, cardboard, wood waste, kitchen grease and general refuse. In addition, there will be some dewatered sewage treatment sludge.

Utilidor

An underground concrete utilidor will connect the substation, camp, mill building, mine portal and truck shop. This ventilated utilidor will be used by mill personnel to access facilities and to route electrical feeders and process and utility pipes.

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 door.

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 door.

Temporary Facilities

A metal covered structure will be constructed at the laydown area to store equipment temporarily during construction. The footprint of the structure will be about 40 m by 20 m.

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.

18.10 WATER TREATMENT PLANTS

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 18 and that the mine water treatment plant will continue to operate until Year 22.

The mine water treatment plant will be capable of treating up to 10,000 m^3/d. Treatment will consist of the following steps:

- metal removal
- biological nitrification and de-nitrification
- polishing.

For underground development in Years -2 and -1, there will be a rental water treatment plant in operation to treat water pumped from underground and from the surface contact water collection pond.

18.10.2 POTABLE WATER TREATMENT PLANT

The new 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 ground well(s) to the storage tank. A packaged booster pumping system will be provided to pump water from the storage tank to the mill building and camp distribution system and supply to underground.

The new potable water treatment plant will supply approximately 120 m^3/d during construction, based on an average usage rate of 300 L/d per person and a crew of 400 people. The existing camp water treatment plant will supply treated water for an additional 180 people during construction.

During operation, the water requirement will be approximately 105 m^3/d to service a crew of 350 people.

18.10.3 SEWAGE TREATMENT PLANT

A sewage treatment plant will service up to 400 people. Sewage will be piped from the camp/dry to the treatment plant. Sewage from underground and truck shop will be stored in a heat traced holding tank. A truck will transfer sewage from the holding tank to the treatment plant. Sludge from the sewage treatment plant will be incinerated. The discharge water will be piped to the lake.

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 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 2014). It is assumed that all of the waste rock generated from the sources noted above will be PAG and, therefore, must be placed more than 1 m below the low water elevation of the lake to limit oxidation of the material. 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.

Overwater geophysical surveys consisting of bathymetric and sub-bottom acoustic profiling were completed on Brucejack Lake to develop bathymetric contours of the lake and to assess the thickness of the lake bottom sediments. Disturbed samples of the lake bottom sediments were also collected to assess the material's strength and behaviour characteristics (BGC 2014). This data was used to facilitate a preliminary stability assessment of the waste rock pile (BGC 2014).

The results of the assessment indicate that, under drained loading, the waste rock pile will have a factor of safety ranging from approximately 1.1 to 1.4 when applying strength estimates to the lake bottom sediments based on the results of laboratory testing. However, when applying possible lower bound strength estimates based on values from literature, the factor of safety ranges from approximately 0.9 to 1.2. These results are, however, based on the assumption that soft, weak lake bottom sediments extend all the way down to bedrock. It is considered possible that denser and stronger sediments may be present below the surface of the lake bottom. Further investigations, consisting of drilling and sampling, should be conducted to confirm this. For better definition of the strengths, undisturbed samples should be collected and in-situ vane shear strength profiling should be completed. This will allow for a more confident estimate of the waste rock pile's stability.

18.11.1 QUARRY

Construction aggregate for the Project will be sourced from a quarry located near the southeast corner of Brucejack Lake. In 2013, site investigations consisting of geotechnical drilling and surface mapping were completed to characterize the rock mass and structural geology of the proposed quarry (BGC 2014). Based on the results of the investigations, recommended slope design parameters were developed with consideration of potential structurally controlled instabilities, and rock mass controlled instabilities (BGC 2014). Geotechnical and regulatory design requirements were also considered. The recommended slope design parameters consist of an average bench face angle of 65°, a maximum bench height of 15 m, minimum bench widths ranging from 9 to 10 m, and a maximum overall height of 75 m. The overall angle of the quarry will be controlled by the bench geometry, and will be dependent on the number of benches utilized. Based on the maximum overall slope height considered, overall slope angles ranging from 45 to 47° are recommended.

18.12 TAILINGS DELIVERY SYSTEM

It is planned to discharge and store tailings in Brucejack Lake below the 40 m depth contour by a method that will minimize the contribution of suspended solids from tailings in the discharge from Brucejack Lake.

18.13 BRUCEJACK LAKE SUSPENDED SOLIDS OUTFLOW CONTROL

Approximately 3.5 Mt of waste rock and 8.7 Mt of tailings are anticipated to be deposited in Brucejack Lake over the projected 18-year mine life. Stringent discharge criteria (based on the Metal Mining Effluent Regulations (MMER)) state that the total suspended solids (TSS) concentrations in the outflow at Brucejack Creek must be less than 15 mg/L (Schedule 4 - Maximum Authorized Monthly Mean Concentration).

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.

Hydrodynamic modelling of Brucejack Lake was carried out by Lorax (2013) to examine the likelihood of the migration of tailings solids into lake surface waters. The results indicated that the potential for elevated TSS levels in surface waters was unlikely if the minimum particle diameter was greater than or equal to 5 μm.

However, it will be necessary to control the TSS concentrations at the outlet of Brucejack Lake to meet the MMER regulations. The current design basis for control of suspended solids includes the following:

- install one or more lines of turbidity curtains at the outlet of the lake to contain suspended solids
- install a flow monitoring weir across Brucejack Creek downstream from the lake outlet to facilitate monitoring

An allowance for site investigation and design of the outflow monitoring weir has been included in the capital cost estimate.

As a contingency to the use of turbidity curtains, an outlet control structure was designed to allow storage and release of lake water in a controlled manner. Review of the storage capacity versus lake level elevation for the outlet control structure indicates that flow from the lake could be stopped for a period ranging from several days (e.g. during freshet) to several tens of days (e.g. during the summer and early fall) depending upon runoff conditions in the lake catchment area (BGC 2014).

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.

The underground communication system is specified in Section 16.0 of this report.

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 near the mill and camp and in close proximity to the interconnection utilidors. Power feeds to the mill building, camp, truck shop and underground will be installed and routed through utilidors.

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 1.2 km of 4.16 kV single wood-pole overhead power lines will be constructed to provide power to outlying buildings such as the concrete batch plant, water reclaim, heli-pad, etc.

The emergency power strategy will employ the following generator sets:

- Four existing 500 kW, 600 V diesel generators will be installed at the main substation and will connect to the main power distribution bus for use as emergency power for operations. Two new 1,250 kW, 4,160 V diesel generators will be purchased for construction power and will also be installed at the main substation and connected to the main power distribution bus to further augment emergency power supply. 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.
- An additional 500 kW, 600 V diesel generator will be purchased for construction activities and 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 two week capacity, including allowance for auxiliary equipment. The fueling station will include loading/unloading pumps and filters.

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, maintenance and emergency vehicle building, cold storage, fuel dispensing system, helipad, incinerator, gatehouse, truck scale 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, potable water treatment plant 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. A wireless system will be installed for communications. An incinerator will be installed within a fenced area.

Maintenance and Emergency Vehicle Building

The footprint of the metal-roofed enclosed maintenance and emergency vehicle building is approximately 75 m by 20 m. This building will contain vehicle maintenance bays and emergency parking vehicle bays.

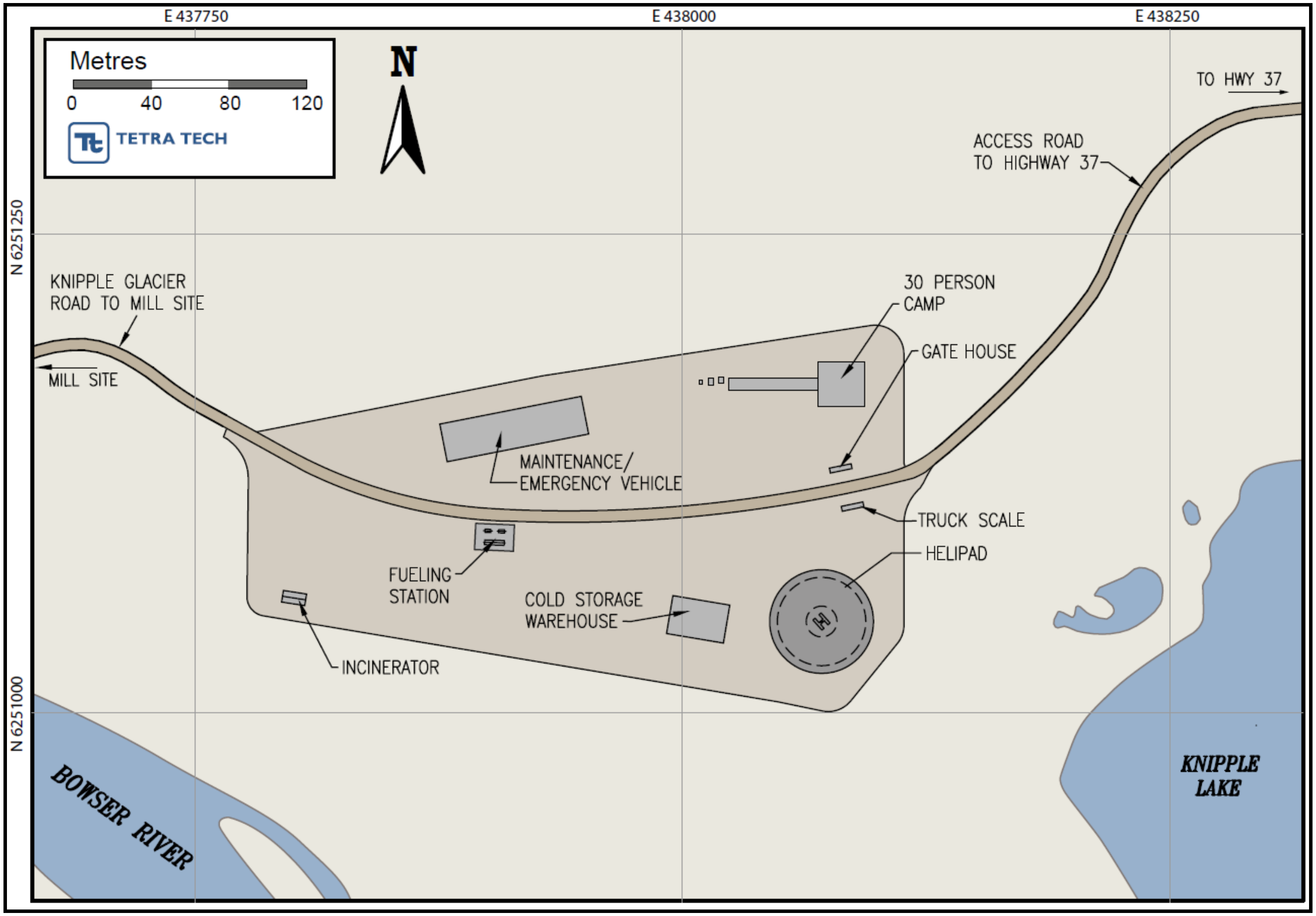
Diesel

Diesel fuel primarily for mobile equipment will be stored in one 50,000 L double-walled tanks located at the laydown area. The fueling station will include a receiving pump, a strainer and delivery pumps, and filters.

Temporary Facilities

A metal roofed enclosed structure will be constructed at the laydown area to store equipment and materials temporarily during construction. The footprint will be approximately 40 m by 20 m.

Figure 18.9 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. No site visit was conducted to visually verify the presence of any obstructions in the take-off/approach obstacle protection surfaces.

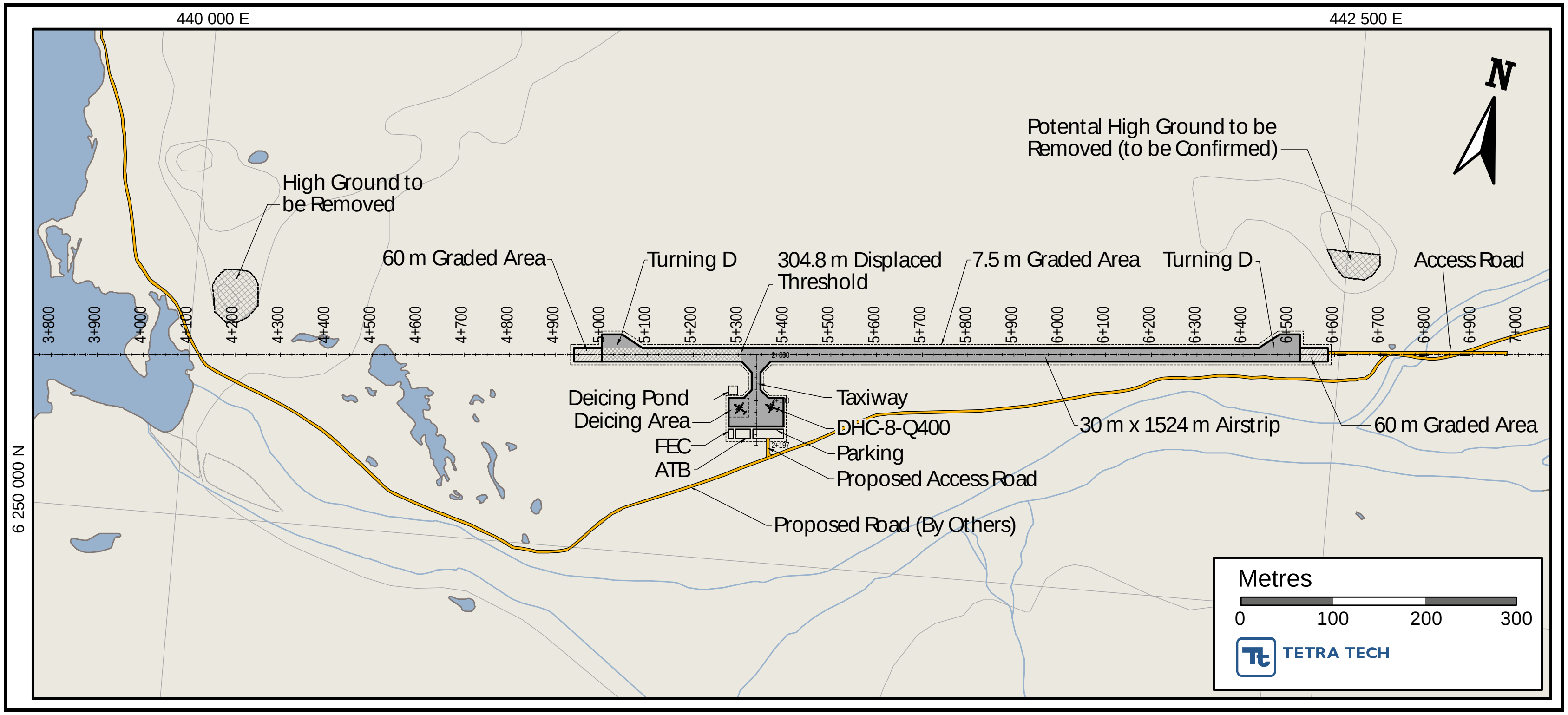
The passenger aircraft used in the design of the aerodrome is the Beechcraft 1900 however, the aerodrome facilities are sized sufficiently to allow DE Havilland Dash 8 turboprops and C-130 Hercules aircraft upon acceptance of the sites by the aircraft operators.

