PREMIER & RED MOUNTAIN GOLD PROJECT

Feasibility Study NI 43-101 Technical Report
British Columbia

Prepared for:
Ascot Resources Limited

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IMPORTANT NOTICE

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Glossary

Units of Measure

Amperes	A
Canadian dollars	$ or C$
Centimetre cubic	cm^3
Centimetre	cm
Cubic feet per minute	cfm
Degree	°
Degrees Celsius	°C
Dry Tonnes	dmt
Feet	ft
Gallons per day	gpd
Gallons per minute	gpm
Gallons	gal
Gram	g
Grams per cubic centimetre	g/cm^3
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Inches	"
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per square metre	kg/m^2
Kilometre square	km^2
Kilometre	km
Kilopascals	kPa
Kilovolt	kV
Kilowatt	kW
Less than	<
Litre	L
Litres per minute	L/min
Mega-annum (1 million years)	Ma
Megavolt ampere	MVA
Megavolt	MV
Megawatt	MW
Metre cubic	m^3

Metre level (relative metres level below surface)	m Level or mL
Metre square	m^2
Metre	m
Metres above sea level	masl
Metres cubic per hour	m^3/h
Micron	µm
Millimetre	mm
Million litres per day	ML/d
Million tonnes per annum	Mt/a
Million tonnes	Mt
Million years (annum)	Ma
Million	M
Ounce (troy ounce – 31.1035 grams)	oz
Ounce per annum	oz/a
Ounce per tonne	oz/t
Part per million	ppm
Percent by mass	%m
Percent mass fraction for percent mass	%w/w
Percent	%
Pound	lb
Pounds per square inch gage	psig
Tonnes per cubic metre	t/m^3
Tonnes per day	t/d
Tonnes per hour	t/h
United States dollars	US$
Volt	V

Abbreviations and Acronyms

Acid Desorption and Refining	ADR
Acid Rock Drainage	ARD
Activation Laboratories Ltd.	Actlabs
All-Terrain Vehicle	ATV
ALS Minerals Laboratory	ALS
American Association of Cost Engineers	AACE
American Smelting and Refining Company	Asarco
American Wire Gauge	AWG
Andesite-Rich Composite (AXXZ) and Silica/Breccia-Rich Composite Samples	CBXX
Arseneau Consulting Services Inc.	ACS
Ascot Resources Limited	Ascot

Association of Professional Engineers and Geoscientists of British Columbia	APEGBC
Atomic Absorption	AA
Atomic Emission Spectroscopy	AES
Ball Mill Work Index	BWi
Banks Island Gold Ltd.	Banks
Base Metallurgical Laboratories Ltd.	BML
BC Water Quality Guidelines	BCWQG
Best Achievable Technology	BAT
Big Missouri	BM
Bird Resource Consulting Corp.	BRCC
Bond Abrasion	Ai
Bond Gold Canada Inc.	Bond
British Columbia Geological Survey	BCGS
British Silbak Premier Mines Limited	BSP
Canaccord Resources Inc.	Canaccord
Canadian Development Expense	CDE
Canadian Exploration Expense	CEE
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Capital Expenditure	CAPEX
Carbon-in-pulp	CIP
Cascade Creek Diversion Channel	CCDC
CDA Guidelines	CDA
CDN Resource Laboratories Ltd.	CDN
Certified Reference Materials	CRM
Close-Circuit Television	CCTV
Coarse Ore Stockpile	COS
Coefficient of Variation	CV
Concrete Reinforced Barriers	CRB
Consolidated Mining and Smelting Company	Cominco
Copper	CU
Cumulative Probability Plots	CPP
Cumulative Tax Credit Account	CTCA
Cut-and-Fill	C&F
Cut-Off Grade	COG
Cyanide	CN
Diamond Drill Hole	DDH
Drop-Weight index	DWT
Effective Grinding Length	EGL
Electromagnetic	EM

Engineering-Procurement-Construction-Management	EPCM
Environmental Assessment Certificate	EAC
Environmental Design Flood	EDF
Environmental Management Act	EMA
Existing Water Treatment Plant	EWTP
Extended-Gravity Recoverable Gold	E-GRG
Falkirk Environmental and Ecologic	Falkirk
Fire Assay	FA
Footwall	FW
Franco-Nevada Corporation	Franco
Front-End-Loader	FEL
General and Administrative	G&A
Global Positioning System	GPS
Gold by Fire Assay	AuFA
Gravity Recoverable Gold	GRG
Gravity/Flotation/Leach	GFL
Half Absolute Relative Difference	HARD
Hangingwall	hw
Heating, Ventilation, and Air Conditioning	HVAC
High-Density Polyethylene	HDPE
High-Density Sludge	HDS
Hydrochloric acid	HCl
IDM Mining Ltd.	IDM
Implicit Modelling Tool in MineSight	MSIM®)
Independent Power Producer	IPP
Induced Polarization	IP
Inductively Coupled Plasma	ICP
Inductively-Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
InnovExplo Inc.	INN
Inspectorate laboratories	Inspectorate
Intensive Leach Reactor	ILR
Internal Rate-of-Return	IRR
International Organization for Standardization	ISO
Inverse Distance Cubed	ID3
Inverse Distance Cubic	ID^3
Inverse Distance Square	ID^2
Inverse Distance Weighting	IDW
Inverse Distance	ID
Jayden Resources Canada Inc.	Jayden Canada

Klohn Crippen Berger	KCB
Knight Piésold Ltd.	Knight Piésold
LAC Minerals Ltd.	LAC
Life-of-Mine	LOM
Light Detection and Ranging	LiDAR
Load-Haul-Dump	LHD
Long Lake Hydroelectric Project	LLHP
Low-Density Sludge	LDS
Marsland Environmental Associates Limited	MEA
Mass Spectrometry	MS
McElhanney Engineering Ltd.	McElhanney
Minarco-MineConsult	MMC
Mine Paste Ltd.	Mine Paste
Mines Act Permit Application	MAPA
Ministry of Energy, Northern Development and Mines	MENDM
Ministry of Natural Resources and Forestry	MNRF
Ministry of Natural Resources	MNR
Ministry of Northern Development and Mines	MNDM
Mountain Boy Minerals Inc.	MBM
Moving Bed Biofilm Reactor	MBBR
Nanika Resources Inc.	Nanika
National Instrument	NI
National Topographic System	NTS
Nearest Neighbour	NN
Net Present Value	NPV
Net Profit Interest	NPI
Net Smelter Return	NSR
Non-Potentially Acid Generating	NPAG
North American Metals Corporation	NAMC
North American Metals, Inc.	NAM
Operating Expense	OPEX
Ordinary Kriging	OK
Palmer Environmental Consulting Group Inc.	Palmer
Parts per Billion	ppb
Parts per Million	ppm
Paul Hughes Consulting Ltd.	PHC
Personal Computer Local / Wide Area Network	PC LAN/WAN
Pinnacle Mines Ltd.	Pinnacle
Pioneer Metals Corporation	Pioneer

Potentially Acid Generating	PAG
Preliminary Economic Assessment	PEA
Preliminary Feasibility Study	PFS
Premier & Red Mountain Gold Project	the Project
Premier Gold Mining Company, Limited	Premier Gold
Premier Gold Project	PGP
Premier/Northern Light	PNL
Prime Engineering	PE
Probable Maximum Flood	PMF
Process Research Associates	PRA
Qualified Person	QP
Quality Assurance	QA
Quality Assurance/Quality Control	QA/QC
Quality Control	QC
Quality Management System	QMS
Red Mountain Project	RMP
Right-of-way	ROW
Rock Quality Designation	RQD
Room-and-pillar	R&P
Royal Oak Mines Inc.	Royal Oak
Sacré-Davy Engineering Inc.	SDE
SAG Mill Comminution	SMC
SAG/Ball	SAB
Seabridge Gold	Seabridge
Sedgman Canada Limited	Sedgman
Semi-Autogenous Grinding	SAG
SGS Minerals Services	SGS
Silver Coin	SC
Sodium Cyanide	NaCN
Sodium Metabisulfite	SMBS
Specific Gravity	SG
SRK Consulting Canada Inc.	SRK
Standard Deviation	SD
Standard Operating Procedures	SOP
Sulfur Dioxide	SO_2
Sulfur	S
Sulfide Sulfur	S=
Surplus Water Management System	SWMS
Tailings Storage Facility	TSF

Tenajon Resources Corp.	Tenajon
Total Organic Carbon	TOC
Total Suspended Solids	TSS
Tournigan Mining Explorations Ltd.	Tournigan
Tudor Gold Corp.	Tudor Gold
Undepreciated Capital Cost	UCC
Underground	UG
Universal Transverse Mercator	UTM
Variable Frequency Drive	VFD
Very High Frequency	VHF
Voice over Internet Protocol	VoIP
Volcanogenic Massive Sulfide	VMS
Warning Level	WL
Water Treatment Plant	WTP
Weakly Acidic Dissociable CN	CN_{WAD}
Weight by Volume	w/v
Weight by Weight	w/w
Whole-Ore Leach	WOL
Wide Water Balance Model And Water Quality Model	WBM/WQM
Work Breakdown Structure	WBS
Wotan Resources Inc.	Wotan
X-ray fluorescence spectrometry	XRF

1 SUMMARY

Ascot Resources Limited (Ascot or the Company) is a Canadian-based exploration and development company based in Vancouver, Canada. Shares of Ascot are currently traded on the Toronto Stock Exchange and the OTCQX. Ascot is focused on re-starting the past producing Premier Gold mine located near Stewart, British Columbia.

This Feasibility Study is based on four underground mining operations feeding 2,500 tonnes per day (t/d) to the existing processing facility at Premier. The four mining operations known as Silver Coin, Big Missouri, Premier and Red Mountain will be sequenced over an 8-year period to initially produce 1.1 million troy ounces (Moz) of gold and 3.0 Moz of silver. The Silver Coin, Big Missouri, and Premier deposits, collectively being named the Premier Gold Project (PGP) are located near the processing facility on the historical Premier Mine site, and the Red Mountain Project (RMP) is located 23 km to the southeast in an adjacent valley. The projects benefit from existing road access, historical under ground mining infrastructure, a mill processing facility, the nearby Long Lake Hydro power plant, a tailings storage facility (TSF), a water treatment plant, and mine waste stockpile infrastructure resulting in a low initial capital refurbishment cost.

Mining will begin at the Silver Coin and Big Missouri deposits followed by the Red Mountain and Premier deposits. Access for production at all the deposits will be through both new and existing side hill portal access using a combination of new ramp development and the refurbishment of existing underground infrastructure. Mining methods will largely consist of low-cost long-hole stoping with limited use of inclined undercut long-hole, room-and-pillar, and cut-and-fill mining methods. Ore will be trucked to the processing facility, and mining waste will be used underground as a combination of rockfill and cemented rockfill.

The existing processing plant will be refurbished within a construction period of approximately 40 weeks. The plant will use conventional crushing, grinding, and gravity circuits followed by a standard carbon-in-leach (CIL) process to produce a gold doré. Plant refurbishment will use a combination of existing, new, and repaired equipment and infrastructure. Ore from the Red Mountain Project (RMP) is both harder and requires finer grind than ore from the other deposits. To optimize gold recovery, a fine grinding-mill and an additional pre-leach thickener will therefore be added to the plant prior to processing RMP ore.

The Project has an existing TSF and Water Treatment Plant (WTP) adjacent to the Long Lake Hydro power plant, which currently supplies Pretium's Brucejack Mine and connects to the BC Hydro grid. Currently, the site receives power via a 25 kilovolt (kV) power line from the town of Stewart. This arrangement would be modified, with a new substation to be constructed adjacent to the processing plant that would receive power from the Long Lake power plant approximately 800 metres (m) south of the processing plant. Power would be distributed to the site from this substation. This Feasibility Study has two key enhancements to the existing infrastructure: the TSF embankment would be expanded by excavating approximately 1.2 million cubic metres (Mm3) of non–acid generating rock material to reshape the slopes of the tailings dam to a 2:1 slope in a series of dam-centerline lifts throughout the mine life; the WTP would be replaced by a high-density sludge lime water treatment plant and an ammonia treatment plant (moving bed bioreactor) to accommodate additional water treatment requirements from the Big Missouri and Silver Coin operations and tailings supernatant.

1.1 Introduction

This Technical Report on the Feasibility Study of the Project has been prepared in accordance with NI 43-101 for Ascot by a team of professional consultants led by Sacré-Davey Engineering Inc. (SDE). SDE was responsible for overall coordination, infrastructure, and the economic evaluation; Bird Resource Consulting Corp. (BRCC) is responsible for the resource estimate of PGP; Arseneau Consulting Services Inc. (ACS) is responsible for the resource estimate for RMP; InnovExplo Inc. and Mine Paste Ltd. (Mine Paste) for mining; Sedgman Canada Limited (Sedgman) for metallurgy and processing; Knight Piésold Ltd. (Knight Piésold) for tailings and surface water management; SRK Consulting (Canada) Inc.(SRK) for the WTP; Paul Hughes Consulting Ltd. (PHC) for geotechnical; McElhanney Ltd. (McElhanney) for access roads; Prime Engineering for the electrical substation; Marsland Environmental Associates Limited (MEA) for geochemistry, hydrology, and water-quality modelling.

1.2 Key Outcomes

The NI 43-101 highlights are as follows:

- The plan uses Proven and Probable Reserves of 6.2 Mt at 5.9 grams per tonne (g/t) Au and 19.7 g/t Ag.
- Low initial capital expenditure of $147 million, including a 9% contingency, and 22% indirect costs.
- Life-of-mine (LOM) sustaining capital of $178 million, including closure capital.
- LOM payable production of 1.1 Moz of gold and 3.0 Moz of silver, with peak production of 172,000 gold equivalent ounces.
- LOM operating costs (C1)* of $145/t processed, or US$642 per payable ounce (oz) produced and LOM all-in sustaining costs (AISC)* of $174/t processed, or US$769 per payable oz produced.
- Base case pre-tax net present value (NPV) 5% of $516 million, internal rate of return (IRR) of 62%, and after-tax payback period of 1.8 years.
- Base case after-tax NPV5% of $341 million and IRR 51%.
- Assuming a spot gold price of US$1,710/oz, sliver price of US$15.32/oz and spot C$ to US$ exchange rate of 0.71, the Project economics increase to an after tax NPV5% of $602 million and IRR 71%.

**See note in Table 1-10*

All currency amounts are referred to in Canadian dollars ($) unless otherwise indicated. All units of measure used in this report are metric or otherwise stated. All supporting documents cited in this report are referenced in Section 27.

1.3 Reliance on other Experts

The Qualified Persons (QP) opinions contained herein are based on information provided by Ascot and by others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for it.

Non-QP specialists relied upon for specific advice includes:

- Onsite Engineering Ltd.
- Prime Engineering
- Soucie Construction Ltd.
- Falkirk Environmental Consultants
- Integrated Sustainability.

1.4 Property Description and Location

The Project in this study is divided into two land holdings. The PGP comprising 8,133 hectares (ha) located approximately 19 km to the northwest of the town of Stewart, British Columbia and the RMP comprising 17,125 ha located approximately 23 km southeast of the PGP. The nearest major centre is Terrace, British Columbia located 327 km to the south which hosts a major airport with numerous daily flights to Vancouver, British Columbia. Figure 1-1 shows the general location of the Project.

1.4.1 Premier Gold Project

The PGP property includes 175 Crown grants, 107 mineral claims, and 3 mineral leases covering a combined area of 8,133 hectares (ha) when overlaps are excluded.

Land Tenure and Acquisition Agreements

PGP is 100% owned by Ascot and was acquired under two separate agreements; the Dilworth option and related Premier asset purchase agreements and Silver Coin Acquisition Agreement

Dilworth Option and Premier Asset Purchase Agreements

The original Dilworth property agreement between Ascot and owners Boliden, R. Kasum, and the estate of J. Wang, was signed in March 2007. Under the original terms, Ascot acquired the right to earn a 100% interest in the Dilworth property by making staged option payments over ten years totalling $6.75 million.

The asset purchase agreement between Boliden and Ascot dated July 31, 2017, facilitated Ascot acquiring the Premier property for payment of $4.8 million in addition to all previously paid option payments of $6.2 million. Both the Dilworth option agreement and Premier asset purchase agreement are subject to a 5% NSR royalty which Ascot has the right to buy back for total payment of $13.7 million. In addition, the property is subject to other smaller historical royalty arrangements.

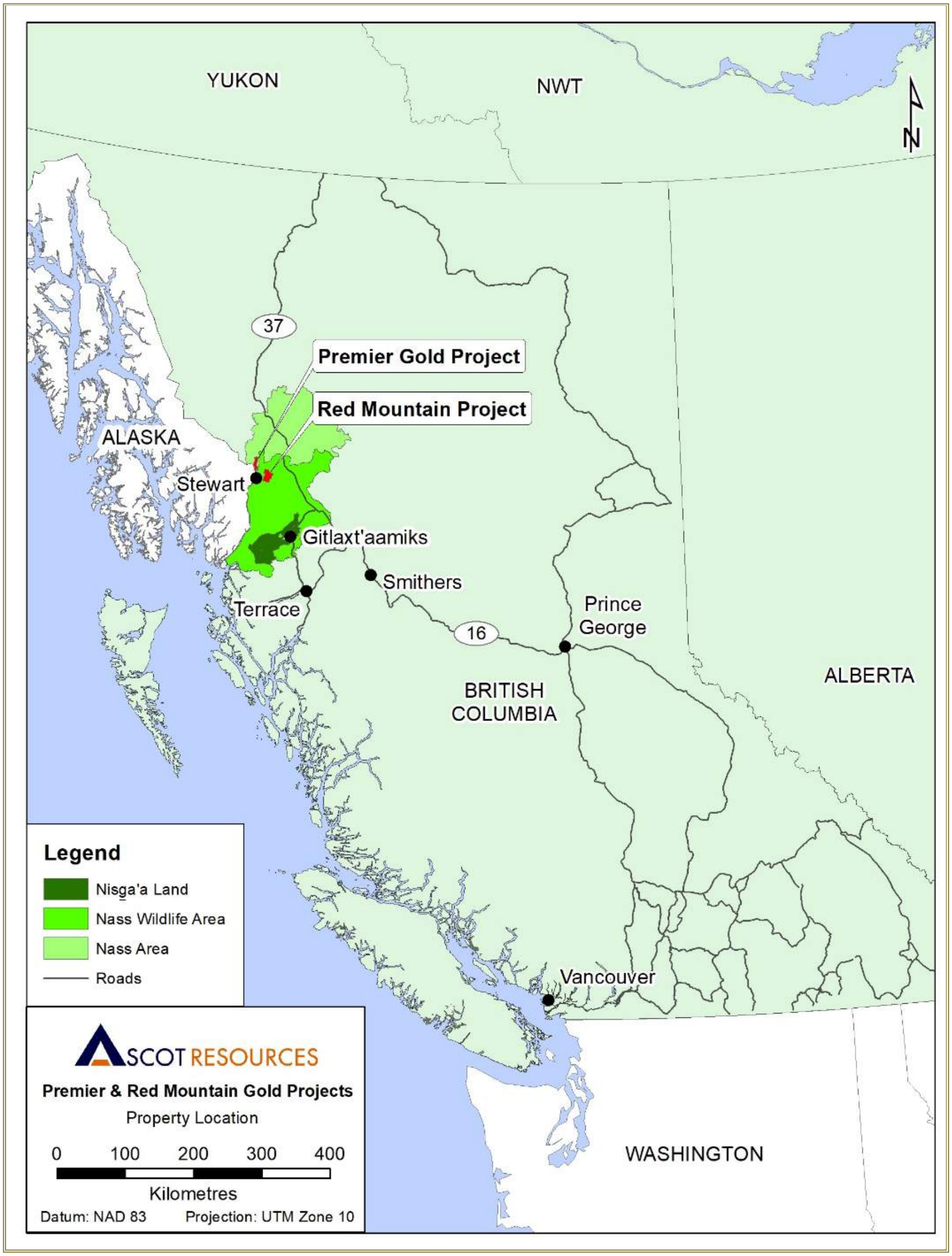


Figure 1-1: Project Location

Silver Coin Agreement

The Silver Coin property is 100% owned by Ascot. Prior to Ascot's acquisition, the property was held under a joint-venture agreement between Jayden Resources (Canada) Inc. (Jayden Canada), a subsidiary of Jayden, and Mountain Boy Minerals Inc. (MBM). Jayden Canada owned 80% of the property, with the remaining 20% owned by MBM. On 29 October 2018, Ascot announced that it had completed purchase of the outstanding shares of Jayden Canada in exchange for 14,987,497 Ascot shares, plus an additional 192,000 Ascot shares for settlement of options and warrants. Concurrent with this, Ascot acquired MBM's 20% interest in exchange for 3,746,874 Ascot shares, plus an additional 48,000 shares for settlement of Jayden options and warrants. A small portion of the Silver coin property is subject to a 2% NSR on the INDI claims pursuant to an earlier purchase agreement with Jayden. The NSR can be bought back for $1,000,000 for each 1% NSR.

1.4.2 Red Mountain Project

The RMP consists of 47 contiguous mineral claims for a total of 17,125 ha. It is located approximately 18km east–northeast of Stewart, British Columbia.

Land Tenure and Acquisition Agreements

RMP is 100% owned by Ascot after acquisition on March 27, 2019, when Ascot announced that it had completed purchase of the outstanding shares of IDM Mining in exchange for 35,078,939 Ascot shares, 715,500 Ascot options, and 4,309,128 Ascot warrants.

RMP is subject to payment of production royalties, payment of an annual minimum royalty of $50,000 on the key Wotan Resources Corp. claim group, a one-time payment upon commercial production, and a gold metal streaming arrangement. There is a 1.0% NSR payable to Franco-Nevada and a 2.5% NSR payable to Wotan Resources Corp. Alternatively, Seabridge may elect to receive a one-time cash payment of $4 million at the commencement of production in exchange for a buy-back of the gold metal stream.

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

1.5.1 Accessibility

PGP is readily accessible along the gravel-surface Granduc Road from Stewart through the town of Hyder, Alaska, and back into British Columbia. Additional access is provided by old haul and forest roads that are accessible by all-terrain vehicle (ATV), snowmobile, or hiking. Several helicopter companies maintain bases in Stewart.

RMP, 23 km to the southeast of the PGP mill, is currently accessible only by helicopter. In 1994, LAC Minerals Ltd. (LAC) partially developed road access up the Bitter Creek valley from Highway 37A for 13 km to the Hartley Gulch–Otter Creek area. Currently this road is passable for only a few kilometres from the highway. The remainder is not passable, as sections have been subjected to washout or landslide activity.

1.5.2 Climate

Located at sea level, Stewart has a coastal rainforest climate, with a yearly precipitation of approximately 1,843 mm, much of it as snow, and an average yearly temperature of 6°C, according to Environment Canada.

Average monthly temperatures are minus 3.7°C in January and 15.1°C in July. Significant snowfall accumulations restrict fieldwork at higher elevations. A weather station was established at the site in 2001.

Climatic conditions at Red Mountain are dictated primarily by its elevation (1,742 m at the centre of the deposit) and proximity to the Pacific Ocean. Temperatures are moderated year-round by the coastal influence. Precipitation is significant in all months, with October being the wettest. Even at sea level, over one-third of the annual precipitation falls as snow. This proportion is greater at higher elevations, where snow may fall at almost any time of the year.

1.5.3 Infrastructure and Local Resources

Stewart reportedly had a population of 37,367 in 2016. The district has a long mining history over the past 100 years. The town provides services, including fuel, groceries, lodging, schools, hospital, air strip and helicopters base and a workforce. Situated at the head of the Portland Canal, Stewart is Canada's northernmost ice-free deep sea port with existing loading facilities for concentrate. Nearby Hyder, Alaska, has a population of approximately 90.

Principal infrastructure at the PGP consists of the following:

- Crush-grind-cyanidation processing plant building for semi-autogenous grinding (SAG) mill and ball mill (removed at time of closure in 1996) with rated capacity of 2,000 t/d up to 3,000 t/d depending on grind size and ore hardness
- Mill, shop, assay laboratory, cold storage building
- Camp and environmental monitoring office at 6 Level
- a 25 kV powerline from Stewart.
- A 31 MW power hydro electric plant owned by Long Lake Hydro Inc. approximately 700 m from the PGP Mill. The plant is connected to the BC Hydro grid and provides power to the Brucejack Mine
- 1.6 megawatt hour (MWh) generator
- Tailings Storage Facility
- Water monitoring and treatment systems, including settling ponds
- Access and site road
- Underground development and portals and waste dumps.

RMP is located approximately 13 km from the BC Hydro transmission line that runs adjacent to Highway 37A. Infrastructure at the site comprises 1,500 m of existing underground decline and drift development that was fully rehabilitated in 2016 and 2017. Surface tote road network, basic surface structures (42-person winter camp buildings, helipad, and waste rock storage areas), a shop, generator building, fuel tanks, and used mobile equipment.

1.6 History

1.6.1 Premier Gold Project

Exploration commenced in the region in the latter part of the 19th century, with the first discoveries in the district occurring in 1898. The Premier mine operated as an underground operation from 1918 to 1968 with short interruptions. In that period of time, the mine produced 1.8 Moz of gold and 41 Moz of silver at average grades of 12.15 g/t Au and 269 g/t Ag.

In 1989, Westmin Resources constructed the current mine site and resumed production utilizing open pit and underground mining until 1996 when production was terminated. Westmin produced 200,000 oz of gold and 2 Moz of silver at average grades of 2.6 g/t Au and 53 g/t Ag.

In 1991, approximately 100,000 tonnes of ore were extracted from the Silver Coin deposit and processed at the Premier mill. The average grade of this material was 8.9 g/t Au and 55 g/t Ag.

1.6.2 Red Mountain Project

Placer mining commenced in the early 1900s in Bitter Creek, downstream from Red Mountain. The Red Mountain deposit was discovered in 1989 by Bond Gold Canada Inc. (Bond). LAC acquired Bond in 1991 and surface drilling on the Marc, AV, and JW zones continued in 1991–1994. Underground exploration of the Marc zone was conducted in 1993 and 1994. In 1995, Barrick acquired LAC, which subsequently optioned the property to Royal Oak in 1996. North American Minerals Inc. (NAMC) purchased the property from the receivership sale of Royal Oak in 2000. NAMC subsequently sold the property to Seabridge in 2002 which optioned the property to Banks Island Gold Ltd. (Banks). Banks terminated the option in 2013, and the property reverted to Seabridge. Seabridge subsequently optioned the property to IDM Mining Ltd. (IDM) in 2014.

No historical production has taken place at Red Mountain.

1.7 Geological Setting and Mineralization

1.7.1 Premier Gold Project

The Premier property is mainly underlain by Jurassic-aged Hazelton Group rocks composed of a thick package of homogeneous andesitic tuffs, lapilli tuffs, and flows interpreted to have formed in an Island Arc setting.

Dykes and sills of Premier porphyry (a quartz-K-spar-hornblende porphyry of intermediate composition) are the most abundant intrusive rocks in the area, and are spatially associated with some mineralized zones, particularly at Premier.

Gold–silver mineralization is hosted within structural zones expressed by quartz breccias, quartz veins and stockwork often within large areas of quartz-sericite-pyrite alteration. Elevated gold and silver values are closely associated with silicification and sericitic alteration. Gold occurs predominantly as electrum, with native gold present locally. Silver occurs in its native form, and in electrum, argentite, and freibergite. The most common sulfides are pyrite, sphalerite with minor galena, and chalcopyrite. The mineral assemblage suggests that the style of mineralization at Premier falls into the intermediate sulfidation epithermal category as neither high-sulfidation minerals (such as covellite or enargite) nor low-sulfidation minerals (such as arsenopyrite and pyrrhotite) have been observed.

1.7.2 Red Mountain Project

The geology of the Red Mountain area is characterized by Upper Triassic to Lower Jurassic metasedimentary and tuffitic units that have been intruded by a multi-phase intermediate intrusive complex. The intrusive rocks show porphyry style alteration with K-spar alteration and tourmaline as well as lower temperature quartz-sericite-pyrite alteration. Gold mineralization is hosted in a series of pyrite rich breccia bodies and stockwork zones associated with the brecciated contact zone at the edge of the intrusive body. The alteration associated with the high grade-gold mineralization is characterized by sericite and silicification.

Eocene intrusions of the Coast Plutonic Complex occur to the west and south of Red Mountain and are associated with high-grade silver-lead-zinc occurrences; gold-silver-bismuth ± copper-lead-zinc mineralization recently identified in the Lost Valley area is likely of Eocene age.

Recent interpretation is that the gold mineralization at Red Mountain is consistent with an intrusive-related hydrothermal system, rather than a porphyry-gold deposit according to previous interpretation.

1.8 Deposit Types

1.8.1 Premier Gold Project

Mineral deposits in the PGP are intermediate-sulfidation epithermal gold-silver deposits with subsidiary base metals. These deposits form at comparatively shallow depths (generally above 1 km depth), often in association with hot-spring activity on surface. Mineralization results from circulation of aqueous solutions driven by remnant heat from intrusive bodies. Where these ascending fluids encounter meteoric waters, and/or as the hydrostatic pressure drops, changes in temperature and chemistry results in precipitation of minerals into fractures, breccias, and open spaces.

Mineralized bodies are structurally controlled veins, stockworks, and breccia bodies, and are broadly tabular with a wide range of orientations. They measure from centimetre scale to many metres in thickness and can often be traced for strike lengths of several hundred metres or even kilometres. Economic minerals comprise native gold and native silver, electrum, silver sulfosalts, and silver sulfides, along with pyrite, sphalerite, and comparatively minor amounts of chalcopyrite and galena. Gold and silver values are quite variable, and average on the order of 5 g/t to 10 g/t Au and 20 g/t to 30 g/t Ag within the historical stopes.

1.8.2 Red Mountain Project

Several models have been presented for the formation of the RMP gold deposits. Rhys et al. (1995) concluded that the setting and style of mineralization is similar to that of many porphyry systems. This was based on data from deep drilling that indicated mineralization and alteration zoning common to traditional porphyry systems. Lang (2000b) suggested that, while the porphyry system zonation was present, the alteration and mineralization was more consistent with a later magmatic-hydrothermal system that overprinted the earlier vertical alteration pattern. Recent interpretation is that the gold mineralization at Red Mountain is consistent with an intrusive-related system, rather than a porphyry-gold deposit.

Incorporating recent suggestions for regional early-Jurassic intrusive-related and magmatic-hydrothermal mineralization in northwest British Columbia, which incorporate mapping and petrographic observations (Lang, 2000a, 2000b), the proposed metallogenic sequence for the Red Mountain property is as follows:

- Approximately 200 Ma, the Hillside porphyry intruded into Stuhini and unconsolidated lower-most Hazelton Group strata. Large rafts of sedimentary rocks are encapsulated in the intrusion; and contact brecciation is between porphyry and sedimentary rocks.
- The Hillside porphyry cools and contracts, causing microfracturing of the porphyry and breccia zones. Early pyrite was deposited into these fractures.
- Ongoing cooling, and alteration of hosts rocks by hydrothermal fluids, with fracturing and brecciation of coarse-grained pyrite veins. Additional coarse-grained pyrite is deposited. The early gold mineralization, including petzite, is deposited as small inclusions in pyrite grains.
- Intrusion of the Goldslide porphyry, including quartz-phyric phase. The intrusion drives a pulse of hydrothermal fluids, primarily containing native gold with local tellurides and sulfosalts, into fractures and rims in the coarse-grained pyrite veins.
- Final infilling of remaining fractures in the coarse-grained pyrite veins with gold minerals, fibrous quartz, calcite, feldspar, and sericite.
- Intrusion of biotite-phyric phase of Goldslide Suite.
- Mid-Jurassic extensional tectonism.
- Cretaceous transpressional tectonics; recumbent folding of mineralization and favourable breccia horizon.
- Intrusion of multiple phases of 57.3 Ma McAdam Point stock; intrusive-related/porphyry gold-molybdenum quartz stockworks and disseminations.
- Remobilization of gold and sulfides at Lost Valley during subsequent thrusting.
- North–south faults with minor offset; pyrrhotite-dominant gold-silver base-metal veins.
- Intrusion of andesite and lamprophyre dykes.

1.9 Exploration

1.9.1 Premier Gold Project

Recent exploration work has been conducted continuously by Ascot since acquisition of the Property in 2007, and has been successful in delineating Mineral Resources at Big Missouri, Martha Ellen, Dilworth, Premier, and Silver Coin.

Large parts of the property are covered by glacial till or younger units of the Betty Creek Formation. It is very likely that additional "blind" mineralization exists on the property. Ascot has established that known mineralization can be detected using modern IP technology and plans to explore the covered parts of the property utilizing geophysical and geochemical techniques.

The next step for Ascot is to continue to drill to increase the Indicated Resource at the PGP. The 2019 drill program at PGP increased by 60% the in-situ gold ounces in the Indicated category. Many areas of the remaining modelled Inferred resources for these three deposit areas require deeper drilling which can be more efficiently drilled from underground. Additional drilling to convert Inferred resources to the Measured and Indicated (M&I) categories is planned to be conducted from underground.

It will be necessary to identify a suitable area to extract a bulk sample to improve grade reconciliation. The mineralization contains a lot of coarse gold, and a tightly controlled bulk sample with narrow drill spacing followed by complete extraction of material should aid in understanding the grade variation within the deposit.

Ascot's exploration budget for the 2020 program at PGP is $4.0 million

1.9.2 Red Mountain Project

Past exploration is summarized in Sections 1, 6, and 10. No exploration was conducted from 2001 to 2012 as the property was on care and maintenance by Seabridge. In 2012, Banks drilled three drill holes in the Marc zone, two of which intersected the Marc mineralized zone, and the third hole was abandoned prior to reaching the Marc zone.

Exploration potential for the property is deemed to be high. Since 1994, when the surface exploration was terminated, the glaciers surrounding the RMP have significantly receded, exposing considerable area that was previously inaccessible. The intrusion system that hosts the current resource has a broad areal extent, and surface prospecting, mapping, geochemistry, geophysics, and drilling have the potential to discover similar deposits. Additional drilling also has the potential to expand the current resource zones.

The exploration programs completed to date are appropriate for the style of the mineralization and prospects located on the Project. There are a number of prospective targets for further exploration on the property.

1.10 Drilling

1.10.1 Premier Gold Project

Drilling on the PGP dates back to 1928. The Ascot database contains a total of 8,029 holes and 875,340 m. 3,406 of these holes representing 138,806 m are from years 1928 to 1941. These cover the entire property, are generally shallow, and have unreliable assay results. They have therefore not been used for resource modelling.

The database used for this Feasibility Study includes 1,879 holes and 152,005 m of legacy drilling from 1974 to 1996 that was predominantly drilled by Westmin. Jayden/MBM also drilled 476 holes and 74,741 m at Silver Coin prior to being taken over by Ascot.

The remainder of the database is comprised of 2,268 holes and 509,789 m drilled by Ascot between 2007 and September 2019.

Most of the legacy holes were selectively sampled in zones of visible sulfide mineralization. No assay Quality Assurance/Quality Control (QA/QC) data is available for these drill holes. Validation work conducted by Ascot personnel has demonstrated that the legacy drilling results in the Premier deposit area are generally reliable and so this data has been used for the Resource Estimates, with some restrictions. Details regarding this validation work are provided in the Data Verification section of this report.

Ascot commenced drilling on the Property in 2007 with drilling in 2007 and 2008 restricted to the Dilworth area. From 2009 to 2014, most of the drilling was on Big Missouri with comparatively modest programs on Martha Ellen and Dilworth, and only minor drilling in the Premier area. Most of the work from 2015 to the end of 2017 was in the Premier area. In 2018 and 2019 Ascot has done in-fill drilling at Premier, Big Missouri, and Silver Coin.

1.10.2 Red Mountain Project

A total of 699 surface and underground diamond drill holes (180,426 m) have tested a variety of targets on the RMP property. The majority of the historical drilling tested the Marc, AV and JW zones.

A total of 406 holes (100,298 m) were drilled by Bond and LAC between 1989 and 1994, and 60 holes totalling 29,671 m were drilled by Royal Oak in 1996. During 2012, Banks Island completed 3 drill holes for 681 m in the Marc zone.

From 2014 to 2018 IDM Mining completed 230 drill holes for a total of 49,667 m on the property. Most of these holes were drilled into known mineralization from the existing underground drift.

1.11 Sample Preparation, Analysis, and Security

1.11.1 Premier Gold Project

Sample preparation for drill samples consists of drying as required, crushing, and selection of a sub-split which is then pulverized to produce a pulp sample sufficient for analytical purposes using standard fire assays and ICP procedures.

Ascot has maintained a fairly consistent program of independent assay QA/QC since 2007. The programs include the addition of Certified Reference Materials (CRM), blanks, and duplicates to the sample stream, as well as pulps sent from the principal laboratory to a secondary laboratory for checks. Control samples are added at a nominal rate of one for every ten samples, with blanks and standards alternated and the grade range of the CRM continually rotated.

Ascot maintains a secure logging and storage facility in Stewart, BC. All sample collection and handling are supervised by Ascot personnel. Collected samples are stored in bags sealed with a zap-strap and the samples are combined in large woven rice bags for shipping. The contents of each sealed rice bag are recorded, and full bags are stacked on pallets and shipped by commercial carrier (Bandstra Transportation Systems Ltd., with a head office in Smithers, BC) to the assay laboratory in Vancouver, BC in secure transport trucks.

The QA/QC for the legacy data at Silver Coin from the MBM/Jayden era is not fully comprehensive but indicates that thought and effort was given to a control system. A review of the available QA/QC indicates acceptable credibility to the data of this era.

Sample preparation, analysis, and security is acceptable for all drilling used in the Resource.

1.11.2 Red Mountain Project

Sample preparation at RMP has followed the similar procedures at PGP; drying as required, crushing, and selection of a sub-split which is then pulverized to produce a pulp sample sufficient for analytical purposes using standard fire assays and ICP procedures.

For all Red Mountain drilling programs samples were under the control of drill contractors and Project staff until they have left the immediate area as it has helicopter access only.

The historical QA/QC for Red Mountain is not as robust as current QA/QC programs. Standard and duplicate coverage is weak for some programs and no blanks were run to test for contamination issues associated with sample preparation on all but the recent IDM drilling programs. However, most of the historical work was carried out between 1993 and 1994 and the program was quite strong and extensive for its time. Additionally, strong check assay programs from some of the earlier years mitigate other weaknesses.

IDM's QA/QC protocols have followed standard industry practices and are deemed adequate for inclusion of the assay data in resource estimation.

1.12 Data Verification

1.12.1 Premier Gold Project

In the QP's opinion, the Ascot drill data have generally been collected in a manner consistent with industry best practice. The assaying used for the Resource Estimate has been carried out at accredited commercial laboratories using conventional industry-standard methods. Ascot has implemented an assay QA/QC program that is also consistent with best practice guidelines.

Due to the lack of information for the legacy drilling at all properties, the data have been verified by an extensive re-assay program of pulps and core. In all cases relevant to legacy drilling the conclusion is that grades within the range applicable to this study have been validated and may be used for Resource Estimation. Portions of Indicated blocks have been down-graded to Inferred in some areas of Silver Coin, Dilworth, and Martha Ellen due to lack of QA/QC for some legacy assays.

Data collection have been updated in 2019 to consist of a comprehensive property-wide database.

The database verification procedures applied by Ascot comply with industry standards and are adequate for the purposes of Mineral Resource estimation. This includes the validation for use of the legacy drill results, for values above 0.3 g/t Au.

1.12.2 Red Mountain Project

Data verification has been carried out by previous operators of the Project including Bond, LAC and NAMC.

During 2000, NAMC cross-referenced and catalogued all data from previous operators. Data that could not be verified were removed from the database (Craig et al., 2014).

The verification programs undertaken on the collected data adequately support the geological interpretations. The analytical database quality therefore supports the use of the data in Mineral Resource estimation.

1.13 Mineral Processing and Metallurgical Testwork

The ores from the Premier and Red Mountain deposits were subject to numerous metallurgical testwork campaigns dating back to the 1980s. The majority of these testwork efforts are referenced in this Feasibility Study, but this section primarily focuses on the selected campaigns that provided the relevant basis for the design that is applicable to the forward mine plans described in this study.

1.13.1 Premier Ore Testwork Overview

Testwork on Premier ores dates back to 1987 when Coasttech lab in North Vancouver, Lakefield Research, and Allis Chalmers were previously engaged to perform the direct ore cyanidation and comminution testwork on the Big Missouri and Silver Coin deposits, which results were the basis for the design of the existing Premier process plant.

In 2015 a metallurgical testwork program on the Premier ores was conducted at the ALS Laboratory in Kamloops, and in 2018, Base Metallurgical Laboratories Ltd. (BML) also conducted a metallurgical testwork campaign. The above-mentioned BML program labelled “Metallurgical Testing on Samples from the PGP – BL0366, November 27, 2018” is most comprehensive in nature, and presents the basis for the process design presented in this report.

Most recently, metallurgical testwork was performed at the SGS facility in Burnaby, and at BML and Kemetco Research facilities, with the aim to address gaps (most notably extended gravity recoverable gold [E-GRG] testwork) from the 2018 BML campaign, and to ensure that the overall testwork can support the feasibility study level of Project definition. Detailed descriptions of the conducted testwork are presented in Section 13 of this report, while this section provides only the significant highlights.

The focus of the testwork for Premier ores was ore characterization, head assays, mineralogy, and ore hardness; hydrometallurgical and dewatering testwork; followed by the subsequent cyanide detoxification testing.

Head Assays, Mineralogy, and Ore Hardness

Head assaying has revealed the presence of recoverable coarse gold, which has warranted investigation into gravity separation inclusion into the overall process flow sheet. The total organic carbon content was low, at levels that would be unlikely to have an effect on the gold dissolution. Sulfur was present at levels of up to 7.8%, indicating sulfur mineralization, of which pyrite was the predominant sulfide mineral.

Comminution testwork was conducted on the Premier ore samples, consisting of Bond Abrasion (Ai), SAG mill comminution (SMC), and rod and ball mill grinding index testing. The results from this testwork have revealed that Premier ores are considered to be:

- Moderately abrasive to abrasive
- Moderately hard to hard from the SAG milling perspective
- Moderately hard to hard from the rod and ball milling perspective.

Hydrometallurgical Testwork

All Premier samples were subjected to gravity testwork, followed by whole-ore leach (WOL) or CIL. In addition, E-GRG testing was completed on the NorthStar, Silver Coin, and Big Missouri samples, which were excluded from this testing during the BML 2018 metallurgical testwork campaign. The Premier ore samples have exhibited acceptable gold recoveries ranging from 18.3% for the Silvercoin samples up to 37.8% Au recovery for the Premier samples, which have shown the best response to gravity testwork.

The combination of a gravity circuit and CIL has produced gold recoveries ranging from 93.5% for the Big Missouri and up to 98.4% for the Premier deposits, which are superior when compared to the gravity/WOL-circuit configuration testwork results.

Dewatering and Cyanide Detoxification Testwork

The dewatering testwork results have revealed that 30 g/t addition anionic flocculant can produce thickener underflow densities of 60% solids weight for weight (w/w). However, additional testwork is required to improve the clarity of the thickener overflow and identify the common flocculant for processing of the both Premier and Red Mountain ores.

Based on the testwork results, the SO_2/Air cyanide destruction process was successful in reducing the (CN_{WAD}) levels to below 1 part per million (ppm) for all the Premier deposits and is the preferred cyanide destruction method for the Project.

Premier Ore Process Flowsheet Selection and Recoveries

With consideration of the comminution characterization testwork, a SAG/Ball (SAB) milling circuit configuration is selected for processing the Premier ores. Based upon the historical and most recent metallurgical testwork, a gravity concentration/intensive leach, followed by CIL, is the recommended process plant configuration for the Premier ores, as it provides the highest overall metal recoveries and best project economics.

The Premier ore testwork data suggests that the overall (gravity with leach) expected gold and silver recoveries of 95.4% and 71.5%, respectively, can be achieved.

1.13.2 Red Mountain Testwork Overview

Lakefield Research performed initial metallurgical testing on Red Mountain samples in 1991, followed by several testwork campaigns in the early and late 1990s that were primarily focused on the cyanide leaching as a sole process for extracting gold and silver from the deposit. Starting with Process Research Associates (PRA) testwork in 2000, and continuing with Gekko Systems' testwork in 2015, the focus has shifted towards a production of the precious-metals rich flotation concentrates. All testwork reports on the Red Mountain ore between 1991 and 2015 have been listed in the reference section of this document and have been summarized in the "NI 43-101 Preliminary Economical Assessment Technical Report for the Red Mountain Project in BC, Canada, JDS, August 25, 2016".

The focus of the JDS study is mainly the 2016/2017 metallurgical testing completed by Base Metallurgical Laboratories in Kamloops, labelled BL0084 and BL0184 testwork program, because outcomes from this testwork present the basis of the recovery method and process design criteria outlined in Section 17 of this document. The main objectives of the 2016/2017 metallurgical test program conducted on variability and composite samples were to:

- Define the metallurgical response of the two available process options—gravity/flotation/leach (GFL) and a WOL cyanidation of the Red Mountain samples
- Generate advanced process engineering data for equipment selection
- Generate tailings samples for environmental testing.

Confirmatory testwork was completed in 2019/2020 at the BML facility, with the intent to fill in any gaps from the previous testwork campaigns; establish fine grinding parameters; assess gold and silver metallurgical recoveries and the efficiency of the suggested cyanide detoxification methods; as well as generate liquid/solid separation data needed for tailings disposal.

Head Assays, Mineralogy, and Gold Deportment

The head assays have revealed that gold, silver, and sulfur were highly variable throughout the variability samples, with the sulfur grades registering up to approximately 19% of the total composition. This is an indication of significantly higher sulfide mineralization presence when compared to the Premier ore samples. The presence of sulfides could possibly predetermine the process selection, as this can result in higher cyanide and oxygen consumption, as well as the requirement for the costlier and more-complex cyanide detoxication process.

The testwork from 2019 has confirmed that pyrite and pyrrhotite, when combined, represent the majority of the above-mentioned sulfide mineralization, which is as high as 35% in the samples tested. This is very important, as pyrrhotite, being highly reactive is prone to oxidation, can have a detrimental effect if a flotation circuit is to be considered. It was also noted that sphalerite was present, but not in the amounts which would warrant its economical extraction.

During this testwork campaign the gold deportment testing was also completed, which concluded that the majority of the gold was unliberated and locked with the pyrite. Also, there was no significant coarse gold present, but rather the majority of the gold particles are very fine (< 10 microns [µm] size), which could possibly rule out gravity concentration as an option. The fact that the gold was very fine and locked with pyrite suggested that a fine/tertiary grinding stage will be required to achieve economical gold and silver recoveries.

Comminution Testwork

Bond crushing index, Bond Ai, SMC, and Ball mill grinding index testing were conducted on the Red mountain ore samples. Results from this testwork have reveal that the ores are considered to be:

- Abrasive and average hardness with the respect to coarse-particle breakage
- Hard from a comminution perspective due to the SMC values being located in the 80th to 95th percentile in the JK database.
- Hard or very hard from the ball milling perspective, Bond ball mill grinding index values were as high as 22.2 kilowatt hours (kWh)/t.

The fine gold particle size distribution and its deportment nature prompted fine-grinding/signature-plot testing at the Glencore certified ALS facility in Kamloops in 2019. The testwork goal was to rectify misleading results from a 2015 testwork campaign and to determine specific energy requirements needed for the fine grinding circuit design.