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 18.10 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 along 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 whether night flights can 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 ATCO-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 157 g/t gold and 1,000 g/t silver.

Gold and silver prices have fluctuated significantly. The current gold and silver prices (as of June 17 2014) together with the last three years and the last five years average prices are shown in Table 19.1.

Table 19.1 Gold and Silver Prices

Metal	Units	Spot	3 Years	5 Years
Gold	US$/oz	1,272	1,523	1,399
Silver	US$/oz	19.7	27.7	25.5

For the feasibility study update, the metal prices used for the base case estimate are US$1,100/oz for gold and US$17/oz for silver.

19.2 SMELTER TERMS

Pretivm contacted a number of smelters and metal traders 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. Currently, it is anticipated that the concentrate will be trucked to Terrace, BC, and then transported by rail to a smelter in eastern Canada.

For the feasibility study update, the indicative terms for the doré and flotation concentrates are 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 of \$184.00/dmt of concentrate is applied. A penalty charge for arsenic in the flotation concentrate is \$9.20 per each 0.1% of arsenic if the arsenic concentration is above 0.2%.

Tetra Tech recommends conducting further marketing studies, including 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 Railway 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. Small amounts of 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 water's 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 onto 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 representative indicative terms, the concentrate may 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. For any North American smelter, 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.

The estimated concentrate transportation cost for the feasibility study update is $181.65/wmt of concentrate.

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 mining, processing, and closure activities.

Environmental management is a corporate priority. It is integrated into all aspects of the organization which includes risk management, efficiency in development, design, and operation of facilities and 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, where practical, 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 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 and subsurface hydrology, aquatics, water and sediment, limnology, and fish habitat. Pretivm has also carried out 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 were 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 2014). There is currently uncertainty regarding the average annual precipitation estimate for Brucejack Lake due to a relatively short dataset and the difficulty of obtaining representative precipitation from a high altitude, steep watershed. Therefore, a range of precipitation values have been utilized for the hydrologic analyses, with average annual precipitation at site assumed to fall within the range of 1,900 to 2,034 mm. Precipitation data reported at the Unuk River Eskay Creek (#1078L3D) Meteorological Service of Canada (MSC) climate station were used to characterize the upper end of this average. 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. The lower end of the estimate, 1,900 mm, is based on the site and regional stream flow data, as described in BGC (2014).

These annual estimates are consistent with Environment Canada (2012) that 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)
		Lower End	Upper End	
January	-9.3	233	249	-9.3
February	-7.8	200	214	-7.8
March	-5.7	169	181	-5.7
April	-1.3	91	97	-1.3
May	3.4	82	88	3.4
June	7.1	63	67	7.1
July	9.0	78	83	9.0
August	8.7	130	139	8.7
September	4.8	193	207	4.8
October	-0.7	231	247	-0.7
November	-7.1	201	215	-7.1
December	-9.1	231	247	-9.1
Average/Total	**-0.7**	**1,900**	**2,034**	**-0.7**

Source: BGC (2014).

Average monthly temperature data used at the Project site are based on output from the ClimateBC model (Wang et al. 2012). 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 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 area 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, 2012, and 2013. 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 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 2013 as part of the baseline program. Results from three 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 1,300 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 the following materials exposed and produced during construction and operation of the underground mine:

- waste rock
- tailings, paste, sludge (from the Water Treatment Plant (WTP)) and ore (stockpile)
- surface materials involving overburden, excavated material and cut and fill materials at:
  - the mine site (including the plant foundation, overburden and roads)
  - proposed air strip (the Bowser Aerodrome)
  - the Brucejack Access Road
  - the proposed non-potentially acid-generating (non-PAG) quarry site.

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.

Waste rock and flotation tailings are planned for underground disposal and deposition in Brucejack Lake. Approximately 1.76 Mt (40%) of the waste rock produced from the underground mine will also be stored in the underground and the remaining 2.52 Mt (60%) will be deposited in Brucejack Lake, predominantly during the construction phase. Similarly, approximately 8.55 Mt (47%) of the flotation tailings will go to the underground workings as paste backfill and 9.52 Mt (53%) will be piped as fluidized tailings to the bottom of Brucejack Lake.

The ARD/ML assessment of different materials involved:

- static tests (acid base accounting (ABA) and elemental analysis)
- mineralogical analyses (optical mineralogy, X-ray diffraction (XRD) and scanning electron microscopy (SEM))
- shake flask extracts (SFEs)
- kinetic tests (humidity cells, subaqueous columns and field barrels).

The following sections provide a summary of the ARD/ML assessment of the different materials. An in-depth discussion of the Brucejack ARD/ML assessment is provided in BGC (2014).

Waste Rock

The waste rock material was distributed across seven spatial geological model units, identified by Pretivm through drill core logs and photographs. Ranked in order of decreasing contribution to the waste rock volume produced by underground mine operations, the seven units are: Volcanic Sedimentary Facies (VSF) – Fragmental – Conglomerate – Bridge P1 – Office P1 – Silicified Cap – P2. Each geological model unit was shown to contain many unique rock types. For the purposes of simplification and identification of lithology-specific trends, samples were also reorganized into the following nine lithology groups: felsic volcanics, meta-sediments, conglomerate, andesite, mafic volcanics, porphyry, siliceous rocks, granitoids, and metavolcanics.

A large number (N=428) of collected waste rock samples were submitted for ABA and elemental composition tests. Paste pH values are circumneutral to alkaline and closely related to the presence of carbonates. Most of the total-S constitutes sulphide-S with only a small contribution from sulphate-S. Insoluble-S can amount up to 12-20% of the total-S and likely represents highly insoluble sulphate minerals (barite, anglesite) or elemental S. A comparison between Sobek neutralization potential (NP) and Carbonate NP values indicates carbonates are the main NP contributors in Brucejack waste rock. Comparison of median neutralization potential ratio (NPR) and net neutralization potential (NNP) values indicates Office P1 is the only geological model unit with a significant excess of NP.

Frequency analyses of NPR values of waste rock samples from the various geological model units show that Office P1 is the only unit that contains predominantly (84%) non-PAG waste rock. The waste rock from the other geological model units is characterized as predominantly (77%-100%) PAG material. The VSF, Fragmental and Conglomerate units account for 84.8% of the total generated waste rock and contain 76.5 to 85.0% PAG material. The three geological model units constitute 86.7% and 87.6% of the waste rock destined for the underground mine and Brucejack Lake, respectively.

ABA data for each lithology group show that paste pH values of the most lithology groups are generally circum-neutral to alkaline. Sulphide is the main sulphur species (greater than 90%), whereas sulphate-S and insoluble-S (less than 10%) make up relatively small components of the total-S content of most samples. Plots of neutralization versus acid potentials (i.e. NP vs AP) show that most of the samples (66 to 92%) from each lithology group are PAG, with the exception of mafic volcanics which are dominantly (83%) non-PAG. Similar to geological model trends, carbonates are the dominant buffering mineral and several logged lithologies contain iron carbonates, shown by higher CaNP versus Sobek NP plots and mineralogical analyses.

The total elemental concentrations of waste rock from geological model units and lithology groups were compared to concentrations in non-mineralized rock (i.e., continental crust). Geological model units and lithology groups were considered to have a significant elemental enrichment if the median values of the elemental concentrations in waste rock were 10 times higher than continental crust values. Significant enrichment was noted for silver, arsenic, cadmium, molybdenum, lead, antimony, selenium, and zinc in waste rock from most geological model units and from most lithology groups. Similar to frequency analyses results pertaining to Silicified Cap units, the siliceous rock lithology (including siliceous veins and breccias) present the highest metal enrichment for several metals (i.e., silver, antimony, cadmium, and molybdenum).

A total of 36 humidity cells were implemented and leachate data was collected from test periods ranging from 26 weeks to 61 weeks. Leachates from the Fragmental and Silicified Cap units have the lowest average pH, as some of these humidity cells produced acidic pHs within months of initiation. For all other geological model units the average pH is circum-neutral. Relative elevated metal concentrations of arsenic, cadmium, copper, lead, selenium, and zinc are often observed in Fragmental and Silicified Cap leachates. Oxyanions (arsenic, antimony, and selenium) show a different leaching behavior than transition metals, as they are more mobile at neutral pH conditions.

The shortest lag times (less than 15 years) in the humidity cell (HC) tests are estimated for waste rock from the Fragmental (3 of 10 HCs), VSF (2 of 13 HCs), Silicified Cap (1 of 2 HCs) and Conglomerate (1 of 3 HCs) units. Materials with the shortest lag times typically have paste pH values below 7 and very low NP values (5 to 15 kg calcium carbonate per tonne) combined with high sulphide-S values (3 to 8%). These materials weather readily and quickly.

Material from the geological model units representing the largest expected waste rock volumes (according to current mine plans) were used in the two subaqueous columns; Fragmental (andesite) and VSF (felsic volcanics). Leachate results from these columns show several metals with elevated concentrations (arsenic, antimony, molybdenum, selenium, and zinc), based on median values.

Eight field barrels were constructed of five (of seven) geological model lithologies; Fragmental, VSF, P2, and Bridge P1. Among those elements with stable or decreasing leachate concentrations trends, arsenic, and selenium are the only solutes with concentration values above maximum CCME guidelines and are highlighted as possible parameters of concern (POCs). The metals copper, cadmium, and zinc also show increasing trends and are considered POCs for waste rock drainage in unsaturated conditions.

Flotation Tailings, Paste, Sludge and Ore

In contrast to the ore and sludge, the acidifying potential (AP) of the tailings and paste is insignificant (AP less than2 kg calcium carbonate per tonne). The Sobek NP of the samples varies between 40 and 90 kg calcium carbonate per tonne, with the exception of high-grade ore and cemented paste materials that show the highest neutralization potential (165 kg calcium carbonate per tonne) and the highest carbonate contents. Ore and sludge materials are the only samples with NPR values less than two and are characterized as PAG materials, whereas tailings samples and paste samples are non-PAG materials.

The elemental composition of ore, tailings, paste, and sludge shows enrichments of silver, arsenic, cadmium, molybdenum, antimony, manganese, and possibly selenium, relative to non-mineralized rock. Copper and zinc are selectively enriched in ore and ore and sludge materials (respectively). Tailings and paste show enrichments of chromium and nickel and sludge shows the highest concentrations of cadmium, lead, and zinc (relative to tailings, paste, and ore).

The tailings material was tested in two humidity cells (T1 and T2). Parameters of concern in the leachates from these humidity cells are identified as arsenic, antimony, molybdenum, and selenium. The concentrations of these oxyanions maintain relatively high values throughout the test. As per tailings static test results that show extremely low AP values, it is evident the AP will be depleted prior to the exhaustion of its neutralization potential. Therefore tailings material used in humidity cell tests is not likely to generate ARD.

The tailings material was also tested in two subaqueous columns (Column 3 and 5). Similar to the leachate chemistry from tailings in the humidity cells, concentrations of arsenic, antimony, molybdenum, and selenium are often elevated in the leachate from the columns. Similar to the humidity cells, estimated lag times of the subaqueous columns are such that they will never generate ARD conditions.

Surface Materials

Twenty-three of the 40 samples from the plant site present NPR values less than 2.0 and are characterized as PAG material. The majority of access road samples has NPR values greater than 2.0 and is generally considered non-PAG materials. Shale material poses the greatest risk to ARD as over half of the shale samples show NPR values below 2.0. Samples taken from the aerodrome and quarry are characterized as non-PAG.

The metals arsenic, silver, mercury, and antimony are found at high concentrations in almost all lithologies sampled. The metals molybdenum, lead, zinc, and cadmium are found at moderate to high concentrations in many samples while only a few lithologies show an exceedance of chromium and nickel.

Concluding Remarks

The preceding discussion of the ARD/ML assessment indicates that waste rock, ore and sludge is primarily PAG material while tailings and paste materials are non-PAG material. Except for the plant site area and shale samples, most of the surface materials are non-PAG materials. Although ARD and ML processes can be significant under optimal weathering conditions, these processes are reduced in a subaqueous environment. The selected management strategies include the subaqueous disposal of waste rock and tailings (with sludge) in Brucejack Lake and storage of waste rock and tailings (as paste backfill) material in the underground workings. These strategies should prevent the potential for ARD, thereby reducing the potential for ML at the site. Although ML is reduced by the selected management strategies, its potential impact on the downstream receiving environment will be evaluated by water quality predictions outlined in Section 20.1.5.

20.1.5 Water Quality

Site-specific water quality models were developed by Lorax Environmental Services Ltd. to evaluate the potential effects of mining activities on surface water quality at the Property. Water quality modelling was conducted using GoldSim, an environmental and engineering mass balance program, based on source terms and water balance information developed by BGC. Water quality predictions covering construction, operations and post-closure mine phases were generated for two locations: Brucejack Lake and the proposed attainment location (monitoring station BJ200mD/S).

Water quality models developed incorporate the effects of several site-wide water quality control measures including:

- a WTP which will treat contact water from the plant site and the underground
- a sewage treatment plant where treated water will be discharged to Brucejack Creek during construction and to Brucejack Lake during operations
- disposal of tailings and waste rock into Brucejack Lake which will act as a natural sedimentation pond where any TSS remaining in suspension will be managed using sediment control curtains.

Hydrodynamic modelling of Brucejack Lake was carried out by Lorax (2013) to examine the likelihood of the migration of tailings solids into lake surface waters. The results indicated that the potential for elevated TSS levels in surface waters was unlikely if the minimum particle diameter was greater than or equal to 5 µm.

Water quality results were compared to Metal Mining Effluent Regulations (MMER) and BC Ministry of Environment (MOE) Water Quality Guidelines (WQGs) for the Protection of Aquatic Life. Base case water quality predictions representing expected flow conditions and average source term values indicate that MMER guidelines will not be exceeded during any of the mine stages and that exceedances of BC WQGs at the attainment location are limited to silver, arsenic, cadmium and zinc. Science-Based Effects Benchmarks (SBEBs) have been proposed for nitrite, silver, copper, lead, cadmium and zinc in order to reflect the naturally-elevated background concentrations. No exceedances of SBEB levels are predicted for these parameters during any of the mine phases, with the exception of arsenic. In the case of As, the predicted exceedances of BC WQGs are relatively minor, but predicted to occur in all mine phases. Although there are no BC WQG limits defined for P within creek systems, water quality predictions indicate that levels within Brucejack Creek will be elevated during the high flow months.

A number of sensitivity cases were modelled in order to assess how water quality predictions vary in response to variations in background flow conditions, groundwater seepage through the underground working and geochemical source terms. Based on the sensitivity case model results, extreme background flow conditions (1 in 100 year events) had minimal impact on water quality predictions, whereas groundwater seepage in the underground had a significant impact on water quality predictions. Based on scenarios considered, the high hydraulic conductivity + high recharge scenario (high seepage flow in the underground) represents the conservative worst case water quality condition. Water quality predictions associated with this scenario led to BC WQG exceedances of chromium and zinc, in addition to the parameters described above.

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).

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 area 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 for 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 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 in the area is minimal and seasonal in nature.

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, 200-year return period rain-on-snow event (220 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 9,600 m^3/d. The system will be scalable such that additional units can be added if required

Average annual groundwater seepage into the underground workings is expected to vary from approximately 3,840 to 6,240 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 or downstream of the lake into Brucejack Creek.

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)

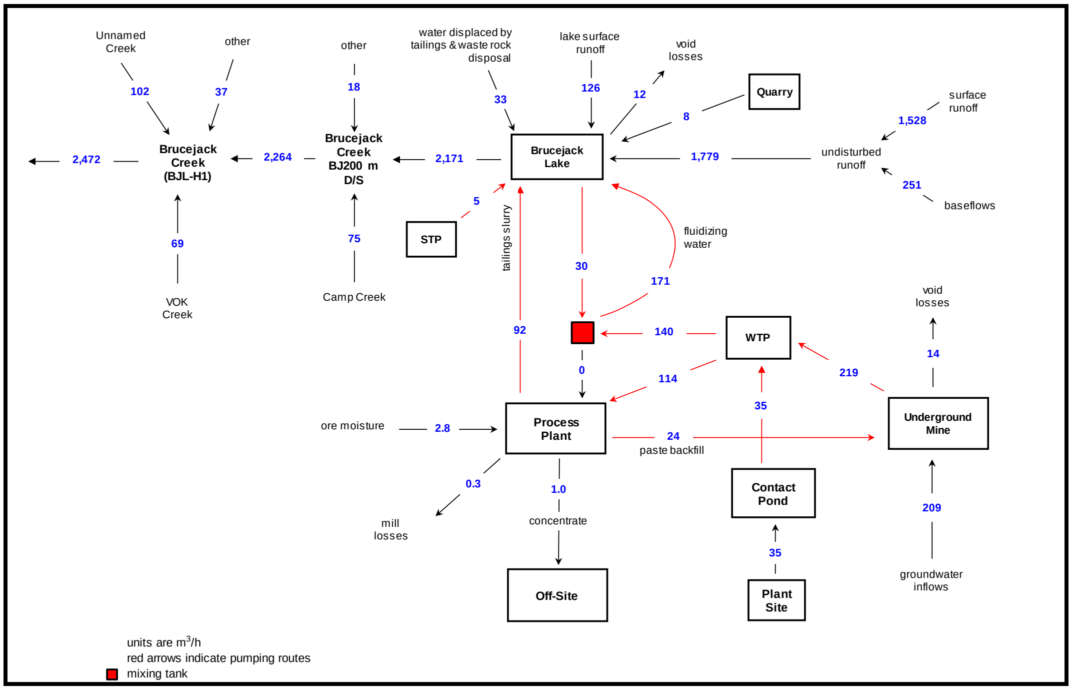
  - 1,245 t/d (43.5% of total production including 5 to 6% bonder) will be deposited in the underground mine in a backfill paste of 65% solids by weight (670 m^3/d of slurry water)

- an average mill loss of about 7 m^3/d
- an average annual precipitation of 1,900 mm and potential lake evaporation and sublimation losses of 167 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/h) 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/m^3 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)



An average annual outflow of 2,472 m³/h from Brucejack Lake has been estimated for the LOM, an average increase of approximately 6.4% above existing conditions (2,324 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 MEM 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. Progressive reclamation will occur during the operation phase. Closure of the mine site at the end of operations 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 material supplies and mobile equipment such as ventilation fans and safety equipment. These will be removed from the site for reuse or will be recycled. 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 removed.

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 water table is not expected to reach the two new mine portals but may occur in the exploration portal. Any seepage water from the existing portal will be monitored post-closure and will be 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. Concrete foundations will be broken up and possibly used as backfill. The above-ground pipes that carry tailings to the lake and the turbidity curtain will be removed and disposed of off-site.

Pad and road surfaces will be ripped to increase water infiltration and reduce the potential for surface erosion and instability. Some soils will be salvaged during the construction phase and will be used to reclaim the pad surfaces. Any areas receiving soils will be re-vegetated using a native seed mix.

The transmission line will be dismantled. The steel poles and the conductors 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 demolition 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 using native seed.

All structures at the Bower Aerodrome, the Knipple Transfer area, and the Tide Staging area will be similarly closed. The footprint areas will be ripped and any soils salvaged for reclamation purposes will be spread and seeded using a native seed mix. The Tide Staging area will be partially reclaimed once the transmission line is constructed. During the closure phase, it will be needed as a staging area to transport the poles and conductors of the dismantled transmission line. Final closure and reclamation of the Tide Staging area will be carried out as soon as the site is no longer required. The cost to dismantle the above structures and to close the various facilities and reclaim the disturbed areas 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 MEM 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 Section 10 and Section 11 orders 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 June 2014. Provincial and federal decisions on the EA process are expected in Q2 2015. 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 20.2 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*

table continues...

BC Government Permits and Licences	Enabling Legislation
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 20.3 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	*Navigation Protection Act*
Navigable Water: sub-aqueous disposal of waste rock and tailings	*Navigation 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. 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 US$746.9 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 (US$ million)
11	Mine Site	21.5
21	Mine Underground	179.5
31	Mine Site Process	53.8
32	Mine Site Utilities	30.5
33	Mine Site Facilities	53.5
34	Mine Site Tailings	3.5
35	Mine Site Temporary Facilities	33.4
36	Mine Site (Surface) Mobile Equipment	14.6
84	Off Site Infrastructure	89.1
Subtotal Direct Costs		**479.4**
91	Indirect Costs	127.5
98	Owner's Costs	71.0
99	Contingency	69.0
Total Initial Capital Costs		**746.9**

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 provide feasibility-level input to the Project financial model.

CLASS OF ESTIMATE, DEGREE OF PROJECT DEFINITION AND ACCURACY

This estimate is a Class 3 feasibility cost estimate prepared in accordance with the standards of the AACE. The estimated degree of project definition is 30%. The accuracy of this estimate is -15%/+20%. There has been 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 2014 and does not include any escalation beyond this date. The quotations used for this feasibility study estimate were obtained in Q2 2014 and have a validity period of 90 days.

21.1.3 Estimate Approach

Currency and Foreign Exchange

The capital cost estimate uses US 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$0.92
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 US dollars using the Project 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 Responsibility Matrix

A team of engineers, procurement specialists, and cost estimators from the following companies contributed to the development of this capital cost estimate:

- Tetra Tech – processing, infrastructure, mine site facilities, mine site temporary facilities, Knipple Transfer Station, borrow quarry, overall preparation of the capital cost estimate
- AMC – underground mining (preproduction, equipment, infrastructure) including mine capital and operating cost estimates
- Aran – paste backfill plant

- ERM Rescan – environmental aspects including closure and tailings delivery system design
- BGC –waste rock disposal and water management
- Valard – off-site transmission line
- BC Hydro – BC Hydro System Upgrade
- EBA – airstrip
- Allnorth – off-site access road and crossings
- Alpine Solutions – avalanche hazard assessment.

21.1.5 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.

21.1.6 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 US$479.4 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 US$127.5 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 that supervises one or more general contractors. Typical items included in the Owner's costs are allowances for home office staffing, home office travel, home office general expenses, field staffing, field travel, general field expenses, and Owner's contingency. Tetra Tech and the Project Owners team reviewed the indirect costs and Owners costs in great detail. Due to the project execution strategy, some costs that are typically carried in "indirects" have been moved to Owner's costs thus the Owner's costs for the Project may appear significantly higher (on a percentage basis) than a "typical" project. The list of Owners costs is exhaustive and includes the Project team costs (home office, facilities in Stewart, mine management team, process group, and finance group), site services group (surface, transfer station, Wildfire Camp, Knipple transfer station, transport vehicles and spare parts), all logistics (incoming freight to Stewart, laydown and tranship across glacier and laydown at mine), IT, health and safety, travel, catering, training, community relations and environmental. Adding to the Owner's costs are allowances made for the site location (equipment and materials for transportation of all materials and personnel to site across the glacier by specialized tracked vehicles), purchase of construction cranes by Owner for later use in operations and allowances for snow removal and transportation of all waste off-site.

The total Owner's cost for the Project is estimated to be US$71.0 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 US$69.0 million.

21.1.7 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.

Design Basis

The following items were referenced during preparation of this estimate:

- equipment list and process flow sheets
- 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
- costs for pre-production mining and mining equipment
- quantities and costs for tailings delivery system, water management, waste rock disposal and geotechnical design

Planning Basis – Execution Strategy and Schedule

The project execution plan (PEP) including scheduling has been considered in the preparation of this capital cost estimate.

This capital cost estimate is based on key milestones dates found in the PEP (refer to Section 24.0).

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 WBS 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 Cdn$117/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 tradesmen, uncertified tradesmen, skilled labourers, and helpers.

The blended labour rate of Cdn$117/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, safety briefings, etc.)
- supervision
- overhead and profit.

Travel and a living-out allowance have been calculated separately and are included in the construction indirect costs section.

PRODUCTIVITY FACTOR

A productivity factor of 1.34 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
- project supervision labour relationship
- job conditions
- construction equipment and tools
- weather
- level of estimate detail.

21.1.8 CAPITAL COST EXCLUSIONS

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

- working or deferred capital
- financing costs
- refundable taxes and duties
- land acquisition
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment , materials, labor or services resultant from a change in project schedule

- warehouse inventories other than those supplied in initial fills
- any project sunk costs (studies, exploration programs, etc.)
- sustaining capital costs (included in the financial model)
- mine reclamation costs (included in financial model)
- mine closure costs (included in financial model)
- escalation costs
- permitting costs
- community relations.

21.2 OPERATING COST ESTIMATE

21.2.1 SUMMARY

The total LOM average operating cost for the Project is estimated at Cdn$163.05/t ore milled, which includes costs for:

- mining
- process
- 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, or other government imposed costs, unless otherwise noted.

A total of 593 personnel are projected to be required for the Project, including an average of 351 personnel for mining operations, 100 personnel for process, 54 personnel for G&A, 78 personnel for surface services, and 10 personnel for the backfill plant and water treatment plant. Less mining personnel requirement is expected in the initial two years and after Year 12. The estimated mining personnel required in Year 18 is approximately 264 personnel.

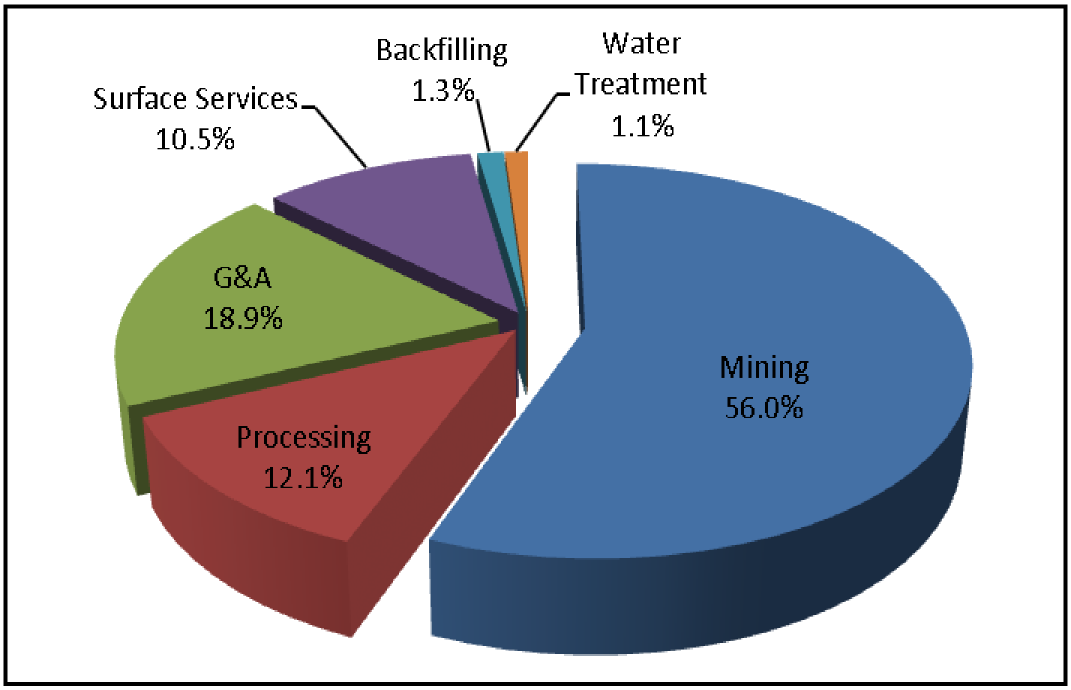
The unit cost estimates are based on the LOM ore production and a mine life of 18 years. The currency exchange rate used for the estimate is 1:0.92 (Cdn$:US$). The operating cost for the Project has been estimated in Canadian dollars within an accuracy range of ±15%. A summary of the overall LOM average 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 (Cdn$/t milled)
Mining*	351**	91.34
Processing	100	19.69
G&A	54	30.87
Surface Services	78	17.18
Backfilling	6	2.11
Water Treatment	4	1.86
Total	**593**	**163.05**

Notes: *Average LOM mining cost including crushing cost, cement cost for backfill and back-hauling cost for the preproduction ore stocked on the surface; if excluding the ore mined during preproduction, the estimated unit cost is Cdn$91.78/t.
**351 workers during Years 3 to 12 and less mining personnel requirement is estimated for the rest of the operation years.

Figure 21.1 Overall Operating Cost Distribution



21.2.2 Mining Operating Costs

The total underground mine operating cost is estimated to be Cdn$1,512 million, or Cdn$91.34/t of ore mined, and includes costs for all underground mine operating activities beyond the pre-production period (Years 1 to 18). The underground mine operating cost is marginally less than the underground mine operating cost reported in the 2013 Brucejack Feasibility Study (Ireland et al. 2013) (Cdn$94.40/t) due to revised production slashing requirements, revised stoping costs in the Brucejack Fault Zone, and lower general mine costs (definition drilling, technical services budget, etc.).

The estimation of the underground mine operating cost is based on a first principles work-up of all key mining activities, using the following inputs:

- mine design and method
- production and development schedule
- waste and backfill schedule
- power, heating and ventilation requirements.

The key mining activities are listed below; associated cost areas include labour, equipment, consumables (ground support, explosive, fuel consumption, etc.), ventilation, dewatering, and mine services:

- waste development –waste cross cut development for stope access
- ore development – ore cross cut development and full-width production slashing for long hole stopes
- long hole stoping
- backfill
- mine general
- mine maintenance.