With consideration of the available testwork data from all three Red Mountain deposits, the estimated specific energy requirement for the fine grinding application is 25.6 kWh. With this in mind, coupled with the testwork data and the fine-grinding circuit throughput, it is estimated that a high-speed stirred mill with an installed power of 3 megawatts (MW) will be required for the tertiary milling application.

Hydrometallurgical Testwork

Based on numerous historical testwork campaigns referenced in this report, two valid process options were pursued further: WOL and GFL. Testwork concluded that recoveries were comparable, but according to the 2016 JDS PEA study, the GFL circuit presented more favorable preliminary capital and operating cost estimates.

The initial focus for flowsheet development centered on the GFL testwork, which was conducted on the Red Mountain variability samples. The mass recovery of a gravity concentrate was considered poor for the majority of samples, with the average recovery for all samples at 11%. The results confirm the presence of minor amounts of gravity-recoverable gold and suggest that the application of the gravity separation circuit will not benefit the process from a metallurgical perspective.

Due to the high initial sulfide content in the samples, the mass recovery of the rougher concentrate was quite high, with a peak value of 48% and an average of 27%. This would likely reduce the advantage of a flotation pre-concentration stage, as significant rougher concentrate mass will require a regrinding and leaching stage, prompting a possible increase in the fine-grinding energy requirement and increased capital costs in the leaching stage.

The rougher concentrates were subsequently reground to 27 µm and leached with cyanide to determine the overall gold and silver extractions. Many of the samples have exhibited significant gold and silver losses to the leach residue, which has caused average lower recoveries for the GFL circuit than what was initially expected. The average gold recoveries were ranging from 67.2% for the JW deposit up to 82.9% for the AV deposit. The silver recovery range was from 50.1% for the JW deposit up to 71.1% for the AV deposit.

It was evident that the overall performance of the GFL process was quite variable and could have been influenced by several factors such as:

- Low initial flotation recovery which could be attributed to high pyrrhotite content that is reactive and susceptible to rapid oxidation
- Poor leach performance attributed to high cyanide consumption and high oxygen demand of the sulfides (pyrrhotite) resulting in insufficient oxygen in the leach
- Uneven rougher concentrate particle-size distribution, as coarser particles tend to exhibit poorer performance.

Due to the inconsistent performance of the GFL circuit, this flowsheet option was not investigated any further, and the focus shifted to WOL testing. The aim of the subsequent WOL testing was to establish which of the CIL or carbon-in-pulp (CIP) configurations would be more suitable for leaching Red Mountain ores. It is worth noting that testwork has proceeded with the Marc and AV deposits, while the JW deposits were excluded.

The Marc and AV composite samples were leached under CIL and CIP configurations, and the effect of factors such as primary grind size, lead nitrate addition, sodium cyanide concentration, pH, and pre-oxygenation were observed.

Ores from the Red Mountain deposit were quite sensitive to the particle grind size compared to ores from the Premier deposit. The highest achieved gold extractions were 93% for the Marc master composite at a P_{80} of 17 µm; and 89% for the AV master composite, at a P_{80} of 16 µm. At the same particle grind size of 37 µm, CIL tests were outperforming CIP tests, which was particularly visible for the Marc master composite samples where a 3% increase in gold extraction was observed.

The effect of lead nitrate was negligible, as final extraction rates were constant, and an increase in sodium cyanide concentration to 2,000 ppm caused an increase in the metals extraction rates; in contrast, a sodium cyanide concentration of 500 ppm was causing a decrease in extraction rates. The testwork concluded that an adjustment of the pH and an application of air and oxygen pre-oxygenation did not have a noticeable effect on the gold and silver extractions.

Dewatering and Cyanide Detoxification Testwork

The dewatering testwork results have revealed that 60 g/t addition anionic flocculant can produce thickener underflow densities of 50% solids w/w. Compared to Premier ores, the higher flocculant dosages and lower densities were due to a much finer leach circuit feed particle size (25 µm vs. 80 µm). Additional testwork to improve the thickener overflow clarity by investigating the effect of pH and addition of coagulant should be explored in the next phase of the project.

Based on the testwork results, the SO_2/Air cyanide destruction process was successful in reducing the CN_{WAD} levels to below 1 ppm for the Red Mountain ores; however, additional testing on the JW samples is still recommended.

Red Mountain Ore Process Flowsheet Selection and Recoveries

The Red Mountain ores are characterized by low amounts of free gold; therefore, the application of the proposed Premier gravity circuit will not yield any benefits in processing of these ores. During the processing of the Red Mountain ore at the Premier mill facility the gravity separation circuit will be bypassed.

The testwork results confirmed that the WOL circuit would yield higher recoveries and encourage a more suitable economic outcome when compared to the GFL. The CIL circuit configuration outperformed the CIP circuit arrangement and is considered to be better suited to lower throughputs.

Based upon the available testwork data, precious metal recoveries for the Red Mountain ores are sensitive to the particle grind size, so therefore a leach feed particle grind size of 25 µm is recommended as the basis of design. To achieve acceptable metal recoveries the integration of a tertiary/fine grinding mill will be required for the process plant. Due to the fine grind size, installation of a 27 m diameter high-rate thickener will be required to achieve an acceptable slurry density for the CIL circuit.

The SO_2/Air cyanide destruction process is recommended for the design basis as the test results demonstrated a successful reduction in the CN_{WAD} concentrations to below 1 ppm for both the Premier and Red Mountain deposit ores.

The estimated expected gold and silver recoveries for the Red Mountain ores are 86.8% and 83.6%, respectively.

1.14 Mineral Resource Estimate

The total Mineral Resource Estimate for the Project includes the resources of both, the PGP area and RMP area.

The resources at the PGP area include the Premier, Big Missouri, Silver Coin, Martha-Ellen, and Dilworth deposits. This work was completed by Susan Bird, P.Eng. (APEGBC) with an effective date of December 12, 2019.

The resource estimation work at the RMP was completed by Dr. Gilles Arseneau, P.Geo. (APEGBC) and the effective date of the RMP Mineral Resource Statement is August 30, 2019.

1.14.1 Premier Gold Project

The Mineral Resources for the PGP have been updated since the previous estimate in January 2019 (Rennie et al., 2019) due to additional drilling and updated geologic interpretation for the Premier, Big Missouri, and Silver Coin deposit areas.

The Resource Estimate is based on 4,623 drill holes for 736,535 m of drilling (Ascot holes account for 509,000 m of that total).

The geological models for all five deposit areas at PGP consist of interpreted shapes of mineralized zones and of post-mineral porphyry dikes and faults. Mineralization within each of the deposits is interpreted to have been emplaced by sub-vertical structures which acted as conduits to fluid flow.

The Mineral Resource Estimate is based on “mineralized percent” block models with 3 m x 3 m x 3 m sized blocks for each area. There are up to two separate mineralized domains allowed within each block, with the domain code and the percent of each domain within the block stored and used in the resource estimation.

Grade shells have been created in each area to confine material at a cut-off grade of approximately 2.0 g/t gold equivalent (AuEq) and a nominal minimum true thickness of approximately 2.0 m. In peripheral areas of the resource where the position and orientation of modeled zones was less defined, thinner intervals at lower grade have at times been used to connect individual intercepts into a coherent zone. Gold and silver grades were interpolated inside each solid domain using 1 m composites, with no sharing of composites between domains. The True Thickness values have also been interpolated inside each domain solid. Mineralized areas above the Resource cut-off of 3.5 g/t AuEq, but with True Thickness values that are less than 2.5 m are not included in the Resource Estimate.

An average bulk density of 2.85 t/m^3 for Premier and 2.80 t/m^3 for the other four deposits were used for all rock types within each block model, based on data collected by Ascot from drill core.

High grade samples were capped at various levels, depending on domain, as described in the text of this report. Composites have been restricted during interpolation at outlier values to limited search distances depending on domain.

The blocks were classified according to CIM (2014 and 2019) definitions as follows:

- All Classified material must be within a potentially mineralized wireframe and have a minimum minable true thickness of 2.5 m.
- Blocks within a wireframe and within an anisotropic search ellipse with dimensions of 100 m x 100 m x 15 m are assigned a preliminary classification of Inferred.
- Indicted blocks are required to have at least one of the following criteria:
  - The average distance to the nearest 3 drill holes is less than 35 m with none further than 35 m, and there are samples from at least 2 "split quadrants", or
  - the average distance to the nearest two drill holes is less than 17.5 m, and there are samples from at least 2 "split quadrants", or
  - the distance to the nearest drill hole is less than 10 m and at least 2 drill holes have been used in the estimate.

The Mineral Resource with an effective date of December 12, 2019 is listed Table 1-1, using a 3.5 g/t AuEq cut-off. CIM definition standards for mineral resources and mineral reserves (CIM, 2014) were followed for the Mineral Resource Estimate.

Table 1-1: Premier Area Resource Estimate at a 3.5g/t AuEq Cut-off – Effective date: December 12, 2019

Class	Deposit	In-situ Tonnage (kt)	In-situ Grades			Metal	
			AuEq (g/t)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
Indicated	Premier	1,298	8.90	8.46	64.20	353	2,680
	Big Missouri	1,116	8.48	8.36	16.90	300	607
	Silver Coin	1,597	7.77	7.61	23.00	390	1,181
	Martha-Ellen	130	5.80	5.47	48.00	23	201
	Dilworth	-	-	-	-	-	-
	Total Indicated	**4,141**	**8.25**	**8.01**	**35.1**	**1,066**	**4,669**
Inferred	Premier	1,753	7.00	6.72	39.80	379	2,243
	Big Missouri	1,897	8.44	8.34	14.70	508	896
	Silver Coin	523	7.19	7.03	23.20	118	390
	Martha-Ellen	653	6.36	6.12	34.30	129	720
	Dilworth	235	6.51	6.13	56.10	46	424
	Total Inferred	**5,061**	**7.45**	**7.25**	**28.7**	**1,180**	**4,673**

Notes: **1.** Mineral Resources are estimated at a cut-off grade of 3.5 g/t AuEq based on metal prices of US$1,300/oz Au and US$20/oz Ag. **2.** The AuEq values were calculated using US$1,300/oz Au, US$20/oz Ag, a silver metallurgical recovery of 45.2%, and the following equation: AuEq = Au g/t + (Ag g/t x 0.00695). **3.** A mean bulk density of 2.85 t/m^3 is used for Premier and of 2.80 t/m^3 for all other deposit areas. **4.** A minimum mining width of 2.5 m true thickness is required to be classified as Resource material. **5.** Numbers may not add due to rounding.

1.14.2 Red Mountain Project

The Red Mountain mineral resource model used a total of 699 drill holes, 230 of which were drilled by IDM between 2014 and 2018.

ACS audited the database used to estimate the Red Mountain mineral resources, and is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the gold mineralization, and that the assay data are sufficiently reliable to support mineral resource estimation.

Grade estimates were based on capped composited assay data. Gold values, depending on the zone, were top cut in a range from 20 g/t to 75 g/t and silver values were top cut in a range from 45 g/t to 500 g/t. For the updated 2019 mineral resource estimate, it was decided to cap raw assays with top cuts for gold and silver on a zone by zone basis. The most significant capping was undertaken in the Marc and AV zones of the deposit.

Block modelling was performed using 4 m x 4 m x 4 m blocks. ACS considers that blocks in the Marc, AV, and JW zones estimated during pass one and from at least 3 drill holes could be assigned to the Measured Category. All other blocks interpolated during pass 1 in the Marc, AV and JW zones were assigned to the Indicated Category. Blocks estimated with at least 3 holes during pass 2 in all zones were classified Indicated. All other estimated blocks were classified as Inferred. Interpolation was by ordinary kriging, or inverse-distance squared methods on smaller or dispersed data sets, with anisotropic search ellipsoids designed to fit the strike and dips of the zones. An extensive QA/QC review was completed on all 2018 and previous exploration work and a comparative analysis was performed on drill hole data, underground bulk sampling and geology. Bulk density was interpolated using inverse-distance squared method where there were sufficient data populations. For zones with sparse data, average values from the data available for a given zone were applied.

Table 1-2summarizes the resource estimate for the Red Mountain Project.

Table 1-2: RMP Mineral Resource Statement Reported at a 3.0 g/t Au Cut-off

	Tonnage (kt)	Grade		Contained Ounces	
		Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
Measured	1,920	8.81	28.30	543.8	1,747
Indicated	1,271	5.85	10.01	238.8	409
Total Measured and Indicated I	**3,190**	**7.63**	**21.02**	**782.6**	**2,156**
Inferred	405	5.32	7.33	69.3	95.5

Note: RMP Resources are reported at a 3.0 g/t Au cut-off for underground long hole stoping.

1.15 Mineral Reserve Estimate

The calculated reserves based on the mine plans at PGP and RMP are shown in Table 1-3.

Table 1-3: Reserves by Category

Reserves by Category	Ore (t)	% of Tonnage	Grade Au (g/t)	Grade Ag (g/t)	Grade AuEq (g/t)	Ounces Au	Ounces Ag	Ounces AuEq
PGP								
Proven	-	-	-	-	-	-	-	-
Probable	3,631,898	100	5.45	19.11	5.69	636,805	2,230,964	663,920
PGP Total	**3,631,898**	**100**	**5.45**	**19.11**	**5.69**	**636,805**	**2,230,964**	**663,920**
RMP								
Proven	2,193,599	86.2	6.68	21.69	6.93	471,368	1,530,052	489,023
Probable	351,234	13.8	5.51	13.76	5.67	62,241	155,340	64,033
RMP Total	**2,544,833**	**100**	**6.52**	**20.60**	**6.76**	**533,609**	**1,685,392**	**553,056**
PGP & RMP								
Proven	2,193,599	35.5	6.68	21.69	6.93	471,368	1,530,052	489,023
Probable	3,983,133	64.5	5.46	18.63	5.68	699,046	2,386,304	727,954
PGP & RMP Total	**6,176,732**	**100**	**5.89**	**19.72**	**6.13**	**1,170,414**	**3,916,356**	**1,216,976**

Notes: CIM Definition Standards were followed for classification of Mineral Reserves
The Qualified Person for the Mineral Reserve Estimate is Frank Palkovits, P.Eng., of Mine Paste
AuEq values for PGP were calculated in the spring 2020 using $1,400/oz Au and $17/oz Ag with no allowance for silver recovery
AuEq values for RMP were completed in the fall 2019 at $1,300/oz Au and $15/oz Ag with no allowance for silver recovery
Based on current mining areas, Silver is an immaterial contributor to overall economic, but is recovered in the mill
Rounding may result in minor differences.

1.16 Mining Methods

Mining methods described herein will be applied at both PGP and RMP. In the case of RMP, the orebody is continuous and sufficiently wide to use a transverse longhole stoping method; whereas at PGP, the ore is along more discrete lenses and tend be narrow, requiring a narrow longitudinal retreat approach to longhole mining. These can be single or multiple sub-levels mined in a block.

The project employed mining methods appropriate to the local conditions at each site, where variations in geotechnical character, grade, ore thickness and ore geometry and inclination were all considered in stope optimization. The target was to develop a coordinated plan to supply 2,500 t/d to the Premier mill, optimizing the production of gold ounces at the lowest operational cost.

The Study's mine plan generally utilizes a combination of three mining methods: longhole (64%), inclined undercut longhole (14%), and room and pillar (12%), with minor amounts of cut and fill (2%) and development ore (8%) to extract the mineral reserves. A particular mining method was chosen based on an economic assessment of each method for a given geometry and geotechnical characteristics depending on its location in the deposit. The stope shapes and mine access development were individually modelled and evaluated to form the final mineable reserve.

Initial mining commences at Silver Coin (1.794 Mt) and Big Missouri (0.809 Mt), followed by RMP (2.545 Mt) and Premier (1.028 Mt). This sequencing allows mobile mining equipment and some fixed assets (electrical and ventilation) to most effectively be remobilized and re-used at different deposits as dictated by mine schedules. The combined operations produce about 6.2 Mt at 5.9 g/t Au and 19.7 g/t Ag over the LOM.

Mining dilution occurs at various rates depending on the mining method and ground conditions based on rock quality in geotechnical domains in the block model. Dilution comes in from a number of sources: planned dilution is material taken within the bounds of a stope layout while unplanned material comes from material outside the stope shape such as the hanging wall and footwall, or minor amounts from backfill. Dilution generally ranges from 10% to 40%. In some cases where two wireframes are very close together, the waste parting between the wireframes was taken providing it was economically justified. Table 1-4 summarizes the annual production from all areas.

Cut-off grades were determined for each site by mining method using a AuEq calculation:

- Initial RMP estimates were done in the fall of 2019 using prices of $1,350/oz Au and $15/oz Ag and had to account for longer trucking distance to the Premier mill. AuEq COGs used to estimate the economic potential of the stopes were: Longhole 3.11 g/t, Inclined undercut Longhole 4.0 g/t, and cut-and-fill 4.1 g/t.
- PGP estimates were done in the spring of 2020 using prices of $1,400/oz Au and $17/oz Ag, resulting in COGs: Longhole 2.85 g/t, Inclined Undercut Longhole 3.44 g/t, room-and-pillar 3.82 g/t, and cut-and-fill 3.44 g/t.

Table 1-4: Annual Production from All Areas

Mine Production	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	LOM Total
Premier	t ('000s)	-	-	-	-	-	227.2	461.5	339.5	-	1,028.2
AuEq Grade	g/t	-	-	-	-	-	5.8	5.6	5.5	-	5.6
AuEq Ounces	oz ('000s)	-	-	-	-	-	42.1	82.8	60.2	-	185.2
Silver Coin	t ('000s)	0.5	305.6	505.5	299.0	451.0	232.7	-	-	-	1,794.4
AuEq Grade	g/t	3.7	5.2	5.1	5.0	5.1	4.5	-	-	-	5.0
AuEq Ounces	oz ('000s)	0.1	51.5	82.1	48.2	73.4	33.6	-	-	-	288.9
Big Missouri	t ('000s)	8.5	305.3	304.8	179.2	11.6	-	-	-	-	809.3
AuEq Grade	g/t	7.1	6.2	7.6	8.7	4.8	-	-	-	-	7.3
AuEq Ounces	oz ('000s)	1.9	61.3	74.7	50.2	1.8	-	-	-	-	189.9
Red Mountain	t (000s)	-	-	86.2	419.6	437.3	438.7	435.6	434.2	293.2	2,544.8
AuEq Grade	g/t	-	-	9.4	7.5	6.4	6.4	7.2	6.2	6.0	6.8
AuEq Ounces	oz ('000s)	-	-	26.1	101.7	90.6	90.1	101.2	86.7	56.6	553.1
Total	**t ('000s)**	**9.0**	**610.9**	**896.5**	**897.8**	**899.9**	**898.6**	**897.1**	**773.7**	**293.2**	**6,176.7**
AuEq Grade	g/t	6.9	5.7	6.3	6.9	5.7	5.7	6.4	5.9	6.0	6.1
AuEq Ounces	oz ('000s)	2.0	112.8	182.9	200.2	165.8	165.8	184.1	146.9	56.6	1,217.0

Note: Ore from development at RMP in Year 2 go to stockpile

A conventional and common mobile mining fleet is shared between the two sites, to reduce spares and capital expense. Development headings and stope accesses used a common approach with key equipment used during preproduction and operations consisting of 2B jumbos, 10-tonne LHDs, 30-tonne mine haul trucks, bolters, shotcreters and production longhole drills.

Mine services such as dewatering, ventilation and electrical reticulation employed a common approach at each site in a similar manner to mobile equipment to standardize pumps, fans and MCC's reducing the required spares and capital expenditure. Underground water handling at both sites employs a conventional series of sumps and pumps to move the water out to settling ponds on surface. At PGP water from Silver Coin and Big Missouri have a common pond, which is then directed by pipeline to the water treatment plant at the historical 6 level at Premier mine.

The ventilation systems were designed to meet BC regulations based on the requirements of engines sizes and utilization. Fresh air is heated by a propane system when required during winter months.

Workforce will consist of technical staff and operations personnel, consisting of miners, mechanics, electricians, and supervision. At peak production the mine department will have 110 people, with about 40 people active at the site at a given time. In some instances, shared technical resources will be based in Stewart supporting both sites.

Personnel will live in the town of Stewart which is an easy drive from both PGP and RMP. Buses will bring staff and operating people to site, in order to limit the number of personal vehicles on the surrounding roads. Some staff and supervision required to move among sites will drive company supplied pickup trucks.

1.17 Recovery Methods

The existing Premier mill facilities, mine, and surface infrastructure have been kept on a care and maintenance regime since 2001. During 2019, an engineering assessment was conducted with a detailed field review of the facilities by a local constructor. The review focussed on the condition of the plant and equipment with the aim of establishing a basis for costing a re-commissioning of the operation. This forms the basis for the Feasibility Study design and execution activities described in this NI 43-101.

The Premier mill facility plans for re-starting will feature a combination of upgrades and returning the existing facilities to an operating condition. The development work assessed the current condition of the equipment and structures, allowing the engineering team to develop a capital cost for the restart of the facilities using a combination of the existing, refurbished existing, and new equipment for each of the following areas.

- Crushing and stockpiling
- Grinding and classification
- Gravity concentration and intensive leaching
- CIL management
- Gold room
- Detoxification and tailings deposition.

Detailed area descriptions, overall process flow sheet, and process design criteria supporting this Feasibility Study work are presented in Section 17.

The existing plant arrangement is suited to a SAB milling flowsheet followed by CIL, which is retained to treat the 2,500 t/d throughput. Over the LOM, the plant will operate 365 d/a to produce gold doré with an overall plant availability of 92%. Up to Q4 of Year 2 the process plant will be exclusively processing ore from the Silver Coin and Big Missouri underground deposits from the Premier lease. Ore will be processed in this order, which aligns with the current published mine plan.

Within two years from start-up, Red Mountain ore from the neighbouring JW, Marc, and AV deposits will be milled through the existing Premier mill facility. The Red Mountain ore types have differing properties from those of

Premier, and as a result, specific circuit modifications are required to the plant design, most notably the addition of a fine-grinding circuit and pre-leach thickening stage, which are described in Section 17.

In Q4 of Year 2 of the mine plan, it is expected that the mill will be fed from either one of the Premier deposits or from the Red Mountain mine by campaigning the ore on a two-week basis (two weeks of processing Premier ore followed by two weeks of processing Red Mountain ore).

Ore processing at the Premier mill begins with primary crushing and stockpiling, followed by SAB milling to achieve a grind size P_{80} of 80 µm (90 µm for Red Mountain ores). An integrated gravity circuit will remove coarse gold for cyanidation in the intensive leach reactor (ILR), with the remainder of the ore to be cyanide leached in a conventional CIL circuit.

Gold will be recovered on carbon, eluted, and then electrowon to produce a silver/gold doré. Gold recovered from the ILR will be electrowon separately to produce a separate gold doré.

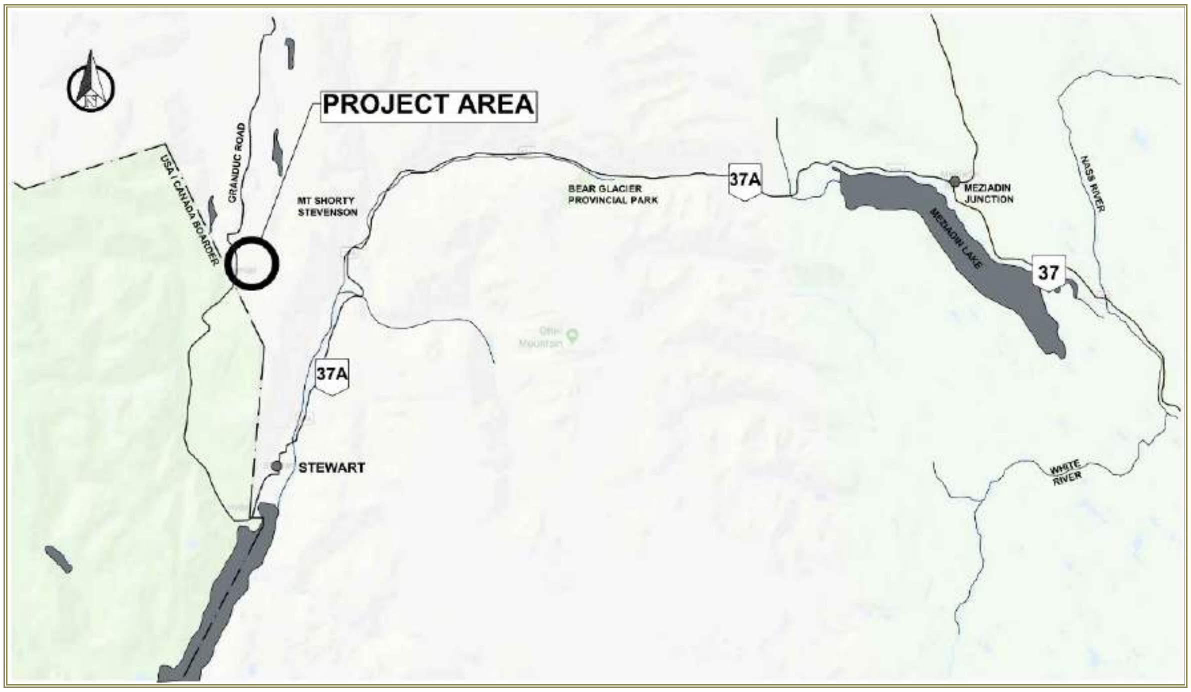
Leached tails will be detoxified in an SO_2/Air cyanide destruction circuit, then thickened using a tailings thickener, which will then be pumped to a TSF approximately 1 to 2 km from the plant. Fresh water required for reagent mixing, gland water, and process water make-up are pumped to the plant from Cascade Creek, while process water is recovered from the TSF decant water and returned to the plant, which will be used for services such as grinding and utility water.

The Premier mill processing circuit will be modified in a Year 2 to process ores from Red Mountain. Gravity-recoverable gold is absent in the Red Mountain ore; therefore, the gravity circuit will be bypassed while processing this ore. The Red Mountain ore gold and silver recovery is sensitive to grind size, and as such a target of 25 µm is required to achieve acceptable precious metals recoveries in the leaching circuit. To achieve the targeted fine grind, a fine-grind mill (high-speed stirred mill) will be installed in the plant. The grinding circuit product will require a thickening stage prior to introduction to the CIL circuit. Based on current and historical testwork, a 27 m pre-leach thickener will be required for this application; the planned upgrades to the base milling circuit design have been described in Section 17.

1.18 Project Infrastructure

1.18.1 PGP Access

Currently, PGP is accessed via Highway 37. Minimal helicopter support is required because the Project still has access roads in place from historical mining operations. Access to the Project site is via the Port of Stewart via the Granduc Road/Highway 37A or Continental North America via Highway 37A, Highway 37, and points beyond (Figure 1-2).



Source: McElhanney, 2020

Figure 1-2: Access Roads to PGP

1.18.2 Mill Site Infrastructure

The process plant consists of ore stockpiling, crushing, conveying, grinding, gravity concentration, leaching, cyanide detoxification, and reagents. Refer to Section 17 for details of the process facilities. The following is a general description of the existing ancillary facilities at the process plant site which needs upgrading or replacing. All the existing facilities require the removal of debris from buildings (such as walls with mould, corroded piping, general garbage and debris, chemical spills, etc.). The infrastructure planned at the process plant to support the mining and processing operations includes:

- Upgraded site and access roads
- Upgrading/replacing the administration facilities, mine dry, truck shop, and maintenance facilities
- Upgrading/replacing elements of the assay laboratory/cold storage building
- Waste water treatment systems
- Solid waste disposal facilities
- Tailings storage facility
- Water management
- Water treatment plant
- Temporary construction camp
- Power supply and distribution system
- Site services

- Fuel
- Propane
- First aid station
- Water supply
- Communication system.

Minor ancillary infrastructure facilities currently in use for the care and maintenance period, such as the bunkhouse, trailers, generators, fuel tanks, trailers, and minor shops are included in the infrastructure on site but require no upgrade or replacement.

The current WTP will be replaced with a new WTP as described in Section 18.6.

Figure 1-3 shows the mill process site infrastructure of the PGP.

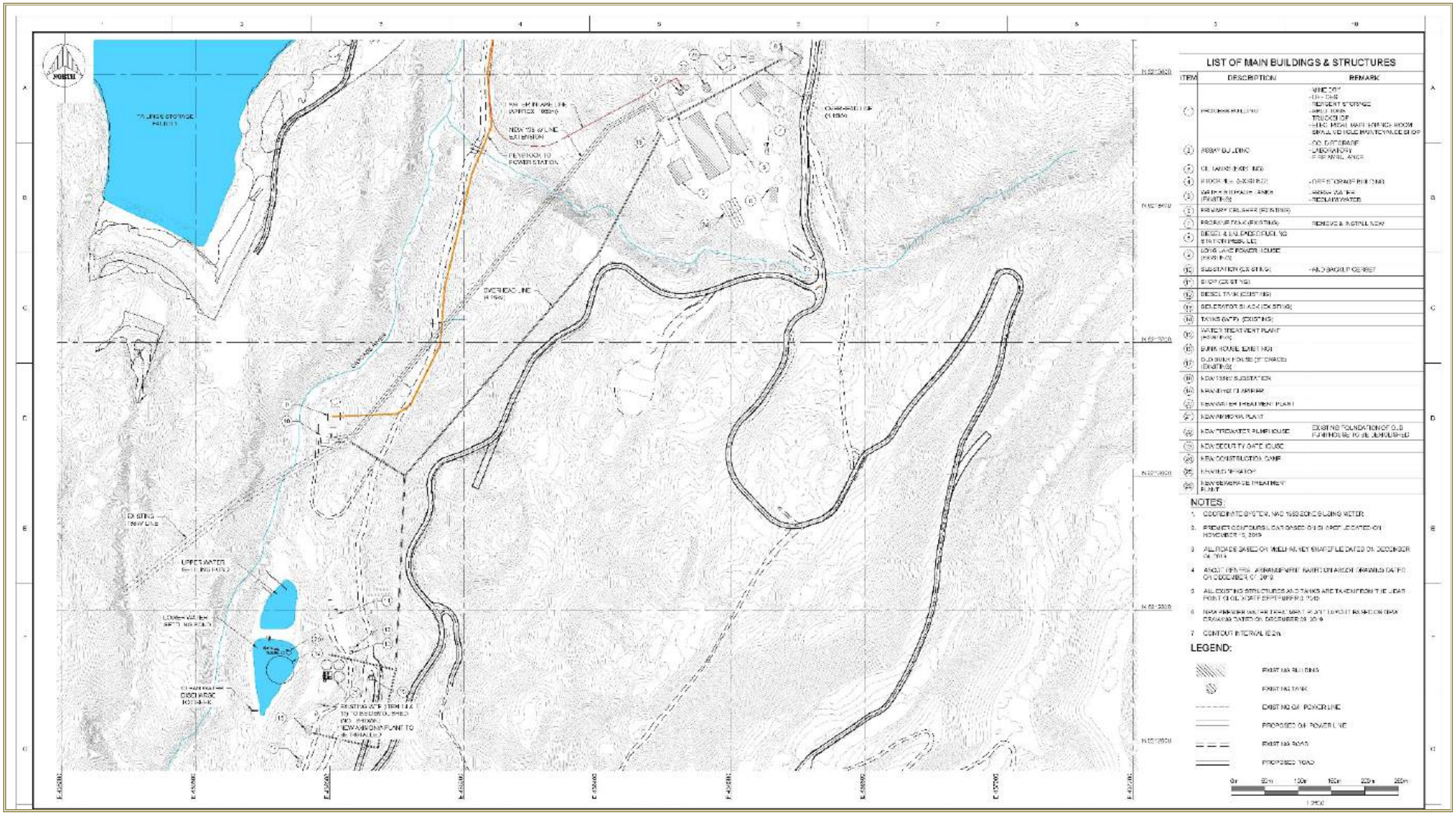


Figure 1-3: Site Layout – Main PGP Facilities

1.18.3 Tailings Storage Facility and Surface Water Management

The principal objectives for the TSF are to provide safe and secure storage of tailings to protect regional groundwater and surface water during operations, and in the long term (post-closure), and to achieve effective reclamation at mine closure. The design of the TSF has taken into account the following requirements:

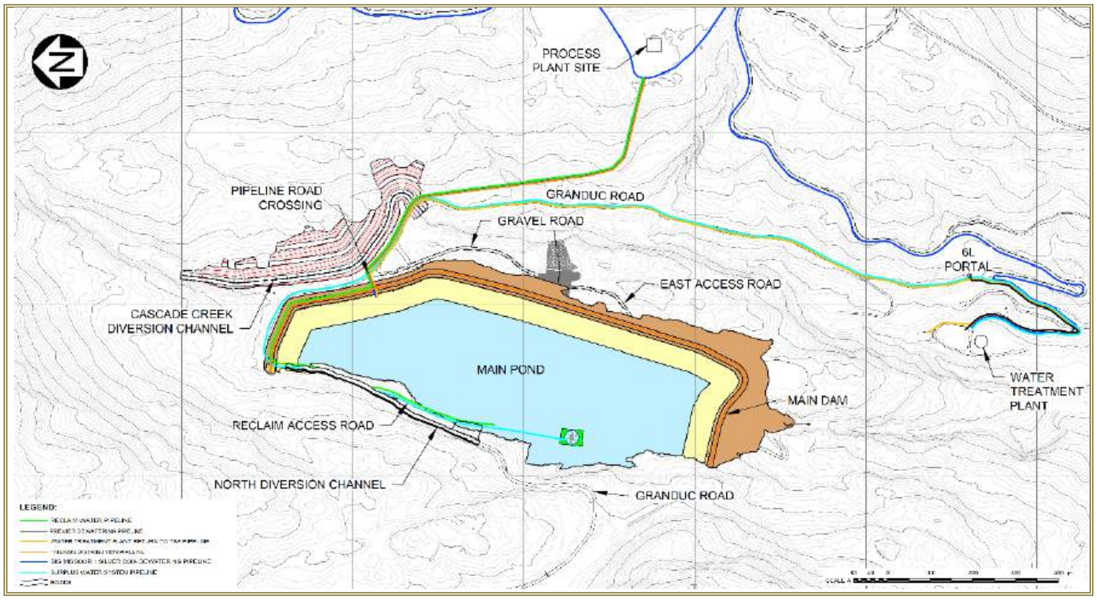
- Permanent, secure, and total confinement of all tailings materials within an engineered disposal facility
- Diversion of non-contact water around the TSF to the maximum extent possible
- Control, collection, and removal of free water from the TSF during operations for recycling as process water to the maximum practical extent
- The inclusion of monitoring features for all aspects of the facility to confirm performance goals are achieved and design criteria and assumptions are met
- Staged development of the facility over the life of the PGP.

The TSF was designed to permanently store tailings generated during the operation of the mine. This will be accomplished by constructing staged embankment raises on the existing TSF, which has been in long-term care and maintenance for over 20 years. The TSF comprises a basin constrained by a rockfill embankment on three sides, and natural topography to the west. The design of the TSF foundations relied on historical site investigation programs from pre-construction and construction periods of the Project, and as-built reports and information. The embankment will be expanded during operations using the centreline method of construction. The final (Stage VI) layout for the TSF is shown on Figure 1-4.

Tailings will be delivered to the TSF in a single stream, in a single overland pipeline. Tailings will be discharged from the embankment crest via spigots spaced along the length of the embankment. Supernatant water will be reclaimed to the plant site for use in processing of ore via a floating pump barge and overland pipeline. The supernatant pond volume will be managed by removing surplus water to the WTP for treatment and subsequent discharge to the environment. The surplus water system pumps will be housed on the same barge as the reclaim water system pumps.

The tailings are characterized as potentially acid generating (PAG). Mitigation to prevent oxidation of the tailings includes: continuous deposition of fresh layers of tailings over the above-water beaches adjacent to each dam section; maintaining a supernatant pond over a portion of the tailings during the operating life; and constructing a cover at closure, once the operational supernatant pond has been removed.

An alternatives assessment, completed in 2019, assessed several tailings disposal locations and technologies, and concluded that tailings storage in the existing Premier TSF was the preferred alternative for tailings management for the Project (KCB, 2019).



Source: Knight Piésold, 2020a

Figure 1-4: TSF General Arrangement (Stage VI – Year 8.5)

1.18.4 Water Management

Site water management involves controlling surface water around the PGP site during the construction, operations, closure, and post-closure phases of the PGP. Water in contact with mine workings or disturbed areas (groundwater inflows from the underground mines; runoff from waste rock, ore stockpiles, quarry areas, tailings, laydown areas, etc.) is considered contact water. Non-contact water is runoff from undisturbed areas, including those areas that are being diverted.

Management of surface water on site will be undertaken by upgrades to existing water diversion structures, construction of the TSF and other infrastructure, selective grading of surfaces, and installation of pump and pipeline systems. The major facilities for contact water management include:

- TSF
- Cascade Creek diversion channel
- Site diversion ditches
- WTP
- Water management pump and pipeline systems.

1.18.5 Water Treatment Plant

Planned water treatment infrastructure for the Project include:

- A moving bed bioreactor water treatment facility for removal of ammonia, cyanide, cyanate, and thiocyanate from tailings supernatant (nominal treatment capacity of 585 kg/d nitrogen and 240 m^3/h)
- A high-density sludge (HDS) lime WTP for removal of dissolved metals and total suspended solids (TSS) (nominal treatment capacity of approximately 720 m^3/h).

Both planned treatment processes are commonly implemented to treat mine water produced at underground gold mines.

Ascot is currently operating a low-density lime water treatment process on site that removes TSS, zinc, and other dissolved metals from mine water draining from the historical Premier underground mine. The planned water treatment processes were selected based on results of a best achievable technology (BAT) assessment of water management and water treatment options. The BAT assessment and water treatment process selection were conducted in collaboration with representatives from Nisg̱a'a Lisims Government.

1.18.6 Power and Electrical

The mill throughput is nominally 2,500 t/d. At this production level, the plant load is estimated to be approximately 15 MW ± 10%. Electrical power will be supplied from a 138 kV tap from the Long Lake Independent Power Producer (IPP) line. Figure 1-3 shows the location of the Long Lake IPP and the route of the 138 kV line to the new Main Substation.

There will be one main transformer feeding the mill site. Each transformer will be base rated at 15 megavolt amperes (MVA), with additional fan-cooled ratings of 20 MVA. Transformers of this size are in the range of 40-tonnes and will be one of the largest loads transported to the site.

The transformer will feed a 4.16 kV secondary bus. Large motor loads (e.g., ball mills) will be served at 4.16 kV. Power will be distributed at 4.16 kV around the site using cables and overhead lines, and additional step-down transformers will be located near remaining loads. Medium-sized motor loads (250 to 5,000 horsepower [hp]) will be served at 4.16 kV. Smaller motor loads will be served at 600 V.

Electrical rooms (housed within the heating and ventilation structure) will be provided at the Premier portals (Big Missouri, Silver Coin, and Premier). These electrical rooms will include motor control, lighting panels, and other electrical equipment necessary for facility operation. The power supplied at the Big Missouri portal will be reticulated underground to the Silver Coin deposit.

The portals will be fed with 4.16 kV, a suitable voltage to feed via cable through the portals to the underground workings, where it will be further stepped down to 600 V to feed the Jumbos and drills.

About 1 MW of power will be reticulated to each portal.

A generator system will provide 4.16 kV of power at the RMP portal, distributed throughout the RMP mine site at this voltage for large electrical loads. A number of centrally located electrical rooms (by others) will transform the 4.16 kV power to 600 V as necessary.

Electrical power will be distributed at 4.16 kV by overhead power line to locations such as the TSF, reclaim, and booster pump stations, fresh-water intake, the WTP, and the existing infrastructure facilities, such as the bunkhouse.

Single-line diagrams and layout drawings can be found in Appendix C.

Underground electrical power distribution equipment will feed the underground ventilation and miscellaneous loads. Power will be delivered to underground operations via a single 4.16 kV underground power cable.

1.19 Market Studies and Contracts

No market studies have been undertaken regarding the sale of gold and silver produced from the mining of gold ore from the project and its processing into doré. The final product will gold and silver doré. The gold and silver doré will likely be transported to a North American based previous metals refinery or sold to precious metals traders. Indicative doré refining terms were obtained from a Canadian refinery in early 2020. Doré can be refined by several refineries throughout North America, and the gold and silver produced sold either domestically or abroad in order to realize the highest available price.

Doré refining charges were estimated at US$0.76/oz. Transportation costs for doré produced were estimated at US$0.54/oz shipped, based on bi-weekly shipping to eastern Canada for refining and sale.

Table 1-5: Refining and Transportation Assumptions

	Unit	LOM
Gold (Au) Payable	%	99
Silver (Ag) Payable	%	99
Refining Charges	US$/oz doré	0.76
Transportation	US$/oz doré	0.54

1.20 Environmental Studies, Permitting, and Social Community Impact

1.20.1 Permitting Process

PGP is currently in care and maintenance with existing permits for continued reclamation and mine water discharge. The site has been maintained in good standing, with reclamation activities and environmental monitoring ongoing. In 2018 and 2019, Ascot undertook additional environmental baseline monitoring and data collection to support permit amendments for the *Mine's Act* and the *Environmental Management Act*, and several ancillary permits, which will be required to bring PGP back into operation. In 2018, Ascot received confirmation from the Nisg̱a'a Lisims Government and both provincial and federal government agencies that PGP will not need to undergo an environmental assessment.

In 2019, RMP received federal approval and issuance of a provincial Environmental Assessment Certificate (EAC). The decision also included a determination of the potential effects of the Nisg̱a'a Final Agreement (2000). RMP will next require issuance of the necessary statutory permits and authorizations to commence construction of the Project. Any changes to the Project description, resulting from coupling activities or toll milling with PGP, will first require an amendment to the RMP EAC before proceeding to detailed design and ensuing permit applications.

1.20.2 Aboriginal and Community Stakeholders

PGP is located in the Nass Area, and RMP is located in the Nass Wildlife Area, as defined in the Nisg̱a'a Final Agreement (2000), a tripartite agreement between the federal government, provincial government, and Nisg̱a'a Nation, which sets out Nisg̱a'a Nation's rights under Section 35 of the *Canadian Constitution Act*. Nisg̱a'a Nation's Treaty rights under the Nisg̱a'a Final Agreement include: establishing the boundaries and the Nisg̱a'a Nation's ownership of Nisg̱a'a Lands and Nisg̱a'a Fee Simple Lands; water allocations; the right of Nisg̱a'a citizens to harvest fish, wildlife, plants, and migratory birds; and the legislative jurisdiction of the Nisg̱a'a Lisims Government. Nisg̱a'a citizens have Treaty rights to manage and harvest wildlife in the Nass Wildlife Area, and to harvest fish, aquatic plants, and migratory birds within the Nass Area. The clarity and certainty provided by the Nisg̱a'a Final Agreement, including Chapter 10, which sets out the required processes for the assessment of environmental effects on Nisg̱a'a Nation Treaty rights from projects such as this one, is a major advantage to development.

The nearest communities to RMP and PGP are the town of Stewart, and the village of Hyder, Alaska. Both communities have a long-standing history with mining projects and have historically been supportive of mining activities. Broader stakeholders may include overlapping tenure holders (such as trapline holders, guide outfitters, and independent power producers), local and regional governments, and government regulatory agencies.

Ascot is committed to meaningful, timely, and transparent engagement and consultation with the Nisg̱a'a Lisims Government, community members, stakeholders, and the public. Ascot will maintain this commitment throughout the proposed development, construction, operation, and closure of the Project.

1.21 Capital and Operating Costs

The capital cost and operating estimates for the PGP are developed to a level appropriate for a Feasibility Study. All capital and operating costs are reported in Canadian dollars (C$) unless specified otherwise. The overall capital cost estimate (with the exception of the WTP and MBBR estimates – which is a Class 4 AACE estimate) meets the American Association of Cost Engineers (AACE) Class 3 requirement of an accuracy range between −15% and +15% of the final Project cost.

PGP benefits from significant existing infrastructure, which helps reduce the initial capital cost. Total initial pre-production capital cost is $146.6 million inclusive of construction indirect costs, engineering-procurement-construction-management (EPCM), contingencies and owners' costs. The sustaining capital is $157.3 million inclusive of mine development capital, road construction to RMP, and process plant modifications for the fine grind and additional pre-leach thickener. The LOM capital expenditure (CAPEX) is $324 million inclusive of closure costs. Underground mining and haulage are anticipated to be completed using an owner-operator development model operating 365 d/a with a leased mobile equipment fleet which is included in operating costs. Table 1-7 presents the Project capital cost breakdown and the costs for each work breakdown structure (WBS).

Table 1-6: Total Project Capital Cost Summary by Area

WBS	Description	Total Cost ($ '000s)		
		Initial	Sustaining	LOM Total
1 Direct Costs		**100,036**	**161,229**	**261,311**
1000	Mining/Dewatering	14,019	110,183	124,202
2000	Overall Site Development	8,187	1,378	9,565
3000	Mineral Processing	35,637	10,266	45,903
4000	TSF	8,659	4,580	13,240
4500	Site-Wide Surface Water Management	7,016	4,695	11,711
4900	Closure and Reclamation	0	20,500	20,500
5000	On-Site Infrastructure	14,038	0	14,038
5800	WTP	12,480	0	12,480
6000	Off-Site Infrastructure	0	9,672	9,672
2 Indirect Costs		**30,457**	**9,392**	**39,849**
9000	Project Indirect Costs	30,457	9,392	39,849
3 Owner's Costs		**3,663**	**204**	**3,867**
9800	Owner's Costs	3,663	204	3,867
4 Contingency		**12,443**	**6,690**	**19,133**
9900	Contingency	12,443	6,690	19,133
Total Project Costs		**146,600**	**177,515**	**324,160**

The total estimated initial capital cost is $146.6 million (and sustaining capital of $177.5 million) is $324.2 million over the LOM. The estimated LOM operating costs are $139.34/t of mill feed.

1.21.1 Operating Costs

LOM operating costs for the Project were developed from first principles for mining, processing, site services, and administration using the mine and processing plans, incorporating development rates, labour, materials, consumables, and certain contract services for a 2,500 t/d processing rate. Processing cost in Year 2 increases by $4.25/t processed due the higher grinding requirements for harder ore from RMP. Table 1-8 shows the breakdown of LOM operating costs.

Table 1-7: Project LOM Operating Costs

Operating Costs	Costs ($/t milled)
UG Mining Cost ($/t milled)	97.00
Processing Cost ($/t milled)	31.05
G&A Cost ($/t milled)	7.93
Site Services ($/t milled)	3.36
Total Operating Costs ($/t milled)	**139.34**

1.21.2 Closure Costs

Mill, TSF, water management, and infrastructure closure estimates have been prepared as of the date of this report. The closure cost for the PGP is $25 million based on estimates summarized in Table 1-9.

Table 1-8: Closure Cost

Description	Closure Cost ($ '000s)
Mining (surface infrastructure)	150
Process Building	7,334
TSF and Water Management	11,845
Access Roads	655
On-Site Infrastructure	476
Directs Subtotal	**20,500**
Owner's Costs for Road Closure	204
Project Indirect Costs	1,051
Contingency	3,300

1.22 Economic Analysis

The economic evaluation of the PGP was carried out using a financial model developed by Ascot based on production schedule, application of operating, capital, sustaining costs, royalties and taxes as discussed in the Economic Analysis section of this document. The financial model generated an estimate of annual pre-tax and post-tax cash flows and Ascot has undertaken an internal review of both pre-tax and post-tax economic evaluation models.