Manpower requirements and labour rates (including benefits and burdens) were developed in consultation with Pretivm and benchmarked against similar projects in the region. The manpower build-up is estimated based on planned mine development and production rates, mine activities, and mobile equipment quantities. A contract workforce is proposed to complete all underground mine development and operating activities during the pre-production period (Years -1 and -2). Unit rates, including labour, in Years -1 and -2 include a contractor premium. Mine management, technical services, and mine maintenance personnel will be an Owner workforce from the beginning of the pre-production period. After the initial pre-production years, all underground mine development and operating activities (apart from vertical development and diamond drilling) are to be undertaken by an Owner workforce and hence, no contractor premiums apply in these years (Years 1 to 18). The Owner workforce will be phased-in during 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 and this is accounted for in the cost estimate through increased labour quantities. Operational personnel build-up occurring in Year -1 (a percentage of the full personnel complement in Year 1, in addition to the personnel required in Year -1) includes:

- technical services and supervisory: 45% of full complement
- production crew: 15% of full complement
- development crew: 35% of full complement
- maintenance crew: 20% of full complement.

As part of the handover, one contractor mining crew consisting of 12 personnel per shift will remain onsite for six months following mine production start-up (Year 1). In this year, the contractor crew is accounted for in the cost estimate as an operating expense and their unit rates include a contractor premium.

Equipment selection and operating costs are based on mine activities and workups of required operating hours, productivity/cycle times, availability, and projected usage. The operating costs for all major mobile equipment have been obtained from supplier quotes and benchmarked against AMC's database of costs for similar recent projects.

The unit costs of all major consumables including explosives, ground support, pipes, and ventilation ducting have been obtained from current supplier quotes.

Mine general costs include mine power, heating, material handling (crushing and conveying), labour (mine management and technical services), technical services consumables, definition drilling, personal protective equipment (PPE), and support equipment costs.

Mine maintenance costs include maintenance labour, fixed plant and electrical maintenance, maintenance overheads, and shop consumables. Mine general and mine maintenance costs are developed based on industry experience and benchmarking.

Mine Operating Costs

The LOM underground operating cost estimate is presented in Table 21.4.

Table 21.4 LOM Underground Operating Costs by Activity

Activity	Total Cost (Cdn$ million)	Unit Cost (Cdn$/t of ore)	Percentage of Total (%)
Waste Development	116.3	7.03	7.7
Ore Development	144.7	8.74	9.6
Long Hole Stoping	507.2	30.65	33.6
Backfill	198.4	11.99	13.1
Mine Maintenance	270.5	16.35	17.9
Mine General	274.5	16.59	18.2
Total	**1,511.7**	**91.34**	**100.0**

The breakdown of mine general costs is presented in Table 21.5. Key consumable costs include:

- power: Cdn$0.053 /kWh at full production and Cdn$0.35/kWh during construction
- propane: Cdn$0.650/L
- diesel: Cdn$1.179/L.

Power consuming items include primary and secondary ventilation fans, mobile drilling equipment, electric scoops, dewatering pumps, workshop services, and underground crusher and conveyors. Propane heating costs are based on the scheduled total mine airflow, which is derived from the planned mine development and production rates. Crushing and conveying operating costs were estimated based on benchmarking data from "Mine & Mill Equipment Costs An Estimator's Guide" (InfoMine 2011). Support equipment costs were derived from the expected usage to complete general mine tasks and unit operating rates supplied by vendors.

Table 21.5 Underground Operating Costs – Mine General Area

Area	Total Cost (Cdn$ million)	Unit Cost (Cdn$/t of ore)
Power	34.6	2.09
Heating	5.1	0.31
Crushing and Conveying	10.6	0.64
Support Equipment	23.0	1.39
Labour – Mine Management	23.8	1.44
Labour – Technical Services	87.5	5.28
Technical Services Consumables	10.9	0.66
Mine General Consumables	49.4	2.99
PPE	3.3	0.20
Freight	13.8	0.83
Definition Drilling	12.5	0.76
Total	**274.5**	**16.59**

Table 21.6 shows the totals per year for the underground mine operating costs over the LOM.

Table 21.6 Annual Underground Mine Operating Costs (Cdn$ million)

Year	Waste Development	Ore Development	Long Hole Stoping	Backfill	Mine Maintenance	Mine General	Total
1	11.6	9.1	24.3	12.4	15.1	16.2	88.7
2	8.4	10.1	24.7	11.1	15.8	15.4	85.6
3	7.5	7.8	26.5	11.2	16.0	15.7	84.7
4	7.2	9.3	26.8	11.0	16.0	15.8	86.1
5	7.0	7.6	30.5	11.6	16.2	16.3	89.1
6	8.3	8.4	32.1	10.8	16.2	16.6	92.3
7	9.1	12.1	32.2	10.8	16.2	16.7	97.1
8	8.1	10.2	32.0	11.0	16.2	16.5	94.1
9	7.1	7.5	30.3	10.9	16.2	16.4	88.4
10	6.1	4.8	28.3	11.2	16.0	15.9	82.3
11	7.6	8.5	27.5	10.0	16.0	16.0	85.5
12	6.8	13.0	28.7	10.8	16.0	16.0	91.4
13	2.6	9.4	31.0	12.1	15.6	15.4	86.2
14	4.1	6.4	32.0	13.2	15.6	15.4	86.7
15	5.0	5.7	32.0	12.8	15.0	15.4	86.0
16	4.1	7.1	31.9	12.2	14.9	15.1	85.3
17	4.2	5.3	25.1	10.3	11.8	13.0	69.8
18	1.3	2.4	11.3	5.0	5.8	6.5	32.3
Total	**116.3**	**144.7**	**507.2**	**198.4**	**270.5**	**274.5**	**1,511.7**

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 Cdn$19.69/t milled or Cdn$18.1 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 16,549,680 t and a mine life of 18 years, or an average annual process rate of 919,400 t.

Table 21.7 Summary of Process Operating Cost

Description	Labour Force	Annual Cost (Cdn$)	Unit Cost (Cdn$/t milled)
Labour Force			
Operating Staff	21	2,475,000	2.69
Operating Labour	40	3,291,200	3.58
Maintenance Labour	39	3,721,700	4.05
Subtotal Labour Force	100	9,487,900	10.32
Major Consumables			
Metal Consumables	-	2,812,500	3.06
Reagent Consumables	-	856,300	0.93
Subtotal Major Consumables	**-**	**3,668,800**	**3.99**
Supplies			
Maintenance Supplies	-	1,946,500	2.12
Operating Supplies	-	925,500	1.01
Power Supply	-	2,071,100	2.25
Subtotal Supplies	**-**	**4,943,100**	**5.38**
Total (Process)	**-**	**18,099,800**	**19.69**

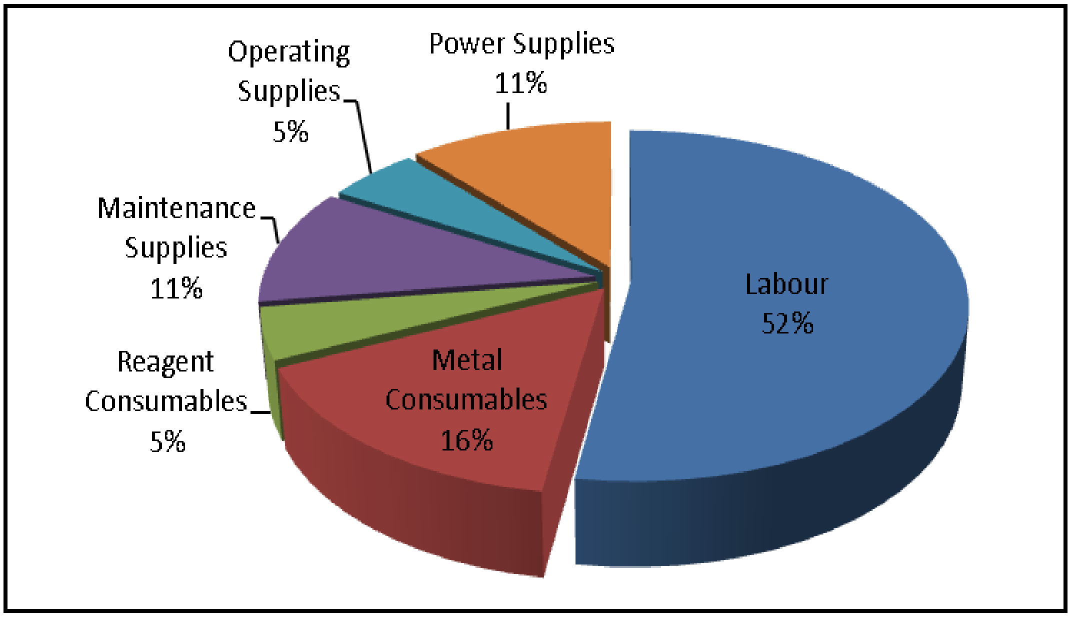
All process operating costs are exclusive of taxes, permitting costs, or other government imposed costs unless otherwise noted. The following items have been included in the estimate:

- 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 approximately 30.4% 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 process equipment capital costs.
- laboratory supplies, building maintenance and other costs, based on Tetra Tech's in-house database and industry experience; the assay including the samples from geological and mining department.
- 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 Cdn$10.32/t milled. A total of 100 personnel are estimated for the process operation, including 21 staff for management and professional services, 40 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 Cdn$3.06/t milled. The metal consumables include mill liners, and mill grinding media.

The estimated reagent cost is Cdn$0.93/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 Cdn$2.12/t milled while the operating supply cost is Cdn$1.01/t milled.

The power cost is estimated at Cdn$2.25/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 Cdn$0.053/kWh.

21.2.4 BACKFILLING OPERATING COSTS

The estimated operating cost for the backfilling plant is Cdn$2.11/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 (Cdn$)	Unit Cost (Cdn$/t milled)
Labour Force			
Operating Labour	4	360,000	0.39
Maintenance Labour	2	195,600	0.21
Subtotal Labour Force	**6**	**555,600**	**0.60**
Major Consumables (Included in Mining Operating Cost)			
Supplies			
Maintenance Supplies	-	326,500	0.36
Operating Supplies	-	779,200	0.85
Power Supply	-	275,400	0.30
Subtotal Supplies	**-**	**1,381,100**	**1.51**
Total (Backfilling)	**-**	**1,936,700**	**2.11**

The estimated labour force cost is Cdn$0.60/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 Cdn$0.36/t milled while the operating supply cost is Cdn$0.85/t milled. The power cost is estimated at Cdn$0.30/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 Cdn$1.87/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 (Cdn$)	Unit Cost (Cdn$/t milled)
Labour Force			
Operating Labour	4	276,400	0.30
Subtotal Labour Force	**4**	**276,400**	**0.30**
Major Consumables			
Reagent Consumables	-	1,147,400	1.25
Subtotal Major Consumables	**-**	**1,147,400**	**1.25**
Supplies			
Maintenance Supplies	-	180,000	0.20
Operating Supplies	-	10,000	0.01
Power Supply	-	100,500	0.11
Subtotal Supplies	**-**	**290,500**	**0.32**
Total (water treatment)	**-**	**1,714,300**	**1.87**

21.2.6 General and Administrative, and Surface Services

Tetra Tech and Pretivm developed the G&A costs, which are estimated to be Cdn$30.87/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
- site medical services
- 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 (Cdn$/a)	Unit Cost (Cdn$/t milled)
G&A Labour Force			
G&A	42	4,627,100	5.03
G&A Hourly Personnel	12	949,200	1.03
Subtotal Labour Force	**54**	**5,576,300**	**6.06**
G&A Expense			
General Office Expense	-	250,000	0.27
Computer Supplies Including Software	-	61,000	0.07
Communications	-	610,000	0.66
Travel	-	150,000	0.16
Audit	-	85,000	0.09
Consulting/External Assays	-	250,000	0.27
Environmental	-	375,000	0.41
Insurance	-	1,343,000	1.46
Regional Taxes and Licenses Allowance	-	401,000	0.44
ERP Purchase (SAP, etc.)		200,000	0.22
Legal Services		100,000	0.11
Warehouse		200,000	0.22
Recruiting; including Relocation Expense	-	500,000	0.55
Entertainment/Memberships	-	50,000	0.05
Medicals and First Aid Supplies	-	100,000	0.11
Medicals and First Aid Contract Services	-	702,000	0.76
Training/Safety		250,000	0.27
Accommodation/Camp Costs	-	7,547,300	8.21
Crew Transportation (Flight and Bus)	-	8,567,800	9.32
Liaison Committee/Sustainability/Sponsorships	-	200,000	0.22
Small Vehicles, included in Surface Services	-	-	-
Satellite Office - Smithers	-	716,000	0.78
Others	-	150,000	0.16
Subtotal Expense	**-**	**22,808,100**	**24.81**
Total	**54**	**28,384,400**	**30.87**

There are 78 personnel required to provide surface services. The total surface services cost is estimated to be Cdn$17.18/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, excluding satellite office operation in Smithers
- 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 (Cdn$/a)	Unit Cost (Cdn$/t milled)
Surface Service Labour Force			
Surface Service Personnel	78	6,806,200	7.40
Subtotal Labour Force	**78**	**6,806,200**	**7.40**
Surface Service Expense			
Potable Water and Waste Management	-	320,000	0.35
Supplies	-	150,000	0.16
Building Maintenance	-	300,000	0.33
Building Heating, Ventilations & Others Loads	-	1,259,000	1.37
Road Maintenance	-	1,440,000	1.56
Avalanche Control (Contract)	-	600,000	0.65
Power Line Maintenance	-	164,000	0.18
Mobile Equipment - Maintenance	-	1,400,000	1.52
Mobile Equipment - Fuel	-	2,499,000	2.72
Airstrip Instrumentation Maintenance	-	20,000	0.02
Helicopter Supports	-	540,000	0.59
Off-site Operation - Transfer Station	-	200,000	0.22
Others		100,000	0.11
Subtotal Expense		**8,992,000**	**9.78**
Total	**78**	**15,798,200**	**17.18**

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 18-year LOM and 16.55 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 34.7% IRR
- 2.7-year payback on the US$746.9 million initial capital
- US$2,251 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:

- 28.5% IRR
- 2.8-year payback on the US$746.9 million initial capital
- US$1,445 million NPV at a 5% discount rate.

As indicated in Sections 19.0 and 21.0 of this report, the base case metal prices (Section 19.0) and exchange rate (Section 21.0) used for this study are as follows:

- gold – US$1,100/oz
- silver – US$17.00/oz
- exchange rate – 0.92:1.00 (US$: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 US$320.62 million.

The mine closure and reclamation cost is US$27.46 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.

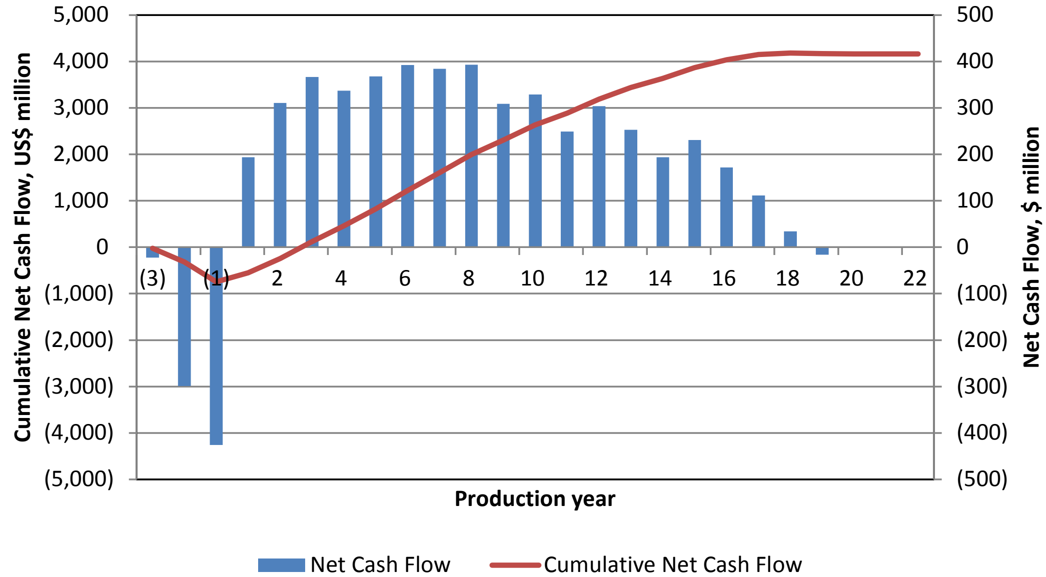
NPV was estimated at the beginning of the three-year engineering, procurement and construction period.

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)	9,762	16,550
Annual Tonnes to Mill ('000)	976	919
Average Grade		
Gold (g/t)	16.348	14.138
Silver (g/t)	11.250	57.686
Total Production		
Gold ('000 oz)	4,972	7,274
Silver ('000 oz)	2,996	27,626
Average Annual Production		
Gold ('000 oz)	497.178	404
Silver ('000 oz)	300	1,535

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
- higher prices case.

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	Higher Price
NCF	US$ million	4,160	2,020	6,353
NPV at 5.0% Discount Rate	US$ million	2,251	985	3,540
Project IRR	%	34.7	20.3	47.0
Payback	Years	2.7	4.4	2.0
Exchange Rate	US$:Cdn$	0.92	0.92	0.92
Gold Price	US$/oz	1,100	800	1,400
Silver Price	US$/oz	17.00	15.00	21.00

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	Higher Price
NCF	US$ million	2,724	1,344	4,134
NPV at 5.0% Discount Rate	US$ million	1,445	620	2,279
Project IRR	%	28.5	16.5	38.7
Payback	Years	2.8	4.5	2.1
Exchange Rate	US$:Cdn$	0.92	0.92	0.92
Gold Price	US$/oz	1,100	800	1,400
Silver Price	US$/oz	17.00	15.00	21.00

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

As referenced in Section 19.0 of this report, the following payment, smelting and refining terms are applied in the economic analysis:

- doré:
  - gold and silver – pay 99.8% of gold and silver content. A refining and transport charge of US$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 of US$184.00/dmt of concentrate is applied. A penalty charge of US$9.20 per each 0.1% of arsenic above 0.2% 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 TRANSPORTATION AND INSURANCE

Doré transportation cost is US$1.00/oz. Concentrate transportation cost is US$181.65/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 followed by operating costs, capital costs, exchange rate and silver price. 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

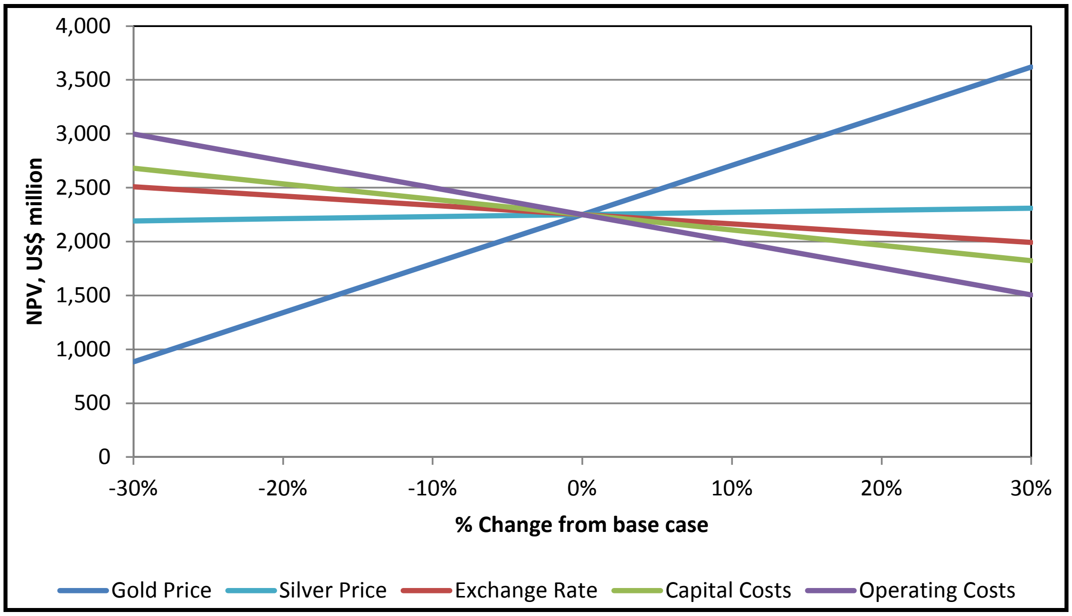


Figure 22.3 Figure 22.3 Pre-tax IRR Sensitivity Analysis

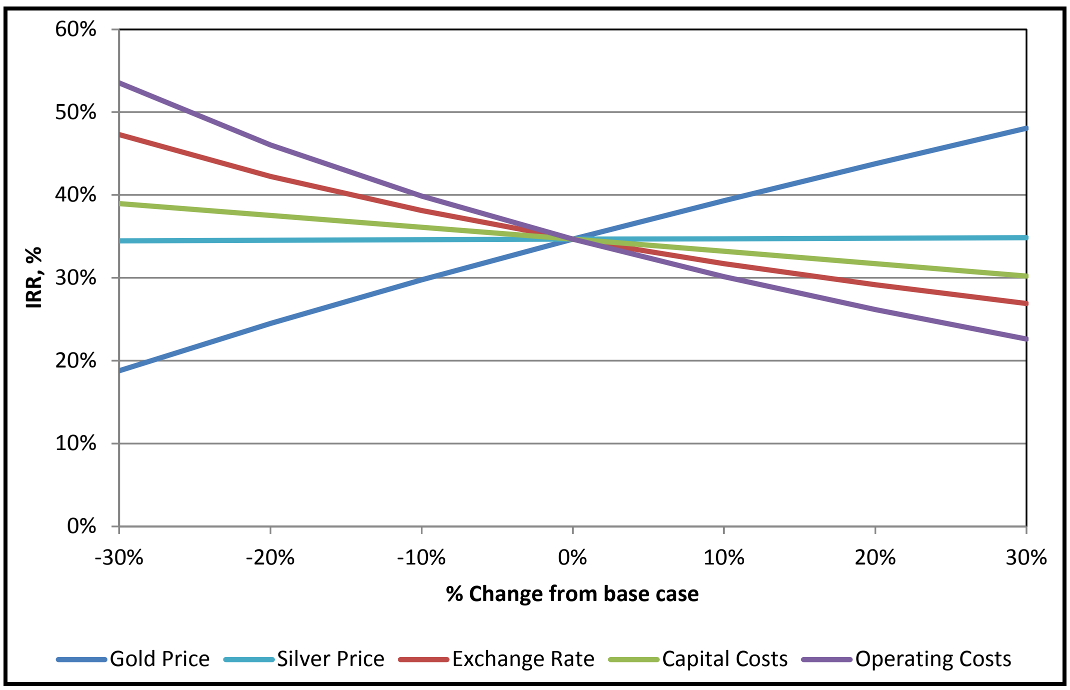
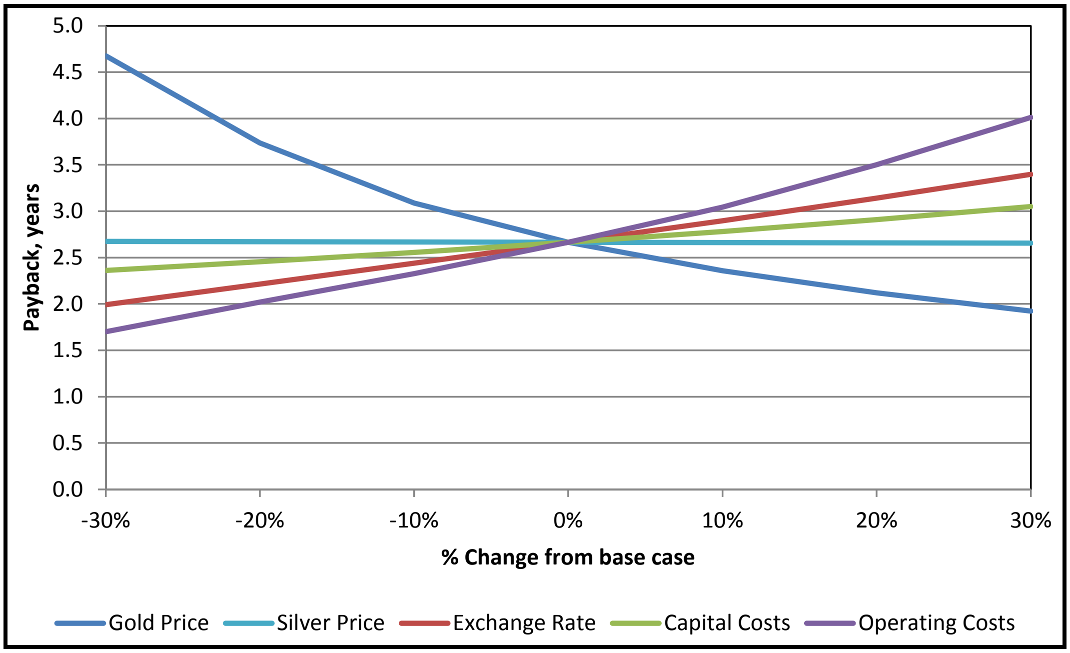


Figure 22.4 Pre-tax Payback Period Sensitivity Analysis



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 US$1,436 million over the 18-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 (US$ million)
Corporate Tax (Federal)	510.52
Corporate Tax (Provincial)	392.76
Provincial Mineral Taxes	532.83
Total Taxes	**1,436.11**

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:	11%
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. Changes from the 2013 Federal Budget phases 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:	This includes costs of land, exploration and mining rights, licenses, permits and leases. Costs are added to a Canadian development expense (CDE) pool, and can be deducted at up to 30% of the balance in 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 (CEE) 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. Pre-production mine development costs incurred subsequent to 2017 will be treated as CDE instead of CEE. The transition will be phased-in beginning in 2015, with 20% of costs being allocated proportionately to CDE and 80% to CEE in 2015, 40% to CDE and 60% to CEE in 2016, and 70% to CDE in 2017 and 30% to CEE 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, preproduction 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) account. 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 preproduction 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 133% of its capital expenditures incurred prior to commencing production to the CEA account if the mine began producing minerals in reasonable commercial quantities before January 1, 2016. The model is calculated on the assumption that as the mine will not commence production until 2017,

that the new mine allowance would not be available to the company.

BC mineral taxes are deductible for federal and provincial income tax purposes.

23.0 ADJACENT PROPERTIES

The following paragraphs describing adjacent properties are based on information which 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 for any of the described adjacent properties except against what has been publicly reported and 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 Gold published a revised prefeasibility study, which resulted in a Mineral Reserve of 2.2 Bt (2.2 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.

In February 2014, Seabridge Gold announced a new Mineral Resource estimate for their recently discovered Deep Kerr Zone on the KSM Property. The Deep Kerr Zone has been estimated as containing an Inferred Mineral Resource of 515 Mt, grading 053% copper and 0.36 g/t gold for 6.0 Blb of copper and 5.9 Moz of gold (using a $20/t NSR cut-off based on economic parameters discussed in their February 18, 2014 press release).

All information for this section has been taken from the Seabridge Gold website (www.seabridgegold.net).

Table 23.1 Mineral Reserve Estimates for the KSM Property as of December 31, 2012

Zone	Mining Method	Reserve Category	Tonnes (Mt)	Average Grades				Contained Cetal			
				Gold (g/t)	Copper (%)	Silver (g/t)	Molybdenum (ppm)	Gold (Moz)	Copper (Mlb)	Silver (Moz)	Molybdenum (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 used to report the Mineral Reserve Figures were defined based on NSR values and the mining method. 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). Several recent surface grab samples (UH-1 through UH-10) returned assays of 4.8 to 63 g/t gold, 18-86 g/t silver, 0.31-2.42% lead, and 1.4 to 6.5% zinc for mineralization hosted in quartz vein stockworks in pervasively altered chlorite-sericite andesitic volcanic and volcaniclastic rocks.

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.

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, including the Owner, 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 Project.

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.

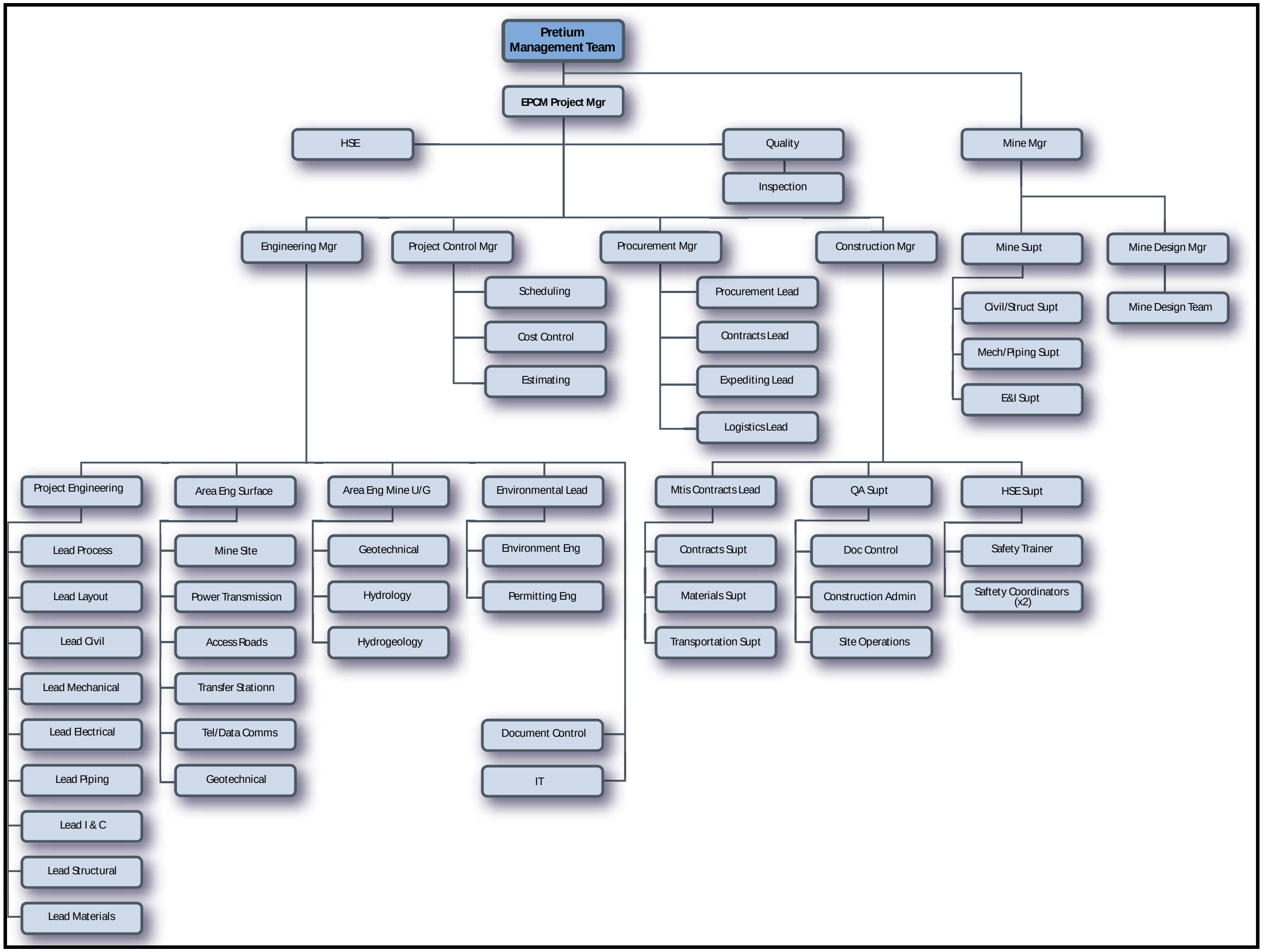
The Project will transition from the study phase to basic engineering in Q3 2014 and will move forward in the following phases:

- 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 and construction of the operating mine, process facilities, and infrastructure in order to provide a fully operational facility by Q3 2017.