Analyses were performed using a discounted cash-flow model prepared by Ascot. This model uses mid-year discounting at a base-case discount rate of 5%. Pre-production period is estimated to be two years. NPV and IRR are calculated based on the start of this two-year period. No provisions are made for inflation and/or future increases in costs, and analysis is presented in constant dollars. The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered production by calculating them from production head grades and tonnage, and mining and processing recoveries.

The following assumption and basis were used in the analysis:

- Commercial production will begin in Q1 2022
- Base case gold and silver prices are US$1,400/oz and US$17/oz, respectively
- The Canadian to United States dollar exchange rate has been assumed to be $0.76
- All costs and sales estimates are in Q1 2020 Canadian Dollars with no inflation or escalation
- All gold and silver are sold in the same year of production
- Working capital costs are estimated based on two months of necessary capital needed (2 months of operating expenses) to fund operating costs prior to revenue starts in the first year of operation, and are constant every year.

- Costs in the analyses exclude sunk costs (e.g., drilling costs, corporate overheads, permitting, government reclamation bond) prior to the commencement of PGP construction. Water treatment plants costs, reclamation bond for closure, and future bonding estimates are not included in the analysis.

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. The LOM summary analysis is shown in Table 1-10.

Table 1-9: LOM Summary

Assumptions	Combined
Gold Price (US$)	1,400
Silver Price (US$)	17.00
Exchange Rate (C$/US$)	0.76
Effective Royalty Rate (%)	4.7
Payable Metals	
Gold Production (koz)	1,059
Silver Production Ounces (koz)	2,964
Mining	
Mine Life (Years)	8
Total Ore Tonnage Mined (kt)	6,177
Processing	
Gold Recovery %	91.4
Silver Recovery %	76.5
Processing Throughput (t/d)	2,500
Average Gold Grade (g/t)	5.89
Average Silver Grade (g/t)	19.72
Capital Expenditure Costs	
Initial CAPEX ($ million)	146.6
Sustaining Capital ($ million)	157.1
Closure Costs ($ million)	25.0
Operating Costs	
UG Mining Cost ($/t Milled)	97.00
Processing Cost ($/t Milled)	31.05
G&A Cost ($/t Milled)	7.93
Site Services ($/t Milled)	3.36
Total Operating Costs ($ million)	**139.34**

Note: C1 includes mining processing, site services, G&A, refining cost, and royalty costs. AISC includes C1 cost plus sustaining capital. C1 and AISC costs are non-GAAP performance measures; see “Non-GAAP Measures and Other Financial Measures” below.

The PGP has an after-tax NPV at 5% (NPV 5%) of $341 million and an after-tax IRR 51%. The pre-tax payback period is 1.7 years, and the after-tax payback period is 1.8 years. Figure 1-5 shows a summary of the annual cash flows and the cash flow model.

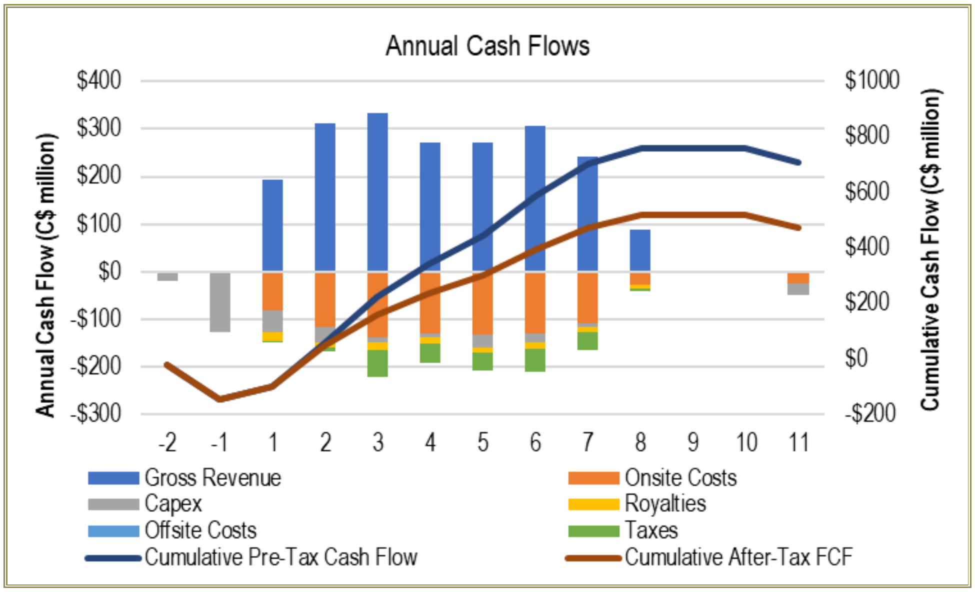


***Figure 1-5:	Annual Cash Flows***

Sensitivities

After-tax economic sensitivities are presented in Table 1-11, illustrating the effects of varying precious metals prices and exchange rates to LOM base-case. Figure 1-6 Additional Project sensitivities are presented in Section 22.

***Table 1-10:	After-Tax NPV (5%) and IRR Sensitivities to Commodity Prices***

	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Au Price ($)	123	167	211	254	297	**341**	383	425	468	510	552
Ag Price ($)	333	335	336	338	339	**341**	342	344	345	346	348
Mining ($)	414	399	384	370	355	**341**	326	311	296	281	266
Process ($)	363	358	354	349	345	**341**	336	332	327	323	318
CAPEX ($)	386	377	368	359	350	**341**	331	322	312	303	294
C$/US$	643	567	501	441	388	**341**	296	256	220	186	155
Au Grade ($)	114	161	206	251	296	**341**	385	429	473	518	562

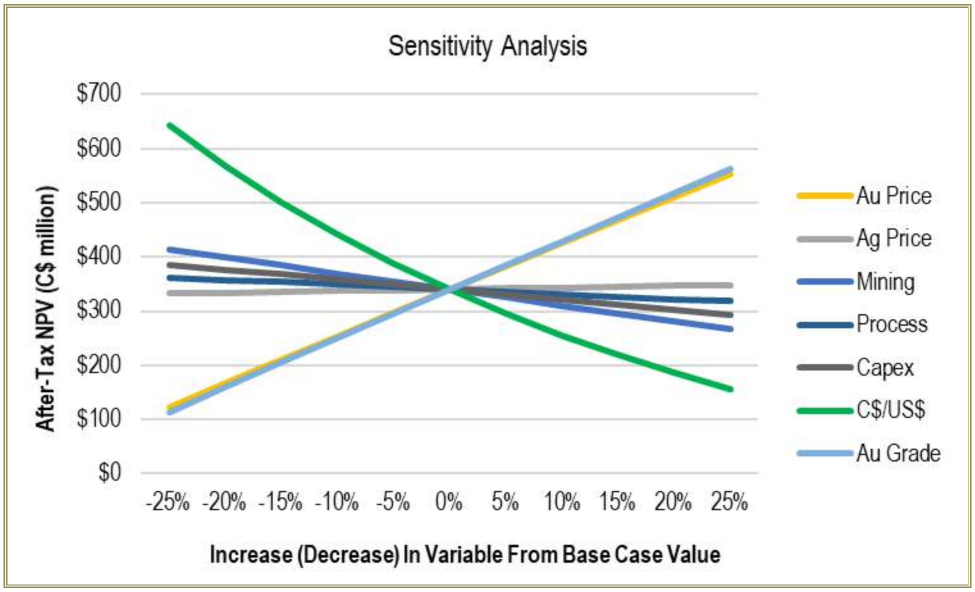


Figure 1-6: Sensitivity Analysis

1.23 Adjacent Properties

The PGP is located at the southern tip of British Columbia's Golden Triangle, host to a large number of epithermal, volcanogenic massive sulfide (VMS)–style, and porphyry copper deposits. The mineralization at the Premier Project is hydrothermal in nature, and there are a number of similar showings and deposits in PGP's proximity. PGP is the largest project in terms of size and contained metal in the Stewart area.

The Scottie Gold Mine is approximately 20 km north of the PGP mill buildings, and is accessed by the Granduc Road along the Salmon Glacier (refer to Section 23). Gold and silver mineralization occur as bodies of massive pyrite and pyrrhotite with accessory sphalerite, chalcopyrite, galena, arsenopyrite, and tetrahedrite in epithermal quartz-carbonate veins. From 1981 to 1984, the mine produced 160,264 tonnes, containing 2,984 kg Au and 1,625 kg Ag (http://minfile.gov.bc.ca). The property is currently held by Scottie Resources Corporation.

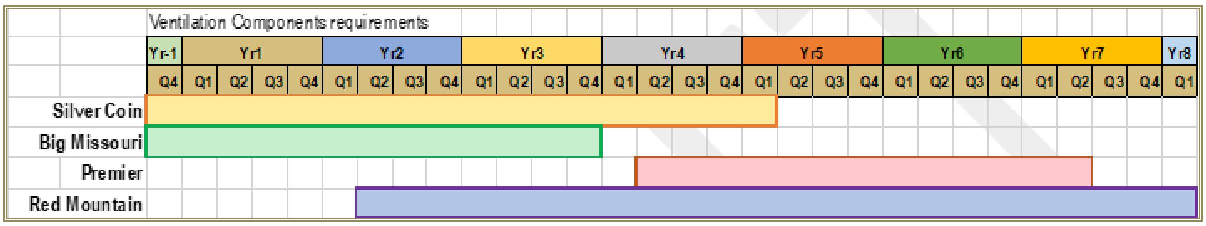
Five kilometres further north lies the Electrum property, which is 60% owned by Tudor Gold Corp. Gold and silver mineralization occurs in epithermal quartz-carbonate veins, stockworks, and breccias hosted in island-arc volcanic rocks (http://tudor-gold.com). Sulfide minerals include pyrite, sphalerite, galena, and chalcopyrite.

The Red Cliff project is a former producing copper and gold property 6 km east of the PGP mill buildings, in the adjacent valley. It is owned 65% by Decade Resources and 35% by MBM. Gold is associated with abundant chalcopyrite and pyrite, most commonly in sulfide-bearing veins within a 30 m to 40 m wide shear that can be traced over 2 km. There are also gold-bearing stockwork zones outside of the vein (https://www.mountainboyminerals.ca).

With respect to Red Mountain there are no adjacent properties relevant to the scope of this report.

1.24 Other Relevant Data and Information

The existing processing facility will be refurbished within a construction period of approximately 40 weeks. Commissioning the refurbished processing facility will take approximately 16 weeks. The preliminary development schedule is presented in Figure 1-7.



Source: InnovExplo Inc. (2020)

Figure 1-7: Mining Development Schedule

The process plant will use conventional crushing, grinding, and gravity circuits followed by a standard CIL process to produce a gold doré. The plant refurbishment will consist of a combination of existing, new, and repaired equipment and supporting plant infrastructure. Prior to ore from RMP being treated, the plant will add an energy-efficient fine grinding mill and an additional pre-leach thickener to accommodate processing of the harder ore feed and the finer grind required for recovery purposes.

1.24.1 Planning and Scheduling

The project execution schedule has been developed for the PGP to achieve production by Q2 2022. The schedule was developed with interaction between the mine development team, the processing and surface infrastructure development team to check for timing of the production ore and interactions with the surface infrastructure and tailings facilities upgrade scopes of work. The schedule was also shared with regional contractors to assess the installation duration and planned sequence of work to develop the works forward from engineering to construction and performance testing of the plant.

The schedule considered the early ordering of equipment using the designs developed for the feasibility study in which key long lead items were tendered. These long lead items were reviewed to the point of a technical recommendation for Ascot.

Construction activities are dependent on receiving an approved *Mines Act* Permit Application (MAPA) and *Environmental Management Act* Permit Application.

The major Project activity durations and milestones are listed below in Table 1-12.

Activities allowing the production and delivery of RMP ore also include the access road permit. Construction of the RMP Road will commence in Year 1 to facilitate road access to the mine portal and the installation of generator power, heating, and ventilation facilities. Red Mountain ore is planned to be delivered to the PGP mill by Q2 Year 2.

The current plan will be to complete this Feasibility Study by Q2 2020 (Year −2) and to proceed to detailed engineering. This will allow Ascot to be in a position to start early works at site in Q1 Year −1. This work will include preparing the mill and ancillary facilities as an accessible workspace/office, early construction of the earthworks for the new WTP, and installation of 138 kV powerline from Long Lake to the new Main Substation.

Table 1-11: Major Milestone Table

Milestones (Completion Dates)	Milestone Start Date	Milestone Completion Date
BCH SIS Review and Approval	Q2 2020	Q4 2020
Detailed Engineering	Q3 2020	Q1 2021
Long Lead Orders – Award	Q4 2020	Q3 2021
MAPA Approved	Q4 2020	Q1 2021
Construction Permits Approved	Q1 2021	Q1 2021
Construction of Plant Site and Infrastructure	Q2 2021	Q1 2022
Plant Commissioning	Q1 2022	Q2 2022
Construct Access Road to RMP portal	Q3 2021	Q4 2022
Commence Mining Red Mountain Development and Mining	Q2 2022	Q3 2023

1.25 Conclusions and Interpretations

This Feasibility Study represents an economically viable, technically credible, and environmentally sound mine development plan for the PGP and RMP. The project benefits significantly from its proximity to existing sources of labour and existing infrastructure. The Feasibility Study–level eight-year LOM plan demonstrates positive economics. Industry-standard mining and processing methods were used in the study and the QPs are not aware of any fatal flaws that encumber the PGP from undergoing further economic studies, permitting, financing, and ultimately development. It is recommended that the Project proceeds to permitting and detailed engineering design.

The most significant potential risks associated with the Project are: design of an effective control systems for grade and dilution uncontrolled mining dilution, operating and capital cost escalation, permit acquisition, reduced metallurgical recoveries, unforeseen schedule delays, changes in regulatory requirements, the ability to raise financing, exchange rate, and metal prices. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

The study indicates that PGP, based on calculated Proven and Probable Reserves (Table 1-3) of 6.2 Mt grading 5.89 g/t Au and 19.72 g/t Ag can support eight years of mine life at a process rate of 2,500 t/d.

Ore will be processed at a process plant designed to achieve expected gold and silver recoveries of 95.4% and 71.5%, respectively for PGP; and 86.8% and 83.6%, respectively for RMP. It is expected that over the LOM approximately 1.17 Moz of gold and 3.92 Moz of silver will be produced.

The initial capital cost of the Project is estimate to be $146.6 million Initial Capital, $177.6 million in Sustaining Capital. The LOM AISC total cost is US$769/oz including C1 costs plus sustaining capital. The project NPV (after-tax) is estimated to be $341 million using a 5% discount rate. The IRR (after-tax) is estimated to be 51% with a payback period (after-tax) of 1.8 years.

1.26 Recommendations

The PGP is well suited for a potential mining operation. A Feasibility Study–level eight-year LOM plan has a positive economics and it is recommended that the Project proceed to permitting and detailed design.

A summary of the suggested work program to support the next phase of the Project includes the following components:

- Continue underground mine design optimization
- The exploration work proposed by Ascot for 2020 should be carried out as detailed in the section below
- Definition drilling should be conducted to upgrade the current Mineral Resource classification where possible
- Execute geotechnical works to support the next phase of engineering
- Execute additional testwork to support process optimizations
- Procurement of long lead items
- Continue to advance the studies and work with BC Hydro necessary to electrify the site and the procurement of long lead electrical equipment
- Continuation of the environmental permitting process
- Continue to proactively engage First Nations, the Stewart Community and secure social licence to operate
- Secure project financing
- Secure permits and authorizations from government and regulatory agencies
- Begin detailed engineering activities.

The next phase of engineering and support work are estimated to be $23.35 million and are derived from the EPCM activities included in the capital costs estimate and suggested recommendations outlined Section 26, Recommendations.

Activity	Cost ($ million)
Exploration	4.0
Testwork and Studies	2.35
Detailed Engineering Activities	17.0
Total	**23.35**

2 INTRODUCTION

Ascot Resources Limited (Ascot, the Owner) is a Canadian-based exploration and development company based in Vancouver, Canada. Shares of Ascot are currently traded on the TSX. Ascot is focused on re-starting the past producing Premier Gold Project (PGP) and the Red Mountain Project (RMP) which both form part of the Premier & Red Mountain Gold Project (the Project) located in British Columbia's Golden Triangle.

The PGP is located 25 kilometres (km) north of the town of Stewart, British Columbia, adjacent to the border with Alaska. PGP can be accessed by road from Stewart and does not require a remote campsite for employees. Three of the deposits are based at PGP (Premier-Northern Lights, Silver Coin, Big Missouri) and the fourth deposit is located at the RMP which is approximately 23 km to the southeast of the PGP Mill. The focus of this NI 43-101 Feasibility Study was on maximizing the Project's economics, which involved optimizing mining methods and development to reduce costs.

Ascot's involvement with the Premier Property dates back to 2007, when the first option agreement was made on the Dilworth property. Two years later, Ascot acquired the Big Missouri-Premier property via a second option agreement. The Silver Coin property, which is adjacent to the Big Missouri property, was acquired in October 2018 from Jayden Resources Inc. (Jayden) and Mountain Boy Minerals Ltd. (MBM).

Ascot's involvement with the RMP began in March 2019 when it finalized its acquisition of IDM Mining. Prior to that acquisition IDM mining had developed the RMP as a stand-alone operation to the point where it had obtained an Environmental Assessment Certificate through the BC Environmental Assessment process and received federal Minister approval through the Canadian Environmental Assessment process.

This Feasibility Study outlines a low capital restart plan to feed the PGP Mill at 2,500 tonnes per day (t/d) to produce approximately 1.1 million ounces (Moz) of gold and 3.0 Moz of silver over eight years. This Feasibility Study is based on a Proven and Probable reserve of 6.2 million tonnes (Mt) from the Project. In addition to the reserves, Ascot has Inferred resources of 5.1 Mt at 7.25 g/t Au at PGP, with approximately 2.2 Mt of this resource material at similar grade, near planned development, which may potentially be converted to reserves during operations.

2.1 Terms of Reference

In 2019, in order to complete this Feasibility Study, Ascot engaged a team of independent consultants coordinated by Sacré-Davey Engineering Inc. (SDE). SDE is responsible for overall coordination, infrastructure, and the economic evaluation; Bird Resources Consulting Company (BRCC) for resources for the Premier property, Arseneau Consulting Services Inc. (ACS) for resources for the Red Mountain property, INNovexplo Inc.(INN) and Mine Paste Ltd. (Mine Paste) for mining and geotechnical; Sedgman Canada Limited (Sedgman) for metallurgy and processing; Knight Piésold Ltd. (Knight Piésold) for tailings and surface water management; SRK Consulting (Canada) Inc. (SRK) for the water treatment plant; McElhanney Ltd. (McElhanney) for access roads; Prime Engineering (PE) for the Electrical substation; and Marsland Environmental Associates Limited (MEA) for environmental studies.

The following individuals, by virtue of their education, experience, and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of the appropriate professional institutions:

- Sue Bird, P.Eng., Geologic/Mining Engineer, BRCC
- Dr. Gilles Arseneau, P.Geo., President, ACS
- Aleksandar Petrovic, P.Eng., Senior Process Engineer, Sedgman
- Frank Palkovits, P.Eng., President, Mine Paste
- Jim Fogarty P.Eng., Senior Engineer, Knight Piésold
- Soren Jensen, P.Eng., Senior Environmental Engineer, SRK
- Brendon Masson, P.Eng., Civil Engineer, McElhanney
- Robert Marsland, P.Eng., Senior Environmental Engineer, MEA
- Shervin Teymouri, P.Eng., BASc., M.Eng., P.Eng., SDE
- Frank Grills, P.Eng., Senior Project Manager, SDE
- Ken Savage, P.Eng., Senior Civil Engineer, SDE.

2.2 Qualified Persons

The following companies contributed to this technical report and provided Qualified Person (QP) sign-off for their respective sections. All supporting documents cited in this report are referenced in Section 27.

Table 2-1: Feasibility Study Sections and Parties Responsible

Section	Title	Responsible Party	Qualified Persons
1	Summary	All parties provided input	All
2	Introduction	Sacré-Davey	Frank Grills, P.Eng.
3	Reliance on Other Experts	Sacré-Davey	Frank Grills, P.Eng.
4	Property Description and Location	BRCC ACS	Sue Bird, P.Eng. Dr. Gilles Arseneau, P.Geo.
5	Accessibility, Climate, Local Resource Infrastructure and Physiography	BRCC ACS	Sue Bird, P.Eng. Dr. Gilles Arseneau, P.Geo.
6	History	BRCC ACS	Sue Bird, P.Eng. Dr. Gilles Arseneau, P.Geo.
7	Geological Setting and Mineralization		
	PGP	BRCC	Sue Bird, P.Eng.
	RMP	ACS	Dr. Gilles Arseneau, P.Geo.
8	Deposit Types		
	PGP	BRCC	Sue Bird, P.Eng.
	RMP	ACS	Dr. Gilles Arseneau, P.Geo.
9	Exploration		
	PGP	BRCC	Sue Bird, P.Eng.
	RMP	ACS	Dr. Gilles Arseneau, P.Geo.
10	Drilling		
	PGP	BRCC	Sue Bird, P.Eng.

Section	Title	Responsible Party	Qualified Persons
	RMP	ACS	Dr. Gilles Arseneau, P.Geo.
11	Sample Preparation, Analysis, and Security		
	PGP	BRCC	Sue Bird, P.Eng.
	RMP	ACS	Dr. Gilles Arseneau, P.Geo.
12	Data Verification		
	PGP	BRCC	Sue Bird, P.Eng.
	RMP	ACS	Dr. Gilles Arseneau, P.Geo.
13	Mineral Processing and Metallurgical Testing	Sedgman	Aleksandar Petrovic, P.Eng.
14	Mineral Resource Estimates	BRCC ACS	Sue Bird, P.Eng. Dr. Gilles Arseneau, P.Geo.
15	Mineral Reserve Estimates	Mine Paste	Frank Palkovits, P.Eng.
16	Mining Methods	Mine Paste	Frank Palkovits P.Eng.
17	Recovery Methods	Sedgman	Aleksandar Petrovic, P.Eng.
18	Infrastructure		
	18.1; 18.4; 18.9; 18.10; 18.11; 18.12; 18.13; 18.14	Sacré-Davey	Frank Grills, P.Eng.
	18.2; 18.3	McElhanney	Brendon Masson, P.Eng.
	18.3; 18.5; 18.6	Knight Piésold	Jim Fogarty, P.Eng.
	18.6.4; 18.8	SRK	Soren Jensen, P.Eng.
	18.7	MEA	Robert Marsland, P.Eng.
	18.2.5; 18.2.6; 18.2.7	Sacré-Davey	Ken Savage, P.Eng.
19	Market Studies and Contracts	Ascot	Shervin Teymouri, P.Eng.
20	Environmental Studies, Permitting and Social or Community Impact	MEA	Robert Marsland, P.Eng.
21	Capital and Operating Costs		
	21.1.1; 21.1.2; 21.1.3; 21.1.7; 21.1.8; 21.1.9; 21.1.10; 21.1.11; 21.1.14; 21.1.15; 21.2.5; 21.3.4; 21.4.1; 21.4.4 to 21.4.10	Sacré-Davey	Frank Grills, P.Eng.
	21.1.6; 21.2.2; 21.3.2; 21.4.3	Sedgman	Aleksandar Petrovic, P.Eng.
	21.1.2; 21.1.4; 21.1.5; 21.2.1; 21.4.2; 21.4.3; 24.4	Mine Paste	Frank Palkovits, P.Eng.
	21.1.9; 21.2.3; 21.3.3; 21.4.11	Knight Piésold	Jim Fogarty, P.Eng.
	21.1.3	McElhanney	Brendon Masson P.Eng.
	21.1.10; 21.18; 21.2.4; 21.4.12	SRK	Soren Jensen, P.Eng.
	21.2.4; 21.3.5; 21.4.4	Sacré-Davey	Ken Savage, P.Eng.
22	Economic Analysis	Sacré-Davey	Shervin Teymouri, P.Eng.
23	Adjacent Properties	MRCC	Sue Bird, P.Eng.
24	Other Relevant Data and Information	Sacré-Davey	Frank Grills, P.Eng.
	24.1 to 24.3	Sacré-Davey	Frank Grills, P.Eng.
	24.4	Mine Paste	Frank Palkovits, P.Eng.
25	Conclusions and Interpretations	All parties provided input	All
26	Recommendations	All parties provided input	All
27	References	All parties provided input	All
28	Certificate of Authors	All parties provided input	All
29	Signature Page	All parties provided input	All

The QPs preparing this Feasibility Study are specialists in geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, infrastructure, capital and operating cost estimation, and mineral economics.

None of the QPs or associates employed in the preparation of this report have any beneficial interest in Ascot. The QPs are not insiders, associates, or affiliates of Ascot. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Ascot and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

2.3 Site Visits and Scope of Personal Inspection

Table 2-2 provides a list of QP site visits conducted to the Project site.

Table 2-2: QP Site Visits

QP	Dates	Accompanied By	Description of Inspection
Sue Bird, P.Eng.	Sep 4-6, 2018 and Jun 17-20 2019		• PGP: • Inspection of the current drilling and drill hole collar locations and survey methods • Verification of historical drill holes • Flyover to obtain the general site geology for all five deposits, as well as examination of outcrops and adits • Discussion of geology and updated structural interpretations including examination of the core foe several mineralized intervals • Discussion with the site geologists of sample preparation, handling, storage, and transportation • Picking of core samples at Silver Coin for re-assay validation of legacy drilling.
Dr. Gilles Arseneau, P.Geo.	Oct 23-24, 2018		RMP: • Verified property access, logistics, audited logging procedures and visited the underground • Collected check samples for validation
Aleksandar Petrovic, P.Eng.	Did not visit site		• QP relied on information provided by other Sedgman staff that have visited the site on several occasions to assess the current condition of the PGP.
Frank Palkovits, P.Eng.	Jun 30, 2018	John Kiernan	• Inspection of existing PGP site access road power, water treatment, adits, Mill, surface infrastructure, and TSF.
Jim Fogarty, P.Eng.	Jun 17-19, 2019	Rex Johnston	• Inspection of PGP's TSF, WTP Settling Ponds, Mill, existing surface water infrastructure, S1 Pit, and Providence Pit.
Robert Marsland, P.Eng.	Jul 16, 2019	Rex Johnston, Kris Kornylo	• PGP's Province pit, S-1 Pit, Dago pit, LLH low level outlet, mill building, assay lab building, 4-L tailings area, TSF, WTP settling ponds.
Soren Jensen, P.Eng.	Jul 16 and 17, 2019	Rex Johnston	• Inspection of the existing PGP WTP, water management infrastructure, existing contact water sources and overall site tour.
Brendon Masson, P.Eng.	Jun 25, 2019	Chris Houston	• Inspection of PGP's bridges and existing site roads.

QP	Dates	Accompanied By	Description of Inspection
Shervin Teymouri, P.Eng.	Did not visit site		• The financial analyst is not required to visit the site.
Frank Grills, P.Eng.	Nov 26-27, 2019	Dave Green, Wing Wong, Bryan Drew, Rex Johnston, Malcolm Cameron	• Inspection of the PGP non-process areas of the mill building, cold storage/assay lab area, propane, and fuel facility, existing WTP, and minor existing infrastructure. Inspection of the primary crusher.
Ken Savage, P.Eng.	Did not visit site		

2.4 Effective Dates

The QP for the resource estimation work at PGP was completed by independent consultant Susan Bird, P.Eng. (APEGBC) with an effective date of the Mineral Resource Statement of December 12, 2019.

The QP for the resource estimation work for RMP was completed by independent consultant Dr. Gilles Arseneau, P.Geo. (APEGBC) with an effective date of the Mineral Resource Statement of August 30, 2019.

The Effective Date of this NI 43-101 Technical Report is April 15, 2020.

2.5 Information Sources and References

The authors, in writing this Report use sources of information as listed in the references Section 27. Reports have been prepared by qualified persons holding post-secondary geology, or related university degree(s), and are therefore deemed to be accurate.

The purpose of this Report is to provide a NI 43-101 compliant Feasibility Study Technical Report for the Project.

Ascot's involvement with the PGP property dates back to 2007, when the first option agreement was made on the Dilworth property. Two years later, Ascot acquired the Big Missouri-Premier property via a second option agreement. The Silver Coin property, which is adjacent to the Big Missouri property, was acquired in October 2018 from Jayden Resources Inc. (Jayden) and Mountain Boy Minerals Ltd. (MBM). The Silver Coin property is host to epithermal gold-silver-bearing veins and breccias similar to those in the rest of the PGP area.

A Mineral Resource Estimate for the PGP was disclosed in January 2019 in a Technical Report by RPA (Rennie, Bird and Butler, 2019). This estimate is summarized in Section 6 of this report and compared to the current estimate in Section 14 by BRCC.

The RMP resources is a compilation derived from the historical work performed by previous operators from 1986 to present, and first principles design and estimate work by ACS.

Data used in the compilation was derived from unpublished historical reports by Bond Gold Inc., (Bond), Lac Minerals Ltd. (LAC), Royal Oak Mines Inc. (ROM), North American Metals Corp. (NAMC), Seabridge and Banks Island Gold Inc. (Banks).

Bond collected primarily exploration data. LAC continued with exploration and conducted numerous engineering studies, which culminated in a draft feasibility study. ROM conducted exploration and during the NAMC program. Detailed studies of mineralization were conducted by NAMC staff in conjunction with consultants during which all drill holes were re-logged within a 20 m shell of the current resource boundary identified in this report. Seabridge was engaged in several Preliminary Economic Assessment (PEA), as well as tailings management facility studies. Banks completed a PEA in 2013 (Baldwin and Jones, 2013).

Engineering and geological information from historical documents was used in this report after determination by ACS that the work was performed at RMP by competent persons or engineering firms. Data derived from engineering companies, consultants, and authors are listed in Section 27.

2.5.1 Units, Currency, and Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort has been made to clearly display the appropriate units being used throughout this technical report. Currency is in Canadian dollars ($) unless otherwise noted.

This NI 43-101 report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QP does not consider them to be material.

2.6 Previous Technical Reports

Refer to Section 27.

2.7 Important Notice

Each QP that assisted in the preparation of this Feasibility Study assumes responsibility for those sections or areas of this report that are referenced in Table 2-1. None of the QPs; however, accepts any responsibility or liability for the sections or areas of this report that were prepared by other QPs.

This NI 43-101 was prepared to allow Ascot to reach informed decisions respecting the development of the PGP. Except for the purposes legislated under provincial securities law: (a) any use of this report by any third party is at that party's sole risk, and none of the QPs (nor any of the companies for whom they work) shall have any liability to any third party for any such use for any reason whatsoever, including negligence, and (b) each of the QPs hereby disclaims responsibility for any indirect or consequential loss arising from any use of this report or the information contained herein. This report is intended to be read as a whole, and sections should not be read or relied upon out of context. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report contains estimates, projections, and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP’s opinion at the time that they are made, but in most cases involve significant risk and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated, or projected. None of the QPs undertake any obligation to update the forward-looking information.

This report contains the expression of the professional opinion of the QPs based on: (i) information available at the time of preparation, (ii) data supplied by outside sources, (iii) conclusions of other technical specialists named in this report, and (iv) the assumptions, conditions, and qualifications in this report. The quality of the information, conclusions, and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project, and are consistent with the intended level of accuracy for a Feasibility Study.

As permitted by Item 12 of Form 43-101F1, the QPs have, in the preparation of this Report, relied upon certain data provided to the QPs by the Owner and certain other parties. The relevant data and the extent of reliance on such data are described in Section 3 of this Report.

3 RELIANCE ON OTHER EXPERTS

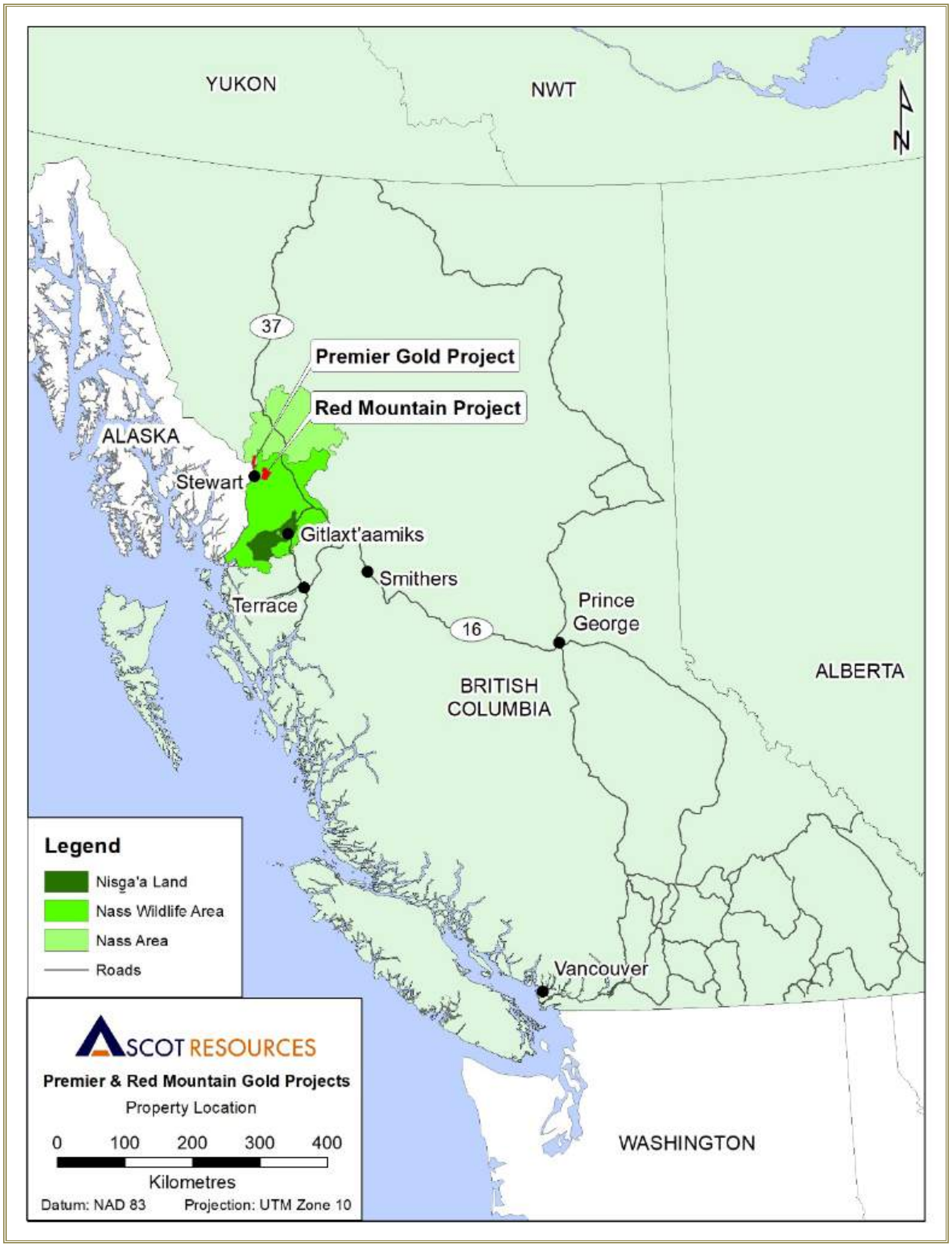
The Qualified Persons (QPs) opinions contained herein are based on information provided by Ascot and by others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for it.

Non-QP specialists relied upon for specific advice includes:

- Onsite Engineering Ltd.—Road Design and Cost Estimation (refer to Section 21)
- Prime Engineering—138 kV Transmission Line from Long Lake (Independent power Producer) and the New Main Substation Costs (refer to Section 21)
- Soucie Construction Ltd., Avalanche Risk Management and Mountain Safety Division (Report for IDM Mining Ltd.)—Avalanche Control Program Recommendations and Operating Costs (refer to Section 21)
- Falkirk Environmental Consultants—Environmental and Permitting (refer to Section 19)
- Integrated Sustainability—Water Treatment Plant Design and Cost
- CGT Industrial Ltd.—Report Assessing Present Condition of the Process Plant.

4 PROPERTY DESCRIPTION AND LOCATION

The Premier Gold Project (PGP) and the Red Mountain Project (RMP) are located in the Skeena Mining Division, in northern British Columbia (BC), Canada, Figure 4-1).



Source: IDM, 2017

Figure 4-1: Premier Gold Project and Red Mountain Location

4.1 Premier Gold Project

The Big Missouri deposit is in the central part of the PGP at latitude (lat.) 56° 7'N, longitude (long.) 130°1'W (UTM coordinates (NAD 83, Zone 9V) are 437785, 6219530 mN. The property lies approximately 20 km north-northeast of Stewart, British Columbia (Figure 4-2). Stewart is situated at the head of the Portland Canal, a 120-km long fjord and is commonly referred to as Canada's northernmost ice-free port. It is 880 km north west of Vancouver and 180 km north of Prince Rupert. Stewart is at the end of Highway 37A, a paved all-weather highway, 347 km from Smithers and 327 km from Terrace. The southern part of the PGP abuts the international boundary between British Columbia, Canada and Alaska, USA.

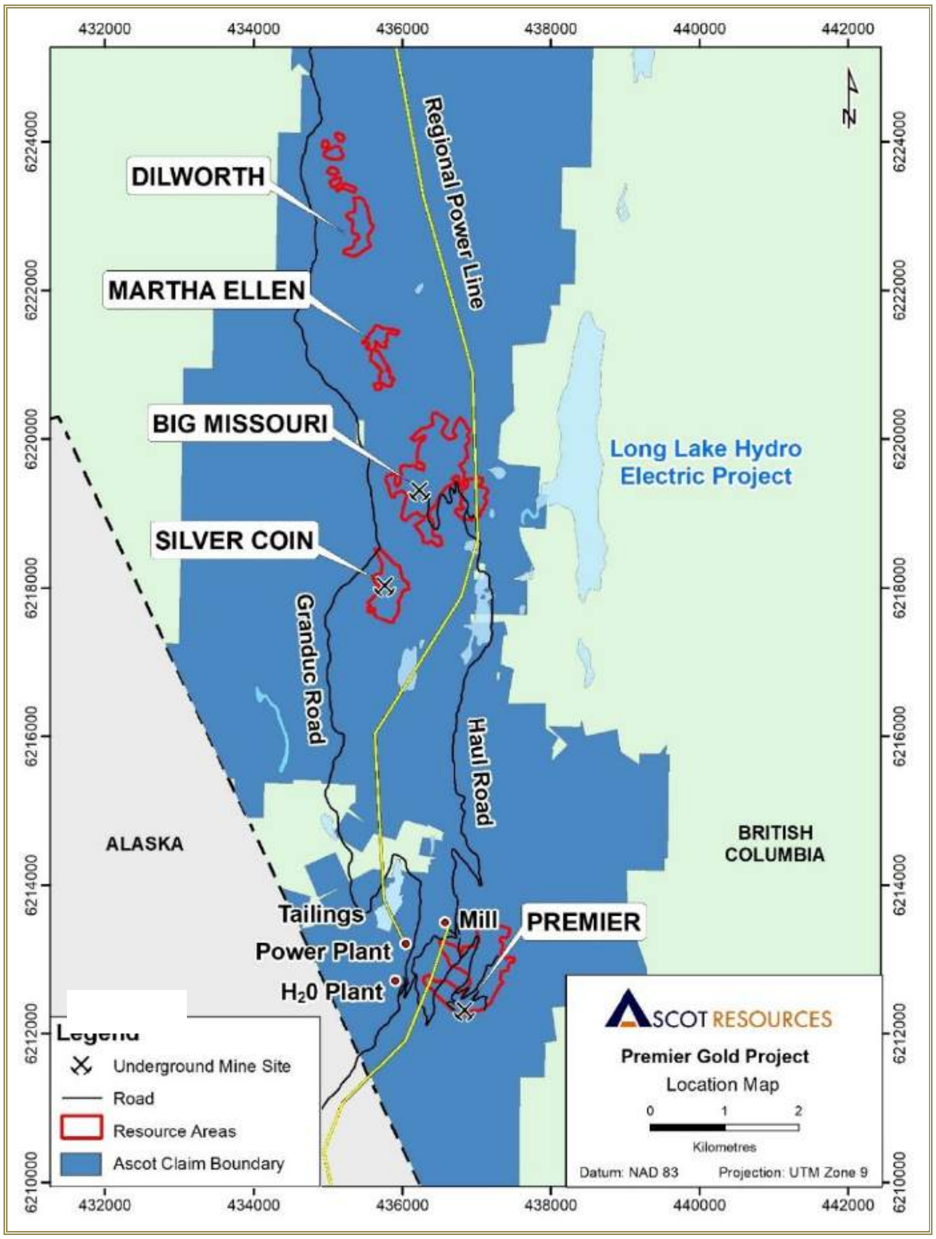


Figure 4-2: Location of the Premier Gold Project

4.1.1 Land Tenure

The Premier Gold Project area extends 22 km in a north–south direction and up to 4 km east–west. It comprises four claim groups, identified as the Premier, Big Missouri, Dilworth, and Silver Coin groups, and includes three mining leases, totalling 392 ha, 175 Crown grants totalling 2,354 ha, and 107 mineral claims totalling 8,907.1 ha. The total area is 8,133 ha when overlaps are accounted for.

The Property is covered by National Topographic System (NTS) map sheets 104A/04 and 104B/01, and BCGS map sheets 104A.001/011/021 and 104B.010/020/030. Coordinates for the area are as follows:

- Premier—lat. 56°4'N, long. 130°1'W (Zone 9V 437703 6213966)
- Big Missouri—lat. 56°7'N, long. 130°1'W (Zone 9V 437785 6219530)
- Dilworth—lat. 56°10'N, long. 130°1'W (Zone 9V 436867 6225095)
- Silver Coin—lat. 56°01'N, long. 130°00'W (Zone 9V 436000 6219000).

The Premier, Big Missouri, Dilworth, and Silver Coin properties are contiguous with one another. The Martha Ellen deposit is located within the Big Missouri Claim group.

Mineral tenure is illustrated in Figure 4-3 and Figure 4-4,and summarized in Table 4-1.

Table 4-1: Land Tenure Summary—PGP

Claim type	Number	Area (ha)
Premier Mineral Claims	46	2,388.05
Premier Mining Leases	3	392.00
Premier Grants, Mineral and Surface Title	13	178.53
Premier Grants, Mineral Title only	128	1,711.50
Big Missouri Grants, Mineral and Surface Title	3	30.46
Big Missouri Grants, Mineral Title only	26	367.66
Big Missouri Grants, Surface Title only	1	10.2
Dilworth Mineral Claims	17	3,624.34
Dilworth Crown Grants, Mineral Title only	3	35.80
Silver Coin Mineral Claims	44	2,892.72
Silver Coin Grants	1	19.50

Ascot's involvement with the Property dates back to 2007, when the first option agreement with Boliden was made on the Dilworth property. Two years later, Ascot acquired the Big Missouri–Premier property via a second option agreement with Boliden. From then until the present time, these agreements have undergone several amendments, but currently have been exercised, giving Ascot 100% ownership. The Silver Coin property, which is adjacent to the Big Missouri property, was acquired in October 29, 2018 from Jayden Resources Inc. and MBM. Details of the property agreements and amendments are provided in the following subsections.

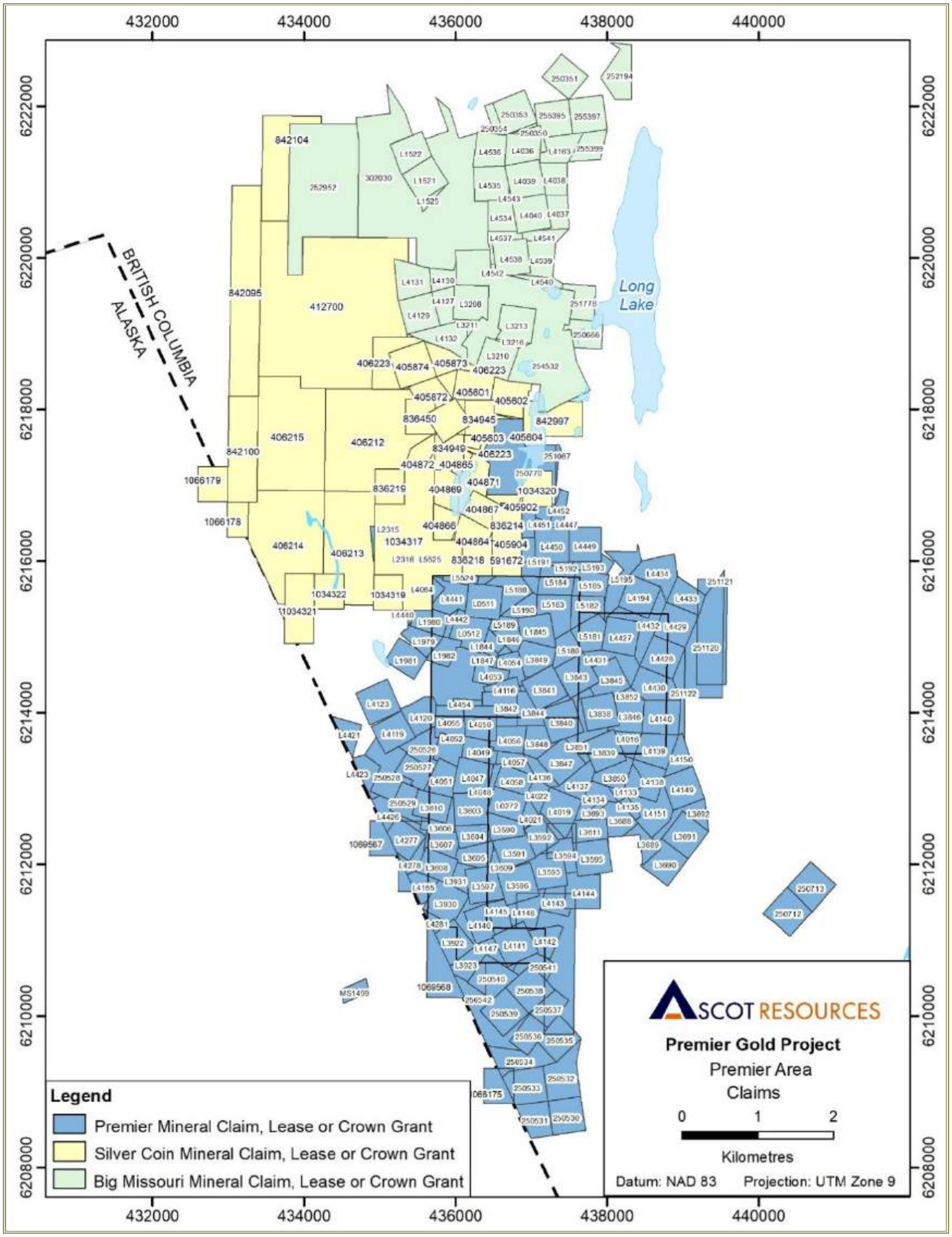


Figure 4-3: Claims for Premier, Big Missouri, Martha Ellen, and Silver Coin

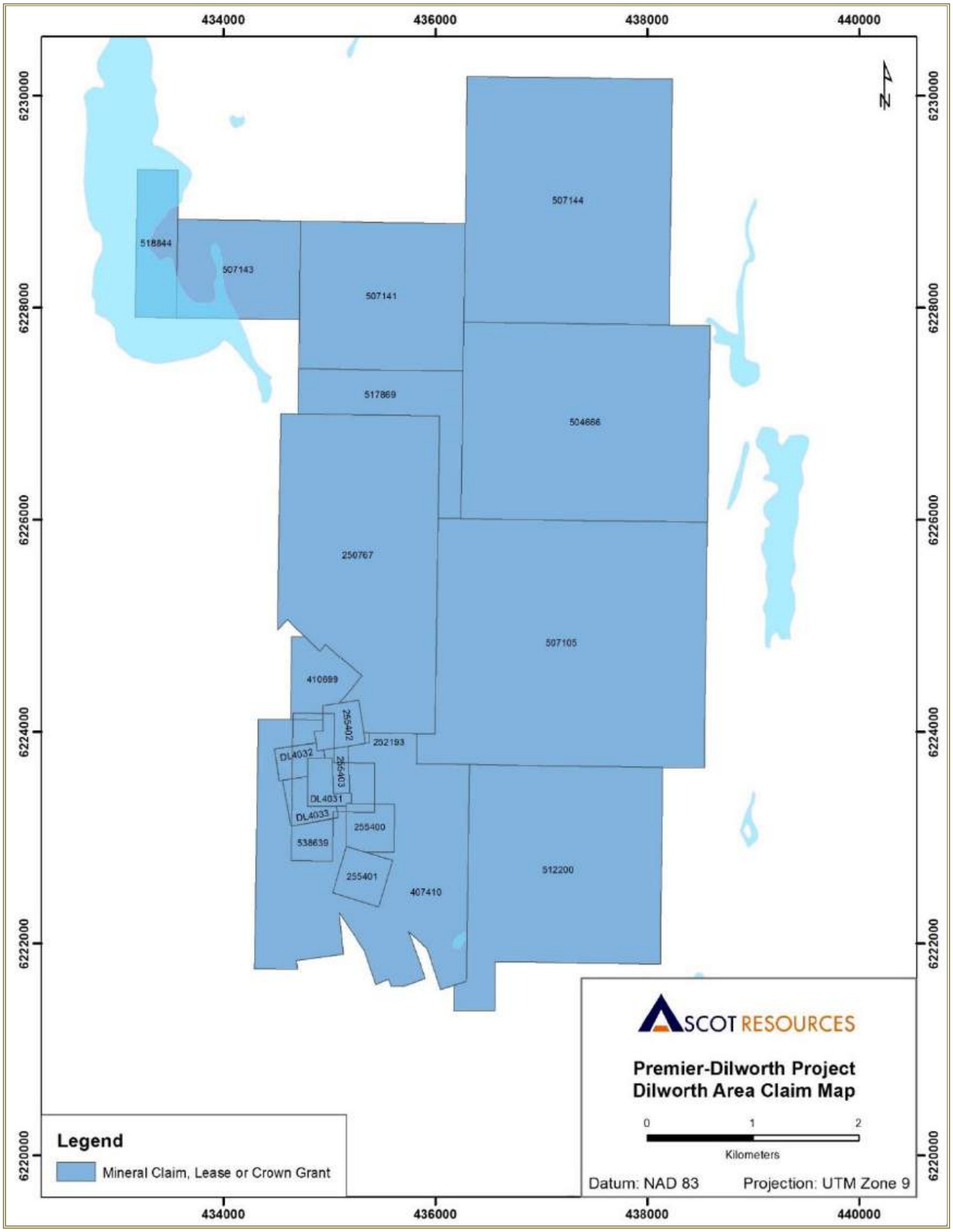


Figure 4-4: Claims for Dilworth

4.1.2 Premier, Big Missouri, Martha Ellen, and Dilworth Option Agreements

The original Dilworth property agreement between Ascot and owners Boliden, R. Kasum, and the estate of J. Wang, was signed in March 2007. Under the original terms, Ascot acquired the right to earn a 100% interest in the Dilworth property, subject to a 5% net smelter return (NSR), by making staged option payments over five years totalling $10.5 million.