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 26-month project construction duration assumes that field activities will commence in June 2015, and Production Start will occur in August 2017.

Table 24.1 Significant Activity Milestone Dates to Project Handover

Year	Quarter	Activity
2014	2	Feasibility Study Update Completion
2014	3	Start of Basic Engineering
2014	3	EPCM Award
2015	2	Start Stage I Early Works Construction
2016	1	Detailed Engineering Completion
2016	1	Start Stage II Full Project Execution Construction
2017	2	Surface Mechanical Completion
2017	2	Underground Mechanical Completion
2017	3	Mine Site Commissioning Completion
2017	3	Production Start

A Level 1 execution schedule is provided in Figure 24.2. A Level 2 schedule was developed during the feasibility study update.

Figure 24.2 Level 1 Execution Schedule

Activity ID	Activity Name	Original Duration	Start	Finish
	Project Major Milestones	254	02-Sep-14	01-Aug-17
	Project Milestones	254	02-Sep-14	01-Aug-17
ML2000	Start Detailed Engineering	0	02-Sep-14	
ML1300	Start Early Work Construction	0	01-Jun-15*	
ML2500	Mine Site Camp, Mill Pad, & Portal Area Site Prep complete	0		21-Sep-15
ML1900	Off Site Access Road Upgrade Complete	0		12-Oct-15
ML1920	Mine Site Operation Camp Complete	0		30-Oct-15
ML1910	Demobilize Above Ground Early Work Contractors	0		01-Nov-15*
ML2400	Start Stage-2 Full Project Execution Construction	0	01-Mar-16	
ML2100	Power Transmission Line & Substation Energized	0		12-Sep-16
ML1400	Above Ground Project Mechanical Completion	0		14-May-17
ML2510	Underground Mechanical Completion	0		31-May-17
ML1600	Start Commissioning	0	01-Jun-17	
ML2200	Complete Commissioning	0		01-Aug-17
	Project Summary Level Schedule	271	23-Jun-14	01-Aug-17
	Permits	231	23-Jun-14	29-May-15
PR1600	Environmental Assessment Process & Permits	231	23-Jun-14	29-May-15
	Engineering and Procurement	629	10-Jul-14	27-Jan-17
PO1000	Procurement for Major Equipment	419	10-Jul-14	22-Mar-16
DE1010	Mine Site Above Ground Detailed Engineering	355	02-Sep-14	11-Feb-16
PO1010	Major Equipment Fabrication and Delivery	550	03-Nov-14	27-Jan-17
KO1000	Contracts - Early Construction Works	105	27-Nov-14	04-May-15
	Stage-1 Early Construction Work	153	01-Jun-15	31-Oct-15
CN1000	Off site Access Road Upgrade	134	01-Jun-15	12-Oct-15
CN1010	Stage-1 Site Preparation	153	01-Jun-15	31-Oct-15
CN6110	Power Transmission Line Stage-1 Construction	120	01-Jun-15	28-Sep-15
CN6120	Knipple Transfer Station - Site Preparation & Camp	145	01-Jun-15	23-Oct-15
CN3500	Construction Facilities Development	137	11-Jun-15	25-Oct-15
CN3313	Mine Site Operation Camp Construction	99	24-Jul-15	30-Oct-15
	Stage-2 Full Project Execution	419	22-Mar-16	14-May-17
CN3310	Mill Building Erection	223	22-Mar-16	30-Oct-16
CN6115	Power Transmission Line Stage-2 Construction	110	05-May-16	22-Aug-16
CN1700	Process Plant Construction	369	11-May-16	14-May-17
CN6123	Knipple Transfer Station Building	117	20-May-16	13-Sep-16
CN3120	SAG Mill, Ball Mill, and Regrind Mill Installation	190	30-Sep-16	07-Apr-17
	Underground Mine Development	701	01-Jul-15	31-May-17
CN3000	Underground Mine Lateral Development	641	01-Jul-15	01-Apr-17
CN3010	Underground Mine Vertical Develpment	213	01-Aug-16	01-Mar-17
CN3020	Underground Infrastructure Installation	244	30-Sep-16	31-May-17
	Commissioning	62	01-Jun-17	01-Aug-17
CM1000	Commissioning	62	01-Jun-17	01-Aug-17
CM1100	Start Full Production	0		01-Aug-17

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.

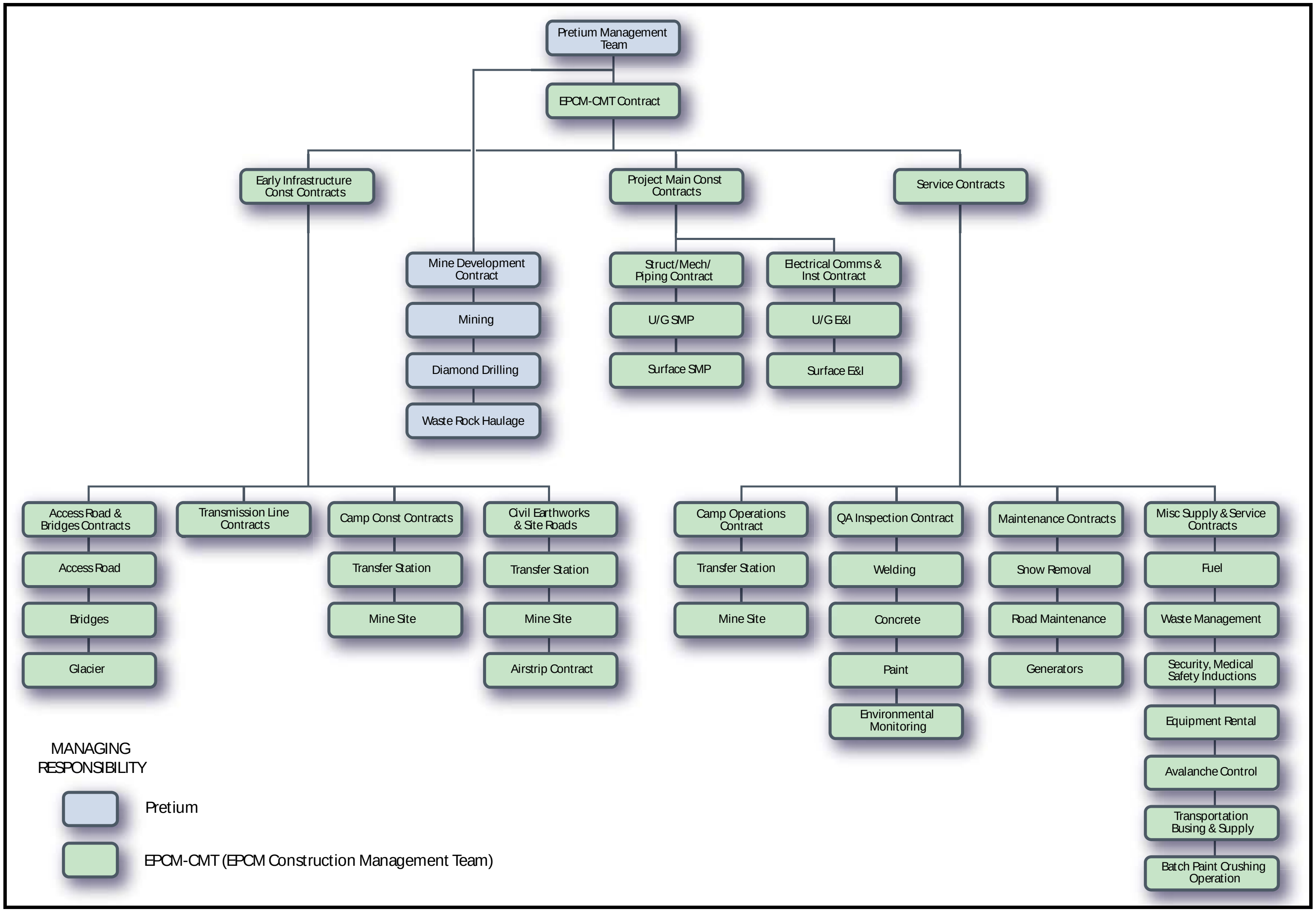
The EPCM contractor will prepare a plan for expediting equipment purchase orders based on the schedule and equipment list. Expediting will be coordinated by the EPCM contractor. Third-party resources may be used to inspect equipment being manufactured in various parts of the world. Purchase orders will 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 70-hour work week, with some double shifts for inside work as required. Crew rotations are planned to be scheduled as three weeks on-site and one week off-site.

There are approximately 2,056,000 man-hours of total construction labour associated with Project construction, excluding engineering. Construction manpower on site will peak at approximately 520 total construction workers, including the construction management team and Owners team.

24.1.7 Construction Camp

A 400-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 Knipple Transfer Station. 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 single occupancy 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 cranes will be certified for cold weather lifts below -20°C. 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 at the mine site will be provided by Pretivm's modular generators, the contractors will provide small portable generators for use in remote locations. 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 line power will be available by Q3 2016 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 update. The recommendations from this review will be implemented during the execution phase of the Project. Additional constructability reviews will be undertaken 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 Material Management System		X		
111800	Fire Alarm System		X		
111950	Mine Site Utilidor		X		
21	**Mine Underground**				
211100	Lateral Development (Capital)			X	
211200	Vertical Development (Capital)			X	
211300	Infrastructure Development (Capital)			X	
211350	Pre-Production Mine Power/Propane			X	
211400	Ventilation Shafts			X	
212	Underground Mining Equipment				
212100	Mobile Equipment			X	
212200	Fixed Equipment			X	
212250	Material Handling System			X	
212300	Ancillary, Survey and Tech Equipment			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, Systems and Mine Services			X	
213200	Underground Explosive Magazine			X	
213250	Mobile Refuge Chambers			X	
213300	Underground Backfill Distribution			X	
213350	Portal Entry and Infrastructure			X	

table continues...

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
213400	Controls and Instrumentation			X	
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	
213700	Canteen facilities			X	
213750	Safety Equipment Storage and Lamps etc.			X	
213800	Surface Works, Ventilation Fans, etc.		X	S	
213850	Backfill Piping and Distribution (Surface)		X	S	
213900	Underground Piping (Air, Water, Fuel)			X	
213950	Underground Fuel Storage and Distribution			X	
31	**Mine Site Process**				
311	Crushing				
311400	SAG Mill Feed Surge Bin and Conveyors		X		
312	Grinding, Flotation and Dewatering				
312200	Primary Grinding		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		
312800	Paste Plant		X		
312825	Slime Flotation		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		

table continues...

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
322300	Gasoline		X		
322400	Lube Oil		X		
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 and Collection Pond		X		
323600	Water Treatment Plant		X		
323800	Gland Water		X		
323900	Reclaim Water from Lake		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				
331050	Mill Building				
331300	Permanent Camp		X		
331700	Truck Shop		X		
331950	Crushing and Batch Plant		X		
331952	Surface Explosive Storage		X		
34	**Mine Site Tailings (Brucejack Lake)**				
341	Tailings				
341100	Tailings Delivery System		X		S
341200	Waste Rock Disposal		X		S
341600	Lake Outlet Control Structure		X		
341800	Environmental		X		
35	**Mine Site Temporary Facilities**				
351	Temporary Facilities				
351150	Temporary Cold Storage		X		
351200	Construction Catering		X		
351300	Construction Laydown Area		X		
36	**Mine Site (Surface) Mobile Equipment**				
361	Surface Mobile Equipment				
361100	Surface Mobile Equipment		X		

table continues...

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
61	**Off-site Infrastructure**				
611	Off-site Infrastructure				
611100	Off-site Power Transmission		X		
611300	BC Hydro Stewart Substation Upgrade		X		
612	Knipple Transfer Station				
612225	Knipple Scale		X		
612250	Knipple Helipad		X		
612255	Knipple Cold Storage		X		
612300	Knipple Transfer Station		X		
612400	Knipple Services and Utilities		X		
612500	Knipple Mobile Equipment		X		
613	Airstrip				
613700	Airstrip		X		
614	Borrow Quarry				
614800	Borrow Quarry		X		
615	Off-Site Access Road				
615	Off-Site Access Road		X		
616	Bowser Camp				
616	Bowser Camp (Existing)	X	S		
617	Avalanche Control				
617600	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		
913	Indirects Off-site Infrastructure				
913100	Off-site Construction Indirects		X		
913500	Off-site Commissioning and Start-Up		X		
913600	Off-site EPCM		X		
913700	Off-Site Vendor Commissioning and Assistance		X		
98	**Owners Costs**				
981	Owner's Costs	X			
981100	Owner's Costs	X			
981200	Owner's Risk and Contingency	X			

table continues...

WBS	Area	Pretivm	EPCM Team	Underground Mining Team	Geotechnical Team
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 high-, medium- and low-level risks associated with overall project execution including HSE risks, production risks, and risks that impact project cost.

- 49 high-level risks
- 135 medium-level risks
- 246 low-level risks.

The high-level risks were grouped and rationalized during the Feasibility Study update and the top ten risks are identified in Table 24.3.

Table 24.3 Top Ten High-level Risks

Threat/Hazard	Mitigation
• Availability of skilled labour, contractors, and equipment causing delays to the project schedule, project cost, and safety of workers.	• Develop a contracting strategy that will capture and retain a skilled workforce. • Conduct contractor prequalification to ensure contractors have sufficient skilled workforce for the project. • Review hiring and training policy to retain a skilled workforce.

table continues...