On June 15, 2009, Ascot announced the signing of an option agreement to acquire a 100% interest in the mineral claims, mining leases, Crown-granted mineral claims, and freehold and surface titles of the Premier Gold Mine held by Boliden in the Premier Gold Camp. The Big Missouri claims were included in this agreement. The original agreement included cash payments totalling $20,300,000 over a period of three years and included a provision that in order to exercise the Premier option, Ascot would also exercise the Dilworth option.

The terms of both of these agreements have been amended several times, with revisions to payment due dates, the payment amounts, and NSRs. On October 17, 2018, Ascot announced that it had fulfilled the current terms of the agreements and acquired 100% of both the Dilworth and Premier properties. In order to fulfill the agreements, Ascot completed payments to Boliden totalling $11,050,000 and agreed to grant a 5% NSR to both Boliden and R. Kasum. Boliden retains the right of first refusal in the event that Ascot wishes to dispose of all or any part of its interest in the Premier property following establishment of the presence of significant base metal mineral reserves. Boliden also retains an option to enter a long-term base metals offtake agreement with Ascot on commencement of commercial production at Premier.

In November 2007, Ascot purchased from F. McEwan three Crown grants that were surrounded by the Dilworth property. The purchase price was 200,000 shares of Ascot, $100,000, and a 1% NSR on the Crown grants. At the time of writing, the payments have been made but the Crown grants have not yet been signed over to Ascot, pending resolution of the estate of Mr. McEwan.

It is noted that in addition to the 5% NSR agreed to Boliden and Kasum, there are a number of other NSR and net profit interest (NPI) obligations attached to certain claim groups from earlier property agreements. The current schedule of NSRs owing on the various claim packages is summarized as follows:

- Kasum Claims (Dilworth Option):
  - 5% NSR to R. Kasum can be purchased for $2.075 million
  - 1% NSR to R. Kasum and the estate of J. Wang, which can be purchased for $1 million
- Boliden Claims (Dilworth Option):
  - 5% NSR to Boliden can be purchased for $2.075 million
  - 1% NSR to Chase Manhattan Bank (now JP Morgan Chase Bank, N.A. [Chase])
  - 5% NPI to Chase
- Boliden Claims (Premier Option):
  - 5% NSR to Boliden can be purchased for $9.55 million
  - 1% NSR to Chase
  - 5% NPI to Chase
- McEwan Claims:
  - 1% NSR to the estate of F. McEwan.

Note that the 1% NSR and 5% NPI owing to Chase result from earlier agreements that predate Ascot's involvement in the Property. Figure 4-5 shows the agreements for these various land packages.

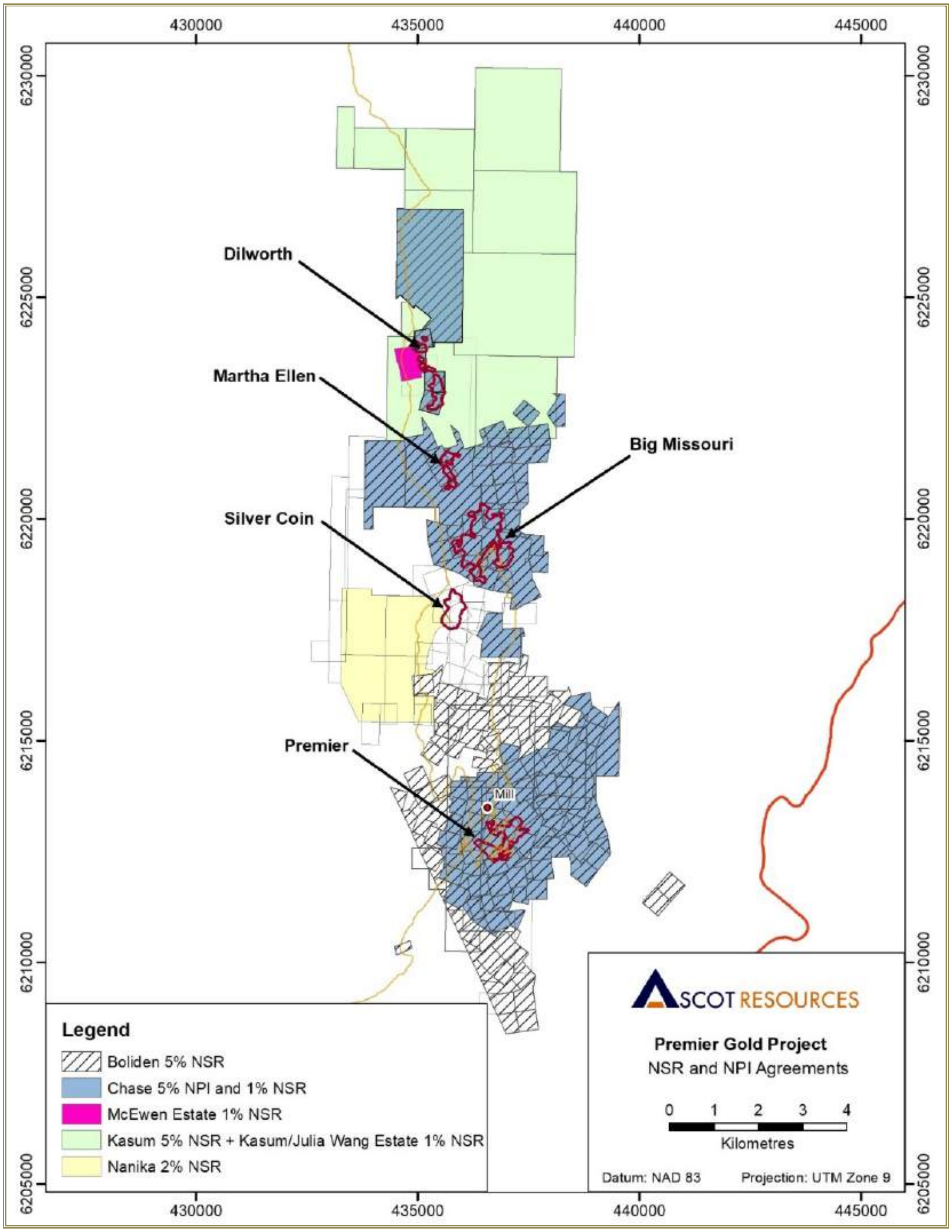


Figure 4-5: NSR and NPI Agreements

4.1.3 Silver Coin Agreement

The Silver Coin property is 100% owned by Ascot. Prior to Ascot's acquisition, Silver Coin was held under a joint venture agreement between Jayden Resources (Canada) Inc. (Jayden Canada), a subsidiary of Jayden, and Mountain Boy Minerals Inc. (MBM). Jayden Canada owned 80% of the Property with the remaining 20% owned by MBM. On October 29, 2018, Ascot announced that it had completed the purchase of the outstanding shares of Jayden Canada in exchange for 14,987,497 Ascot shares, plus an additional 192,000 Ascot shares for settlement of options and warrants. Concurrent with this, Ascot acquired MBM's 20% interest in exchange for 3,746,874 Ascot shares, plus an additional 48,000 shares for settlement of Jayden options and warrants.

Nanika Resources Inc. (Nanika) retains a 2% NSR on the INDI claims pursuant to an earlier purchase agreement with Jayden. The NSR can be bought back for $1,000,000 for each 1% NSR.

4.1.4 Property Commitments

The PGP property encompasses mineral claims, Crown grants, and mining leases, all of which have different annual requirements to maintain tenure. Mineral claims require either completion of exploration or development work (assessment work) above a certain minimum value or a payment of cash. The value of assessment work required to hold a mineral claim for one year is on a scaled rate, which depends on the age of the claims. For the first two years, the work required is $5.00/ha/a; in years three and four, $10.00/ha/a; Year 5 and Year 6, $15.00/ha/a; and thereafter, $20.00/ha/a. If the total value of the work done exceeds the amount required for the current year, the balance can be applied to subsequent years.

Crown grants require an annual payment of taxes to the provincial government in the amount of $1.25/ha. Ascot reports that all taxes for the Crown grants are current and paid to July 2, 2019. The due date for the next tax payment is July 2, 2020.

Ascot owns three mining leases, two of which expire on December 17, 2020, and the third, which has recently been renewed, on December 14, 2048. The leases require an annual fee paid to the Provincial Government of $20.00/ha. Ascot reports that the mining lease fees have been paid for the current year.

4.2 Red Mountain Project

Red Mountain is in northwestern British Columbia, approximately 18 km east–northeast of Stewart (Figure 4-1) and 23 km southwest of the Premier mill. The Project is at lat. 55°57'N and long. 129°42'W, between the Cambria Ice Field and the Bromley Glacier, at elevations ranging between 1,500 and 2,000 m. The area is characterized by rugged, steep terrain, with difficult weather conditions typical of the north coastal mountains. Access to the site is presently by helicopter from Stewart with a flight time of 10 to 15 minutes. In the 1990s a road was pioneered from Highway 37A up the Bitter Creek valley to the base of Red Mountain. Two portions were washed out in 2011 and the road is mostly overgrown with vegetation.

The deposit is located under the summit of Red Mountain at elevations of between 1,600 and 2,000 m. The site is drained by Goldslide Creek, which flows southwest to the flank of the Bromley Glacier, and by Rio Blanco Creek. Both creeks are tributaries of Bitter Creek, which in turn is a tributary of the Bear River. The Bear River drains into tidewater just east of Stewart, on the Canadian side of the Portland Canal. The mouth of the Bear River is 1.5 km east of the Canada—USA boundary.

4.2.1 Mineral Title

IDM Mining Ltd. (IDM), a wholly owned subsidiary of Ascot, owns a 100% interest in 47 contiguous claims that comprise an area of 17,125.2 ha (Table 4-2 and Figure 4-6).

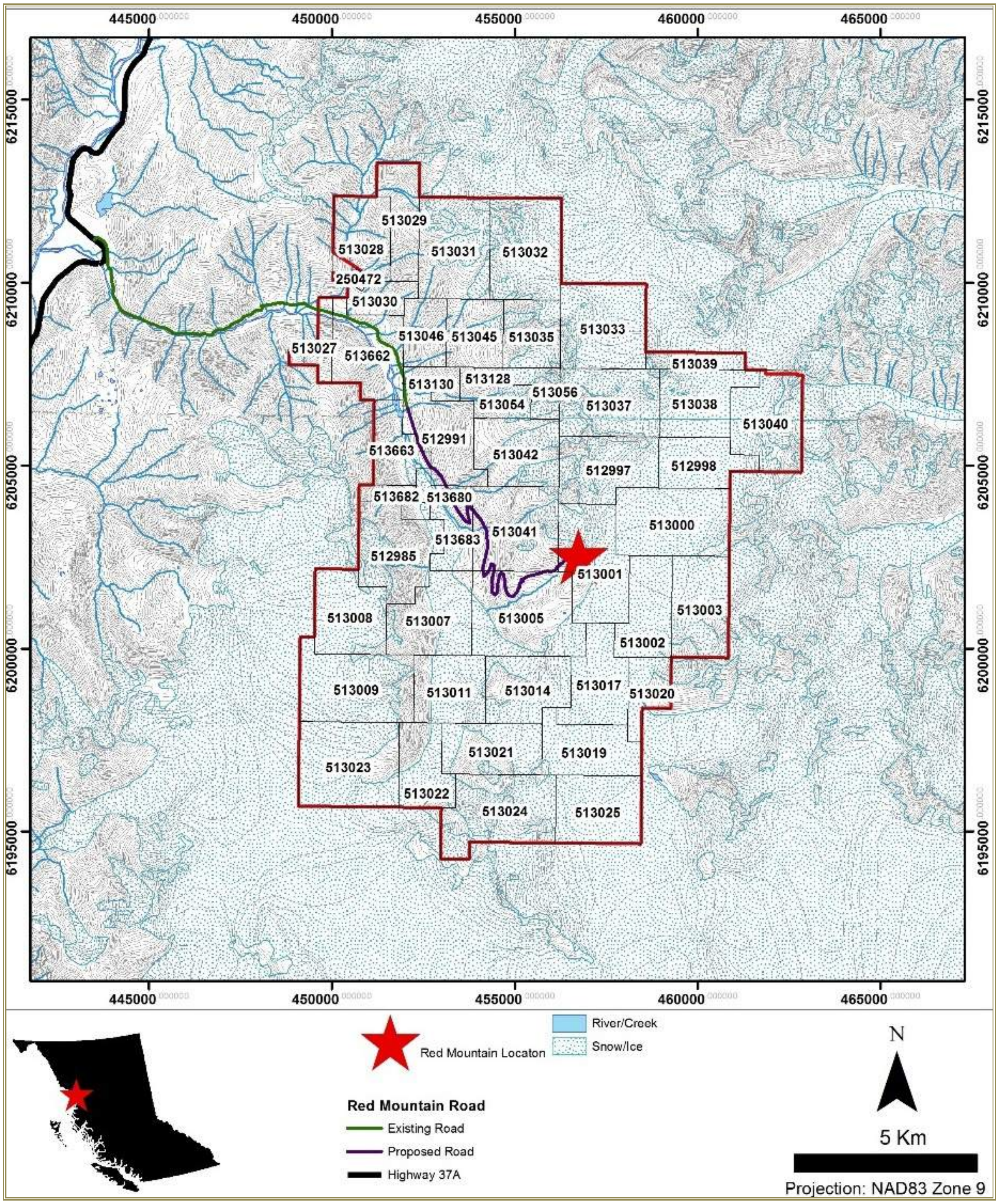
All claims are in good standing until May 9, 2023 according to documents provided by IDM, and information from the Government of BC (2020).

Table 4-2: Red Mountain Claims

Tenure Number	Tenure Type		Hectares	Ownership (%)
512997	Mineral	CLAIM	452.4	100
513001	Mineral	CLAIM	525.1	100
513028	Mineral	CLAIM	361.4	100
513040	Mineral	CLAIM	470.4	100
513046	Mineral	CLAIM	217.0	100
513054	Mineral	CLAIM	180.9	100
513662	Mineral	CLAIM	434.0	100
513002	Mineral	CLAIM	362.3	100
513024	Mineral	CLAIM	580.5	100
513045	Mineral	CLAIM	289.3	100
513130	Mineral	CLAIM	108.5	100
513007	Mineral	CLAIM	452.8	100
513017	Mineral	CLAIM	380.5	100
512985	Mineral	CLAIM	488.8	100
513005	Mineral	CLAIM	670.2	100
513014	Mineral	CLAIM	398.7	100
513019	Mineral	CLAIM	380.7	100
513031	Mineral	CLAIM	542.1	100
513032	Mineral	CLAIM	542.2	100
513033	Mineral	CLAIM	542.4	100
513038	Mineral	CLAIM	398.0	100
513009	Mineral	CLAIM	597.8	100
513021	Mineral	CLAIM	380.7	100
513056	Mineral	CLAIM	144.7	100
513022	Mineral	CLAIM	308.2	100
513023	Mineral	CLAIM	634.4	100
513680	Mineral	CLAIM	90.5	100
512998	Mineral	CLAIM	307.6	100
513027	Mineral	CLAIM	126.6	100
513029	Mineral	CLAIM	289.1	100
513030	Mineral	CLAIM	162.7	100
513682	Mineral	CLAIM	108.6	100
513000	Mineral	CLAIM	579.3	100

Tenure Number	Tenure Type		Hectares	Ownership (%)
513025	Mineral	CLAIM	435.4	100
513035	Mineral	CLAIM	289.3	100
513037	Mineral	CLAIM	506.5	100
513663	Mineral	CLAIM	253.3	100
513683	Mineral	CLAIM	181.0	100
513011	Mineral	CLAIM	362.4	100
513008	Mineral	CLAIM	416.5	100
513020	Mineral	CLAIM	199.3	100
513003	Mineral	CLAIM	434.7	100
513039	Mineral	CLAIM	126.6	100
513128	Mineral	CLAIM	36.2	100
512991	Mineral	CLAIM	416.2	100
513041	Mineral	CLAIM	543.1	100
513042	Mineral	CLAIM	416.2	100
Total Hectares			**17,125.2**	

Source: IDM, 2014



Source: IDM, 2014

Figure 4-6: Red Mountain Claim

4.2.2 Royalties, Agreements, and Encumbrances

Royalties, Metal Stream, and Other

The Red Mountain Gold Project is 100% owned by IDM, a wholly owned subsidiary of Ascot, and is subject to the payment of production royalties, an annual minimum royalty of $50,000 on the key Wotan Resources Corp. (Wotan) claim group, one payment upon commercial production, and a gold metal streaming arrangement.

Bond International Gold Inc. (Bond) assembled most of the existing Red Mountain property package in 1989 by way of three option agreements (these three options were exercised, and the claims purchased by Bond Gold's successor, LAC Minerals [LAC]). The agreements each provide for a royalty in the form of an NSR, and one of them, the Wotan agreement, has an area of influence.

In 1995, Barrick Gold Corporation (Barrick), the successor to LAC, sold the property to Royal Oak Mines Inc. (Royal Oak) and was granted a 1% NSR on all the then existing claims. In November 2013, Barrick transferred all its right, title, and interest in the 1% NSR to Franco-Nevada Corporation (Franco).

As a result, the bulk of the property has stacked NSR royalty obligations, ranging from 2.0% up to 6.5%. Certain peripheral, non-core claims that were staked by Bond or LAC carry a 1.0% NSR and three non-core claims staked by Royal Oak are royalty free.

The mineral resources in this Feasibility Study are subject to two royalties: 1.0% NSR payable to Franco and a 2.5% NSR payable to Wotan.

The mineral resources in this Feasibility Study are also subject to a gold metal stream whereby Seabridge Gold (Seabridge) may acquire up to 10% of the annual gold production from the Property at a cost of US$1,000/oz up to a maximum of 500,000 oz produced (50,000 oz to Seabridge). Alternatively, Seabridge may elect to receive a one-time cash payment of $4 million at the time of commencement of production in exchange for the buy-back of the metal stream.

Underlying Agreements

The principal agreements governing the Project are listed below, along with a summary of the more salient provisions and identified shading in Figure 4-7.

- Wotan Agreement: dated July 26, 1989 between Bond, Wotan, and Dino Cremonese granting Bond an option to acquire seven mineral claims. IDM is obligated to pay Wotan an uncapped 2.5% NSR royalty on production from these claims, and from any other properties within a 2 km area of influence extending from the boundaries of these claims. By October 31 of each year, a minimum royalty of $50,000 is payable. All minimum royalties paid from inception are deductible, once production is attained, against the NSR royalty amount otherwise payable.
- Krohman Sinitsin Agreement: IDM is obligated to pay Darcy Krohman and Greg Sinitsin a 1.0% NSR royalty on production from claims 513128 and 513130. IDM may buy out the royalty at any time for $500,000.
- Harkley Fegan Scott Agreement: Option agreement dated September 26, 1989 between Bond, Harkley Silver Mines Ltd., Stephen Fegen and Wesley Scott, as amended by letter agreement dated

September 30, 1992 between LAC and Harkley Silver. IDM is obligated to pay Harkley Silver an uncapped 3.0% NSR royalty on production from claims 513042 and 513054.

- Franco Agreement: Separated royalty agreement dated May 25, 2017 between Franco and IDM granting an uncapped >1.0% NSR royalty on production from then existing claims sold by Barrick to Royal Oak. Franco is entitled to receive an additional $10.00/oz cash production payment on all ounces of gold produced from the property in excess of 1,850,000 oz.
- Seabridge Agreement: Option agreement dated April 15, 2014 between IDM and Seabridge Gold Inc., granted IDM the right to acquire 47 mineral claims. IDM is obligated to pay Seabridge a one-time $1.5 million upon the commencement of commercial production, and Seabridge also retained a gold metal stream on the RMP to acquire 10% of the annual gold production from the Property at a cost of US$1,000/oz up to a maximum of 500,000 oz produced (50,000 oz to Seabridge). Alternatively, Seabridge may elect to receive a one-time cash payment of $4 million at the commencement of production in exchange for the buy-back of the gold metal stream (the obligations described in this paragraph are referred to collectively as the “Secured Obligations”).
- Mortgage Charge and Security Agreement: Security Agreement dated May 25, 2017 between IDM and Seabridge, granting Seabridge as general and continuing collateral security for the payment of the Secured Obligations: a security interest in the mineral claims comprising the Project (together with leases made in replacement thereof); and all records, rights, and permits relating to or in connection with the mineral claims.
- Benefits Agreement: On March 19, 2019, Ascot’s wholly owned subsidiary IDM Mining Ltd., entered a benefits agreement with the Nisga’a Nation in which the Nisga’a nation will provide ongoing support and continued consultation for the development of a future gold mine. The Nisga’a nation will participate in the economic benefits of RMP through training, employment business opportunities, and financial payments. The summary of benefits outlined in the agreement are: cash payments tied to permitting, RMP financing, and production milestones, as well as annual funding during production as a percentage of provincial mineral tax; the training and employment committee to provide pre-employment training, advancement training, local sourcing, and establish success-based initiatives including business opportunities related to the development and operation of RMP.

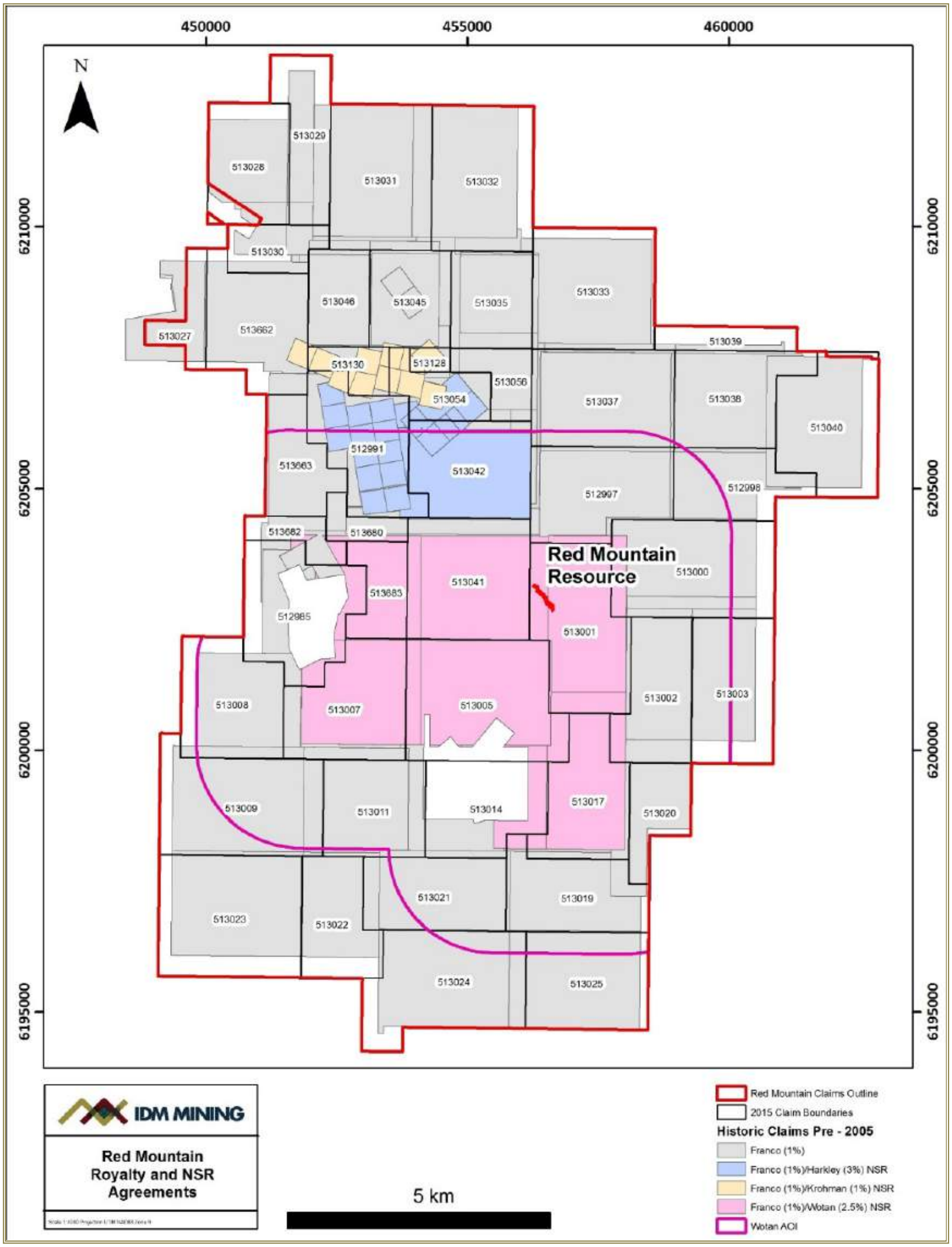


Figure 4-7: Red Mountain Royalty and NSR Agreements

4.2.3 Environmental Liabilities and Permitting

Environmental Liabilities

A $1,000,000 cash reclamation bond has been posted with the provincial government against the property, and can be recovered pending closure and remediation of certain environmental requirements, including the following:

- Reclamation and closure of approximately 50,000 tonnes of development waste rock that may be potentially acid-generating
- Closure of the decline portal
- Removal of equipment from the site.

Fuel, when used, is stored in containment at site and there is no record of any fuel spills. Water quality samples are collected monthly from Goldslide Creek and Bitter Creek as part of the baseline program. No hydrocarbons have been noted in lab analyses.

Required Permits and Status

Provincial approval and issuance of an Environmental Assessment Certificate was received on October 5, 2018 and federal approval of the Environmental Impact Statement (EIS) was received on January 14, 2019. IDM will continue to engage the appropriate provincial agencies to confirm future permitting requirements to progress the Project.

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

5.1 Accessibility

The Premier Gold Project (PGP) is readily accessible from Stewart along the gravel-surfaced Granduc Mining Road from Stewart, BC, through the town of Hyder, Alaska, and back into British Columbia. Three deposits are located at PGP and the fourth deposit is located at the Red Mountain Project (RMP), approximately 23 km to the southeast of the PGP mill. The Big Missouri deposit area is approximately 28 km from Stewart via the Granduc Mining Road, Premier Mine Road, and then Big Missouri Haul Road. From the Granduc Road, the Premier Mine and Big Missouri Mine roads provide further access to the central part of the PGP Property. Additional access is provided by old haul and skidder roads that are accessible by all-terrain vehicle (ATV), snowmobile, or hiking. Several helicopter companies maintain summer bases in Stewart.

The Red Mountain Project (RMP) is currently accessible only by helicopter. Road access up the Bitter Creek valley from Highway 37A was partially developed for 13 km by LAC Minerals Ltd. (LAC) in 1994 to the Hartley Gulch–Otter Creek area. Currently this road is passable for only a few kilometres from the highway. The remainder is not passable, as sections have been subjected to washout or landslide activity.

5.2 Local Resources and Infrastructure

Stewart reportedly had a population of 494 in 2013. The town provides services including fuel, groceries, lodging, helicopters, and a workforce. Situated at the head of the Portland Canal, Stewart has a deep seaport and loading facilities, and is Canada's most northerly ice-free port. Nearby Hyder, Alaska, has a population of approximately 90.

Principal infrastructure at the PGP consists of the following:

- Crush-grind-cyanidation processing plant building—semi-autogenous grinding (SAG) mill and ball mill (both were removed at time of closure in 1996) with rated capacity of 2,000 t/d up to 3,000 t/d depending on grind size and ore hardness
- Mill, shop, assay laboratory, cold storage building
- Camp and environmental monitoring office at 6 Level
- 1.6 MWh generator
- Water Treatment Plant (WTP)
- Tailings Storage Facility (TSF)
- Water monitoring and treatment systems, including settling ponds
- Access and site roadways
- Underground development and portals and waste dumps.

At Red Mountain, a surface tote-road network, basic surface structures (camp buildings, helipads, and waste rock storage areas), a shop, generator building, fuel tanks, and used mobile equipment remain from previous exploration activities and have been rehabilitated by IDM Mining Ltd. (IDM). Water is readily available from both surface and underground sources. As well, mineralized zones have been bulk sampled in the Marc zone, accessed from 1,500 m of existing underground decline and drift development that was fully rehabilitated in 2016 and 2017.

5.2.1 Availability and Sources of Power

Currently the PGP Site is served by a 25 kV powerline from Stewart. This arrangement will be modified with a new substation to be constructed adjacent to the processing plant that will receive power from the 31 MW power plant partly owned by Long Lake Hydro Inc. This facility is approximately 700 m from the mill and adjacent to the WTP built to supply Brucejack Mine (Pretium Resources Inc.) Long Lake power plant approximately 800 m south of the processing plant. Power will be distributed to the site from this substation. For further detail on site power, refer to Section 18.

5.2.2 Availability and Sources of Water

Fresh water sources are readily available from the nearby Cascade Creek. For further details on water sources refer to Section 18.

5.2.3 Availability and Sources of Personnel

The PGP is in close proximity to communities in the Stewart and Terrace. The PGP assumes the workforce for PGP will come from surrounding communities: Stewart, Terrace, and Smithers. The local transportation network allows the Project to easily access labour sources and technical expertise from major centres in western Canada.

5.3 Climate

Located at sea level, Stewart has a coastal rainforest climate, with a yearly precipitation of approximately 1,843 mm, much of it as snow, and an average yearly temperature of 6°C, according to Environment Canada. Average monthly temperatures are minus 3.7°C in January and 15.1°C in July. Significant snowfall accumulations restrict fieldwork at higher elevations. Conditions at PGP are very similar to Stewart. A weather station was established at the site in 2001.

Climatic conditions at Red Mountain are dictated primarily by proximity to the Pacific Ocean and its altitude, 1,742 metres above sea level (masl) at the centre of the deposit. Temperatures are moderated year-round by the coastal influence. Precipitation is significant in all months, with October being the wettest. Even at sea level, over one-third of the annual precipitation falls as snow. This proportion is greater at higher elevations, where snow may fall at almost any time of year.

The heavy snowfall, steep terrain and frequently windy conditions present a challenging combination. Blizzard conditions are frequent in the immediate vicinity of Red Mountain during winter, and avalanches pose a significant threat in the Bitter Creek valley and in the upper Bear River valley through which Highway 37A passes.

5.4 Physiography

The PGP is located along the eastern margin of the Coast Mountains. The Salmon River and Salmon Glacier bound the Property to the west. In the southern part of the Premier property, Bear Ridge forms a height of land bounding the property to the east, while in the north, Mount Dilworth, elevation 1,660 m, dominates the Dilworth property. The lowest elevations are approximately 200 m in the eastern part of the Salmon River valley. The Salmon Glacier occupies the Salmon River valley to the west of the northern part of the Property. The Mt. Dilworth icefield covers a significant part of the Dilworth property.

The elevation around the main exploration areas at Big Missouri varies from 900 m to 1,100 m and the terrain is variable, ranging from gently rolling to rugged (Kirkham and Bjornson, 2012). The lower elevations on the Property are moderately forested with hemlock and low brush. Mid-elevations are blanketed with heather and thick moss, with some small trees. Higher elevations are mostly vegetation free, except for moss and lichens (Christopher, 2009).

The Red Mountain property covers a large range of elevation, from approximately 250 m splice at the bottom of Bitter Creek valley to 2,557 m at the peak of Otter Mountain. Approximately 55% of the property is permanently covered by ice fields and glaciers. The eastern and southwestern parts of the property are covered by the Cambria ice field, the most prominent feature of the property. The Bitter Creek valley runs roughly north–south through the western part of the property. The valley and its flanks along with several ice-free mountain tops and nunataks are the principal areas available for exploration on the property.

6 HISTORY

6.1 Prior Ownership

6.1.1 Premier, Big Missouri, Martha Ellen, and Dilworth

Exploration commenced in the region in the latter part of the 19th century, with the first discoveries in the district occurring in 1898 (McConnell, 1913). Prospectors looking unsuccessfully for placer deposits turned to hard-rock exploration, and staked the first claims along Bitter Creek, located northeast of present-day Stewart, BC. At that time, the border between Alaska and British Columbia had not been formally established and these initial claims in the district were staked under US mining law.

Claims were first staked in 1904 on the Big Missouri deposit, located 8 km north of the Premier area (Kirkham and Bjornson, 2012). Prospecting and development were conducted by Big Missouri Mining Co. Ltd. until 1927, when the property was acquired by Buena Vista Mining Co. Ltd. (http://www.stewartbc.com). Consolidated Mining and Smelting Company (Cominco) subsequently took over the property, commencing production in 1938. Wartime economic pressures caused the mine to be shut down in 1941.

The first claims over the present Premier property were staked in 1910 by the Bunting brothers and W. Dilworth (Brown, 1987) and still form part of the present-day land holdings. Salmon-Bear Mining Co. conducted development work on the property until 1914, when the property was optioned to a group based in New York. Following the completion of underground development that did not produce positive results, the option was dropped. Work resumed in 1918, and Premier Gold Mining Company, Limited (Premier Gold) was incorporated early the following year to undertake exploration. American Smelting and Refining Company (Asarco) acquired a 52% interest in the property from Premier Gold in 1919 by agreeing to finance the development work. All ore produced was shipped directly to a smelter in Tacoma, Washington, until 1921, when a 200 t/d mill was completed. In 1926, the mill throughput was increased to 400 t/d, and again in 1933 to 500 t/d. Despite this, from 1924 to 1931, 45% of the production was direct-shipped to the smelter (Brown, 1987).

The Indian Mine, 5 km north of Premier, was first staked in 1910. A tram line from the property to the Premier mill was completed in 1951, but commercial production ceased in 1953 due to low metal prices.

Mining and development work continued on various showings in and around the Premier property until 1936, when Premier Gold, Sebakwe and District Mines Ltd., and B.C. Silver Mines Ltd. merged to form Silbak Premier Mines Limited (Silbak Premier). This effectively consolidated a collection of adjacent and contiguous claims and workings into a much larger block. Continuous production took place on the property up to 1953, when low metal prices forced a temporary closure. A fire destroyed the mill and other surface infrastructure in 1956. Intermittent mining and development activity extended into the 1970s under various lessors and management groups.

Silbak Premier underwent a name change to British Silbak Premier Mines Limited (BSP) in 1977, and in 1983 optioned a 50% interest in the property to Westmin. Canaccord Resources Inc. (Canaccord) earned 18.75% of Westmin's interest by funding exploration drilling in 1986 and 1987. Pioneer Metals Corporation (Pioneer) purchased controlling interest in BSP in 1987, amalgamating the two companies the following year.

Westmin acquired the Big Missouri property in 1978 from Tournigan Mining Explorations Ltd. (Tournigan). Government of BC (2020) reports that in 1987 the ownership of the entire Premier-Dilworth-Big Missouri property was: 50.1% Westmin; 40.0% Pioneer; 9.9% Canaccord; with Tournigan holding a 5% NSI. This ownership arrangement was via a joint venture agreement between the various stakeholders. Pioneer and Canaccord subsequently defaulted and forfeited their interests, giving Westmin 100% ownership.

After undertaking a drill program, Westmin built a mill and started operations on the old Silbak Premier property in 1989 (http://www.ascotgold.com). Production from open pit and underground began in March 1989 and continued to 1996. The mill capacity was 2,850 t/d and incorporated a carbon-in-leach (CIL) circuit for gold and silver extraction, followed by zinc cementation of the precious metals and smelting of a doré product. Reported metallurgical recoveries were 91% for gold and 45% for silver. Production to 1996 totalled approximately 260,000 oz of gold and 5.1 Moz of silver (Westmin, 1997).

In 1998, Boliden purchased Westmin and assumed ownership of the properties. Ascot acquired its interest through an option agreement with Boliden in 2007. Terms of this agreement have evolved over time, and the current property ownership is described in more detail in the section of this report titled Land Tenure.

6.1.2 Silver Coin

The history of the Silver Coin property is largely derived from the Silver Coin technical report by Minarco-MineConsult (MMC), dated April 13, 2011.

The Silver Coin project includes the historical Terminus, Silver Butte, and Silver Coin properties. The Terminus property includes the Silver Coin 3 and 4 mineral claims. The Silver Butte property includes the Winer, Big Missouri, and Kansas claims. The Silver Coin property includes the Silver Coin, Idaho, Idaho Fraction, and Dan Fraction mineral claims.

The Silver Coin group of claims was located in 1904 along the Big Missouri Ridge. The property was owned by the Noble family from the 1930s until 2003. In the early 1930s, a short adit was completed on the Dan showing. A number of pits were excavated on the Silver Coin and Idaho claims in the late 1930s. In 1967, Granduc Mines Ltd. cleared the adit on the Dan showing and completed sampling and trenching.

MBM first acquired a 100% interest in the Silver Coin property in 2003. Along with the Silver Coin property, MBM held a 55% interest in the adjacent Dauntless property. The following year, MBM sold 51% of its respective property interests to Pinnacle Mines Ltd. (Pinnacle) in exchange for exploration expenditures of $1.75 million over a three-year period. In 2006, these terms were fulfilled, and Pinnacle earned the 51% ownership. Later that same year, Pinnacle and Tenajon Resources Corp. (Tenajon) concluded an agreement wherein Pinnacle could earn up to 60% of the Kansas claim—a Crown grant surrounded by the Silver Coin claims. Under the terms of the original Silver Coin sale agreement, MBM retained the right to participate and acquire 49% of Pinnacle's interest in the Kansas claim.

In July 2009, MBM and Pinnacle entered into a purchase agreement under which Pinnacle could increase its ownership of the Project to 70% by paying MBM $440,000. A further 10% interest could be acquired by Pinnacle by spending $4 million on exploration. On completion of this deal, Pinnacle held 80% of the Silver Coin Project, and MBM held 20%.

In June 2010, Pinnacle changed its company name to Jayden Resources Inc. (Jayden).

6.1.3 Red Mountain

Placer mining commenced in Bitter Creek at the base of Red Mountain at the turn of the 20th century, but significant work on the current deposit began in 1988 when Wotan Resources Inc. (Wotan) staked claims in 1988 and optioned the property to Bond Gold Canada Inc. (Bond Gold) in 1989. The pre-1988 exploration history is outlined below:

- 1899/1902, discovery and small-scale mining of placer gold in Bitter Creek.
- 1912–1919 and 1940, Hartley Gulch Area, three adits developed, grades to 0.79 oz/t Au found.
- 1915 Shipment to Trail, BC, of 15 tons of hand-sorted ore from the Silver Tunnel (Roosevelt #1 claim on Roosevelt Creek). Smelter returns averaged 0.26 oz/t Au, 101 oz/t Ag, 34% Pb and 8% Zn.
- 1965 Hartley Flats, 4.8 tons of hand-cobbed ore from adits, shipped to Trail.
- 1965 Discovery of molybdenite mineralization and visible gold at McAdam Point—rock sampling, geological mapping, hand trenching, and diamond drilling (one 70 m AX hole). Rock sampling yielded an average of 0.475% MoS_2 over 137 m. One of the trenches yielded values of up to 64.45 g/t Au over 0.61 m.
- 1966–1973, rehabilitation and extension of the underground workings at the Silver Tunnel vein on Roosevelt #1 claim; production of about 5,000 tonnes of unknown grade. The ore was processed at the Adam custom mill on lower Bitter Creek.
- 1976, Jack Claims (central and southern portions of Red Mountain) staked by J. Howard and optioned to Zenore Resources Ltd.
- 1977–1978, Zenore Resources Ltd., logging and resampling of the 1967 drill core (samples assayed for molybdenum only), geological mapping, petrographic studies, rock geochemistry (assayed for copper, molybdenum, and gold).
- 1978–1980, Falconbridge Nickel Mines Ltd., reconnaissance program for porphyry copper-molybdenum targets in the Stewart, BC, area.
- 1987–1988, Chuck Kowall, working with a BC Government Prospector Assistance grant, prospected and acquired ground in the Goldslide and Willoughby Creek drainages and brought the area to the attention of Bond Gold.
- 1988–1989, staking of Red Mountain by Wotan and optioned to Bond Gold.

In 1989, gold mineralization at the Marc and Brad zones was discovered. Lac acquired Bond Gold in 1991. Surface drilling on the Marc, AV, and JW zones continued from 1991–1994. During 1993 and 1994, an underground decline was completed, with three crosscuts through the Marc zone for bulk sampling, which also provided access for underground drilling (Smit and Sieb, 1995). In 1995, LAC was acquired by Barrick Gold Corp, which subsequently sold the property to Royal Oak Mines Inc. (Royal Oak) in 1996. North American Metals, Inc. (NAM) purchased the property from the receivership sale of Royal Oak in 2000. They subsequently sold the property in 2002 to Seabridge, who optioned the property to Banks Island Gold Ltd. (Banks) in 2012; minor surface work and three surface core holes were completed. Banks terminated the option in 2013 and the property reverted to Seabridge. Seabridge subsequently optioned the property to IDM in 2014. Details of the exploration program IDM carried out are given in Section 9.6.

6.2 Exploration and Development History

The main events of the PGP's history prior to Ascot's involvement are summarized in Table 6-1.

Table 6-1: Summary of Property History for the PGP

Year	Operator	Exploration
1886	United States Army Corps of Engineers	First report of activity in the area was a survey undertaken by the United States Army Corps of Engineers.
1898	Prospectors	Prospectors first trekked inland from the head of the Portland Canal to Meziadin Lake in search of placer gold. Their search failed but later attempts by prospectors through the Klondike area started an influx of settlement in the area.
1904		Big Missouri claims, 8 km north of Premier, were staked.
1905	Stewart Bros.	Post office was established in Stewart by two brothers, John and Robert Stewart.
1907		Townsite of Stewart incorporated.
1910		Population of Stewart almost reached 2,000 and later experienced a population high of more than 10,000. Premier was first discovered by Charles Buntin and William Dilworth. The Indian Mine, located on Indian Ridge, 5 km north of Premier, was also discovered.
1917–1918		Population of Stewart decreased rapidly in First World War and only three people remained in town during winter of 1917–1918.
1918–1968	Various	The Silbak Premier mine reported to have produced 7.3 Mt of gold-silver-lead-zinc-copper mineralization almost continuously with minor amounts from 1976 to 1979 and 1989 to 1996. Original production was from underground mining operations.
1927–1942	Various	The Big Missouri deposit reported to have mined 768,941 tonnes yielding 58,383 oz Au and 52,676 oz Ag using underground mining methods.
1952–1953		The majority of the Indian Mine mineralization was produced in 1952 and transported by a two-mile aerial tramline for concentration at the Premier mill. The mine closed in 1953 due to low metal prices.
1972	Consolidated Silver Butte Mines Ltd.	Acquired Big Missouri claims.
1973	Giant Mascot Mines Ltd.	Option–11 holes drilled in 1974 on the Province claim.
1976	Tournigan Mining Explorations Inc.	Acquired the Big Missouri property from Silver Butte.
1976	Tapin Copper Mines	Option–8 holes drilled and IP survey completed.
1978	Westmin Resources Ltd. (formerly Western Mines Ltd.)	Acquired the Big Missouri property from Tournigan.
1979		Westmin commenced exploration on the properties.
1982		Westmin acquired the Silbak Premier property.
1988–1989		The new 2,000 t/d Premier mill facility was constructed.
1989		Westmin brought the Premier mill to operation after the consolidation of the Premier mining camp. It acquired a 100% interest in Premier and Big Missouri, as well as partial interest in the Indian and Silver Butte mines. The Premier pit and the S1 and Dago zones at Big Missouri were mined using open pit mining methods.
Dec. 1996		The Premier mill was closed due to low metal prices. The Property has been under care and maintenance since closure in 1996. From 1989 to 1996, Premier Gold was reported to produce 3,039,680 tonnes grading 0.085 oz/t Au and 1.67 oz/t Ag. At the time of the mill closure in 1996, the Property was reported to contain 350,140 tonnes of ore grading 7.19 g/t Au, 37.7 g/t Ag, and 1.6% Zn. Note that this estimate predates NI 43-101, is historical in nature, and should not be relied upon.