Threat/Hazard	Mitigation
• Inadequate supply of consumables for construction and operation causing delays to project schedule and production resulting in a potential increase project cost and loss of production.	• Detailed evaluation of supply logistics and consumable consumptions. • Pre-qualification of suppliers. • Develop contingency plans.
• Inadequately trained mine rescue and underground emergencies.	• Ensure training policy includes emergency response training. • Develop emergency plan for construction and operations. • Update operations and maintenance procedures in compliance with BC *Mines Act*.
• Vehicle collisions.	• Prepare traffic management plan. • Site orientation and driver training. • Prepare standard operating procedures for site vehicle operations. • Procedural controls for personnel movement around site and safe work procedures.
• Insufficient geological/geotechnical information causing delays in schedule, increased costs, and loss in production.	• Ground control management. • Stress measurement and structural model. • Additional drilling once design is closer to final.
• Explosion/detonation causing personnel incidents, damage to equipment, delays in schedule and production.	• BC *Mines Act* and federal *Explosives Act* procedural controls. • Enforcement of safe work polices and personal protective equipment (PPE) requirements. • Develop emergency plan.
• Inadequate refuge chamber and supplies.	• Review of refuge chamber design. • Develop emergency plan. • Site health and safety inspection. • Procedures.
• Paste quality and barricade failure causing delays in production.	• Review barricade design and installation procedures. • Standard operating procedures for paste backfill.
• Avalanche hazard.	• Develop procedural controls and emergency plan.

A Risk Management Plan (RMP) will be prepared during project execution. The RMP will detail the structure and method by which project risks will be identified and plans implemented to eliminate or mitigate the risks. The RMP will present a process to ensure risks addressed during the study phase of the Project are mitigated during project execution and that ongoing risks identified during execution are captured and addressed accordingly.

The Project risk and opportunity management process is focused on the identification of risks and opportunities associated with realizing maximum project value. Technical risk management, including HAZOPs and design review processes, are managed by the engineering discipline. A formal HAZOP will be conducted during the detailed design.

Design reviews will be conducted during the detailed design process and will include internal discipline reviews, inter-discipline reviews, and construction reviews. Technical risks will be mitigated and preferably designed out by elimination, by alternative engineering design, or by engineering control measures. Risks associated with design change will be mitigated by change management controls that will ensure document revision approvals and identification on the risk register. Engineering change management controls will ensure field engineering changes are properly documented, risk assessed, and approved before implemented.

The Project risk and opportunity management process will be overseen by the EPCM Contractor. These personnel can be external to the project team and responsible for the original facilitation and recording of risk and opportunity assessment workshops and ongoing monitoring and review of the Risk Register. A formal risk assessment will be conducted during project execution and ongoing risks identified during execution will be managed by the continuous management of the Risk and Opportunities Register.

Risk management will be included in the Project Monthly Report and used to communicate key issues associated with the Project.

Construction risk assessments are reviewed along with the constructability plan and schedule before the construction contractors mobilize to site. This assists in the alignment of the construction contractor and the EPCM Contractor site construction team.

The risk management plan will ensure:

- a process where action outcomes are recorded and managed to ensure they are properly closed out
- risks and action plans are reviewed monthly on an individual basis and updated in the monthly report
- risk reviews are undertaken at strategic stages of the Project life cycle.

The Project will be audited to ensure compliance with the RMP.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCE

An updated Mineral Resource estimate has been prepared for the VOK Zone at the Brucejack Property of Pretivm located near Stewart, BC. The Measured, Indicated and Inferred Mineral Resource estimates, effective December 2013, are intended for use in a feasibility study for a bulk high-grade underground mining scenario.

Prior to the update of the Mineral Resource, Pretivm completed an underground bulk sample program to confirm the style of mineralization and grade tenor. This 10,000 t bulk sample was processed through a mill and the resultant reconciled metal was used to investigate the local accuracy of the November 2012 Mineral Resource estimate within the VOK. The study findings were also assessed to determine whether the estimation methodology could be improved for the December 2013 Mineral Resource.

Based on the bulk sample comparisons, Snowden concludes that the November 2012 Mineral Resource estimate was a good indicator of the contained metal within the VOK deposit and suitable for mine planning studies based around bulk underground mining. The model however was not locally accurate at the 10 m block scale due to over smoothing. As a result further test work was undertaken to adjust the estimation methodology for the December 2013 Mineral Resource, to produce an estimate that is more locally responsive.

A comparison of the updated Mineral Resource to the previous November 2012 Mineral Resource and the bulk sample results shows that the updated estimate is more locally accurate than the previous Mineral Resource. The updated Mineral Resource underestimates the north-south mineralization which may result in additional ounces if more of these features are discovered during mining.

The December 2013 Mineral Resource confirms the contained metal represented by the November 2012 Mineral Resource (within adequate limits) and also extends the Mineral Resource based on additional drilling. Comparing the 2012 and 2013 estimates shows that the 2013 estimate contains slightly less tonnes at a higher grade, whilst retaining the metal locally, which is a response to the reduction in smoothing during grade estimation. In addition to the improvements in the model, the comparison with hard data (attained from the processing of the bulk sample) has increased confidence in the Mineral Resource as a result of confirmation of the style of mineralization, domain boundaries and grade tenor.

25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

The test results from all the test programs, including the bulk sample processing runs, 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 ores, 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 and scavenger 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.

25.3.1 MINING RISKS

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.

- Mineralized material produced from development activities during the pre-production phase that is below the Cdn$180/t NSR cut-off value has been considered as waste. There may be an opportunity to stockpile this material for future processing depending on both the mine and mill ramp up performance. Considering sunk costs, a marginal NSR cut-off value of $40/t to cover milling costs and some rehandling costs would be appropriate.
- A conservative view has been taken for the mining recovery and suitable stope dimensions in proximity to the Brucejack Fault. Further drilling as level development approaches the Fault Zone may assist in the identification of areas where less conservative assumptions could provide better results.
- If development advance targets are not achieved during the pre-production phase, the ramp-up schedule to full production may be compromised.
- The ability to achieve full production in the early years of mine life is dependent on the development of infrastructure for both the VOK lower block and the VOK middle block. This infrastructural waste development program must be realized in an environment of competing priorities.
- The Brucejack orebody hosts multiple economic lenses in close proximity to each other creating both mine sequence related and geotechnical complexity. Definition drilling and Mineral Resource modelling must precede mining in a timely manner or the loss of resources and/or an increase in ore dilution could occur.

25.4 Project Infrastructure

25.4.1 Avalanche Hazard Assessment

An avalanche hazard assessment was completed for the Project. Mine site facilities and access routes are exposed to approximately 14 paths or areas, and the preliminary transmission line alignment crosses several avalanche paths. Avalanche magnitude and frequency varies depending on location. 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 a number of mining exploration trails exist near the southernmost portion of the transmission route.
- 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. Detailed topography and further engineering along the route could well provide the basis for lengthening the spans and decreasing the number of structures, resulting in lesser costs for the line.
- 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 based on site investigations completed from 2011 to 2013 (BGC 2014). Recommended allowable bearing pressures were provided for foundations on bedrock and on structural fill. Geotechnical recommendations have also been provided for 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 other 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 waste rock in Brucejack Lake (BGC 2014). 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 results of a stability assessment indicate that, under drained loading, the waste rock pile will have a factor of safety ranging from 0.9 to 1.4 depending on the strengths applied to the lake bottom sediments. These results are, however, based on the assumption that soft, weak lake bottom sediments extend all the way down to bedrock. It is considered possible that denser and stronger sediments may be present below the surface of the lake bottom. Further investigations, consisting of drilling and sampling should be conducted to confirm this. For better definition of the strengths, undisturbed samples should be collected and in-situ vane shear strength profiling should be completed. This will allow for a more confident estimate of the waste rock pile's stability.

The dumping platform will require ongoing monitoring throughout the life of the mine to assess whether it is safe for personnel and equipment to be operating in this area. Safe working procedures will be developed specifically for this area and visual monitoring for signs of deformation should be completed continuously while work is actively being conducted on the waste rock pile.

Quarry

Geotechnical design criteria have been recommended for the bench and overall slope scales to support development of the proposed quarry located near the southeast corner of Brucejack Lake. The recommended criteria are based on data collected during site investigations completed in 2013, consisting of geotechnical drilling and surface mapping.

Monitoring of the quarry slopes during development should be conducted to reduce the likelihood of slope instabilities impacting the quarry and/or the main mine site access road. Verification and validation of the slope design, and assumptions, will also be required to determine if as-built design modifications are needed.

25.4.5 BRUCEJACK LAKE OUTFLOW MONITORING AND SUSPENDED SOLIDS CONTROL

Hydrodynamic modelling of Brucejack Lake indicated that the potential for elevated TSS levels in surface waters was unlikely if the minimum particle diameter was greater than or equal to 5 µm (Section 20.1.5; Lorax 2013). However, it will be necessary to control the TSS concentrations at the outlet of Brucejack Lake to meet MMER regulations. One or more lines of turbidity curtains installed across the lake outlet are proposed to mitigate elevated TSS in the lake outflow. To facilitate outflow monitoring, a weir will be installed across Brucejack Creek downstream from the lake outlet and above the confluence with Camp Creek.

Preliminary design (BGC 2013) and geotechnical site investigation (BGC 2014a) for an outlet control structure, capable of stopping flow from the lake temporarily, have been completed as a contingency. Additional studies for this structure have considered storage volume and retention time for varying size runoff events (BGC 2014b) and outlet channel stability (BGC 2014c).

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. The contact water pond will be sized to contain runoff from the 24-hour, 200-year return period rain-on-snow event (220 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 3,043 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 and other losses within the plant (approximately 9 m^3/d). Process water will be sourced from:

- treated underground seepage water
- ore moisture (approximately 5% by weight)
- reclaim from the lake.

Average annual groundwater seepage into the underground workings is expected to vary from approximately 3,840 to 6,240 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 2,472 m^3/h from Brucejack Lake has been estimated for the life of mine, an average increase of 6.4% above existing conditions (2,324 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 1 to 7 years of mine life, ranging between 4,100 m^3/d and 4,600 m^3/d. The rate of inflow to the underground workings is predicted to increase to an annual average peak of approximately 6,500 m^3/d in year 8, with the initiation of development of the WZ resource. During years 9 to 18 of mine life, predicted annual average inflows range between 5,200 and 5,500 m^3/d, before decreasing slightly and ranging between 4,900 and 5,200 m^3/d for the final four years of mine life. The overall average flow for the entire simulated mining period is 4,900 m^3/d.

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 (S.A. Run 1) resulted in higher inflow estimates, with predicted inflows increasing by a factor of approximately 2.4 to approximately 11,700 m^3/d on an average annual basis. The peak inflow associated with the increased hydraulic conductivity scenario is predicted to be approximately 14,400 m^3/d in Year 8 of operations. Decreasing the hydraulic conductivity by a factor of five (S.A. Run 2) resulted in a commensurate decrease in inflow (factor of 0.5), with annual inflows averaging approximately 2,300 m^3/d. The peak inflow associated with the decreased hydraulic conductivity scenario is predicted to be approximately 3,500 m^3/d in Year 8.

The high K and combination sensitivity simulations (S.A. Runs 1, 12, and 14) yielded the highest inflow estimates, averaging approximately 11,700 m^3/d, 14,600 m^3/d, and 14,700 m^3/d, respectively. The annual average peak flows and factor increases associated with the high K sensitivity scenarios were 14,400 m^3/d (S.A. Run 1; factor of 2.2), 17,400 m^3/d (S.A. Run 12; factor of 2.7), and 19,100 m^3/d (S.A. Run 14; factor of 2.9).

It is worth noting that while the high K sensitivity scenarios are considered conservative from a feasibility perspective (i.e., they result in the highest rates of groundwater inflow to the underground workings), none are supported by the model calibration results.

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 400 m in the mine footprint in year 12 of mining operations, after which the water table starts to recover. The cone of depression associated with this dewatering draw down has an areal extent of approximately 2 km by 3 km.

25.4.8 Tailings Delivery System

The tailings placement system for Brucejack Lake will be 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 has focused on closing the adits, removing all site infrastructure, reclamation using native species at the mine site, removing the transmission line, closing the access road including removing the bridges and culverts and revegetation with native species, and similarly removing the infrastructure and carrying out reclamation of the disturbed areas at the Bowser Aerodrome, Knipple Transfer Area, and the Tide Staging Area.

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 Q2 2014.

25.5.1 GEOCHEMISTRY

The main objective of the ML/ARD assessment program is to provide an understanding of the geochemical characteristics of the materials that are disturbed, excavated or exposed as a result of the planned mining activities. The main conclusions of the ML/ARD assessment can be summarized as follows:

- The ABA assessments of waste rock according to geological model units and lithology groupings both show the majority of waste rock at Brucejack is PAG material, with the exception of one geological model unit and one lithological group:
- The Office P1 unit contains predominantly non-PAG waste rock, as per results from frequency analyses conducted on waste rock static tests.
- Mafic volcanics are generally non-PAG material, as 83% of samples submitted for static testing present NPR values greater than 2.
- The VSF, Fragmental and Conglomerate units account for 85% of the total generated waste rock and contain 77 to 85% PAG material. These three geological model units constitute 87% and 88% of the waste rock destined for the underground mine and Brucejack Lake, respectively.
- Due to the absence of a clear distinction in ABA characteristics between lithology groups or geological units it is difficult to propose recommendations for waste rock segregation. Exceptions to this general observation are the andesites from the VSF and Fragmental units that clearly have different ABA characteristics.
- Materials with the shortest lag times (less than 15 years) typically have paste pH values below 7, very low NP values (5 to 15 kg $CaCO_3$/t) and high sulphide-S values (3 to 8%) and weather readily and quickly.
- The elements As, Cd, Cu, Pb, Se and Zn are considered parameters of concern (POCs), based on leachate concentrations from humidity cells and field barrels containing waste rock.
- Subaqueous columns with waste rock show elevated leachate concentrations of As, Sb, Mo, Se and Zn, which may be a concern under more reducing conditions.
- Only ore and sludge samples are characterized as PAG materials, whereas tailings and paste samples are considered non-PAG materials.
- Based on the humidity cells and subaqueous column tests, tailings are not expected to generate ARD.
- POCs in the leachates from humidity cells and subaqueous columns with tailings are As, Sb, Mo and Se.

26.0 RECOMENDATIONS

26.1 GEOLOGY

Snowden makes the following recommendations:

- Extend a ramp down to the 1,260 m level and open up that level to provide access to complete high density definition drilling down dip of the current underground drilling and along trend to the east.
- Extend a ramp up to the 1,410 m level and open that level to provide access to complete high density definition drilling up dip of the current underground drilling and along strike to the west.
- Extend the 1,260 m level approximately 400 m to the east and complete resource definition drilling of the far eastern Inferred Resources.
- Take into account orientation bias associated with variable vein directions in the mineralized stockwork system, when planning further drilling programs.