Table 6-2 is a recent chronological summary of exploration efforts on Red Mountain from 1988 to 2017.

Table 6-2: Red Mountain 1988 to 2017 Exploration Summary

Year	Exploration
1988–1989	Staking of Red Mountain by Wotan Resources Inc.
1989	Red Mountain and Wotan properties optioned to Bond Gold. Discovery of gold-silver mineralization by drilling in the Marc zone (3,623 m); airborne electromagnetic (EM) and magnetic survey.
1990	Bond Gold exploration of Marc zone and adjacent area (11,615 m of drilling).
1991	LAC acquired 100% of Bond Gold. A 2,400 m drill program was completed on the Marc and AV zones.
1992	Results of a 4,000 m drill program by LAC increased Red Mountain resources and indicated excellent potential for expansion.
1993	28,800 m of surface drilling defined the Marc, AV, and JW zones; identification of the 141 zone. An underground exploration adit allowed bulk sampling of the Marc zone. 8,600 m of underground drilling completed in the Marc zone.
1994	LAC completed a 350 m extension of the main decline, 30,000 m of underground drilling and 16,000 m of surface drilling.
1995	Barrick Gold acquires LAC. No exploration work completed by Barrick. Royal Oak purchases the Project from Barrick.
1996	Royal Oak undertakes exploration to explore for additional reserves. Extended underground drift by 304 m and completed 26,966 m of surface and underground drilling.
2000	North American Metals Inc. purchased the property and Project assets from Price Waterhouse Coopers; completed detailed relogging of existing drill core and constructed a geological model for resource estimation purposes and exploration modelling.
2002–2012	Seabridge purchases property, completes two Preliminary Economic Assessment (PEA) studies
2012-2013	Banks Island Gold options property, three surface drill holes completed. LiDAR survey and metallurgical work completed. Published PEA study.
2014	IDM enters into Option Agreement with Seabridge Gold. 12 core holes completed. Surface mapping and sampling completed, with emphasis on Cambria zone.
2016	Underground workings dewatered and rehabilitated. IDM drilled 11 surface holes and 51 underground holes totalling 8,123.44 m, completed surface rock and channel sampling, discovered gold mineralization at Lost Valley. Updated PEA was published. Metallurgical, geotechnical, and hydrological work completed.
2017	IDM drilled 11 surface and 105 underground holes totalling 29,299.26 m, completed surface rock and channel sampling, constructed a new geological model, and published a Feasibility Study. Claim ownership transfers from Seabridge to IDM Mining. Environmental Application and Environmental Impact Statement submitted.
2018	IDM drilled 40 underground holes totalling 10,021.81 m. The Project Environmental Assessment Certificate was received.

6.3 Previous Mineral Resource Estimate

A Mineral Resource Estimate for all five deposits of the PGP was announced in December 2018 by Ascot. Table 6-3 shows a summary of this estimate.

Table 6-3: Previous Resource Estimate for PGP

Class	Deposit	In-Situ Tonnage (t x '000s)	In-Situ Grades			Metal	
			AuEq (g/t)	Au (g/t)	Ag (g/t)	Au (oz x '000s)	Ag (oz x '000s)
Indicated	Premier	1,250	7.18	6.97	30.20	280	1,214
	Big Missouri	539	8.34	8.19	20.50	142	355
	Silver Coin	859	8.16	8.01	20.50	221	566
	Martha-Ellen	130	5.80	5.47	48.00	23	201
	Dilworth	-	-	-	-	0	0
	Total	**2,778**	**7.64**	**7.46**	**26.15**	**666**	**2,336**
Inferred	Premier	1,740	6.12	5.95	24.20	333	1,354
	Big Missouri	2,250	8.38	8.25	18.40	597	1,331
	Silver Coin	1,160	7.93	7.78	22.10	290	824
	Martha-Ellen	654	6.36	6.12	34.30	129	721
	Dilworth	235	6.52	6.13	56.10	46	424
	Total	**6,039**	**7.35**	**7.18**	**23.97**	**1,395**	**4,654**

Source: Ascot, 2018

Several resource estimates were completed in the past for Red Mountain at a 3 g/t Au cut-off. Any mineral resource estimates prepared prior to 2001 do not follow the requirements of NI 43-101. Mineral resources stated in Table 6-4 are stated only for historical completeness and should not be relied upon, as they are superseded by the mineral resources presented in Section 14.

Table 6-4: Historical Resource Estimates at Red Mountain

Date	Company	Classification	Tonnes	In-Situ Grade (Au g/t)	In-Situ Grade (Ag g/t)	In-Situ Contained (Au oz)	In-Situ Contained (Ag oz)
1992	LAC	NA	2,500,000	12.8	38.1	1,028,800	3,062,300
1993	LAC	NA	2,511,000	11.3	29.8	912,200	2,405,700
1994	LAC	NA	2,500,000	10.0	-	803,700	-
1994	LAC	NA	2,399,644	9.6	-	740,640	-
1994	LAC	NA	2,401,855	10.5	-	810,820	-
1995	LAC	NA	3,653,854	7.7	-	904,500	-
1995	LAC	NA	1,938,084	9.7	-	604,400	-
1996	ROM	NA	3,143,880	5.69	22.87	575,273	2,094,770
1997	ROM	NA	2,736,000	5.16	20.72	453,573	1,822,357
1998	ROM	NA	2,457,840	6.31	18.06	498,507	1,427,789
2001	NAMC[1]	M&I	1,594,000	7.80	29.27	400,000	1,499,700
2001	NAMC[1]	M&I	346,000	7.45	12.36	82,900	137,700
2002	Seabridge[1]	M&I	1,594,000	7.80	29.27	400,000	1,499,700
2002	Seabridge[1]	Inferred	346,000	7.45	12.36	82,900	137,500
2008	Seabridge[2]	M&I	882,400	10.55	31.85	299,300	903,500

Date	Company	Classification	Tonnes	In-Situ Grade (Au g/t)	In-Situ Grade (Ag g/t)	In-Situ Contained (Au oz)	In-Situ Contained (Ag oz)
2008	Seabridge[2]	Inferred	191,020	10.25	15.22	62,900	93,500
2013	Banks[3]	M&I	1,612,000	8.4	28.30	432,000	1,440,000
2013	Banks[3]	Inferred	807,000	5.4	10.20	140,000	260,000
2014	JDS[3]	M&I	1,454,300	8.15	29.57	380,900	1,382,800
2014	JDS[3]	Inferred	332,900	7.69	12.72	82,300	136,200
2016	ACS	M&I	1,641,600	8.36	26.00	441,500	1,379,800
2016	ACS	Inferred	548,100	6.10	9.00	107,500	153,700
2017	ACS	M&I	2,074,700	8.75	24.80	583,700	1,655,700
2017	ACS	Inferred	324,700	6.21	10.1	64,800	105,500
2018	ACS	M&I	2,771,300	7.91	22.75	704,600	2,026,800
2018	ACS	Inferred	316,000	6.04	7.6	61,400	72,000

Source: ACS (2018) with modifications. [(1)] 0 g/t Au cut-off; [(2)] 6 g/t Au cut-off; [(3)] 3 g/t Au cut-off. The 2001 NAMC resource was the base for the 2014 JDS resource.

6.4 Past Production

The Silbak Premier mine produced gold-silver-lead-zinc-copper ore intermittently from 1918 to 1996 from both open pit and underground mines. Historical production during the peak years of operation (1918 to 1952) totalled 2 Moz of gold, 42.8 Moz of silver, 54 Mlb of lead, 17.6 Mlb of zinc, 4.1 Mlb of copper, and 177,785 lb of cadmium. The Big Missouri deposit produced 847,612 tons of ore from underground from 1927 to 1942. Metal production totalled 58,383 oz of gold, 52,676 oz of silver, 3,920 lb of zinc, and 2,712 lb of lead. The S1 and Dago zones at Big Missouri property were mined using small open pits. In the Dago pit, 384,000 tonnes of ore grading 1.2 g/t Au and 10.0 g/t Ag were produced in 1988 and 1989. In 1990, a total of 304,000 tonnes of ore grading 2.4 g/t Au and 10.0 g/t Ag were produced in the S1 pit.

Westmin conducted extensive exploration from 1979 to 1996 on the Premier and Big Missouri properties. A 2,000 t/d mill facility was put into operation in 1989 and was closed in 1996 due to low metal prices. Premier Gold mine's total production amounted to 5.6 million tons grading 0.331 oz/ton Au and 7.117 oz/ton Ag from 1918 to 1987 and 3 million tons grading 0.085 oz/ton Au and 1.67 oz/ton Ag from 1989 to 1996. At the time of the mill closure in 1996, the Property reportedly had remaining reserves totaling 350,140 tonnes grading 7.19 g/t Au, 37.7 g/t Ag, and 1.6% Zn.

In the area of the Silver Coin property, a short adit was driven on massive galena veins in the Terminus zone (the present Silver Coin 2 claim) during the 1930s. Work continued intermittently with little documentation. Also, in the early 1930s, a short adit was driven on the Dan zone in the area of the Dan Fraction claim. Several small open pits were excavated on the property, including pits on the Silver Coin and Idaho zones.

Between 1987 and 1994, Tenajon and Westmin completed approximately 1,220 m of underground drifting on three levels, 103 m of crosscutting on one level, and 130 m of Alimak raising at Silver Coin. In 1991, Westmin mined the Facecut-35 zone producing 102,539 tonnes at an average grade of 8.9 g/t Au and 55.50 g/t Ag. Mining was primarily by sub-level retreat with a minor amount of benching. Base metal rich–low gold sections of the Facecut-35 Zone were not mined. No base metal values were recovered as the ore was processed using a cyanide leach process at the Premier mill, 5 km south of Silver Coin. Recoveries reportedly averaged 92.9% for gold and 45.7% for silver. Westmin estimated that 111,000 tonnes of material grading 0.61 g/t Au, 29 g/t Ag, and 3.46% Zn were directed to the tailings pond. Sampling in 2004 by MBM and Jayden (then Pinnacle) indicated that the mine tailings from the Facecut-35 zone averaged 0.72 g/t Au, 31.2 g/t Ag, 0.388% Cu, 0.48% Pb, and 3.61% Zn in two samples (Stone et al., 2007).

No historical production has taken place at Red Mountain.

7 GEOLOGICAL SETTINGS AND MINERALIZATION

7.1 Premier Gold Project

7.1.1 Regional Geology

As summarized by Alldrick (1993), the Stewart mining camp is underlain by Upper Triassic to Lower Jurassic rocks of the Hazelton Group that formed in an island-arc setting. The volcanic pile largely comprises subaerial calc-alkaline basalts, andesites, and dacites with interbedded sedimentary rocks. Lateral variations in volcanic rock textures indicate that the district was a regional paleo-topographic high with a volcanic vent centered near Mount Dilworth. Early Jurassic calc-alkaline hornblende granodiorite plutons of the Texas Creek Plutonic Suite represent coeval, subsidiary magma chambers emplaced 2 km to 5 km below the stratovolcano. From these plutons, late-stage two-feldspar porphyritic dikes cut up through the volcanic sequence to feed surface flows (locally called Premier Porphyries). Following the cessation of volcanism and subsidence, this succession was capped unconformably by the Middle Jurassic Mt. Dilworth and Salmon River formations, followed by later Upper Jurassic-Cretaceous marine-basin turbidites of the Bowser Lake Group.

Mid-Cretaceous tectonism was characterized by greenschist facies regional metamorphism, east-northeast compression, and deformation. It produced upright north-northwest trending en echelon folds and later east verging, ductile reverse faults, and related foliation.

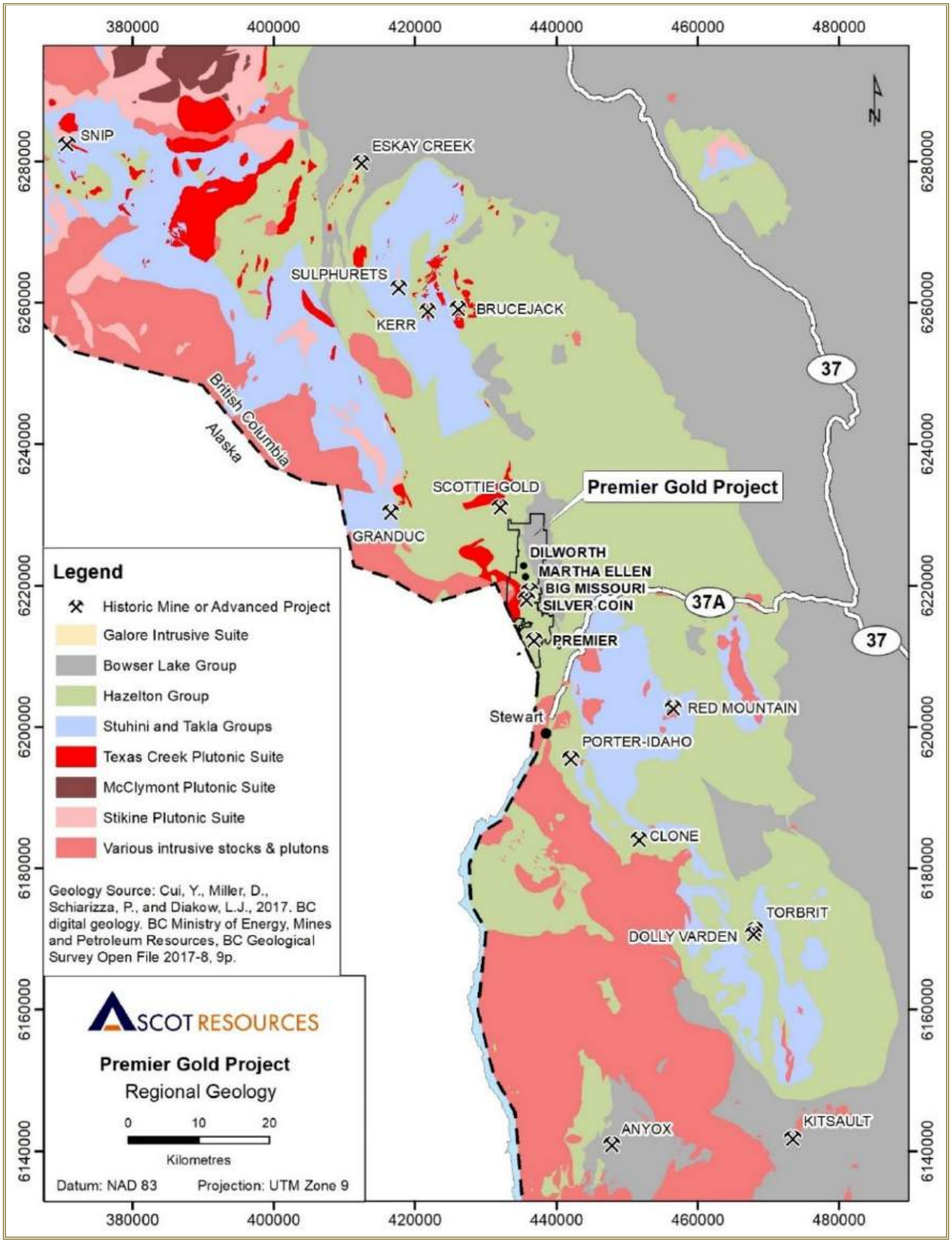
Calc-alkaline biotite granodiorite of the Coast Plutonic Complex intruded the deformed arc rocks during the Mid-Tertiary. The batholith, stocks, and differentiated dikes of the Hyder Plutonic Suite were emplaced over a 30-million-year period from Early Eocene to Late Oligocene. Regional geology is illustrated in Figure 7-1.

7.1.2 Local and Property Geology

Rocks of the Hazelton Group host most of the significant deposits and occurrences within the Property. Regional mapping by Alldrick (1993) and others determined that the entire Hazelton Group package between the Salmon Valley and Mount Dilworth was a north- to northwest-striking, steeply east dipping succession, younging to the east.

Recent work by Ascot demonstrates that at PGP's Silver Coin and Big Missouri deposits the stratigraphy is dipping steeply to the west and younging in that direction. The andesitic volcanics at the Premier deposit are massive flows and show no discernable stratigraphic orientation from the extensive drill database. The westerly dip of strata at Silver Coin and Big Missouri may be a local phenomenon, if Alldrick's observations are correct in a regional sense.

The overall PGP geology is illustrated in the plan map of Figure 7-2.



Source: Gagnon, 2012

Figure 7-1: Premier Gold Project Regional Geology

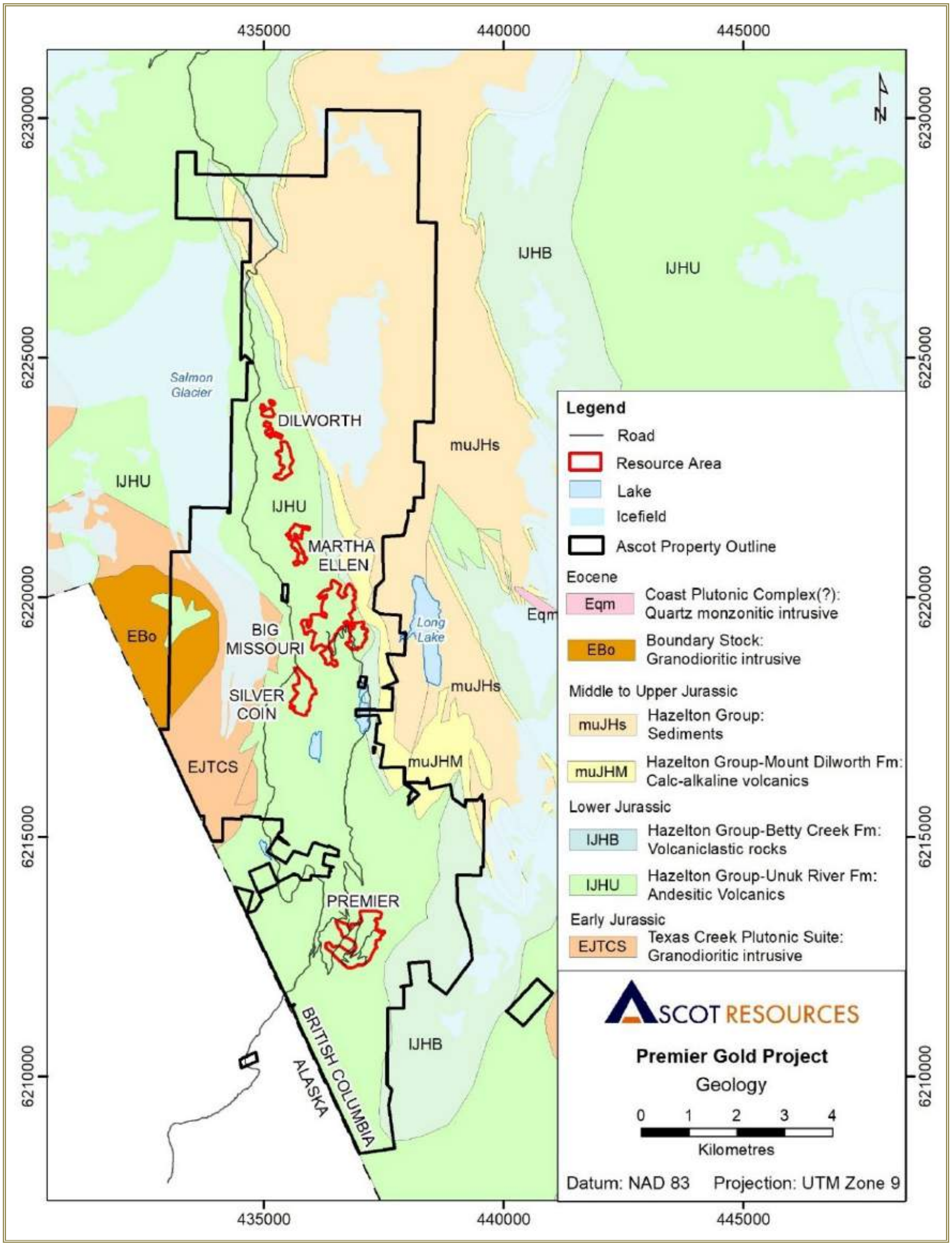


Figure 7-2: Property Geology – Premier, Big Missouri, Silver Coin, Dilworth, and Martha Ellen

Premier

On the Premier property, the Unuk River andesite is the oldest component of the Betty Creek Formation (Figure 7-2). These rocks on the east side of the Salmon Glacier occupy the west limb of a large synformal fold whose steeply inclined north-northwest trending axis passes beneath the Mount Dilworth icefield. This large F1 structure belongs to a phase of regional-scale deformation that resulted in tight isoclinal folds in both the volcanic and in the less competent sedimentary rocks (Alldrick, 1993). However, Ascot's extensive drilling at Premier has not encountered evidence of folding as described by Alldrick. Many units can be traced from drill hole to drill hole with little to no evidence of folding within the rock.

Alldrick (1993) stated that: "Like Big Missouri to the north, the Silbak Premier mine and several nearby showings are all in the Upper Andesite Member of the Unuk River Formation". Nelson (2018) has since suggested that the Unuk River Formation is andesites within the Betty Creek Formation and not a clear formation or member in itself. The black tuff facies, used as a marker in the Big Missouri area, is missing in the Premier area where the main sequence includes medium- to dark-green, moderately to strongly foliated andesitic ash tuff, lapilli tuff, and crystal tuff. The andesites at Premier are darker green and more strongly chloritized. Siltstone members within the Unuk River andesite can be mapped and used to evaluate movement on structures.

Dikes of Premier porphyry are the most abundant intrusive rocks at Premier and are spatially associated with some mineralized zones, particularly at Premier. At Big Missouri and Silver Coin, Premier porphyry has been observed in very small amounts and only at depth. The mineralized zones in these deposits are hosted in andesite with no spatial association to intrusive rocks.

Mid-Cretaceous tectonism was characterized by greenschist regional metamorphism, east-northeast compression, and regional deformation. Mid-Tertiary biotite granodiorite, representative of the Early Eocene to Late Oligocene Hyder Plutonic Suite of the Coast Plutonic Complex, caused further deformation.

Alldrick (1993) described four distinctive alteration envelopes that developed around the Premier mineralization as important guides for exploration. These are:

- Siliceous alteration consisting of siliceous envelope that may extend up to a few metres from major siliceous breccia bodies
- Sericite alteration (potassic) with pyrite, silica, and potassium feldspar
- Carbonate alteration
- Chlorite alteration (propylitic) resulting in darker green colour than in metamorphic greenschist.

Ascot work has shown that gold mineralization occurs in quartz-carbonate breccias and stockwork, mostly in andesite and sometimes hosted by Premier porphyry. The main alteration mineral is sericite, which typically forms an envelope around stockwork veins and breccia bodies. The formation temperature of the mineralization is too low to generate potassic alteration, and neither secondary biotite nor potassic feldspar has been observed in the alteration assemblage. Adularia is very hard to identify in hand specimens and may be present, albeit not as a major component. Chlorite alteration in the andesitic rocks is ubiquitous, and it is hard to say if any of it is related to the mineralizing event. Tertiary dykes often display an envelope of secondary chlorite.

Big Missouri

The central part of the Big Missouri deposit is dominantly hosted in the Upper Andesite Member of the Unuk River andesites. However, mineralization is also hosted in the underlying Upper Siltstone Member of the Betty Creek Formation in the west, and in the overlying tuffaceous units of the Betty Creek Formation in the east at the Dago and Unicorn areas. These stratigraphic associations are difficult to determine, as alteration masks many of the primary textures of these units. The area is further complicated by a series of east-directed thrust and reverse faults that offset mineralized zones. Recent drilling has also resulted in the recognition of the Premier porphyries in this area, including numerous sills and lenses of Premier porphyry along the eastern portion of the zone. These locally contain alteration and mineralization similar to the Premier area (Ascot Geologists, personal communication, 2018).

The alteration and showings associated with the Big Missouri deposit encompass a strike length of 2,200 m north-south by approximately 1,400 m east-west, across strike (Kirkham and Bjornson 2012). This area includes numerous historic occurrences including the Day, Big Missouri, S1, Calcite Cuts, Golden Crown, Dago, Creek, Unicorn, and Northstar zones. The mineralized area is associated with coincident Au, Ag, Pb, and Zn soil anomalies and a strong K and Th/K anomaly on airborne radiometric surveys.

Previous mining from select portions of this system includes underground mining of Big Missouri, and small open pits on Province, S1, and Dago showings. These historic showings, which were originally isolated, are now considered to be part of a single continuous mineralized system. The system consists of gently west to gently east dipping sheet-like stacked zones of silicification, quartz stockwork, and quartz breccia bodies.

Silver Coin

The Unuk River andesites which underlie most of the Silver Coin property and host most of the gold mineralization are part of a generally massive and monotonous volcanic-volcaniclastic sequence that lacks layering that would provide details on the strike of the stratigraphy or the presence of folds (Ray, 2011). Property geology is shown in Figure 7-2.

A north-south striking fault system has divided the Silver Coin property into different geologic areas:

- An area on the east side of the claim group that is bounded by the Cascade Creek Fault Zone
- An area located between the Cascade Creek Fault Zone and the Anomaly Creek Fault that is dominated by andesitic volcanic rocks
- The central portion of the claim block consisting of west-dipping andesite units hosting the majority of the mineralization at Silver Coin
- The Western part of the claim block west of the Granduc road consisting of andesitic rocks and Texas Creek granodiorite.

The sequence of predominantly andesitic volcanic and volcaniclastic rocks which constitutes the fault blocks described above was subsequently cut by numerous intrusive bodies of subvolcanic, porphyritic andesite, and less-numerous bodies of aphanitic dacite.

To the south of the graben, Texas Creek granodiorite and andesitic pyroclastic rocks crop out on the former Silver Coin Crown-granted claims (Stone and Godden, 2007). Foliated andesite is the most common rock type, with only a few outcrops of sheared limey argillite. The main features in the Silver Coin project area are lineaments striking northwest and northeast, which strongly influence the topography over most parts of the property. The lineaments are interpreted as zones of intense fracturing, probably with shearing on the N20°W set and possibly on the N25°E set.

The eastern portion of the Silver Coin property immediately to the west of the Cascade Creek Fault contains a silicified and mineralized fault zone that is up to 75 m wide, hosted within andesitic volcanic rocks, carrying 3% to 5% disseminated euhedral pyrite. The mineralized zones occur along a regional deformation zone extending from the former Big Missouri Mine through the Silver Coin 3 and 4 claims and south towards No Name Lake.

The last major geologic event in the area of the Silver Coin property was emplacement of the Jurassic granodioritic Texas Creek Batholith (Alldrick, 1993). Apophyses derived from this batholith intruded the metamorphosed Jurassic-Triassic volcano-sedimentary rocks along the Anomaly Creek Fault system.

The Anomaly Creek Fault has been interpreted as a right-lateral, oblique-slip structure of unknown displacement. The North Gully Fault has been interpreted as a reverse fault, the displacement of which is probably not large (the alteration zones on both sides of the fault do not appear to be significantly offset). The nature of movement on the North Gully Fault is not well understood, since little work has been done across the areas in which the structure is developed.

There are 20 different mineralized zones which have been identified on the Silver Coin property, and these are likely fault-separated portions of several larger or longer zones. Gold is generally associated with silicification and sericite alteration. Gold generally occurs as electrum with associated sulfide minerals pyrite and sphalerite, with minor amounts of galena and chalcopyrite.

Martha Ellen

The Martha Ellen deposit is located adjacent to the northwest end of the Big Missouri zone. Kirkham and Bjornson (2012) describe this deposit as a gently southwest-dipping zone which, based on showings, soil anomalies, and drilling, is approximately 1,400 m along strike (north-south) and 600 m to 800 m across strike.

The deposit is made up of sheet-like lenses of quartz stockwork and quartz breccias with a thickness of 40 m to 60 m. The deposit is hosted in the Upper Andesite member of the Unuk unit. Quartz-sericite-pyrite alteration is not as well developed as at Big Missouri. The gold and silver values are within quartz veins and quartz breccias containing pyrite, sphalerite, and minor chalcopyrite. The eastern portion of the zone is in contact with a large lobate body of Premier porphyry which contains altered and mineralized structures. This zone of mineralization is very similar in style to the western part of the Big Missouri area and is likely a fault offset, northerly strike extension of the Big Missouri zone. A large northeast linear reflects the Hercules fault, a late, left-lateral fault structure between these two zones that is interpreted to offset both stratigraphy and mineralization to the present location.

A wide swarm of Eocene-age Portland Canal granodiorite dikes intrudes the Martha Ellen zone, striking east-southeast and dipping south-southwest.

Dilworth

The Dilworth deposit is located on strike starting 500 m from the northwest end of the Martha Ellen zone. The zone is the northwest extension of the Martha Ellen deposit, but the intervening area is disrupted by an extensive northwest-striking Eocene multiphase dike swarm known as the Portland Canal dike swarm. Kirkham and Bjornson

(2012) describe this zone as being a gently northeast-dipping zone, which, based on showings, soil anomalies, and drilling, is approximately 1,800 m along strike (north-south) and 600 m to 800 m across strike.

The deposit comprises sheet-like lenses of quartz stockwork and quartz breccias with thicknesses ranging from 40 m to 200 m, dipping gently to moderately to the northeast. The Dilworth deposit is hosted in the Upper Andesite member of the Unuk unit. Underlying upper siltstones, exposed to the west on the Granduc Road, have yet to be encountered in drilling. Quartz-sericite-pyrite alteration is strongly developed particularly in the Yellowstone, Occidental, and Forty Nine areas. The gold and silver values are within quartz veins, quartz stockwork, and quartz breccias containing pyrite, sphalerite, and minor galena with a higher Ag/Au ratio than generally seen in the other areas. The eastern portion of the zone is within and adjacent to a large lobate body of Premier porphyry which also contains altered and mineralized structures and appears to also have a moderate northeast dip. This zone of mineralization is very similar in style to the western part of the Martha Ellen and is likely the strike extension of the Martha Ellen zone.

Mapping of the Dilworth area by Gerry Ray in 2008 revealed several important features, including the mineralized area occupying the western limb of a large northwest-striking F1 synform. He noted hydrothermal brecciation producing the mineralized multiphase quartz breccia bodies, associated with quartz stockwork and pervasive silicification. These are surrounded by areas of pervasive sericite and kaolin alteration and bounded by propylitically altered andesites. Some veining has undergone ductile isoclinal folding related to Cretaceous deformation and Gerry Ray noted several west dipping east verging thrust faults as seen in the Big Missouri area. He also noted a number of east striking late faults often occupied by Eocene Portland dikes, but also containing earlier mineralized quartz veins and quartz stockwork indicating that these were also early structures.

7.1.3 Mineralization—General

Alldrick (1993) interprets the 200 mineral occurrences in the Stewart district as forming during two distinct mineralizing events that were characterized by different base and precious metal suites. One mineralizing episode occurred in Early Jurassic time and the other in the Eocene. Both metallogenic epochs were brief, regional-scale phenomena.

The Early Jurassic mineralization such as the Big Missouri and Premier deposits were deposited in andesitic to dacitic host rocks at the close of volcanic activity, at about 185 Ma (Alldrick 1993). These deposits have regional zoning patterns that are spatially related to plutons of the Texas Creek suite and to their stratigraphic position within the Hazelton Group volcanic-sedimentary sequence. The Early Jurassic hydrothermal system is interpreted to have acquired its characteristic suite of silver, gold, zinc, lead, and copper from magmatic fluids. Early Jurassic deposits include gold-pyrrhotite veins, veins carrying silver, gold, and base metals, as well as stratabound pyritic dacites. Gold-pyrrhotite veins formed adjacent to the subvolcanic plutons during late magma movement. Epithermal base and precious metal veins and breccia veins were formed along shallower faults and shears, and in hydrothermal breccia zones along the contacts of subvolcanic dikes. Stratabound pyritic dacites are barren-fumarole and hotspring-related deposits that formed on the paleosurface from shallow groundwater circulation within hot dacitic pyroclastic sheets.

Panteleyev (1986) and Alldrick (1993) consider Big Missouri to be an epithermal deposit. Recent work by Ascot (Kirkham and Bjornson 2012) describes mineralization as gently discordant to stratigraphy and analogous to the Premier mineralization, which is classified as a low sulfidation epithermal system with some affinities to polymetallic vein systems. The understanding of the Big Missouri system has advanced a great deal with drilling to define the resource. Diagnostic features of the deposit include quartz veins, stockworks, and breccias carrying gold, silver, electrum, argentite, and pyrite, with lesser and variable amounts of sphalerite, chalcopyrite, galena, rare tetrahedrite, and sulfosalt minerals. The mineralization commonly exhibits open-space filling textures and is associated with volcanic-related hydrothermal to geothermal systems in a high-level (epizonal) to near-surface environment.

With new drilling, the series of formerly isolated occurrences were shown to be a large continuous mineralized system offset by a series of east directed thrusts. The western deeper part of the system in the Big Missouri Province area is more base metal (Pb and Zn) rich and crosscuts argillites of the Upper Siltstone Member, and persists through the Upper Andesite Member of the Unuk River unit. The mineralization on the eastern side of the Big Missouri deposit in the DagoUnicorn area displays higher silver contents due to sulfosalts and is associated with low sulfide silicification ±barite and chalcedony migrating into the higher units of the Betty Creek Formation that overly the Unuk River unit. This is very similar to the distribution of mineralization seen at the much more studied Premier deposit, but on a much larger scale. Due to its gently dipping orientation, the outcrop expressions of the Big Missouri deposit cover an area of greater than 3.0 km^2.

Brown (1987) described the mineralization at Premier as occurring in four broad styles: both a low- and high-sulfide type, with stockwork and breccia variants of each. Each style is described as an end member of a continuum between various types of mineralization. High-sulfide mineralization is defined as containing 15% or more sulfides. These mineralization styles are summarized in Table 7-1.

In a 1990 PhD thesis, McDonald categorized the Premier mineralization by relative age, as defined by crosscutting relationships between mineralized features. Veins and breccias were grouped as early, middle, and late stages, with the middle stage further divided into precious- and base metal-rich subgroups.

Early stage breccias consist of rounded to angular fragments of andesite in a dark green aphanitic pyrite matrix. This matrix is composed of intergrown pyrite, chlorite, sericite, quartz, and calcite, with local diffuse patches of chalcedony and potassium feldspar. Earlier workers defined this style of occurrence as “in situ” or “crackle” breccias. Clast abundance ranges from less than 25% to 90%. Where the fragment proportion is lower, the clasts are more rounded to irregular, poorly defined and patchy in distribution. Breccias with a higher proportion of fragments are more angular and display a lower degree of rotation.

These breccias are cut by the early stage veins, which are in turn cut by the middle-stage stockwork veins. The early stage veins comprise banded quartz-chlorite with pyrite on the margins, and occur as steeply dipping, northwest-striking en echelon clusters coincident with foliation. Vein thickness ranges from 0.5 cm to 7.0 cm but is more commonly 1.0 cm to 3.0 cm. Pyrite content varies up to 10% of the veins, and chlorite ranges from 15% to 20% at the 250 m elevation (6 level) to 5% at the 570 m elevation (2 level).

Table 7-1: Premier Vein Styles

Type of Mineralization		Mineralogy	Textures	Host Lithology	Notes
Low Sulfide	Stockwork	py, sph, gln	Quartz veins	Porphyry	Variable alteration
	Breccia	Ag-sulfosalts, native Ag	Siliceous breccia, late fractures filled with native Ag	Altered porphyry	Bonanza ore; silicification, K-feldspar alteration
		Disseminated py, sph, gln	Siliceous breccia	Porphyry and andesite	Altered porphyry and andesite clasts
High Sulfide	Stockwork	py, sph, gln	Veinlets	Porphyry	Grades into siliceous breccia
	Breccia	py	Pyrite veinlets and stockwork	Andesite	High grade Au, low Ag
		py, sph, gln, ± cpy	Breccia	Andesite	Galena rimming andesite fragments, disseminated pyrite, interstitial sphalerite
		Sph, gln, py, ± tet	Breccia, vuggy	Altered porphyry	Silicified angular clasts, some with quartz rims
		py	Podiform to layered	Andesite/porphyry contact	Deformational layering

Source: Brown, 1987
Notes: py = pyrite, sph = sphalerite, gln = galena, cpy = chalcopyrite, tet = tetrahedrite.

Middle stage stockwork veins and breccias tend to have a higher metal content, and encompass precious and base metal-rich variants. Veins are 0.5 cm to 5 cm in thickness, occurring as irregular networks to planar sheets, at times forming breccias in dilatant zones, and encompassing wall rock fragments. These structures crosscut early stage breccias and quartz-chlorite-pyrite veins and are themselves crosscut by late stage quartz-chlorite-calcite and quartz-ferrocalcite veins. Fragments of early stage veins and breccias are contained in middle stage breccias. Most commonly, precious metal-rich veins predate and are cut by base metal-rich veins.

Among the precious metal-rich middle stage veins and breccias, McDonald (1990) identified five sub-classes (Types 1 to 5). Listed in order of earliest to latest, these are:

1. Quartz + potassium feldspar + calcite ± pyrite
2. Quartz + potassium feldspar + albite with precious metal minerals
3. Precious metal-rich breccias
4. Ferrocalcite + quartz
5. Calcite + quartz.

Veins of Type 1 listed above are poorly defined and discontinuous in the core of the breccia bodies, becoming more planar and distinct within 2m to 3m into the margins. They are 0.5cm to 2cm wide, and consist of fine-grained intergrowths of quartz, potassium feldspar, albite, and calcite with irregular concentrations of fine-grained pyrite and chlorite intergrowths.

The Type 2 veins are planar to slightly warped, measure 0.5 cm to 3 cm wide, and dip steeply, oriented subparallel to the precious metal-rich veins (Type 3). Vein minerals comprise quartz and potassium feldspar with local patches of albite, barite, rhodochrosite, and anhydrite. Sulfide content is typically below 5% and consist of pyrite, sphalerite, chalcopyrite, and galena, with isolated grains or aggregates of polybasite, argentiferous tetrahedrite, freibergite, native silver, electrum, pyrargyrite, and argentite.

Precious metal-rich breccias form in andesite and porphyry bodies in sharply defined or fault-bounded dilatant zones, flanked by more planar veins. Fragments on the breccia margins are typically angular to slightly rounded clasts of wall rock or earlier veins and breccias, becoming more rounded, siliceous, and less clearly defined towards the interior. The breccia matrix is predominantly quartz with, again, less than 5% sulfide minerals. Economic minerals include isolated aggregates of sphalerite, galena, polybasite, pyrargyrite, acanthite, tetrahedrite, freibergite, native silver, gold, and electrum with accessory pyrite. The predominant gangue mineral is quartz (sometimes as chalcedony); the intensity of silicification and proportion of matrix in the total rock mass diminishes with distance outwards from the core of the breccia bodies.

The ferrocalcite-quartz veins (Type 4) are light brown in colour, sharply defined, measuring 2 cm to 8 cm wide, and are observed to crosscut the earlier precious metal-rich veins. Pyrite is rarely present and occurs along the vein margins.

The latest phase of the precious metal-rich middle stage veins and breccias are calcite-quartz breccia bodies (Type 5). These are narrow (5 cm to 20 cm), bodies comprising fragments of andesite and earlier middle-stage breccia in a matrix that can contain fine-grained pyrite, sphalerite, and galena.

McDonald (1990) also identified five subtypes of the base metal-rich veins and breccias (Subtypes 1 to 5). From oldest to youngest, these are:

1. Quartz + calcite ± chlorite ±, and pyrite ± potassium feldspar
2. Pyrite + quartz + galena ± calcite ± galena
3. Quartz + barite + albite + calcite + base and precious metals
4. Base metal-rich breccia
5. Pyrite + precious metals.

The veins of Subtype 1 are steeply dipping, irregularly branching veins averaging 3 cm in thickness, and offsetting earlier stage structures. They display a crude banding of minerals, consisting of a core of intergrown quartz and potassium feldspar with varying amounts of pyrite and chlorite along the margins.

The Subtype 2 veins are also steeply dipping, but planar and erratically distributed, varying in thickness from 1 cm to 3 cm. Vein minerals are 40% to 60% pyrite, with 10% to 20% quartz, and the remainder calcite, potassium feldspar, albite, and minor galena.

Quartz-barite-albite-calcite-sulfide veins (Subtype 3) are planar to branching steeply oriented networks varying in width from 1cm to 3cm and occurring up to 2 m from the margins of breccia bodies. They have been observed, through crosscutting relationships, to both pre- and post-date middle-stage precious metal-bearing veins. Vein mineralogy consists of quartz, calcite, and minor barite, with 20% to 45% combined pyrite, sphalerite, chalcopyrite, and galena. Relatively minor components include pyrrhotite, argentiferous tetrahedrite, native silver, electrum, and arsenopyrite.

The base metal-rich breccias (Subtype 4) consist of a core of sulfide-cemented clasts flanked by parallel vein networks, or alternatively, combinations of planar and branching veins intermingled with wall rock clasts. The breccia matrix is very similar in composition to the Subtype 3 veins described above with sulfide minerals occurring as irregular aggregates and planar bands.

Breccia clasts are typically altered host rock fragments, rounded in the central portions and becoming more angular and interlocking towards the margins. Relict textures are visible in some fragments, although the original minerals have been replaced by alteration products. Where quartz-sericite alteration is dominant, the clasts become light-coloured and indistinct. Many fragments have been fractured and filled with calcite and coarse-grained pyrite with minor sphalerite and galena. Fragments often contain veinlets which transect or terminate at the rims of the clasts, and some have rinds of quartz, chlorite, and pyrite. Contacts of the breccia bodies are normally faulted, and as such are quite abrupt.

The last phase of the middle-stage veins comprises very small en echelon arrays of veinlets measuring up to 6 cm long and 2mm thick. These veinlets are predominantly composed of quartz and pyrite, with significant amounts of galena, sphalerite, native silver, polybasite, and electrum.

The late stage veins are generally barren and are observed to crosscut the economic mineralization. McDonald (1990) recognized three sub-types, listed below in order of age:

1. Quartz – calcite – sericite
2. Quartz – chlorite – calcite
3. Quartz – ferrocalcite.

Early stage breccias are observed to be most abundant in the upper portion of the mine, above approximately the 350 m elevation (4 Level), and especially above 2 Level (570 m elevation). Most of the early stage veins occur at or below 4 Level and are best developed at the 250 m elevation (6 Level).

Middle-stage veins and breccias comprised the bulk of the ore bodies in the mine and are generally well developed throughout. They are observed to be comparatively more precious metal-rich in the upper and the north-easterly striking (Main Zone) portions of the deposit. In the northwesterly striking western portion (West Zone) of the mine and the lower parts, base metal-rich veins and breccias predominate.

McDonald (1990) applied these observations along with analytical work to define broad zonations in both silicate and metallic minerals. The proportions of quartz, calcite, and orthoclase were observed to be consistent throughout the mine. In the Main Zone of the deposit, chlorite and albite are more abundant below approximately 350 m in elevation (4 Level). Barite and sericite appear to be more abundant from 4 Level up to 50 m above 2 Level (570 m elevation). In the West Zone, chlorite is more abundant below approximately 440 m elevation (3 Level), with sericite, albite, and barite more abundant above 3 Level.

Base metal minerals are most abundant between 4 and 5 Levels (300 m to 350 m elevation), diminishing rapidly from 5 Level to surface, and less so downwards to 6 Level. Precious metal minerals were observed to increase in proportion above 4 Level, with a significant increase above 2 Level. Relative proportions of precious metal minerals decline from 4 Level to 6 Level. Precious metal abundances are historically higher at the intersection of the West and Main zones, and slightly higher in the Main Zone than the West Zone. Silver-to- gold ratios and overall silver contents are observed to diminish with depth from a high of 150:1 near surface to a low of 5:1 below 3 Level.

Mineralization—Premier

The Premier/Northern Lights zones form roughly parallel curvilinear planes with a strike that varies from northeast at their eastern edge to northwest at the western edge as illustrated in Figure 7-3 which is a 3-D view of the modelled Premier area Quartz Breccia Zones and Tertiary dykes.

The wireframes used to construct the block model for interpolation at the Premier area follow the mineralizing structures as shown in Figure 7-4, with each sub-area of Premier and Northern Lights areas denoted by its name. The southern structure consists of Premier Main, Obscene, Lunchroom, West, 609, and 602; the Northern Lights zones are Prew, Ben, and Northern Lights Main.

The dip of these zones is sub-vertical near surface, flattening at depth to a dip of 20° to 40°. The zones are defined by breccias and stockwork formation in a host of mainly andesitic volcanic rocks and, less frequently, Premier porphyry. These breccia bodies and stockwork zones are the expression of two mineralized fault planes that converge towards the northeast, as illustrated in Figure 7-4. The projection of the intersecting faults converges with the Long Lake strike-slip fault and it appears likely that these faults are step-over structures between the regional Long Lake Fault and the Cascade Creek Fault to the west. These step-over faults are thought to be part of an inverse flower structure in response to a local jog in the regional strike-slip fault system. Ascot is of the opinion that future exploration to the north and the south could establish the presence of additional faults and confirm the geometry of a negative flower structure.

Contained within this broader structural and mineralogical envelope are high-grade zones which have supported underground mining throughout the history of the mine. The modelled zones within the envelope (Figure 7-4) form curviplanar tabular bodies with a thickness ranging from 2 m to greater than 10 m. Grades within these zones average greater than 3 g/t AuEq and locally can reach grades of one or two orders of magnitude higher. The zone orientations are typically slightly oblique to the dip of the main envelope and may represent tension gashes within the main fault plane. Mineralization formed due to intensified temperature and pressure gradients developed within the dilatant zones, which facilitated precipitation of metals from hydrothermal fluids.

Figure 7-6 through Figure 7-11 are cross sections through the different parts of the Premier deposit, illustrating the general geometries described above, with Figure 7-5 showing the location of each section for the Premier area. Figure 7-6 is a cross section through the 602 and 609 zones which shows the interpreted mineralized bodies within the broader corridor of alteration, quartz breccia, and stockwork. Figure 7-6 is a cross section through the Premier Main and Obscene zones, near the heart of historical mining activity. The geometry of the interpreted zones is seen to be similar to the old stope outlines. The cross section in Figure 7-7 shows the relationship between the Ben and Prew zones, demonstrating that they are essentially continuous with one another. Figure 7-6 and Figure 7-7 also illustrate the anastomosing nature of the individual structures hosting the mineralized bodies.