The budget for phase 1 of the program consists of:

- $9.0 million for development of 500 m of access ramp and lateral drift
- $3.5 million for drilling of 15,000 m of underground drilling off the access drift.

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. The recommended test work should include additional metallurgical test work to confirm the metallurgical response of the samples to the established process flowsheet, including locked cycle tests on low grade materials. The samples used for the 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 $150,000.

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 makes the following mining-related recommendations:

- Average LOM operational costs have been estimated at Cdn$163.05/t. The NSR cut-off value of Cdn$180/t used in estimating Mineral Reserves has an average Cdn$16.95/t operational margin for ore mined. Currently, the deposit has not been optimized with regards to overall project value or in terms of other key metrics such as cash flow using cut-off value. AMC recommends further study work on NSR cut-off value optimization. Budgetary estimates for this optimization work are Cdn$40,000.
- Opportunities to improve the grade profile in the early years of operation should be investigated through further analysis. This analysis has an estimated cost of Cdn$30,000.
- Opportunities exist to reduce the ventilation requirement by adopting the CANMET ventilation recommendations for the underground diesel equipment fleet. This could lead to savings in mine power and air heating costs. The estimated cost for this work is Cdn$10,000.
- Further test work is recommended to identify the potential benefit of light classification of paste backfill by cycloning. Cycloning has the potential to remove clays and result in the production of a coarser particle size distribution to improve the filtering process and give a higher-quality paste. The estimated cost for this work is Cdn$10,000.
- Further test work using locally available binders, preferably slag based cement, should be undertaken to determine the appropriate cement dosages for paste backfill. Both local or international suppliers of high slag based cement capable of supplying the required quantities should be contacted. The estimated cost for this work is Cdn$35,000.

26.4.1 GEOTECHNICAL

BGC has identified the following key risks and opportunities with regards to the rock mechanics assessment:

- An authoritative structural geology model for the Project area has not yet been 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. This would allow optimization of excavation and pillar dimensions and associated ground support.

- 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.

- The proposed portal site has moved approximately 60 m to the west since the feasibility study site investigations were completed. Therefore, there is no site-specific geotechnical data available for the proposed location shown in the feasibility study update mine plan. The portal excavation design and ground support recommendations assume that the rock mass at the proposed locations is of similar quality as the rock mass encountered in the two drill holes completed at the original feasibility study portal site. Overburden thickness estimates based on those holes should not be used for material takeoff estimates for the new location.

BGC makes the following recommendations for additional rock mechanics assessment work:

- Eight to ten 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, the West Zone ramp, and the new proposed portal site. 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 any ground deformation

observations recorded during development or sampling activities. 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 hanging walls, 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 distribution of K in the model and inform fault-related sensitivity analyses. Additional packer testing is recommended for additional geotechnical boreholes targeting the Brucejack Fault as part of the detailed design.
- 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 potential lake outlet structure. The next phase of modeling should include the effects of this structure, if it is carried into the detailed design, which, depending upon how it is operated, could alter the elevation of Brucejack Lake and potentially local groundwater flow conditions.

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. These data could also be used for numerical flow model calibration at subsequent project design stages in support of optimizing water treatment plant sizing and/or permitting.

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 26.1.

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 AR5 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 must be used to manage the operational risk for personnel around snow avalanches to the transmission line. Snow avalanches, particularly on the east side of the Salmon Glacier valley, may pose a risk to the transmission line. In this area, towers will be located outside the snow avalanche zones to limit the risk of impact from snow avalanches.
- 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 truck shop, operations camp, substation, 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.

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 $380,000.

26.5.4 WASTE ROCK DISPOSAL

The results of the stability assessment conducted on the waste rock pile are generally based on the assumption that soft, weak lake bottom sediments extend all the way down to bedrock. It is possible that denser and stronger sediments may be present below the surface of the lake bottom. Further investigations, consisting of drilling and sampling should to be conducted to confirm this. For better definition of the strengths, undisturbed samples should be collected and in-situ vane shear strength profiling should be completed.

The dumping platform will require ongoing monitoring throughout construction and operation to assess whether it is safe for personnel and equipment to be operating in this area. Safe working procedures will need to be developed specifically for this area. Procedures utilized at existing or pre-existing operations that disposed of waste rock in bodies of water, such as the Eskay Creek Mine, should be considered.

QUARRY

To support subsequent levels of design, BGC recommends the following:

- Point load testing and laboratory analyses of drill core obtained from the quarry to further evaluate the intact strength of the bedrock. This will provide blast designers with more detailed information for their designs.
- Trial crushing to assess fines generation from the quarry rock types.
- Based on available information for the site, the quarry is located between two areas identified as avalanche hazard zones. A study should be completed to assess if the quarry location is susceptible to potential avalanches.

The estimated cost for this work is $155,000.

26.5.5 Tailings Delivery System

ERM Rescan recommends the following work for the tailings delivery system:

- 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.
- If discharge of tailings as a paste to Brucejack Lake is contemplated then a design needs to be developed including investigation of the behaviour of paste flow under water and the transport of tailings fines to the surface during lake turnover. The design should include the configuration of a paste discharge system, system hydraulics and flushing, start-up and shut-down procedures.

26.5.6 Brucejack Lake Outflow Monitoring Weir

Hydrotechnical design and site investigation for the level control and flow monitoring weir are proposed to be completed during 2014. The estimated cost for this work is $150,000.

26.5.7 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.
- There is currently uncertainty with the watershed area reporting to BJL-H1. If the watershed area was only 8.5 km^2, the implication is that average precipitation at site is on the order of 2560 mm, rather than the current estimate of 1900 mm to 2040 mm. This difference in precipitation would not invalidate the water balance model or water quality results summarized here-in, as the model is calibrated to streamflow, not precipitation. However, confidence in the site precipitation estimates is important for evaluating peak flows and runoff volumes for drainage ditches and collection ponds. Therefore, it is recommended that a site visit be conducted in June 2014 to evaluate runoff patterns at the east end of Brucejack Lake. The purpose of the site visit would be to try and confirm the watershed area reporting to BJL-H1.

The estimated cost for this work is $105,000.

26.5.8 Mill Site Layout

The required construction schedule may impact the earthworks design in the mill/camp/truck shop areas. The current schedule requires the permanent camp to be constructed in the latter part of the 2015 construction season. This will require the earthworks program to be complete early in 2015, such that foundations and camp building erection can be accomplished. Tetra Tech recommends completing a trade-off study to assess potential cost and or schedule savings by lowering the nominal pad elevation of the camp, mill, and truck shop pad. Information required to complete the trade-off study include:

- further defined schedule constraints
- coordination with mining to review re-alignment of the portals.

The estimated cost for this recommended work is $50,000.

26.6 Environmental

Rescan recommends additional work to further develop the Project. This work includes:

- The completion of 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 five year, detailed reclamation and closure plan in conjunction with *Mines Act* permitting. The detailed reclamation and closure plan will require a final site plan and a detailed description of the various facilities and activities that will occur in the first five years of the Project starting with the Construction phase,
- 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 ROM waste (tailings and waste rock) and the waste rock from the plant site (construction) destined for (1) the underground mine and (2) Brucejack Lake, will need to be sampled for an assessment of the ARD potential (PAG or non-PAG material).

- On-site monitoring of quarry excavation will identify any changes to bulk lithology and (potentially) its ML/ARD characteristics. In scenarios of observed changes additional samples of excavated quarry material will be collected and submitted for static testing, to continually validate its characterization as a non-PAG rock source for construction at the Project site and update (if required) quarry source terms used in the water quality model.

- Ongoing kinetic tests (of waste rock and tailings) need to be continued to provide better steady state chemistry data for an update of the source terms used in the water quality model.
- The sampling and chemical analysis of underground seeps (groundwater) and sumps (mine water) should continue during mining for an update of the source terms used in the water quality model.
- The sludge produced by the upgraded Water Treatment Plant (WTP) should be tested (static tests, mineralogy, SFEs and columns) and the data should be used to validate the ML/ARD characteristics for the detrital component of sludge, as well as characterize the secondary mineral precipitate component of sludge materials. Additionally, this data will inform whether the tailings source term used in the water quality model will need to be updated to account for the incorporation of sludge materials with tailings destined for subaqueous disposal in Brucejack Lake.

The estimated cost for this work is $1,315,000.

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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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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 (except 1.8.1 to 1.8.3), 1.13, 1.14, 2.0, 3.0, 18.1, 18.3, 18.8, 18.9, 18.10, 18.13, 18.14, 18.17.2, 18.7.3, 24.0, 26.5.8, 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

David Ireland, C.Eng., P.Eng.
Senior Project Manager
Tetra Tech WEI Inc.

28.2 LYNN OLSSEN, MAUSIMM(CP)

I, Lynn Olssen, MAusIMM(CP), of West Perth, Australia, do hereby certify:

- I am a Senior Principal 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 Update on the Brucejack Project, Stewart, BC”, dated June 19, 2014 (the “Technical Report”).
- I am a graduate of The University of Western Australia, (1993). I am a member in good standing of the Australasian Institute of Mining and Metallurgy, member number 206421. My relevant experience is over 20 years in the mining industry including 7 years as a mine geologist working on gold deposits and 13 years as a consultant during which time I have worked on numerous resource estimates for stockwork and narrow vein gold deposits. 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 16, 2013 for five days.
- I am responsible for the preparation of Sections 1.2, 1.3, 1.4, 4.0 to 12.0, 14.0, 23.0, 25.1, 26.1, 27.10, 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 co-authored the Technical Report entitled “Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC”, dated June 21, 2013. Also, 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 30th day of June, 2014 at West Perth, Australia

Lynn Olssen, MAusIMM(CP)
Senior Principal 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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, 27.11, 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

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 ERM Rescan with a business address at 1111 West Hastings Street, 15th Floor, Vancouver, British Columbia, V6E 2J3.
- This certificate applies to the technical report entitled "Feasibility Study and Technical Report Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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), 27.7 (except 27.7.1 and 27.7.2) 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

____
Pierre Pelletier, P.Eng.
Division Managing Director
ERM Rescan

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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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, 26.5.7, 27.6, 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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, 27.12, 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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, 25.3.1, 26.4 (except 26.4.1 and 26.4.2), 27.8, 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Kamloops, British Columbia

Catherine Schmid, M.Sc., P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

28.10 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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 seven 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 seven days.
- I am responsible for Sections 18.2, 18.11, 25.4.3, 25.4.4, 26.5.3, 26.5.4, 27.4, 27.5, 27.5.1, 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 had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Kamloops, British Columbia.

Brent McAfee, P.Eng.
Geotechnical Engineer
BGC Engineering Inc.

28.11 MICHAEL CHIN, P.ENG.

I, Michael Chin, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Chief 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (the "Technical Report").
- I am a graduate of the University of Alberta (Bachelor of Science in Civil Engineering, 1986). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #1172. 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 28.11 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 co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

Michael Chin, P.Eng.
Chief Civil Engineer
Tetra Tech WEI Inc.

28.12 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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 28.12 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 co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Squamish, British Columbia

Brian Gould, P.Eng.
Senior Avalanche Specialist/Engineer
Alpine Solutions Avalanche Services

28.13 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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 28.13 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 co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

Michael Paul Wise, P.Eng., MBA
Director, Project Development
Valard LP

28.14 PAUL GREISMAN, PH.D., P.ENG.

I, Paul Greisman, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Technical Director of ERM Rescan with a business address at 1111 West Hastings Street, 15th Floor, Vancouver, British Columbia, V6E 2J3.
- This certificate applies to the technical report entitled "Feasibility Study and Technical Report Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (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, and 28.14 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 co-authored the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

Paul Greisman, Ph.D., P.Eng.
Technical Director
ERM Rescan

28.15 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 Update on the Brucejack Project, Stewart, BC”, dated June 19, 2014 (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 28.15 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 co-authored the Technical Report entitled “Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC”, dated June 21, 2013.
- 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 30th day of June, 2014 at Thunder Bay, Ontario

Wayne E. Scott, P.Eng.
Mining Divisional Manager, Electrical
Tetra Tech WEI Inc.

28.16 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 Update on the Brucejack Project, Stewart, BC”, dated June 19, 2014 (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 28.16 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 co-authored the Technical Report entitled “Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC”, dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

Ali Farah, P.Eng.
Lead Mechanical Engineer
Tetra Tech WEI Inc.

28.17 GEORGE ZAZZI, P.ENG.

I, George Zazzi, 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (the "Technical Report").
- I am a graduate of the British Columbia Institute of Technology (Diploma of Mining Engineering, 1993) and the University of British Columbia (BASc in Metallurgical Engineering, 1989). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #28636. I have worked as a Mining Engineer for a total of 21 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, 16.10 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 had prior involvement with the Property that is the subject of the Technical Report. I assisted with the completion of the Technical Report entitled "Feasibility Study and Technical Report on the Brucejack Project, Stewart, BC", dated June 21, 2013.
- 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 30th day of June, 2014 at Vancouver, British Columbia

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George Zazzi, P.Eng.
Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd

28.18 TREVOR CROZIER, M.ENG., P.ENG.

I, Trevor Crozier, M.Eng., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Principal 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (the "Technical Report").
- I am a graduate of the University of British Columbia (B.A.Sc., 1992; M.Eng. (Geological Engineering), 2003). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #22194. My select relevant experience with respect to hydrogeology work includes the KSM Project (BC), the Donlin Gold Project (AK), the Eagle Gold Project (YT), Red Lake Gold Mines (ON), Gibraltar Mine (BC), Los Helados (Chile), and La Coipa (Chile). I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 5, 2013 for one day.
- I am responsible for Sections 16.6, 18.13, 20.1.5, 25.4.5, 25.4.7, 26.4.2, 26.5.6, 27.3, 27.7.1, 27.13, 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 30th day of June, 2014 at Vancouver, British Columbia.

Trevor Crozier, M.Eng., P.Eng.
Principal Hydrogeological Engineer
BGC Engineering Inc.

28.19 SHARON BLACKMORE, M.SC., P.GEO.

I, Sharon Blackmore, M.Sc., P.Geo., of Vancouver, British Columbia, do hereby certify:

- I am an Intermediate Hydrogeochemist 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 Update on the Brucejack Project, Stewart, BC", dated June 19, 2014 (the "Technical Report").
- I am a graduate of the University of Western Ontario (B.Sc. 2003; M.Sc. (Geology), 2005). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #37460. My relevant experience with respect to ML/ARD and hydrogeochemistry includes work for the Brucejack Project, Los Helados (Chile), the Antamina Mine (Antamina, Peru), Buckreef Project (Tanzania), and Red Lake Gold Mines (Red Lake, Ontario). I have been involved in the mining and exploration industry since 2001 and continuously since 2006. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 2013 for two weeks.
- I am responsible for Sections 20.1.4, 25.5.1, 26.6.1, 27.7.2, 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 30th day of June, 2014 at Vancouver, British Columbia.

Sharon Blackmore, M.Sc., P.Geo.
Intermediate Hydrogeochemist
BGC Engineering Inc.