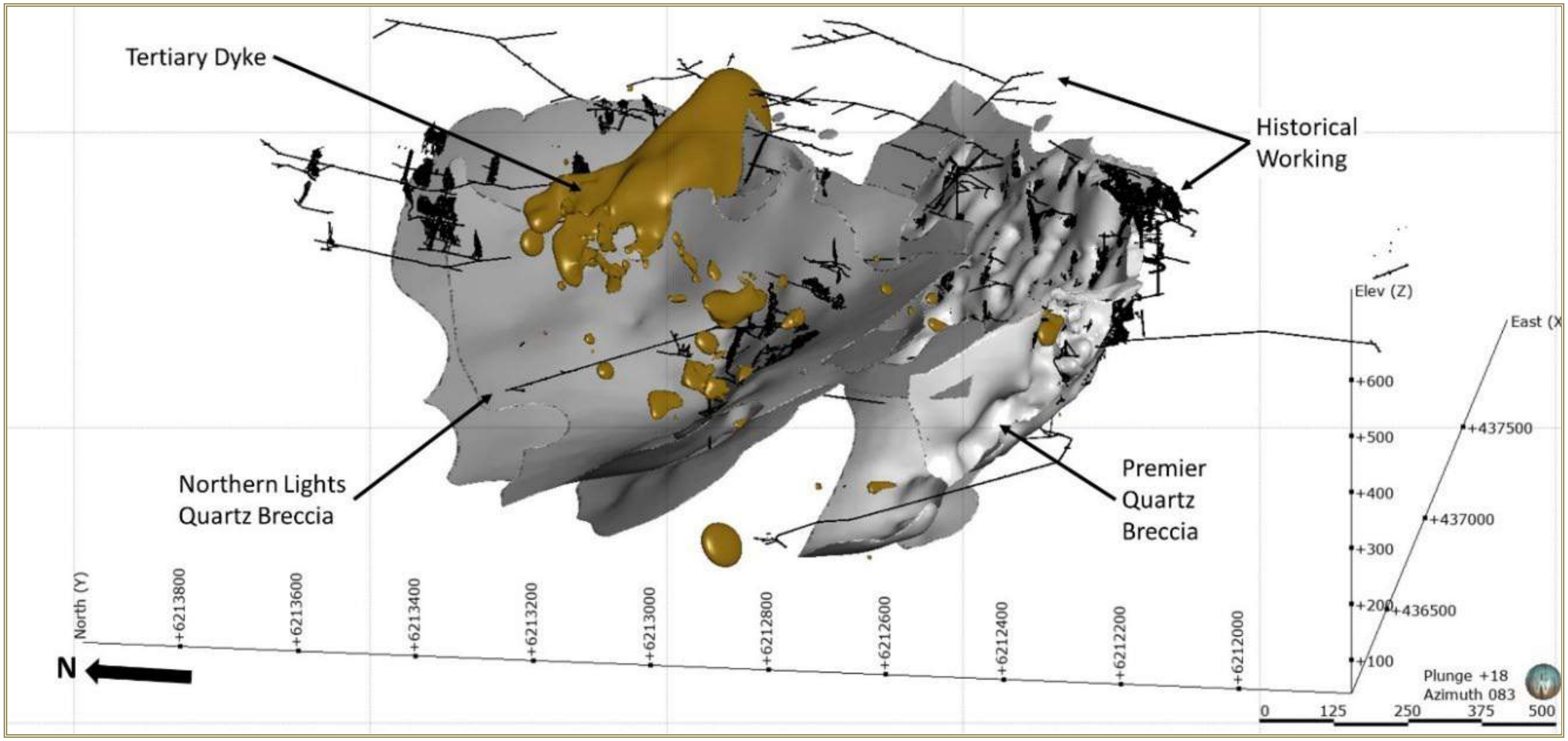


Figure 7-3: 3-D View of Geology and Structure Controlling Mineralization—Premier

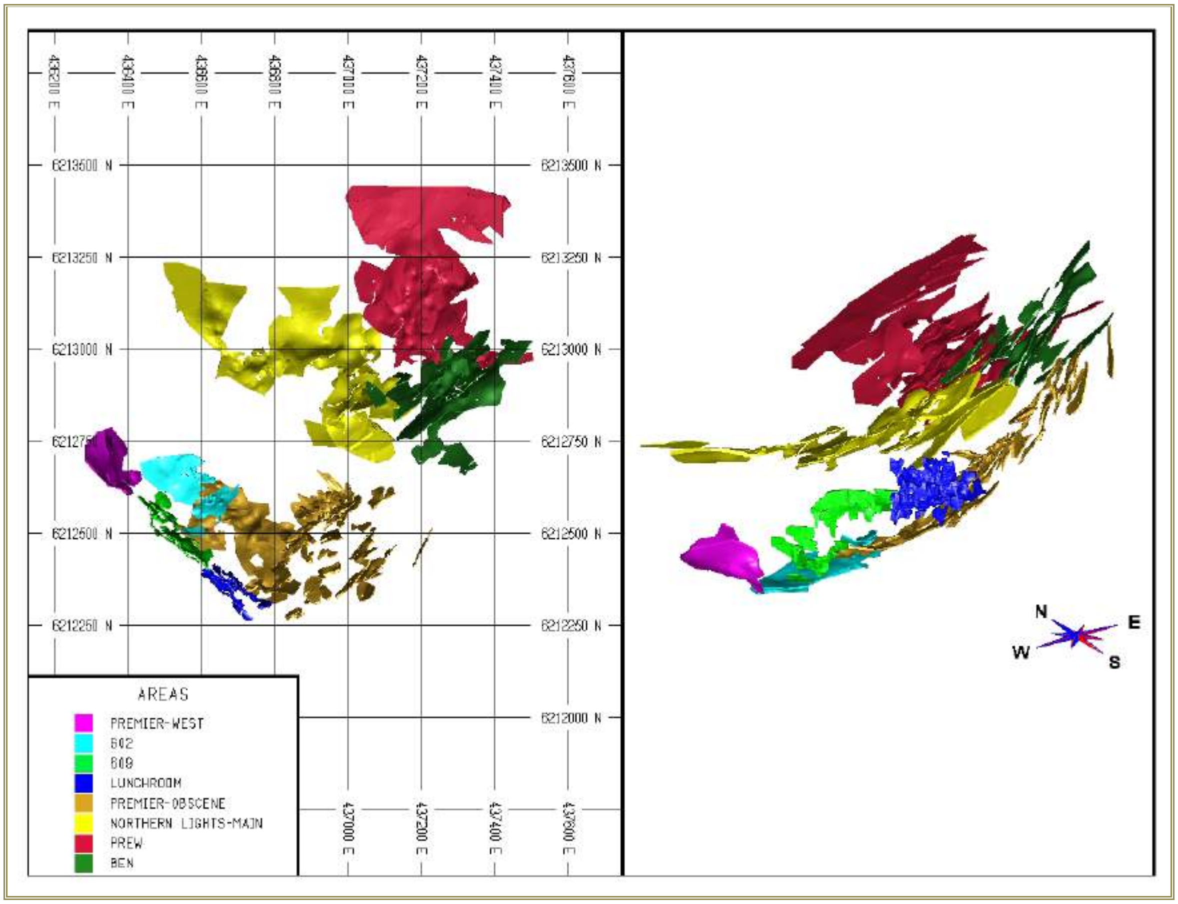


Figure 7-4: Plan and 3-D View of Potentially Mineralized Wireframes—Premier

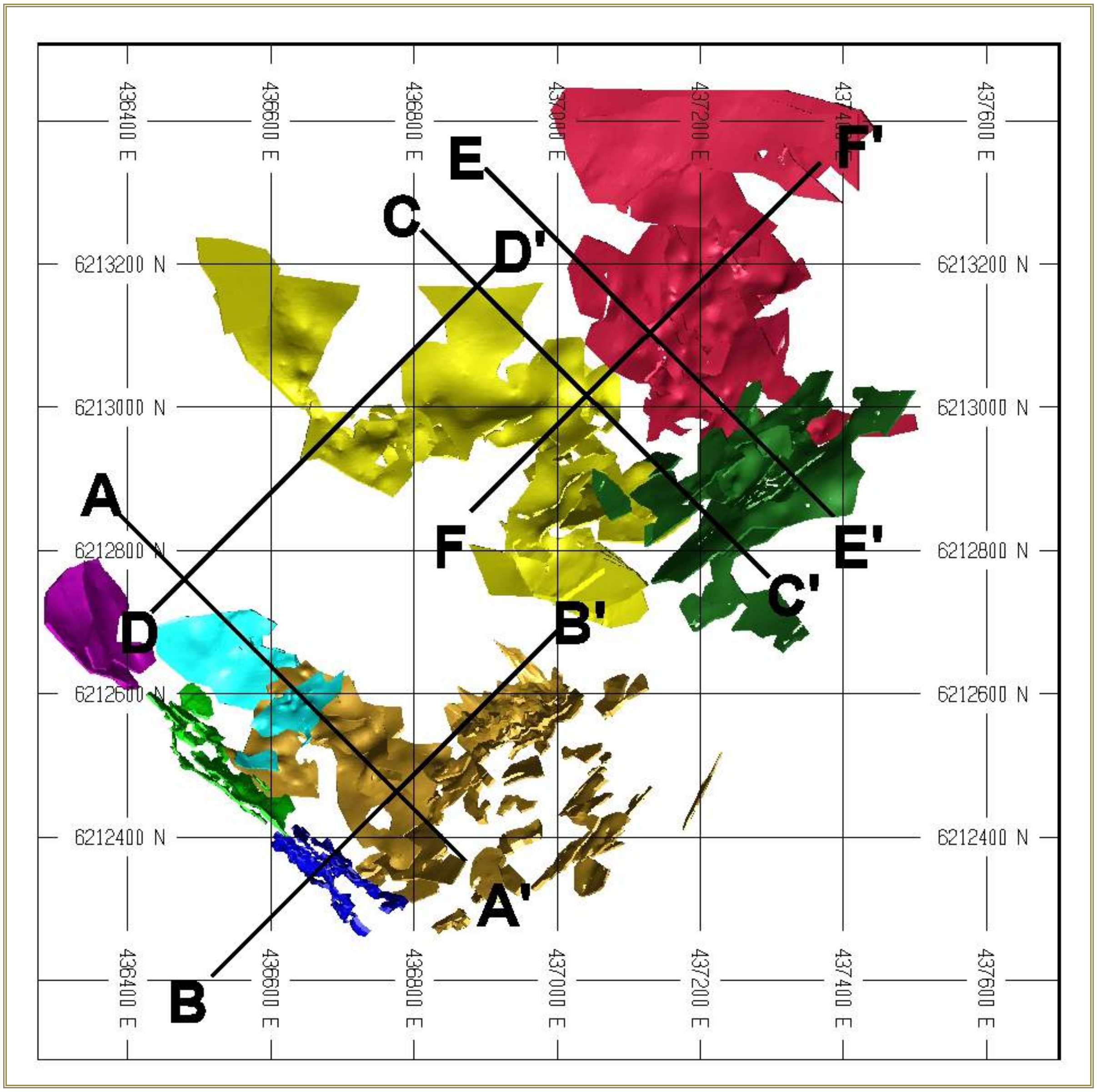


Figure 7-5: Premier—Plan Map of Section Locations for Sections A-A' through F-F'

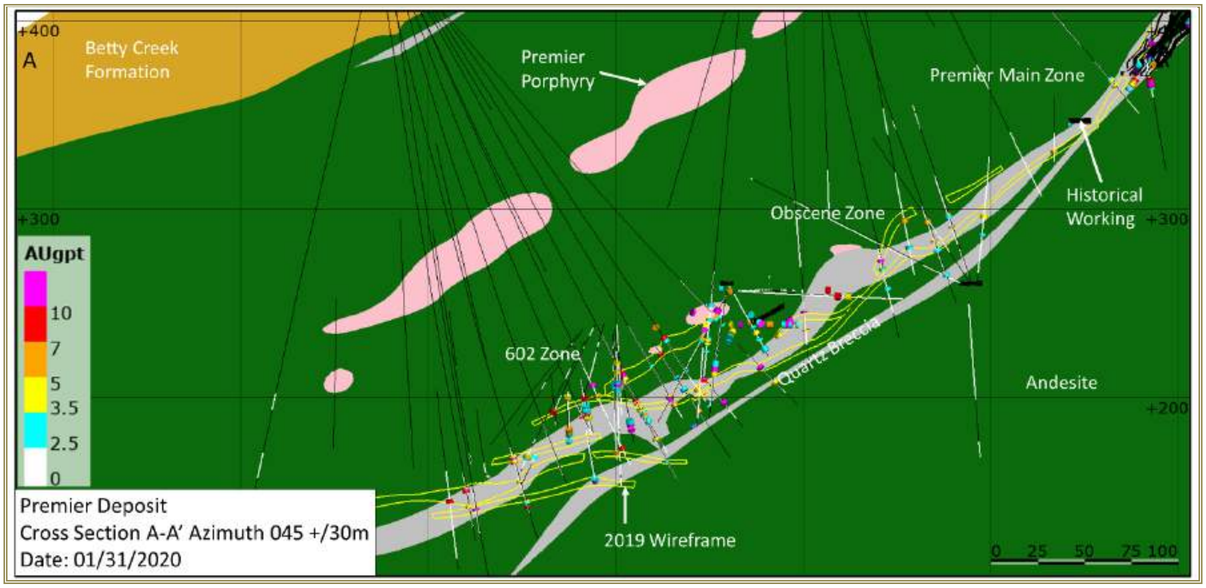


Figure 7-6: Premier—Section A-A' – 602, Obscene and Main Zones

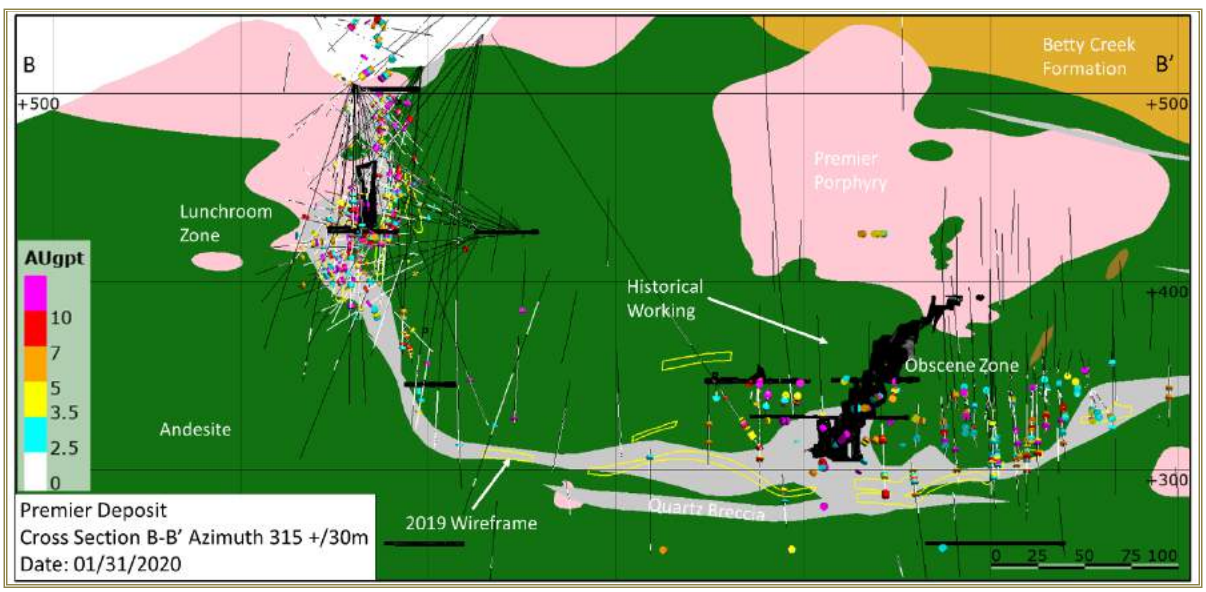


Figure 7-7: Premier—Section B-B' – Lunchroom and Main Zones

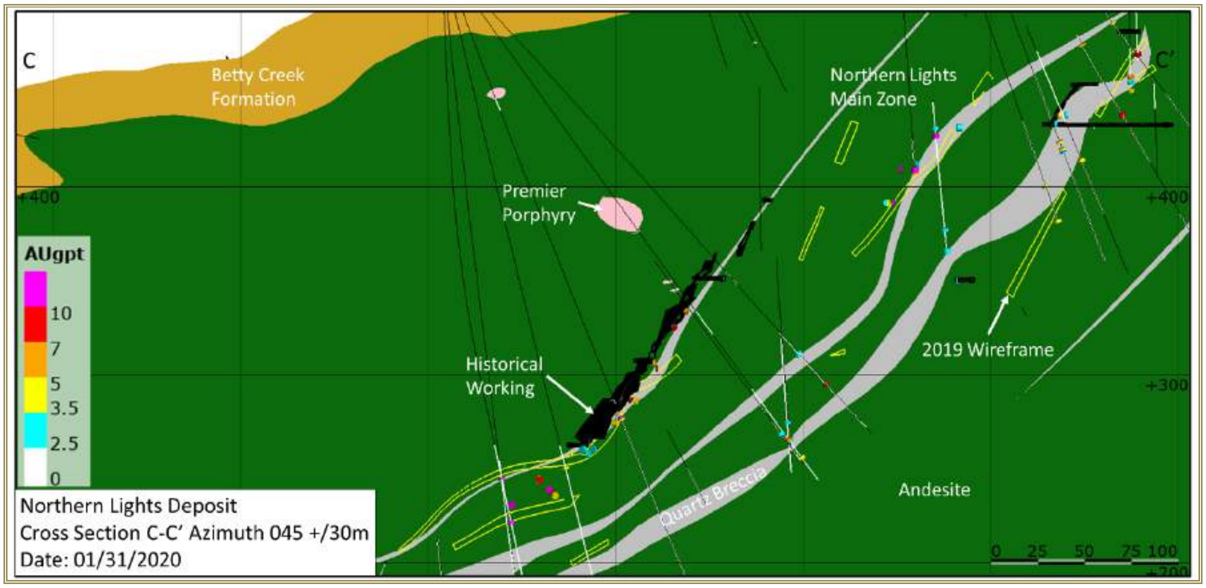


Figure 7-8: Northern Lights—Section C-C' – Northern Lights and Ben Zones

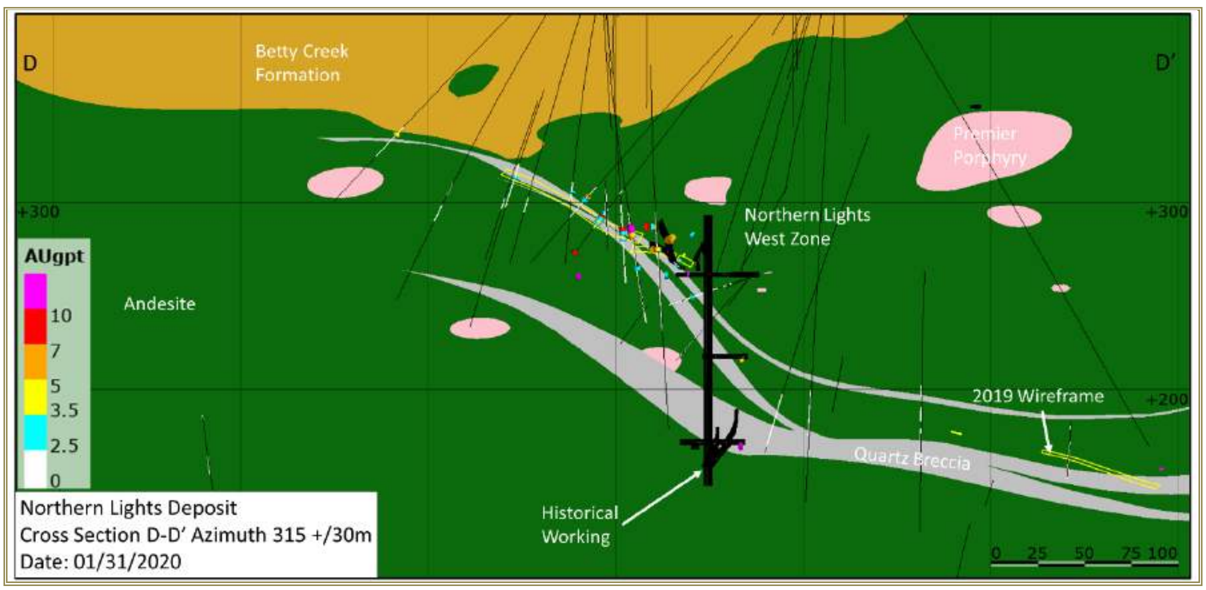


Figure 7-9: Northern Lights—Section D-D'—Northern Lights West Zone

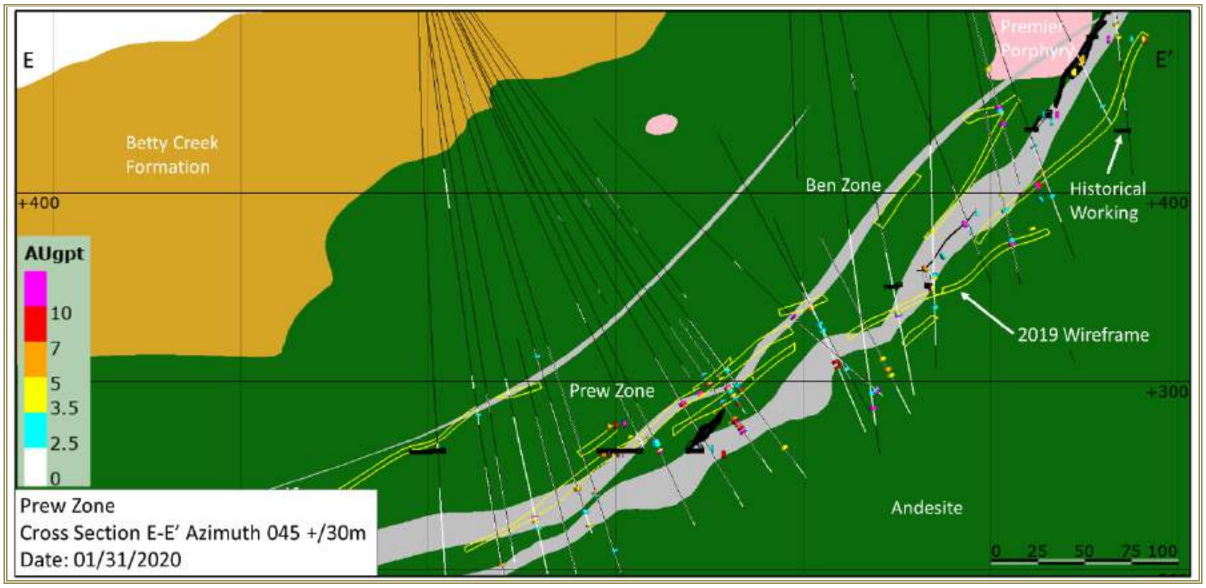


Figure 7-10: Northern Lights—Section E-E'—Prew and Ben Zones

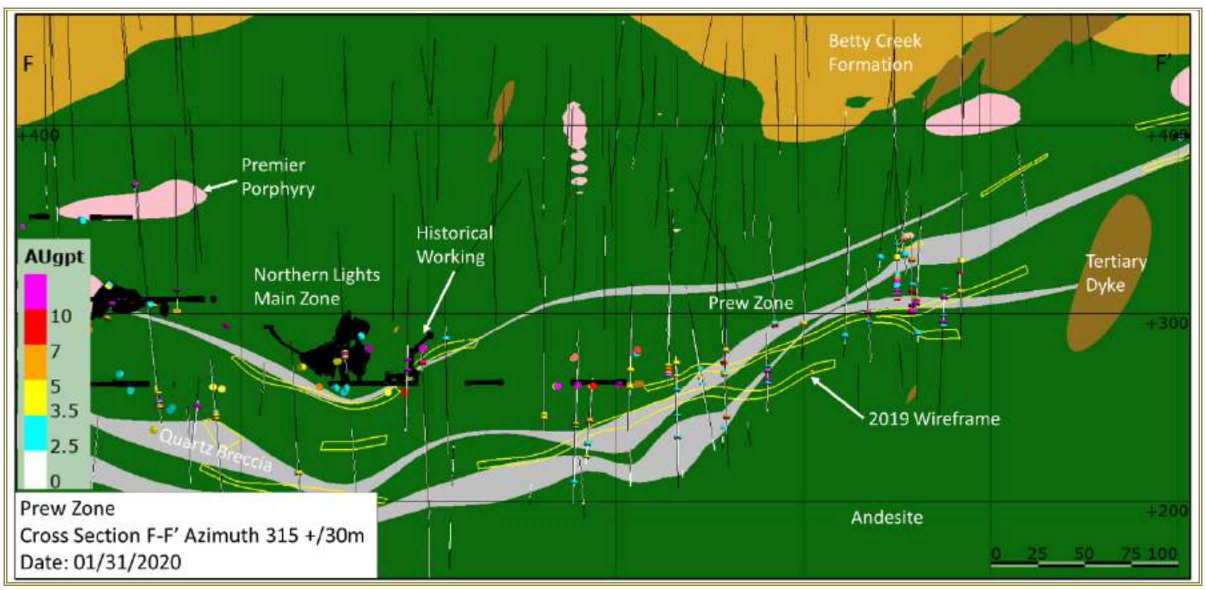


Figure 7-11: Northern Lights—Section F-F'—Northern Lights Main and Prew Zones

Mineralization—Big Missouri

Mineralization at Big Missouri is structurally controlled. It consists generally of moderately dipping quartz breccia structures crosscut by Tertiary dykes and three major faults—the Union Creek, Jain, and Cascade Creek faults—as illustrated in Figure 7-12. Potentially mineralized wireframes have been created to follow the geology and constrain mineralization to above about 1g/t AuEq where continuous structure can be defined. The wireframes used to construct the block model for interpolation at the Big Missouri area follow the mineralizing structures as shown in the plan and 3-D view of Figure 7-13.

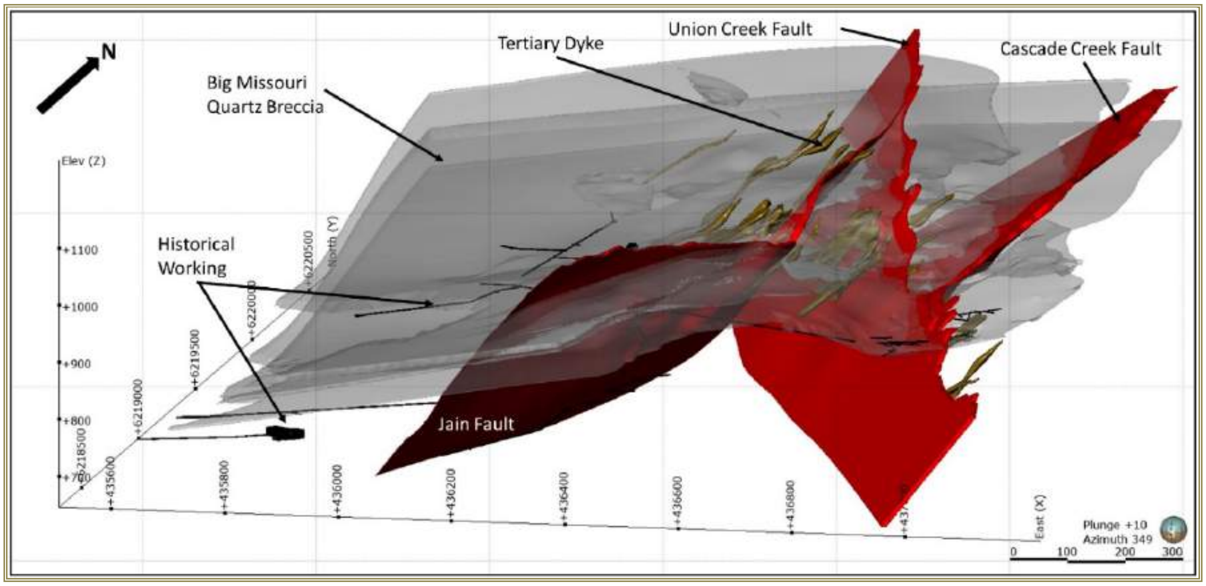


Figure 7-12: 3-D View of Geology and Structure Controlling Mineralization—Big Missouri

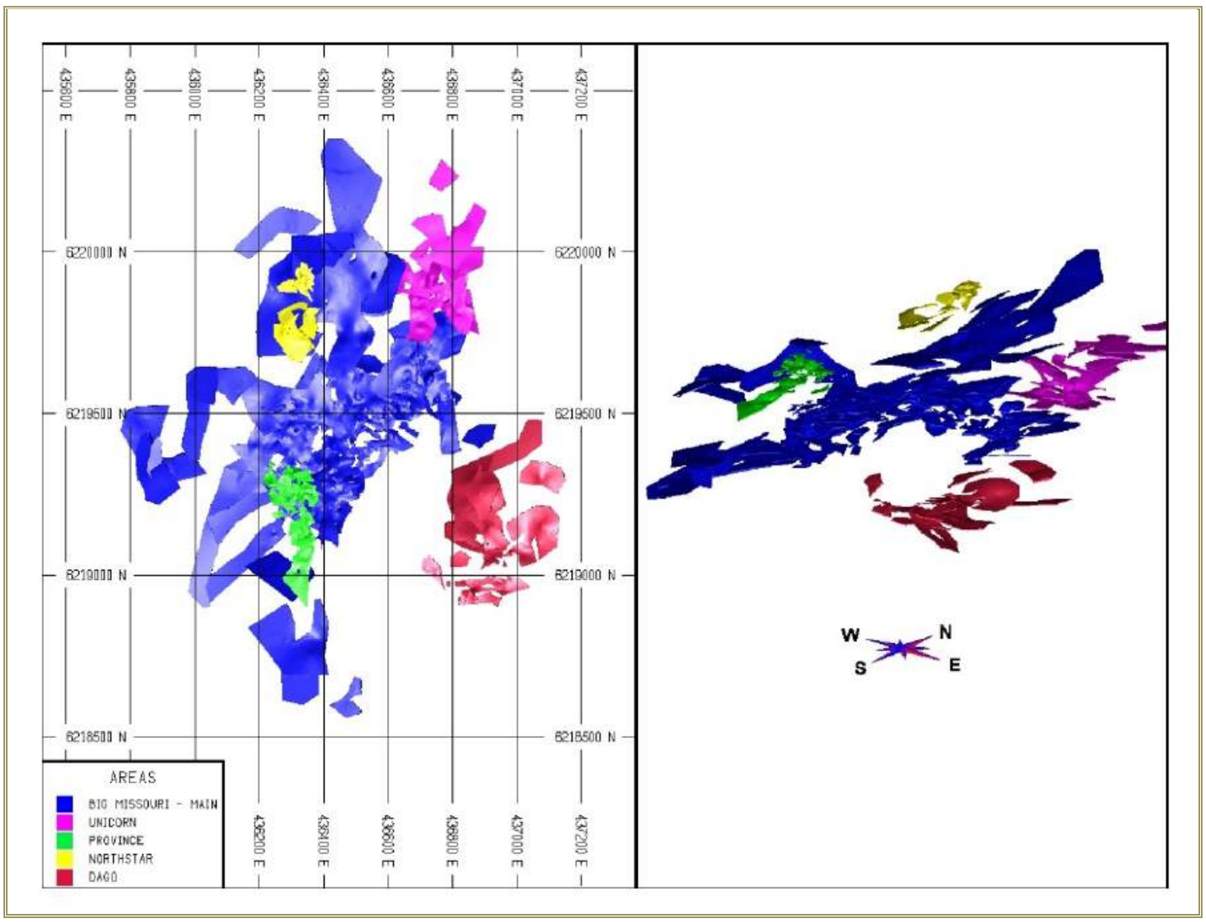


Figure 7-13: Plan and 3-D View of Potentially Mineralized Wireframes—Big Missouri

Figure 7-14 and Figure 7-15 are cross sections through the different parts of the Big Missouri deposit, illustrating the different zones of the deposit including the Big Missouri Main zone, the near-surface Province zone (Figure 7-14), the Unicorn zone (Figure 7-15), and in long section of Figure 7-16 the Northstar zone. Drill holes with gold grade, quartz breccia, faults and dykes are shown on the sections, as well as the wireframes used in interpolation.

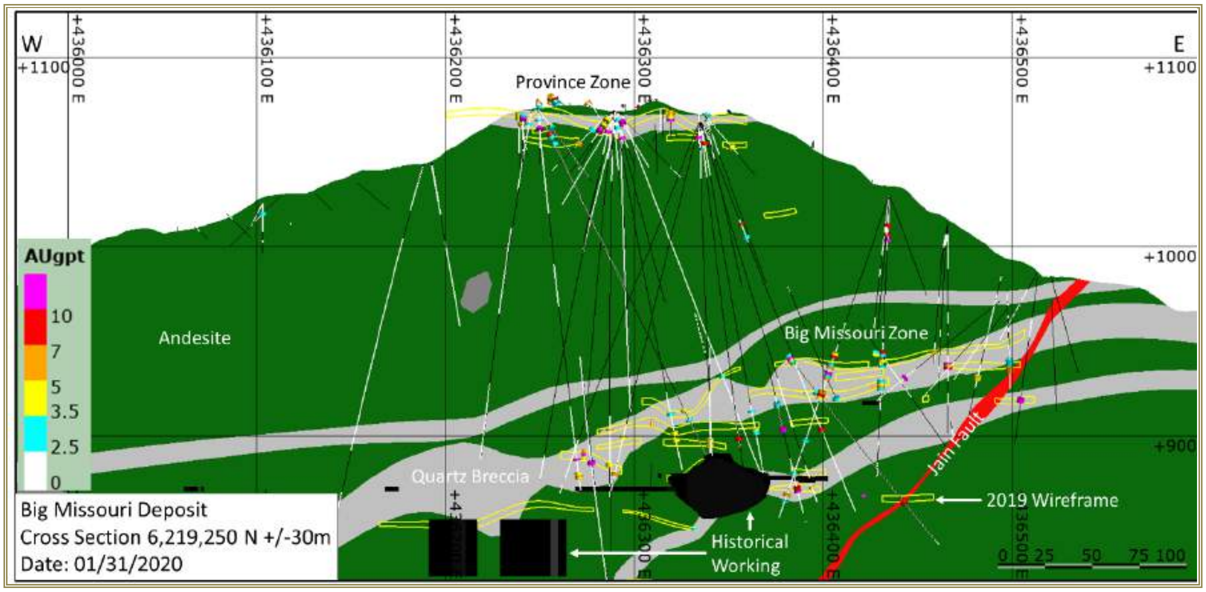


Figure 7-14: Big Missouri—Cross-Section—6,219,250N

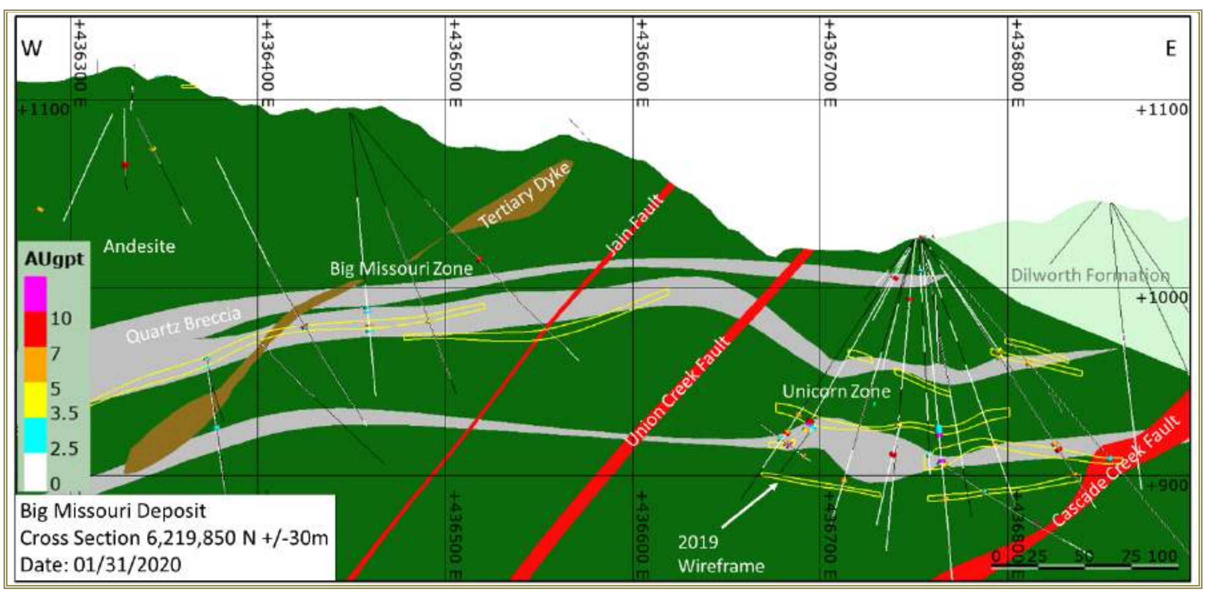


Figure 7-15: Big Missouri—Cross-Section—6,219,850N

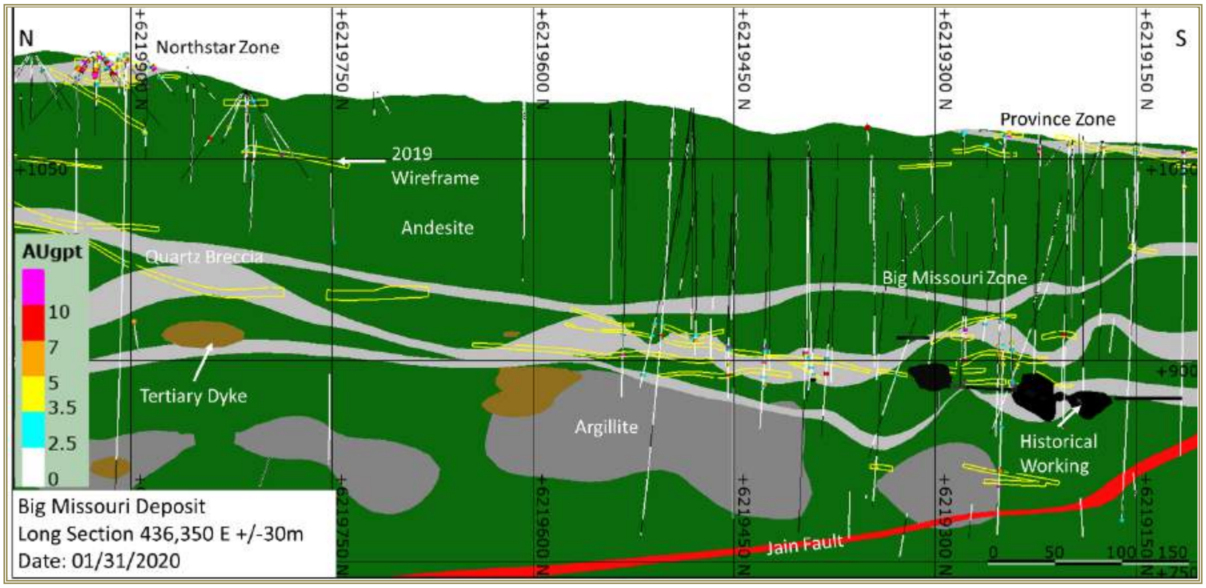


Figure 7-16: Big Missouri—Long Section—436,350E

Silver Coin

Mineralization at Silver Coin is generally confined to the west of the Anomaly Creek Fault and proximal to the quartz breccias as illustrated in Figure 7-17.

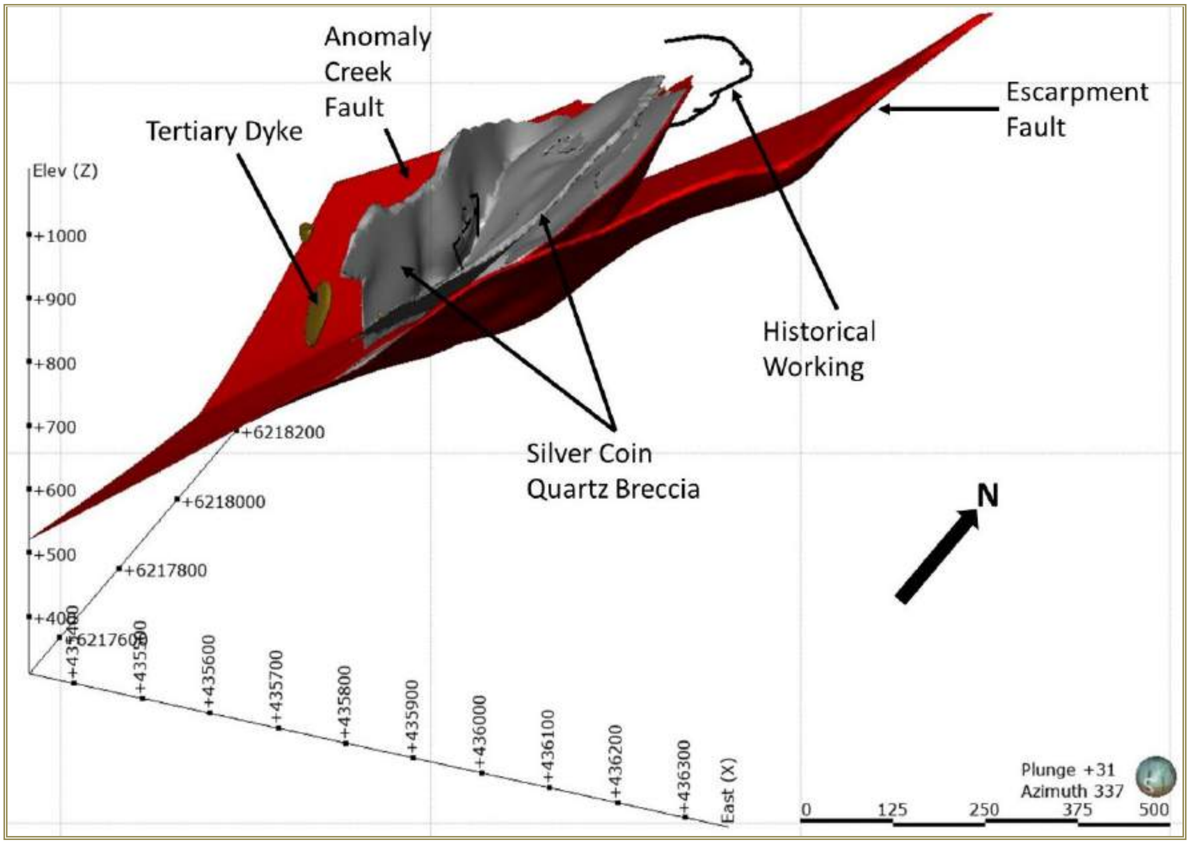


Figure 7-17: **3-D View of Geology and Structure Controlling Mineralization—Silver Coin**

Similar to elsewhere at PGP, mineralization is considered to have formed due to intensified temperature and pressure gradients developed within dilatant zones facilitating the precipitation of metals from hydrothermal fluids. Figure 7-18 and the cross-section of Figure 7-19 illustrate the wireframes used for interpolations, showing the relationship of the wireframes with the quartz breccia bodies.

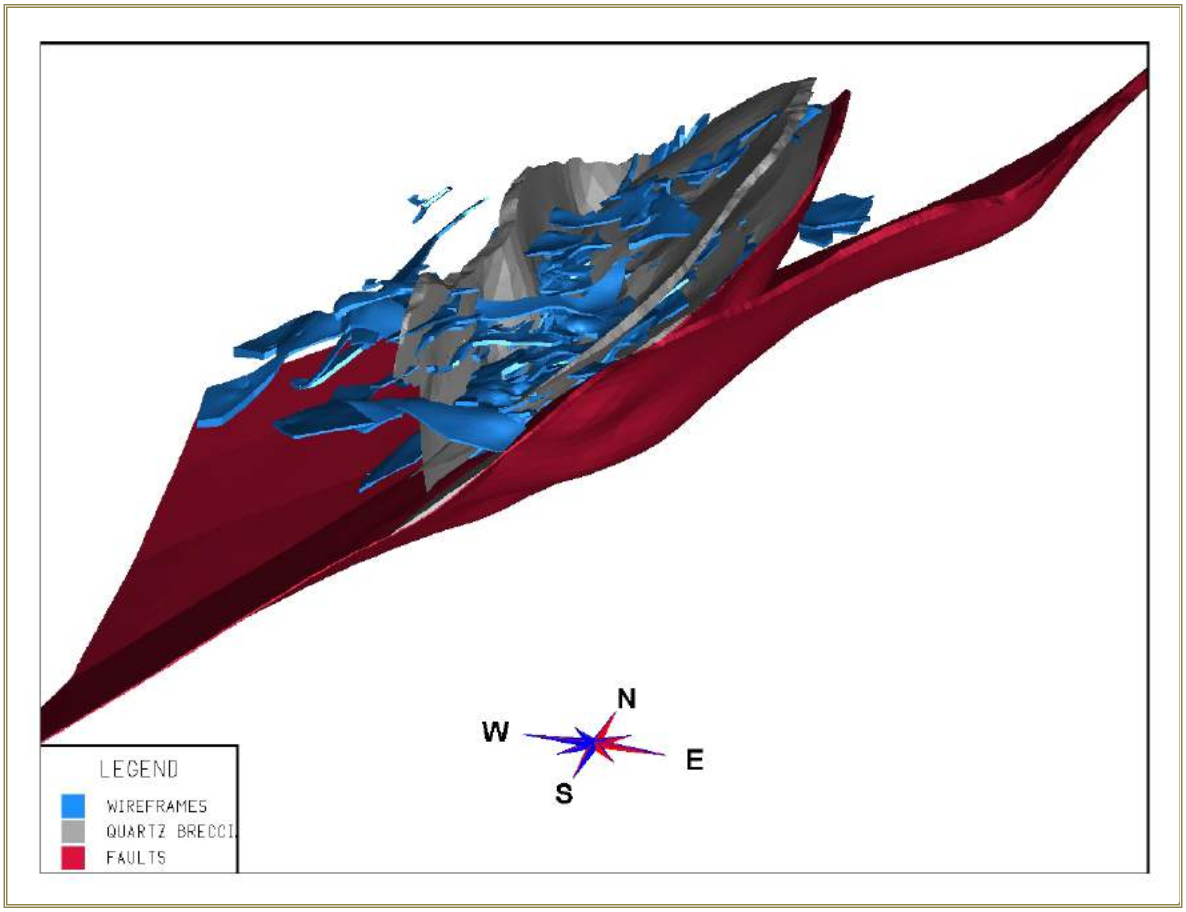


Figure 7-18: Plan and 3-D View of Potentially Mineralized Wireframes—Silver Coin

The vast majority of mineralization is in lenses between the major breccia bodies which form a "V" structure. The lenses are generally parallel to the breccia, or forming haloes more shallowly west-dipping to the main zones due to tension cracks allowing fluid flow. Shallowly dipping higher grade zones peripheral to the breccia are also, less commonly, observed.

The long section of Figure 7-20 illustrates the continuous nature of the quartz breccia in a north-south direction and the Silver Coin mineralization's association with both the breccias and the faulting.

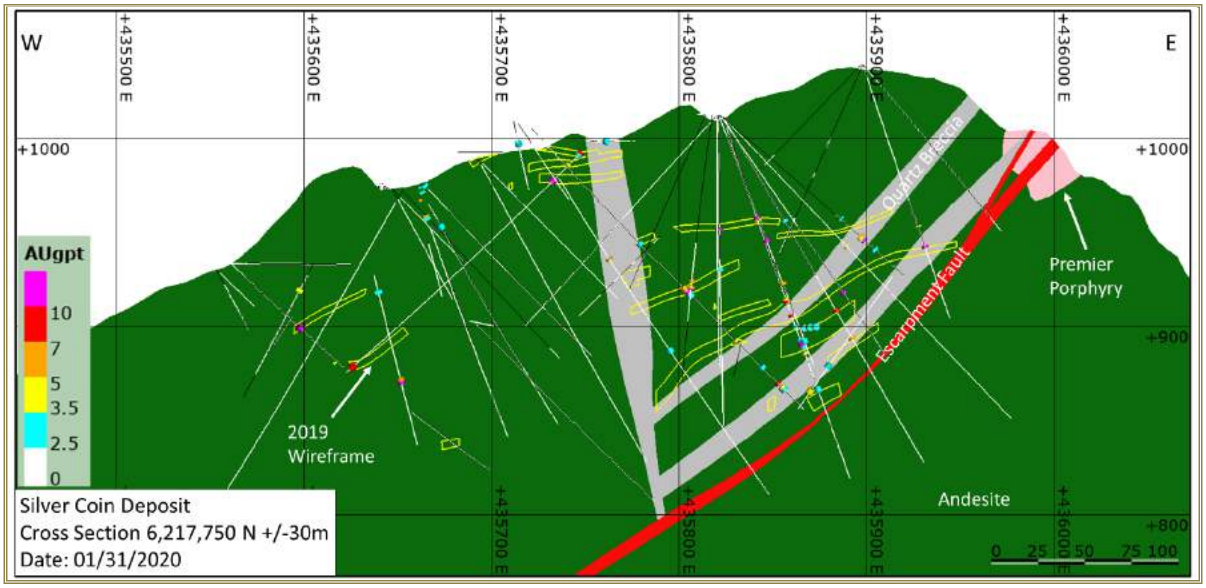


Figure 7-19: Silver Coin Cross Section—6,217,750N

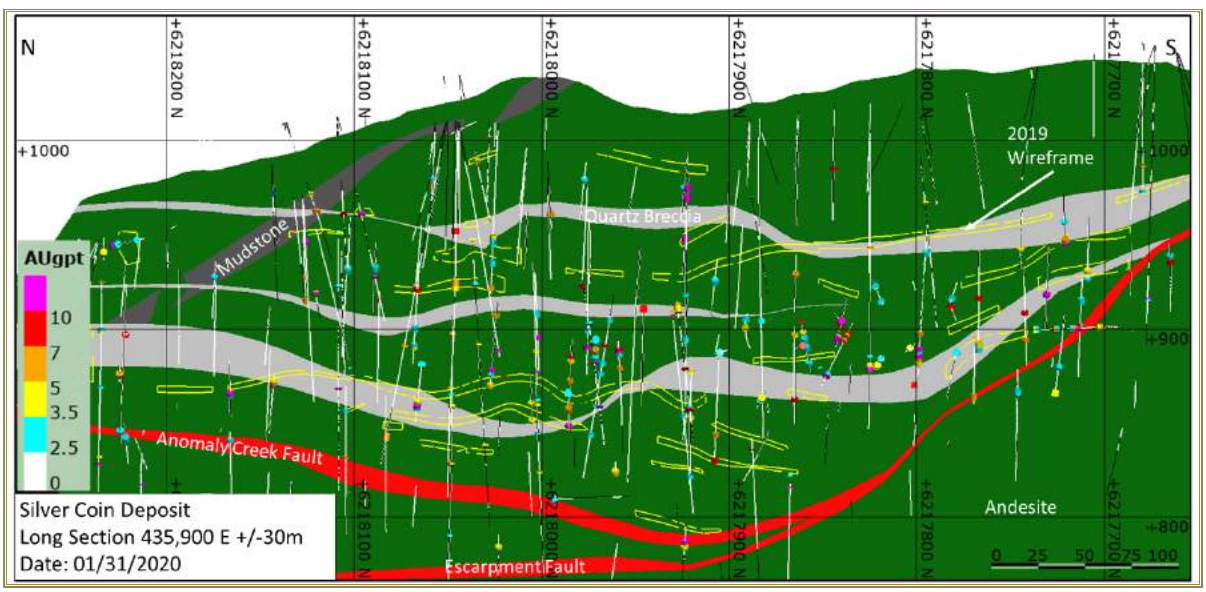


Figure 7-20: Silver Coin—Long Section—435,900E

Mineralization—Martha Ellen and Dilworth

Mineralization at Martha Ellen and Dilworth are likely northern extensions of the Big Missouri deposit. Martha Ellen is fairly flat-lying (similar to Big Missouri), and Dilworth, further north, is shallowly to moderately east dipping. Both of these deposits are crosscut by post-mineral porphyry dykes, as shown Figure 7-21 to Figure 7-23.

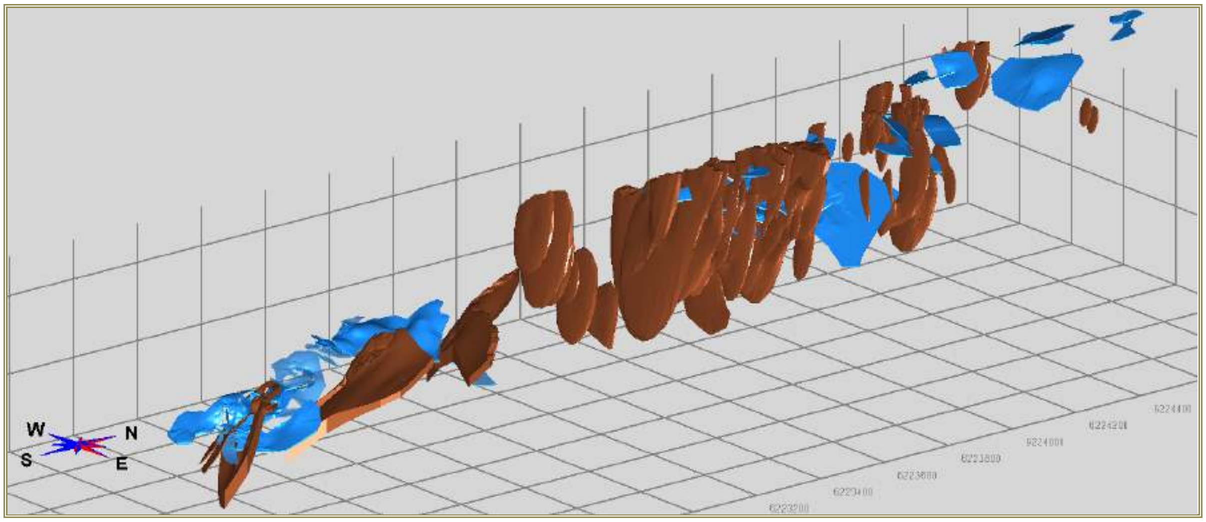


Figure 7-21: **3-D View of Structure Controlling Mineralization (Blue) and Porphyry Dykes (Brown)—Martha Ellen and Dilworth**

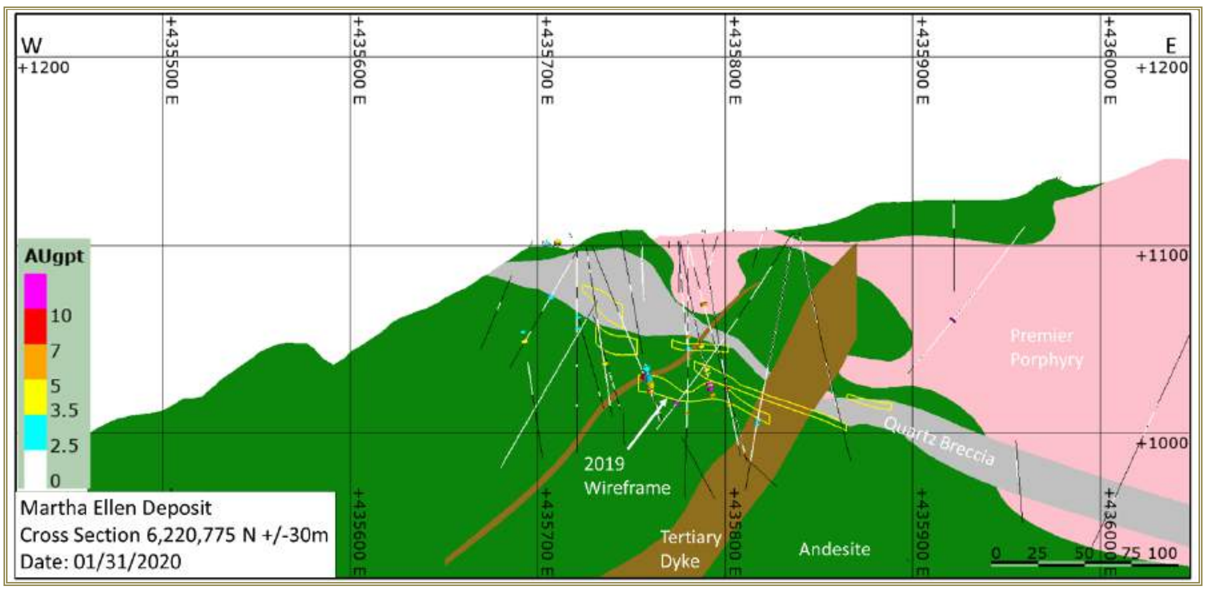


Figure 7-22: Martha Ellen Cross Section—6,220,775N

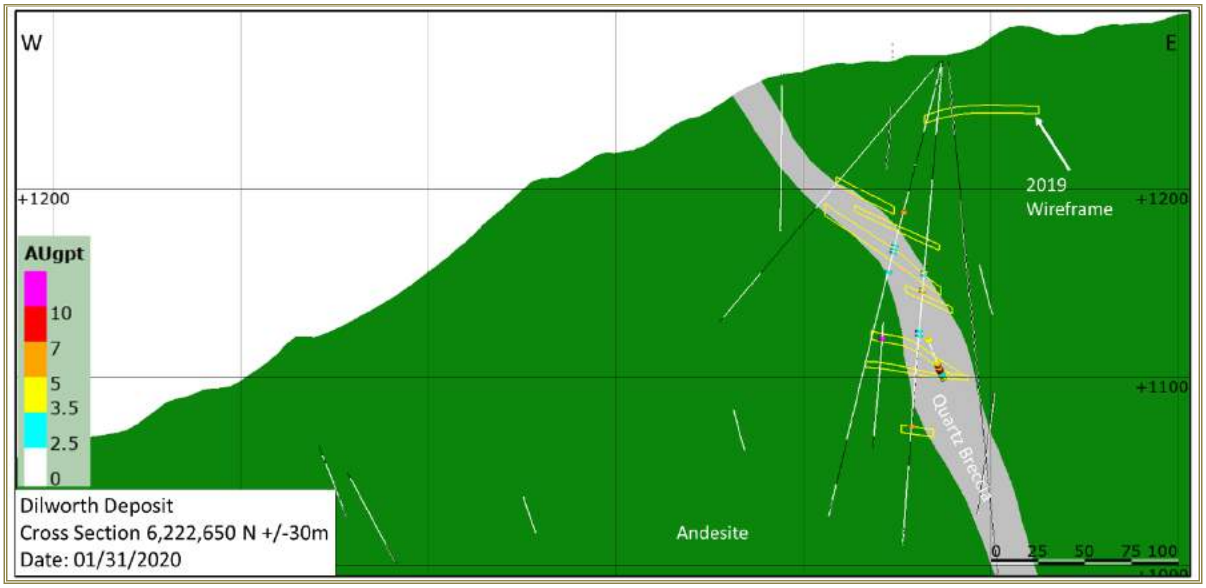


Figure 7-23: Dilworth Cross Section—6,222,650N

7.2 Red Mountain

This section discusses the geology of the Red Mountain area. It includes the regional geology, a discussion of the tectonic history, property geology, a description of the mineralized zones, and presents a model for deposit formation based on observed geology and gold distribution

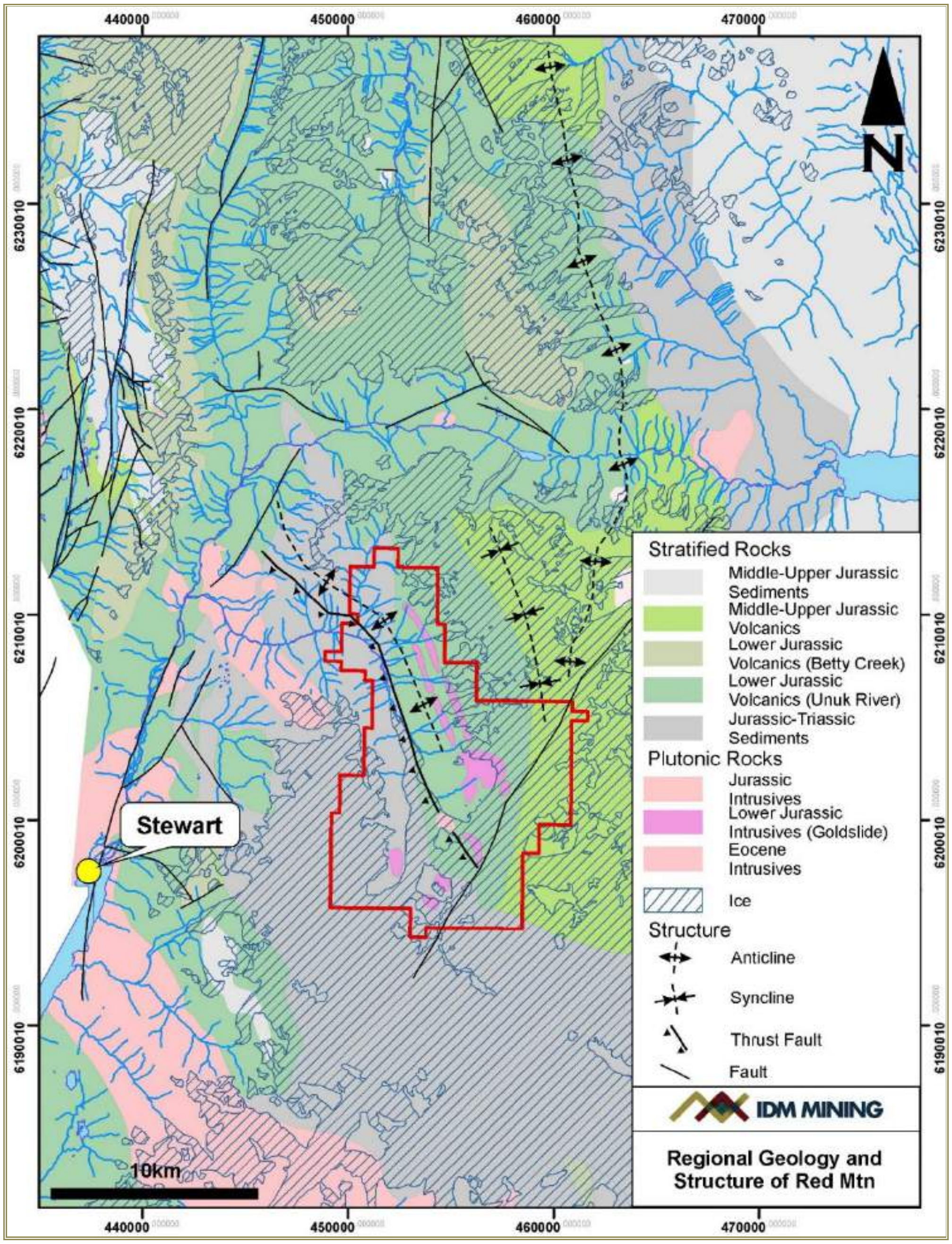
7.2.1 Regional Geology

The regional geology of the Red Mountain area has been described by Greig et al. (1994), Alldrick (1993), and Rhys et al. (1995). The following description is drawn from these sources.

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt (Figure 7-24). There are three primary stratigraphic elements in Stikinia and all are present in the Stewart, BC, area: Middle and Upper Triassic clastic rocks of the Stuhini Group, Lower and Middle Jurassic volcanic and clastic rocks of the Hazelton Group, and Upper Jurassic sedimentary rocks of the Bowser Lake Group. Regional metamorphic grade is typically lower greenschist facies, locally to middle-greenschist. On the Red Mountain property, the Lisa Nunatak area exhibits moderate crenulation cleavage, suggesting a higher degree of regional metamorphism.

Intrusive rocks in the Red Mountain region range in age from Late Triassic to Eocene and form several suites. The Stikine plutonic suite comprises Late Triassic calc-alkaline intrusions that are coeval with the Stuhini Group rocks. Early to Middle Jurassic plutons are roughly coeval with the Hazelton Group rocks and have important economic implications for gold mineralization in the Stewart area, including the Red Mountain resources (referred to as the Goldslide Suite). Intrusive rocks of this age are of variable composition (Rhys et al., 1995). Eocene intrusions of the Coast Plutonic Complex occur to the west and south of Red Mountain and are associated with high-grade silver-lead-zinc occurrences; gold-silver-bismuth ± copper-lead-zinc mineralization recently identified in the Lost Valley area is likely Eocene age.

Structurally, Red Mountain lies along the western edge of a complex, northwest-southeast-trending, doubly-plunging structural culmination, which was formed during Cretaceous tectonic shortening. At this time rocks of the Stuhini, Hazelton, and Bowser Lake Groups were folded and/or faulted, with up to 40% shortening in a northeast-southwest direction. The Red Mountain deposits lie at the core of the Bitter Creek antiform, a northwest-southeast trending structure created during this deformation event (Greig, 2000).



Source: IDM (2016)

Figure 7-24: Red Mountain Regional Geology

7.2.2 Local Geology

The tectonic history of northwestern British Columbia in the Red Mountain area is described below.

200 Ma—Early Jurassic

The Quesnelia and Slide Mountain terrains have already docked with ancestral North America. Stikinia is separated from continental North America by Cache Creek oceanic crust, which is being subducted at both under North America and the western edge of Stikinia. Another subduction zone exists on the eastern edge of Stikinia. Above this subduction zone the Red Mountain gold deposits are formed in an oceanic volcanic arc.

170 Ma—Middle Jurassic

Stikinia has docked with North America. The Bowser Basin has just formed and is getting initial basin fill from Cache Creek rocks in the east, which were placed on top of the Stikine terrain by back-thrusting during docking, and from Stikinia rocks in the west. A lack of intrusive rocks suggests there is no active subduction west of Stikinia at this time or that if present it is so far to the west that no influence is felt.

145 Ma—Early Cretaceous

The Alexandria terrain rocks, and formation of the Skeena fold belt starts. This event folded the rocks of the Stuhini, Hazelton, and Bowser Lake Groups.

65 Ma—End of Cretaceous

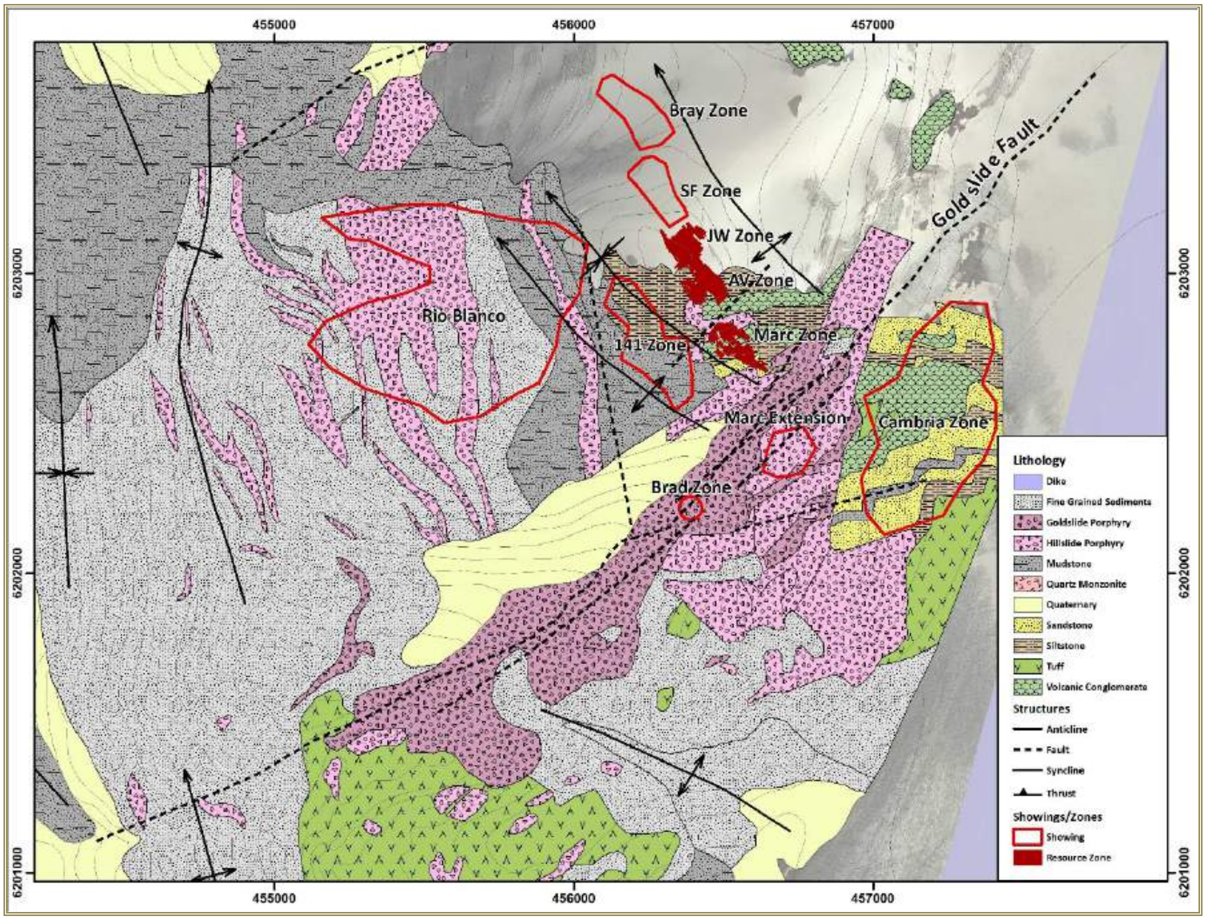
Deformation of Stikine terrain rocks is complete, resulting in folded and doubly plunging structural culminations. The Red Mountain deposits have been rotated from a vertical (?) orientation to a westerly dipping, northerly plunging orientation in the eastern limb of the Bitter Creek antiform. The Alexandria terrain has been intruded by plutons of the Coast Plutonic Complex.

20 Ma—Miocene

Extension along north-northwest and northeast trends forming large- and small-scale structures. Locally at Red Mountain this can be equated to formation of the Rick Fault and other property-scale faults, offsetting the mineralized zones.

7.2.3 Property Geology

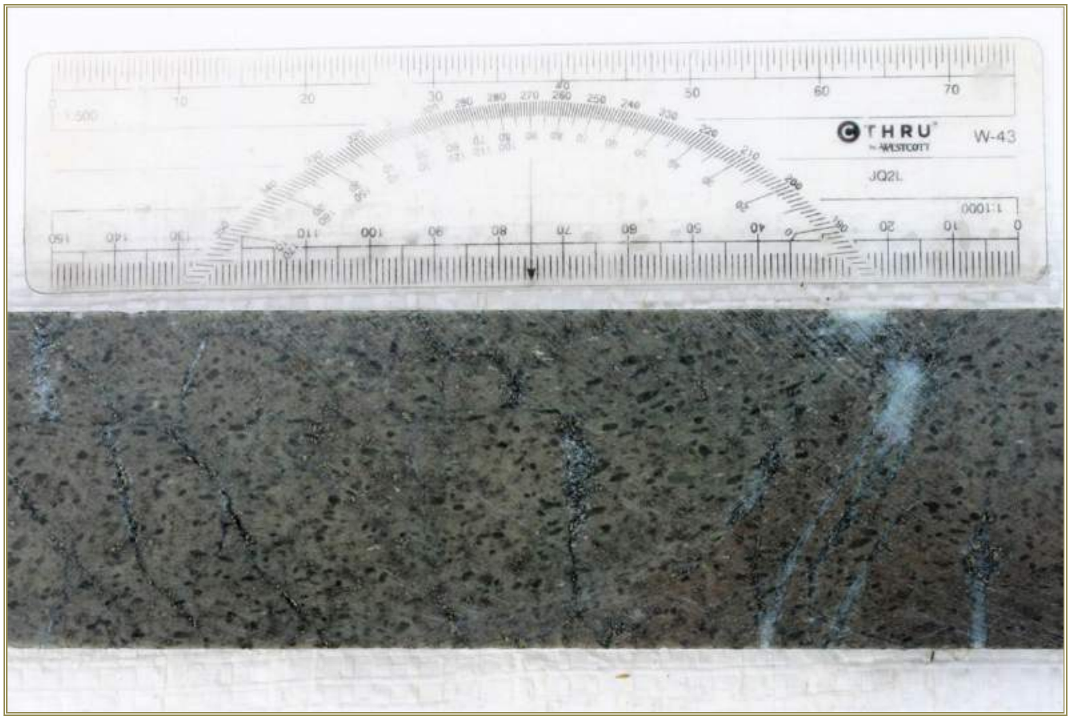
Property geology is shown on Figure 7-25. The oldest rocks, Middle to Upper Triassic mudstone, siltstone, and chert of the Stuhini Group outcrop over about two thirds of the mapped area. The Triassic rocks grade upward into Lower Jurassic Hazelton Group clastic and volcaniclastic rocks, which outcrop in the northeastern portion of the area depicted in the map. The unconformity is not definitely identified on the property, however a pyroxene-phyric volcaniclastic breccia exposed west of the summit of Red Mountain has been suggested as the location of this unconformity. Rocks of both groups are folded about axes, which plunge towards 345° and dip steeply to the southwest. The Goldslide suite of intrusions, which have been identified from the Lisa Nunatak area of the south, through Red Mountain to the Hartley Gulch area to the north, are suggested to follow along an earliest Jurassic back-arc basin growth fault, with thick sequences of bimodal, andesite-dominated volcanic rocks to the east of this structure. The approximate contact between Triassic sedimentary rocks and Jurassic rocks runs parallel to the projected trace of the Bitter Creek antiform. This structure has been mapped by Greig et al. (1994) to the northwest of the map area. Hazelton Group volcaniclastic rocks on the southwest limb of this structure were likely much thinner and may have been eroded away.



Source: IDM (2017)

Figure 7-25: Red Mountain Property Geology

At least four phases of the Early Jurassic- Goldslide suite of intrusions have been identified in the map area. The Hillside porphyry (Figure 7-26), a fine to medium-grained hornblende and plagioclase porphyry, occurs near the summit of Red Mountain and along the ridge to the southeast of the summit. The medium- to coarse-grained hornblende ± quartz-phyric Goldslide porphyry (Figure 7-27), is distinguishable from the Hillside porphyry by mineralogy and phenocryst size. It is exposed along the Goldslide Creek valley, extending from the surface expression of the Marc Zone to the southwest for two kilometres. Rhys et al. (1995) report a Pb/U date of 197.1 ± 1.9 Ma for a sample of Goldslide. Finally, sills of the biotite porphyry intrude Upper Triassic sedimentary rocks on the west side of Red Mountain. Biotite porphyry Goldslide porphyry by the small size of hornblende and plagioclase phenocrysts (Rhys et al., 1995).



Source: DM (2017)

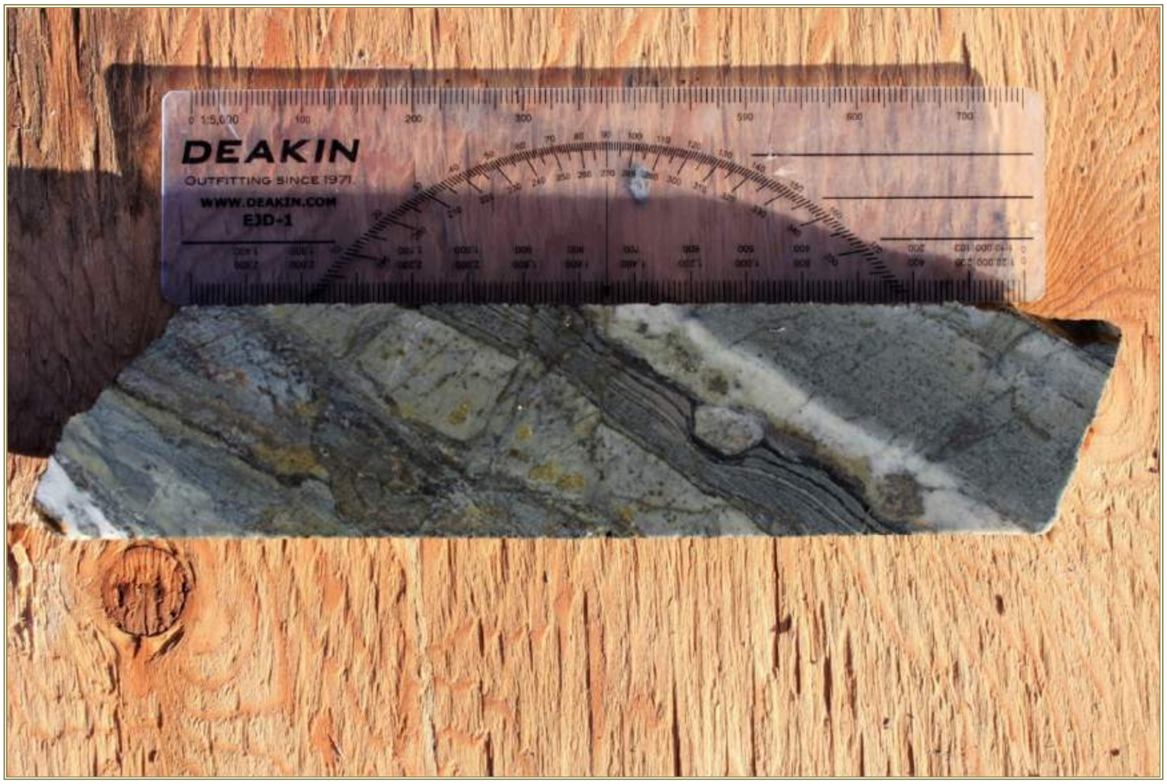
Figure 7-26: Hillside Porphyry

Source: IDM (2017)

Figure 7-27: Goldslide Porphyry with Chlorite-Epidote Alteration

A fourth intrusive phase of the Goldslide suite was identified during the 2017 program, primarily in the area of Smit/Bray zone on the north side of the resource area and outcropping along the Rio Blanco (northwest) side of Red Mountain. This phase has medium-size hornblende needles, with an aphanitic, siliceous matrix. The importance of this unit to the mineralization system is not known.

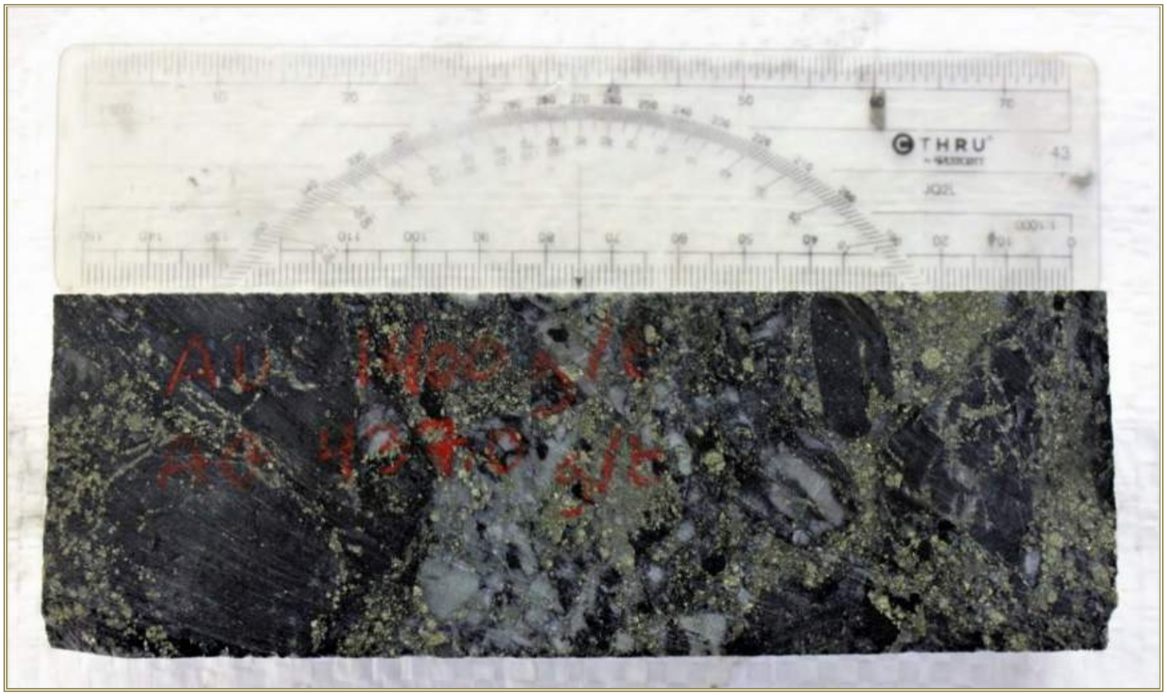
Bedded rocks occurring proximal to the mineralized zones vary in composition, ranging from siliciclastic-dominant siltstones to sandstones, in areas there is minor graphite, to bedded ash tuffs, with local mixed to transition zones (Figure 7-28). Volcanic units often host lapilli to agglomeritic clasts.



Source: IDM (2017)

Figure 7-28: Example of Bedded Tuff with Fine Muddy Laminae

Multiple types of intrusive and hydrothermal breccias have been identified throughout the property, and strongly disrupted bedding is common along the contacts of the intrusions, particularly in association with the Hillside porphyry. Mineral resources at Red Mountain generally follow early, syn-intrusive breccias of multiple types. Breccia types include contact breccias, crackle and mosaic breccias, and intrusive breccias. These are best exposed in underground workings. The Hillside porphyry contains large rafts of the sedimentary rocks ranging in size from 1 m or 2 m to several tens of metres. Figure 7-29 shows an example of a strongly mineralized breccia, containing mudstone and Hillside porphyry clasts.



Source: IDM (2017)

Figure 7-29: Strongly Mineralized Mudstone—Hillside Porphyry Breccia

A Tertiary intrusion, the McAdam Point stock, is exposed in the Lost Valley area adjacent to the Bromley Glacier. It is a medium- to coarse-grained biotite quartz monzonite to biotite monzogranite. Recent, preliminary U-Pb age dates average 56.5 Ma (Travis Murphy, personal communication, 2019). The Lost Valley stock has several intrusive phases, with sharp contacts between coarse and fine phases of quartz monzonite observed in several locations. Several dykes of monzonite have been traced further to the south through the Lost Mountain area and suggest a continuation of the main body at depth, under a mantle of hornfelsed metasedimentary rocks. Multiple phases of narrow gold-silver-molybdenum ± base metal veins have been recently discovered in this area.

Structural deformation at the property scale is consistent with the observations at the regional and tectonic scales. Folds have been mapped in the entire Triassic-Jurassic succession with north- to northwest-plunging axes and generally steeply dipping limbs. Fold traces can be complicated and difficult to trace, particularly near intrusive contacts (Rhys et al., 1995). The timing suggests that the folds are a manifestation of the Cretaceous Skeena Fold Belt deformation. Recent interpretation suggests that the breccia sequence that hosts mineral resources at Red Mountain have east-verging antiform/synform pairs with amplitudes on the order of 200 to 300 m, generally with gentle northwest plunges.

A series of north-to-south striking strike slip faults have been directly observed in Lost Valley, most notably where they truncate the andesitic/lamprophyre dykes, meaning that this movement is happening after the emplacement of the Lost Valley intrusion. These strike-slip faults can then be traced for several kilometres across the property and occur as parallel structures spaced about 400 m apart. Sympathetic structures, such as Riedel shears, normal and reverse faults have been observed propagating from these faults, with some evidence of north-south striking zones of pyrrhotite-dominated sulfide veins, primarily at the Cambria zone and Lost Mountain, and potentially associated with the 050 Fault.

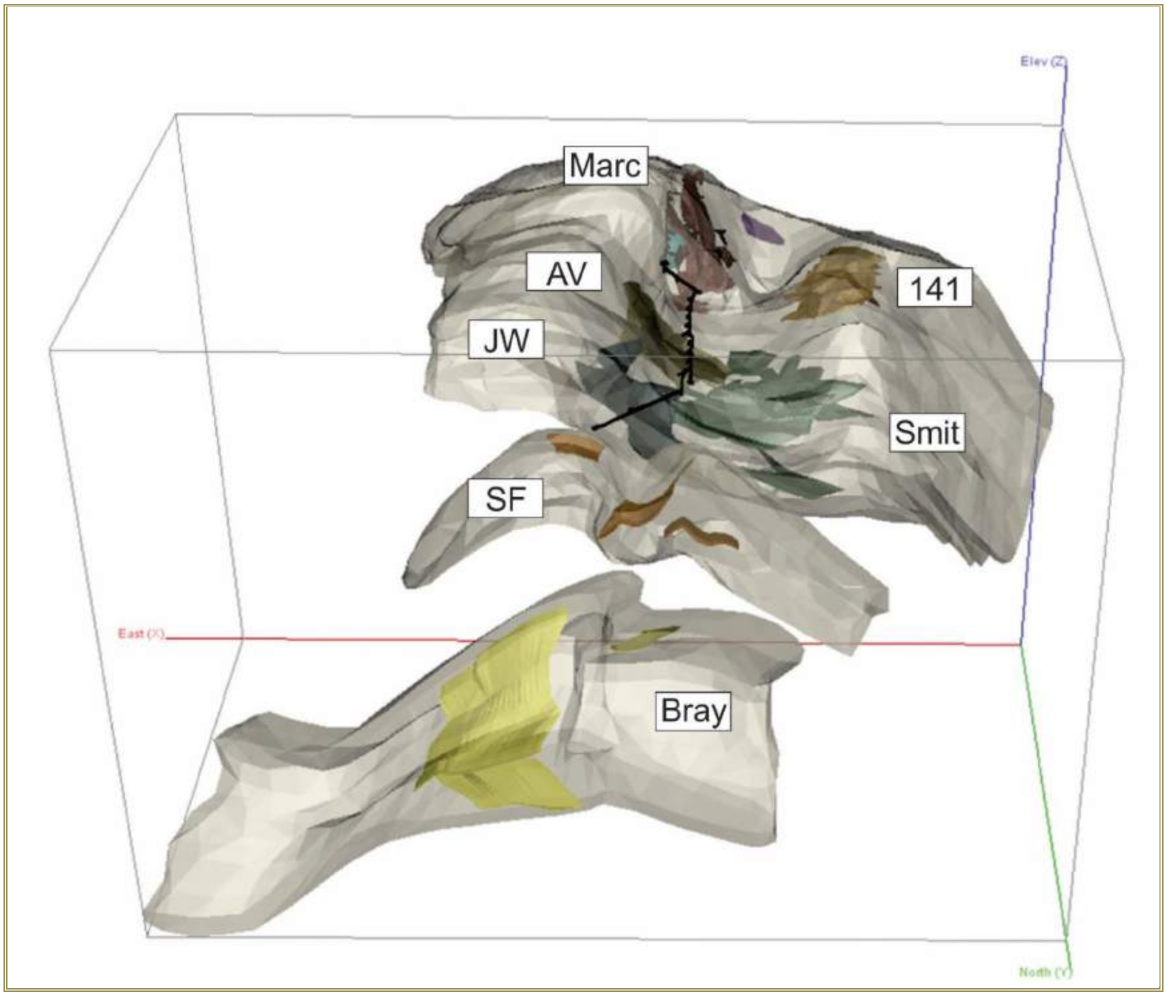
Post-mineralization faulting has affected all rock units at Red Mountain. Rhys et al. (1995) recognized two phases of faulting: northeast striking, steeply northwesterly dipping faults, and north-to-northwest trending faults. Faults of the former group are those that offset the mineralized zones, such as the Rick Fault. The latter group are noted by Rhys et al. (1995) to contain more gouge and have broader alteration envelopes than the former.

Updated Geologic Model

Recent studies and reinterpretation of surface and underground mapping and surface and underground drill hole data have led to the development of a new geologic model for the Red Mountain gold deposits. The model is based on a brecciated horizon favourable for gold mineralization, defined by a complex and variable sequence of breccias that generally follow contacts between sedimentary units and Hillside porphyry. Geologic interpretation simplified the brecciation into two main types: siltstone-mudstone-dominant and Hillside porphyry-dominant. Both breccias are generally monomictic and clast supported, often displaying only limited clast displacement or rotation and jigsaw (mosaic) textures. These breccias are hydrothermal in origin, forming during the earliest Jurassic intrusive event. They likely formed at a shallow-crustal level, intruding poorly consolidated (wet) sediments. Pebble dykes have been commonly identified near the breccia horizon, particularly hosted in sediment rafts.

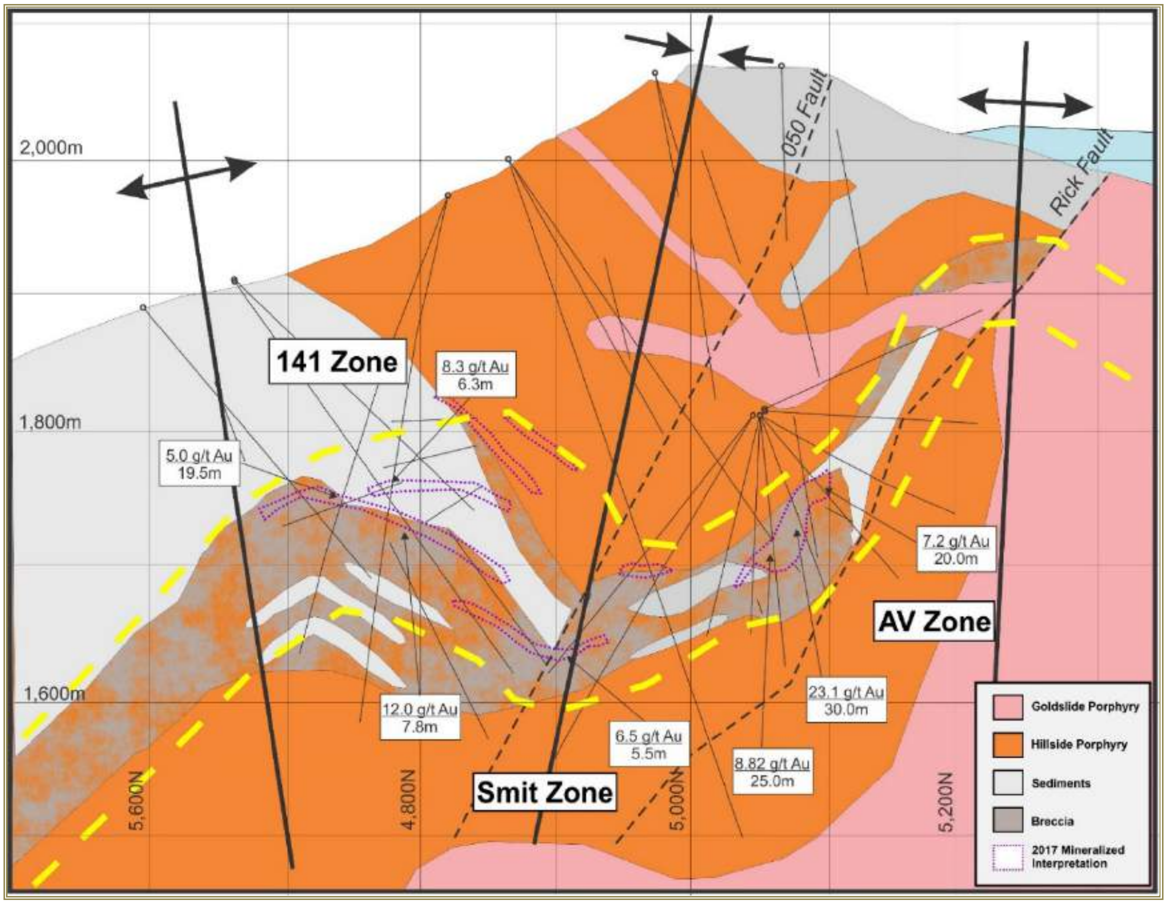
The hydrothermal breccias are accompanied by widespread alteration of potassium feldspar, silica and sericite with pyrite mineralization ranging from disseminated and stockwork to massive mineralization accompanied by lesser pyrrhotite mineralization. The zones that host high-grade mineralization (>3.0 g/t Au) are generally concordant with and hosted within both types of breccias, as well as unbrecciated Hillside porphyry and sedimentary rocks. A definitive origin for the host structure has not been identified, potentially due to a strong alteration overprint. Thicker and higher-grade mineralization is often associated near the contact with discordant bodies of quartz-phyric Goldslide porphyry.

The favourable breccia horizon was clearly folded during mid-Cretaceous transpressional tectonics, resulting in relatively high amplitude (~200 m) gently northwesterly plunging folds. An isometric view of the interpreted, folded horizon is shown in Figure 7-30, looking southeast. The folds have been cut by northeasterly trending normal faults, down-dropping the horizon to the northwest (note that diagram is in mine grid where north is 315°). Figure 7-31 shows cross-section 1350N, illustrating the geometry of the favorable breccia horizon and the location of several of the mineralized zones.



Source: IDM (2017)

Figure 7-30: Red Mountain—Current Geological Model of Folded Favourable Breccia Horizon and Mineralized Zones

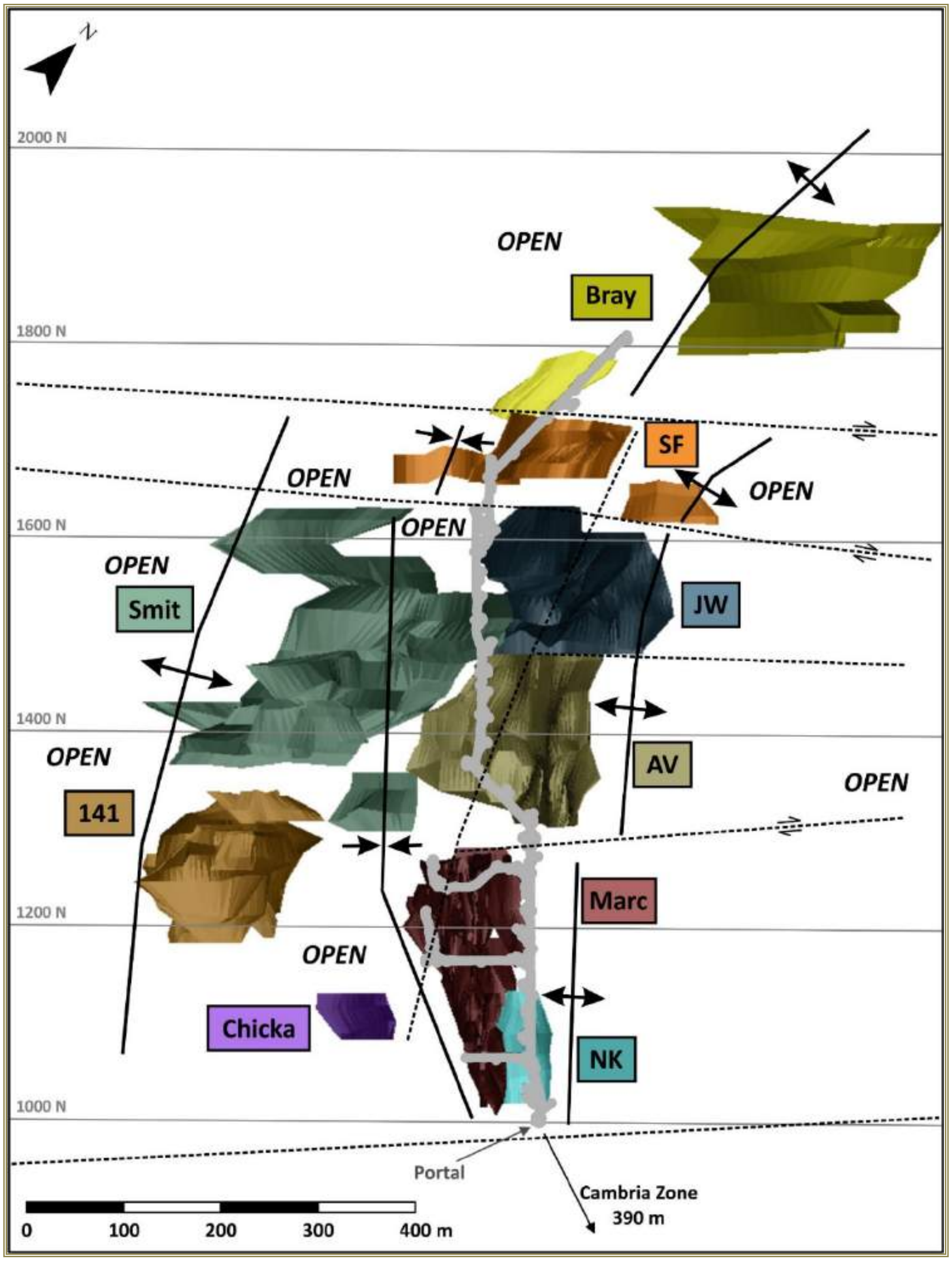


Source: IDM (2017)

Figure 7-31: Red Mountain—Cross-Section 1350N Looking West

Mineralized Zones

The mineralized zones consist of crudely tabular, northwesterly trending and variably dipping gold-bearing iron sulfide stockworks. A plan of the zones with names is given in Figure 7-32. Pyrite is the dominant sulfide, however locally pyrrhotite is important. The stockworks zones are developed primarily within the Hillside porphyry and brecciated Hillside, with volumetrically less in the rafts of sedimentary and tuffaceous rocks and sediment-dominant breccias. Although, locally anomalous gold values are present within the Goldslide porphyry, significant gold-bearing sulfide stockwork zones have not been found in this rock unit, although Goldslide porphyry in proximity to mineralized zones.



Source: IDM (2017)

Figure 7-32: Plan View of Mineralized Stockwork Zones

The stockwork zones consist of pyrite disseminations, irregular coarse-grained pyrite veins and stockworks with pale, strongly sericite-altered rocks (Figure 7-33). Potassium feldspar, silica, calcite, and iron-carbonate alteration are common, particularly in the Hillside porphyry host. Pyrite vein widths vary from 0.1 cm to approximately 200 cm, widths of 1 to 3 cm are most common. The veins are variably spaced, average 2 to 10 per metre, and generally comprise from 4% to 10% of any drill intersection. The veins are very often heavily fractured or brecciated with infillings of fibrous quartz and calcite. Orientations of veins in the stockworks are variable; however, sets with northwesterly trends and moderate to steep northeasterly and southwesterly dips have been identified in underground workings (Rhys et al., 1995).

The pyrite veins typically carry gold grades ranging from ~3 g/t to greater than 100 g/t. Gold occurs in grains of native gold, electrum, petzite, and a variety of gold tellurides and sulfosalts (Barnett, 1991). These mineral grains, which are typically 0.5 µm to 15 µm, occur along cracks in pyrite grains, within quartz- and calcite-filled fractures in pyrite veins, and to a lesser extent, as inclusions within pyrite grains.

The stockwork zones are surrounded by more widespread zones of disseminated pyrite and pyrrhotite alteration. Each of these sulfides, which also occur as sparsely distributed stringers, comprise about 1.5% to 2.0% of the wall rocks to the stockwork zones. The most striking feature is that while disseminated pyrite occurs within the stockwork zones the disseminated pyrrhotite abruptly disappears, often over distances of less than a metre, at the edges of the bleached pyrite stockwork zones. Locally disseminated pyrrhotite does occur within the pyrite stockwork, but generally only in peripheral areas where bleaching and pyrite vein density is weak.

Source: IDM (2017)

Figure 7-33: Typical Coarse-Grained Pyrite Veins

The stockwork zones are also partially surrounded by a halo of red sphalerite. Sphalerite comprises 0.5% to 4.0% of the rock, and generally is more abundant in the footwall portions of the zones and in the sedimentary units. Generally, the sphalerite/pyrrhotite halo hosts lower-grade gold values (0.5 to 5.0 g/t Au).

8 DEPOSIT TYPES

8.1 Premier Gold Project

Mineral deposits at the PGP are intermediate-sulfidation epithermal gold-silver deposits with subsidiary base metals. These deposits form at comparatively shallow depths (generally above 1 km depth), often in association with hot spring activity on surface. Mineralization results from circulation of aqueous solutions driven by remnant heat from intrusive bodies. Where these ascending fluids encounter meteoric waters and/or as the hydrostatic pressure drops, changes in temperature and chemistry results in precipitation of minerals into fractures, breccias, and open spaces.

Mineralized bodies are structurally controlled veins, stockworks, and breccia bodies, and are broadly tabular with a wide range of orientations. They measure from cm-scale to many metres in thickness and can often be traced for strike lengths of several hundred metres or even kilometres. Economic minerals comprise native gold and native silver, electrum, silver sulfosalts, and silver sulfides, along with pyrite and sphalerite and comparatively minor amounts of chalcopyrite and galena. Gold and silver values are quite variable and, while averaging in the order of 5 g/t Au to 10 g/t Au and 20 g/t Ag to 30 g/t Ag within the historical stopes.

8.2 Red Mountain Project

Several models have been presented for the formation of the Red Mountain gold deposits. Rhys et al. (1995) concluded that the setting and style of mineralization is similar to that of many porphyry systems. This was based on data from deep drilling that indicated mineralization and alteration zoning common to traditional porphyry systems. Lang (2000b) suggested that while the porphyry system zonation was present the alteration and mineralization was more consistent with a later magmatic-hydrothermal system that overprinted the earlier vertical alteration pattern. Recent interpretation is that the gold mineralization at Red Mountain is consistent with an intrusive-related system, rather than a porphyry-gold deposit.

Incorporating recent suggestions for regional early-Jurassic intrusive-related and magmatic-hydrothermal mineralization in northwest BC, which incorporate mapping and petrographic observations (Lang, 2000a, 2000b) the proposed metallogenic sequence for the Red Mountain Property is as follows:

- Approximately 200 Ma, the Hillside porphyry intruded into Stuhini and unconsolidated lower-most Hazelton Group strata. Large rafts of sedimentary rocks are encapsulated the intrusion; contact brecciation between porphyry and sedimentary rocks.
- The Hillside porphyry cools and contracts. The contraction causes microfracturing of the porphyry and breccia zones. Early pyrite was deposited into these fractures.
- Ongoing cooling, and alteration of hosts rocks by hydrothermal fluids, with fracturing and brecciation of coarse-grained pyrite veins. Additional coarse-grained pyrite is deposited. The early gold mineralization including petzite is deposited as small inclusions in pyrite grains.

- Intrusion of the Goldslide porphyry including quartz-phyric phase. The intrusion drives a pulse of hydrothermal fluids primarily containing native gold, with local tellurides and sulfosalts into fractures and rims of the in the coarse-grained pyrite veins.
- Final infilling of remaining fractures in the coarse-grained pyrite veins with gold minerals, fibrous quartz, calcite, feldspar, and sericite.
- Intrusion of biotite-phyric phase of Goldslide Suite.
- Mid-Jurassic extensional tectonism.
- Cretaceous transpressional tectonics; recumbent folding of mineralization and favourable breccia horizon.
- Intrusion of multiple phases of 57.3 Ma McAdam Point stock; intrusive related/porphyry gold-molybdenum quartz stockworks and disseminations.
- Remobilization of gold and sulfides at Lost Valley during subsequent thrusting.
- North-south faults with minor offset; pyrrhotite dominant gold-silver-base metal veins.
- Intrusion of andesite and lamprophyre dykes.

9 EXPLORATION

9.1 Premier Gold Project

Exploration work conducted by Ascot from 2007 to 2011, inclusive, is described in detail in a Technical Report by Kirkham and Bjornson (2012). This report is publicly available on SEDAR. Exploration activity from 2012 to 2017 was almost exclusively diamond drilling with the exception of a LiDAR survey that was carried out in 2014. The drilling work for this period is described in Section 10 Drilling. A summary of exploration work conducted by Ascot prior to 2012, excluding drilling, is provided in Table 9-1.

Table 9-1: Summary of Ascot Exploration Work (excluding Drilling) from 2007–2019

Year	Area	Type of Work	Comments
2007	Dilworth	Surface sampling	83 channel, 371 chip, and 29 grab samples
2008	Dilworth	Surface sampling	75 stream sediment, 540 chip, 84 grab, and 590 soil samples
	All	Airborne geophysics	469 line-km EM and magnetometer (Mag), 504 line-km gamma ray spectrometer
	Dilworth	Geological mapping	1:2,000 scale
2009	Premier, Big Missouri	Surface sampling	786 chip and 26 grab samples
2010	Premier, Big Missouri	Surface sampling	383 chip, 133 channel, and 4 grab samples
2018	Premier, Big Missouri, Silver Coin	Wireless IP	14,700 line-m of ground IP

At the beginning of 2018, Ascot began to research means of exploring the entire land package effectively and more cheaply than by systematic grid drilling. Ascot personnel used the current multi-element assay database to estimate modal sulfide contents of sphalerite, galena, chalcopyrite, and pyrite from assayed zinc, lead, copper, and sulfide. The pyrite content was then plotted in 3-D which indicated that the zones of gold mineralization were accompanied by higher amounts of disseminated pyrite. One of the more effective geophysical methods for detection of disseminated pyrite is Induced Polarization (IP), and so a 1,200 m test line of pole-dipole IP at 50 m spacing was run over the western edge of the Premier and Northern Lights zones, covering known zones of gold mineralization.

Figure 9-1 is a location map over the southern part of Ascot's property showing the layout of the IP survey as completed in 2018. In the opinion of Ascot geologists, the image clearly demonstrates that the areas of high chargeability coincide with known gold mineralization.

Following the success of the test survey, Ascot ran additional profiles to the north and south of Premier and between Big Missouri and Silver Coin (Figure 9-1). The entire program encompassed a total length of 14,700 line-m of IP profiles.

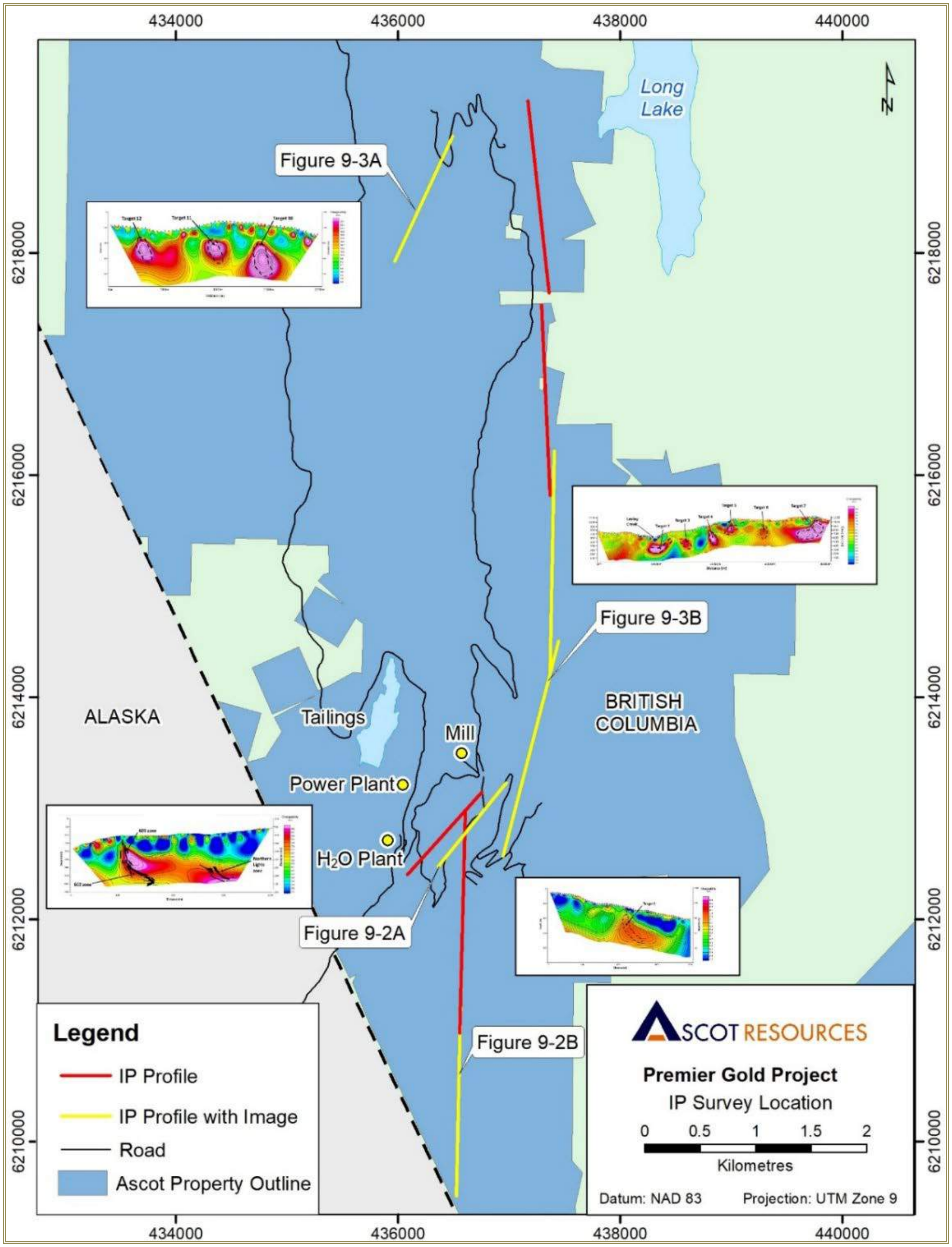


Figure 9-1: IP Survey Location Map from 2018

In 2019, Ascot completed additional IP profiles throughout the property, adding to the inventory of IP anomalies. The IP coverage is still rudimentary and will have to be filled in during 2020 in priority areas. Large parts of the property have not yet been covered by IP.

Figure 9-2a is a profile made to the south of Premier showing a previously unknown chargeability anomaly. The absolute chargeability is somewhat lower in intensity (7mV/V versus 10mV/V) than observed at Premier but the geometry of the anomaly is similar. The inversion sections of chargeability in Figure 9-3 show several previously unknown anomalies in the area to the north of Premier (Figure 9-3a) and in the area between Big Missouri and Silver Coin (Figure 9-3b). Many of these anomalies are of similar strength and character as the anomalies generated from known mineralization at Premier.

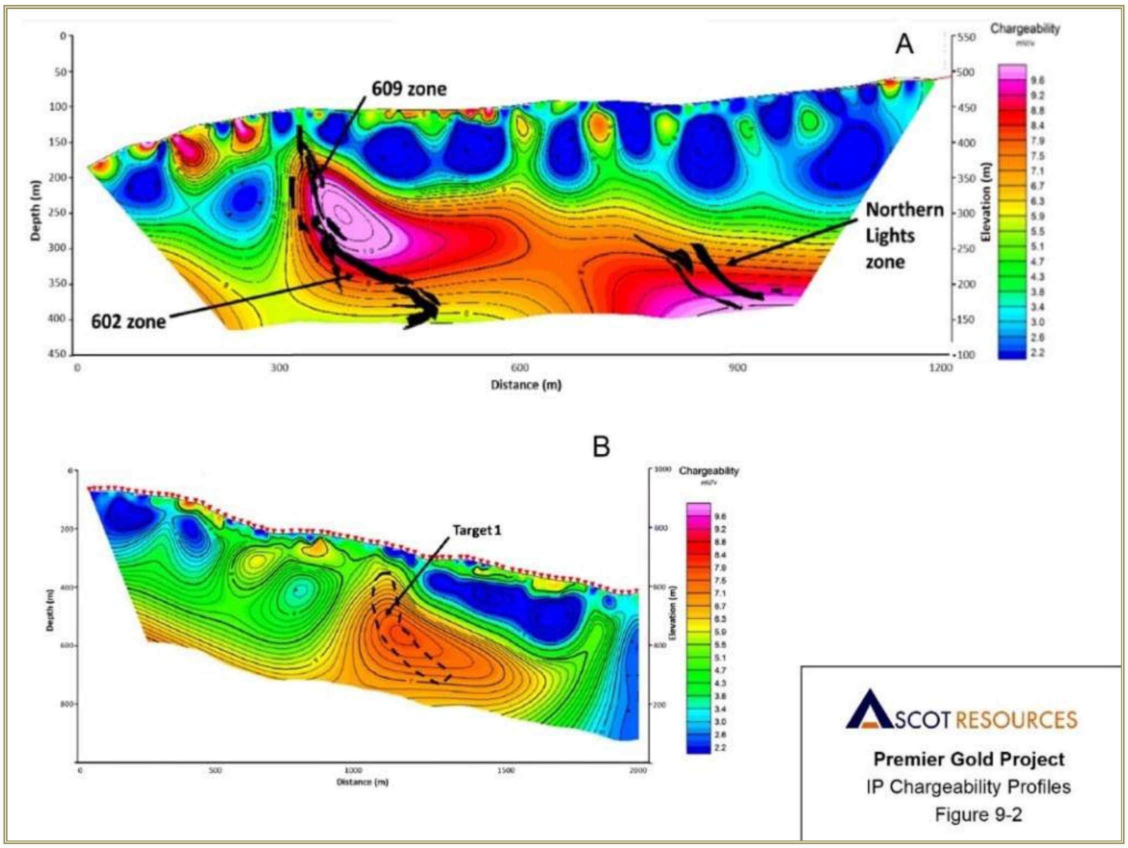


Figure 9-2: IP Chargeability Profiles—Premier Area

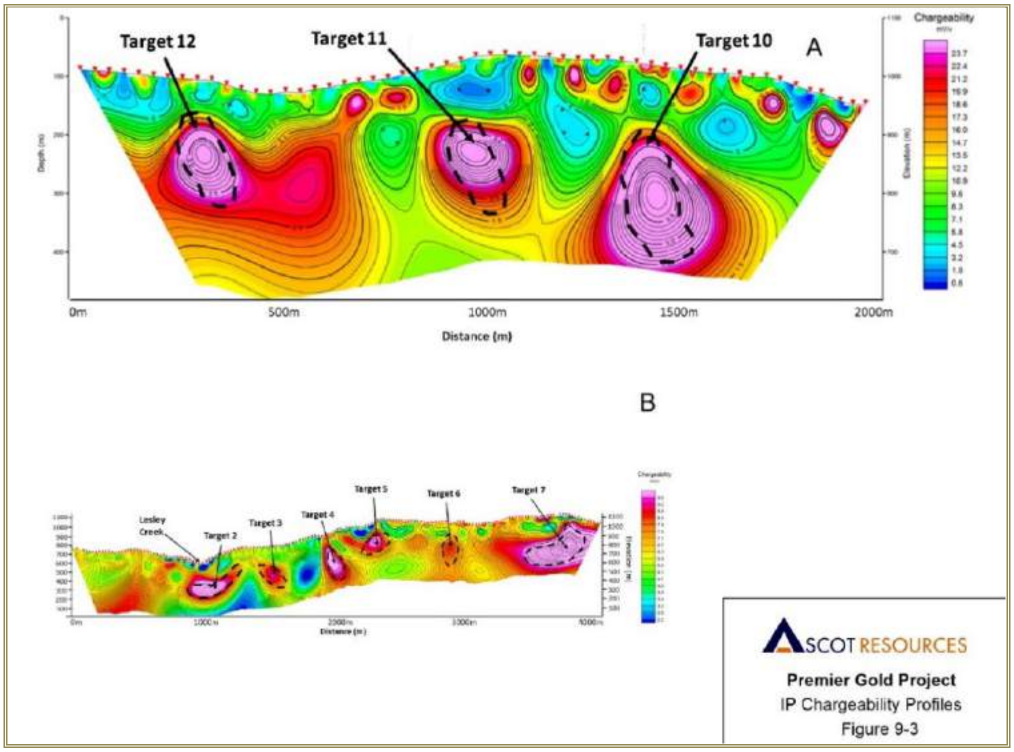


Figure 9-3: IP Chargeability Profiles—North of Premier Area

9.2 2020 Exploration Program

In 2020, Ascot is planning to complete 10,000 m of diamond drilling from surface at the western extension of Premier following up encouraging results from 2019. The Company also plans to conduct induced polarization ground geophysical surveys in various parts of the property. Grassroots mapping and sampling is planned for the northern and eastern parts of the property aiming to identify new zones of mineralization away from the known resource areas.

Additional drilling is budgeted in order to follow up existing and new IP anomalies on the property. The budget for the planned 2020 exploration program is summarized below in Table 9-2.

Table 9-2: 2020 Exploration Budget

Category	Drilling (m)	Cost (C$)
Mapping and Sampling		200,000
Geophysics		
IP		800,000
Exploration Drilling		
Premier West	12,000	1,800,000
IP Targets	8,000	1,200,000
Total	**20,000**	**4,000,000**

The QP agrees with the opinions of Ascot geologists and considers the planned expenditures to be warranted.

9.3 Exploration at Red Mountain

Past exploration is summarized in Sections 1, 6, and 10. No exploration was conducted from 2001 to 2012 as the property was on care and maintenance by Seabridge. In 2012, Banks drilled three drill holes is the Marc zone, two of which intersected the Marc mineralized zone and the third hole was abandoned prior to reaching the Marc zone.

9.3.1 Property Grids

All data in the Red Mountain Gemcom database, including the drill hole orientation data, has two sets of coordinates, and if applicable, two different azimuths. One set comprises UTM grid coordinates and azimuths, for which the north direction is 0.5° west of true north. The second set of coordinates and azimuth is for a local mine grid where the north direction has been rotated 45° to the west. Mine grid north is therefore parallel to the trend of the stockwork zones, and the vertical section orientation at 090°-270°mine grid is perpendicular to the trend of the stockwork zones.

All work for the current resource estimation has used mine grid coordinates and orientations.

9.3.2 Geological Mapping

Geological mapping at a variety of scales from prospect scale to property scale, was carried out by Bond and LAC employees and consultants in order to understand lithological, structural and mineralization relationships. More recently IDM has completed additional mapping of areas exposed due to receding glacial ice.

9.3.3 Geochemical Sampling

Soil, grab, and rock sampling has been, and still is, used to evaluate mineralization potential and generate targets for ongoing exploration programs and core drilling. The Project database contains approximately: 2,200 soil samples, 6,250 rock samples and 890 whole rock samples.

9.3.4 Geophysics

LAC carried out a number of geophysical surveys were completed on the property between 1990 and 1994 for use to vector in on mineralization and generate targets for exploration drilling. Methods have included:

- Surface IP, UTEM, VLF, and magnetics
- Airborne magnetics, EM, and radiometrics
- Downhole IP, magnetics, and UTEM.

9.3.5 Petrology, Mineralogy, and Research Studies

A significant number of research studies have been completed over the years by the various property owners on the Red Mountain Gold Project. These include:

- Structural studies (regional, property, and zone scales)
- Petrographic, alteration, and mineralogical studies
- Deposit genesis and metal distribution studies
- Age dating studies.

9.3.6 IDM Exploration Programs

After acquiring an option on Red Mountain in 2014 IDM commenced exploration on the property, including soil sampling (546 samples), rock sampling (440 samples), channel sampling (241 samples) and 12 diamond drill holes totalling 2,223.0 m (McLeod, 2014). Additionally, historical core was re-logged, and 68 infill samples taken in areas of strong alteration and mineralization.

Soil sampling focused on extending the 1994 grid to the north up the Bitter Creek Valley, while rock samples were collected in all areas there was a rock sampling and channel sampling focus areas that have become exposed by receding glaciers including Lost Valley, Lost Mountain, and the Cambria zone. Mineralized samples requiring additional follow up were collected in many areas and resulted in the identification of several new mineralized showings. Two of these, the Oxlux and Wyy Lo'oop showings in the Cambria zone were assessed by preliminary drilling in 2014.

The 2016 exploration program included underground rehabilitation and drilling, which consisted of 51 holes totalling 6,385.44 m. The drilling program was designed to upgrade the mineral resource classification and to expand resources, as well as to collect samples for metallurgical, geotechnical, and hydrological evaluation. Surface rock sampling, consisting of 509 samples, focused on mineralized exposures in Lost Valley, which were later tested five surface holes. Five additional surface drill holes were also completed to test extensions of the 141 zone and one hole tested the extension of the Brad zone. Finally, additional samples were collected historical core in the Marc and 141 zones.

The 2017 exploration program was focused mainly on underground drilling and was designed to upgrade and expand the mineral resource and test additional targets including the Bray and SF zones, which lie to the north of the main resource zones. Surface sampling, consisting of 709 rock samples, was completed on the surface expression of the Marc zone, new drill road exposures in the 141 zone and second portal area, and during prospecting in the Rio Blanco and Lost Mountain areas.

The 2018 exploration program included underground drilling consisting of 40 holes totalling 10,021.81 m, designed to upgrade and expand the existing resource and test new targets including the Marc–141 gap. Surface rock and channel sampling, consisting of 386 samples was focused on mineralization in Lost Valley, but also tested the Cambria, Cambria West, and Meg zones as well as targets in the Red Mountain Bowl.

9.3.7 Exploration Potential

Exploration potential for the property is deemed as high. Since 1994, when the surface exploration was terminated, the glaciers surrounding the Red Mountain Gold Project have significantly receded exposing considerable area that was previously inaccessible. The intrusion system that hosts the current resource has a broad areal extent and surface prospecting, mapping, geochemistry, geophysics, and drilling have the potential to discover similar deposits. Additional drilling also has the potential to expand the current resource zones.

9.3.8 Comment

The exploration programs completed to date are appropriate for the style of the mineralization and prospects located on the Project. There are a number of targets prospective for further exploration on the property.

10 DRILLING

10.1 Drilling at Premier Gold Project

10.1.1 Legacy Drilling

Drilling on the Premier Gold Project (PGP) dates back to 1928 and the Ascot database contains a total of 8,029 holes and 875,340 m; 3,406 of these holes representing 138,806 m are from years 1928 to 1941. These cover the entire property, are generally shallow, and have unreliable assay results. They have therefore not been used for resource modelling.

The database used for this Feasibility Study includes 1,879 holes and 152,005 m of legacy drilling from 1974 to 1996 that was predominantly drilled by Westmin. Jayden / MBM also drilled 476 holes and 74,741 m at Silver Coin prior to being taken over by Ascot.

Most of the legacy holes were selectively sampled in zones of visible sulfide mineralization. No assay Quality Assurance/Quality Control (QA/QC) data is available for these drill holes. Validation work conducted by Ascot personnel has demonstrated that the legacy drilling results in the Premier deposit area are generally reliable and so this data has been used for the Feasibility Study, with some restrictions. Details regarding this validation work are provided in the section of this report entitled Data Verification.

Some details regarding the work done during this period can be obtained from the BC government MINFILE website. Several Assessment Reports have been filed on the Property in order to fulfill land tenure requirements or as support for obtaining government grants. There are at least seven reports which span the period from 1979 to 1996. The records are far from complete, and only provided information on 48 diamond drill holes spread among the Premier, Big Missouri, Silver Coin and Big Missouri prospects.

Westmin was the operator for the work recorded in the Assessment Reports reviewed. Except for the period 1974 to 1976, the holes were drilled from surface, and in all but one case, were NQ-size (47.6 mm core dia.). The one case where BQ (36.4 mm) was drilled was when the hole traversed some broken or caved ground and it was necessary to reduce size in order to advance. All the holes were logged for lithology and alteration. In only one instance was there a reference to geotechnical logging, and in one other report it was stated that all the core was photographed, and the photos sent for storage in Westmin's Vancouver office.

A drilling contractor, Boisvenu Diamond Drilling, of Delta, BC, was noted as having done the work in reports dated 1987, 1995, and 1997. In these cases, it was also reported that the drill was a Boyles 56A rig. In two reports, the type of drill was reported (Boyles 56A and Longyear 38) but not the contractor.

Survey methods were not usually reported. In two reports, it was stated that the collars were not surveyed but were located using detailed orthophotos. Downhole survey methods were mentioned in two reports: Sperry Sun in 1994, and Tropari in 1996. It is possible to identify the holes where downhole surveys were performed from the database records. Generally, these tend to be longer surface holes, as opposed to the underground holes. It is further noted that there are markedly fewer downhole surveys in holes drilled prior to 1988, but they are fairly common thereafter.

The drilling history is summarized in Table 10-1 to Table 10-5.

Table 10-1: Drilling History—Premier

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
1928-41	Silbak Premier, Northern Lights, Sebakwe	3,406	138,805.80	31,534	60,555.80	44
1980	Westmin	20	2,336.46	439	439.76	19
1981	Westmin	34	4,697.47	965	1,886.75	40
1983	Westmin	18	2,253.30	448	771.21	34
1984	Westmin	22	2,575.28	751	1,170.26	45
1985	Westmin	57	3,052.86	1,303	2,094.74	69
1986	Westmin	104	9,626.53	3,414	5,542.11	58
1987	Westmin	196	17,235.94	4,725	7,492.98	43
1988	Westmin	104	10,782.60	3,798	5,382.48	5
1989	Westmin	33	3,387.30	1,133	1,493.35	44
1990	Westmin	59	4,454.30	1,712	2,535.49	57
1991	Westmin	18	1,871.90	561	564.55	30
1992	Westmin	53	1,046.94	782	934.34	89
1996	Westmin	192	15,142.91	7,550	8,662.75	57
1980-1996	Westmin Total	910	78,463.79	27,581	38,970.77	50
Total		**4,316**	**217,269.59**	**59,115**	**99,526.57**	**46**

Note: Pre-1980 drilling has not been used in the resource estimate

Table 10-2: Drilling History—Big Missouri

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
1974	Silver Butte (Giant Mascot opt)	11	254.36	no Au/Ag		
1976	Tournigan (Tapin opt)	8	177.80	49	77.30	43
1978	Westmin	11	629.42	261	383.13	61
1979	Westmin	7	971.74	336	494.89	51
1980	Westmin	44	2,213.84	854	1,380.84	62
1981	Westmin	47	1,899.12	590	1,084.48	57
1982	Westmin	70	2,627.73	800	1,466.57	56
1984	Westmin	6	283.46	122	185.40	65
1986	Westmin	30	1,260.98	507	826.04	66
1987	Westmin	47	4,612.85	1,238	1,929.14	42
1988	Westmin	86	8,457.25	2,320	3,355.77	40
1989	Westmin	14	1,696.12	411	654.01	39
Total		**381**	**25,084.67**	**7,488**	**11,837.57**	47

Table 10-3: Drilling History—Silver Coin

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
1982	Esso	22	1,374.69	481	849.76	62
1983	Esso	13	1,679.81	356	754.48	45
1986	Tenajon	4	996.27	252	354.56	36
1987	Tenajon	23	3,902.33	1,446	1,836.00	47
1988	Tenajon	58	7,593.06	2,623	3,472.20	46
1989	Tenajon	32	4,337.00	1,613	2,348.90	54
1990	Tenajon + Westmin	120	11,252.40	5,723	6,514.29	58
1993	Westmin	88	2,678.90	1,564	2,207.58	82
1994	Westmin	62	3,506.67	2,413	3,496.02	100
2004	Jayden/MBM	39	3,137.00	1,428	2,281.54	73
2005	Jayden/MBM	64	7,973.55	3,123	7,600.82	95
2006	Jayden/MBM	115	24,221.41	9,987	23,669.22	98
2007	Jayden/MBM	15	2,691.50	925	2,639.30	98
2008	Jayden/MBM	88	12,228.94	4,437	12,023.52	98
2009	Jayden/MBM	7	1,038.15	330	990.45	95
2010	Jayden/MBM	25	3,808.81	1,862	3,022.78	79
2011	Jayden/MBM	109	17,468.42	12,921	16,676.45	95
2017	Jayden/MBM	14	2,173.45	1,066	1,981.03	91
Total		**898**	**112,062.36**	**52,550**	**92,718.90**	**83**

Note: Jayden was called Pinnacle Mines Ltd. prior to June 2010.

Table 10-4: Drilling History—Martha Ellen

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
1981	Westmin	2	96.01	13	24.35	25
1982	Westmin	16	772.81	151	278.00	36
1983	Westmin	17	996.10	192	331.40	33
1986	Westmin	30	911.35	324	510.50	56
1987	Westmin	43	2,543.57	933	1,462.55	57
1988	Westmin	36	3,033.90	1,067	1,540.50	51
1996	Westmin	9	2,156.04	415	338.81	16
Total		**153**	**10,509.78**	**3,095**	**4,486.11**	**43**

Table 10-5: Drilling History—Dilworth

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
1981	Westmin	13	625.45	124	221.30	35

10.1.2 Ascot Drilling

Ascot commenced drilling on the PGP in 2007, and to September 2019 drilled 2,268 holes totaling 509,789 m of which an average of 45% was assayed. During 2007 and 2008, drilling was on the Dilworth area. From 2009 to 2014, most of the drilling was on Big Missouri with comparatively modest programs on Martha Ellen and Dilworth, and only minor drilling in the Premier area. Most of the work from that time up to the end of 2017 was in the Premier area. In 2018 and 2019 Ascot has done in-fill drilling at Premier, Big Missouri, and Silver Coin.

Ascot drill programs are summarized in Table 10-6 to Table 10-10.

Table 10-6: Ascot Drilling—Premier

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
2009	Ascot	20	1,693.69	687	772.87	46
2012	Ascot	1	313.03	16	36.43	12
2013	Ascot	4	801.32	114	248.02	31
2014	Ascot	149	32,541.12	5,904	10,252.42	32
2015	Ascot	198	40,867.68	8,153	13,948.43	34
2016	Ascot	279	69,112.47	7,095	12,087.21	17
2017	Ascot	359	113,465.41	15,033	25,254.39	22
2018	Ascot	53	16,900.06	1,667	2,738.42	16
2019	Ascot	58	12,755.61	2,264	3,462.86	27
Total		**1,121**	**288,450.39**	**40,933**	**68,801.05**	**24**

Table 10-7: Ascot Drilling—Big Missouri

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
2009	Ascot	18	3,956.67	2,526	3,012.05	76
2010	Ascot	52	17,385.67	11,672	17,187.97	99
2011	Ascot	144	34,979.66	18,146	33,025.78	94
2012	Ascot	93	23,218.30	10,546	20,405.29	88
2013	Ascot	76	13,595.93	5,239	10,337.66	76
2014	Ascot	20	4,380.47	1,315	2,513.87	57
2017	Ascot	10	1,947.97	488	781.05	40
2018	Ascot	194	29,860.76	6,946	11,661.34	39
2019	Ascot	156	25,871.63	7,459	11,910.48	46
Total		**763**	**155,197.06**	**64,337**	**110,835.49**	**71**

Table 10-8: Ascot Drilling—Silver Coin

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
2018	Ascot	13	2,626.44	820	1,305.32	50
2019	Ascot	81	10,919.85	4,267	7,078.17	65
Total		**94**	**13,546.29**	**5,087**	**8,383.49**	**62**

Table 10-9: Ascot Drilling—Martha Ellen

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
2009	Ascot	10	1,821.01	1,196	1,711.92	94
2010	Ascot	4	603.81	316	603.81	100
2012	Ascot	54	8,784.66	3,886	7,690.20	88
2013	Ascot	49	7,095.49	2,383	5,047.91	71
2017	Ascot	10	3,442.72	618	1,160.60	34
2018	Ascot	10	605.36	190	270.73	45
Total		**137**	**22,353.05**	**8,589**	**16,485.17**	**74**

Table 10-10: Ascot Drilling—Dilworth

Year	Operator	Holes	Metres	Intervals Assayed	Assayed (m)	Assayed (%)
2007	Ascot	36	5,037.20	2,989	3,465.62	69
2008	Ascot	63	10,910.88	5,669	8,978.11	82
2010	Ascot	12	3,751.79	2,342	3,731.08	99
2011	Ascot	6	1,353.00	698	1,253.12	93
2012	Ascot	19	4,938.84	2,131	4,346.02	88
2013	Ascot	17	4,250.14	1,578	3,082.82	73
Total		**153**	**30,241.85**	**15,407**	**24,856.77**	**82**

Drill hole locations in plan view are illustrated for all drilling in each area in Figure 10-1 to Figure 10-5. Representative sections of the drill holes with respect to the geology can be found in Figure 7-6 to Figure 7-11; Figure 7-14 to Figure 7-16; Figure 7-19 and Figure 7-20; and Figure 7-22 and Figure 7-23. Representative sections of the drilling with respect to the block model can be found in Figure 14-20 to Figure 14-30.

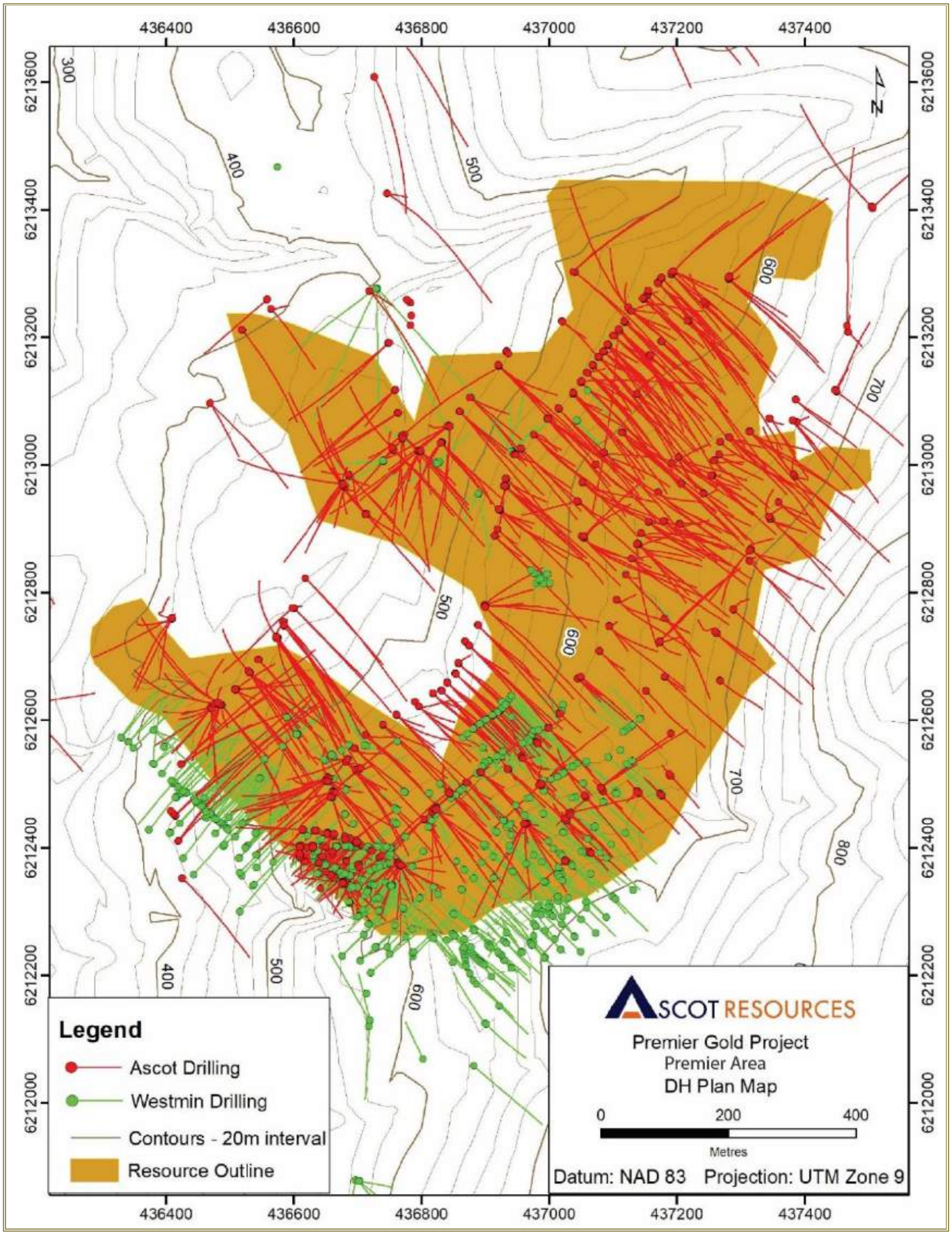


Figure 10-1: Drill Hole Plan—Premier

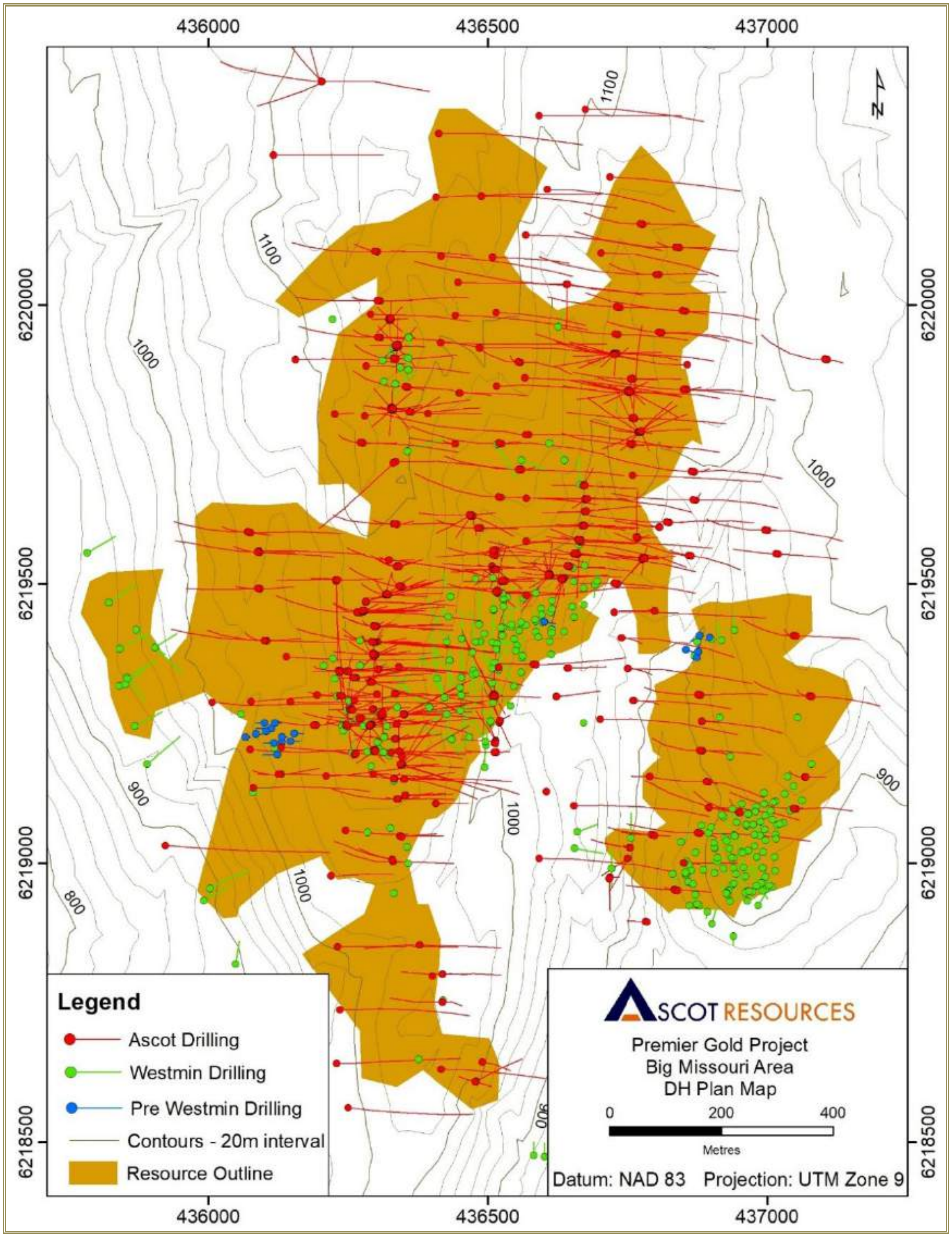


Figure 10-2: Drill Hole Plan—Big Missouri

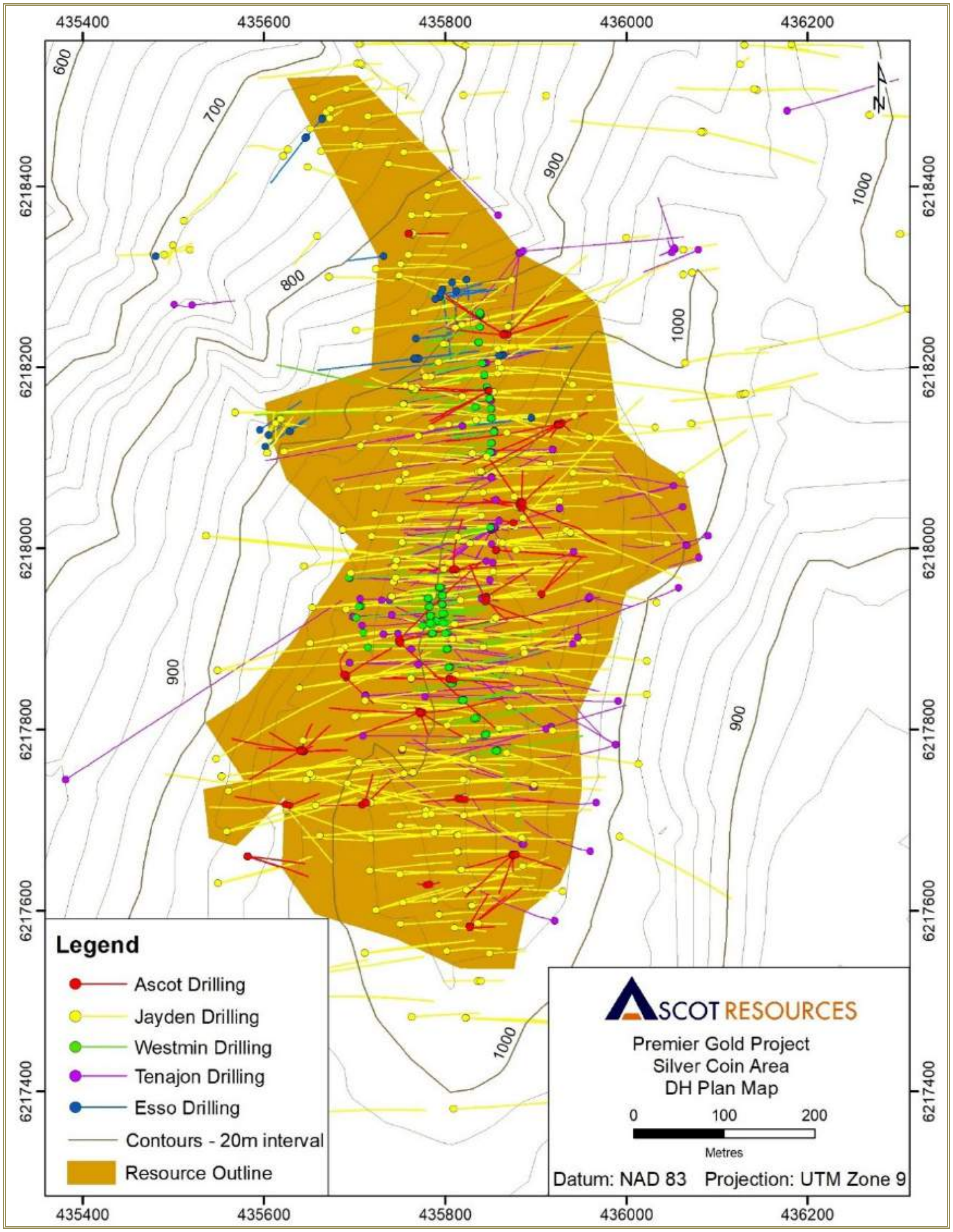


Figure 10-3: Drill Hole Plan—Silver Coin

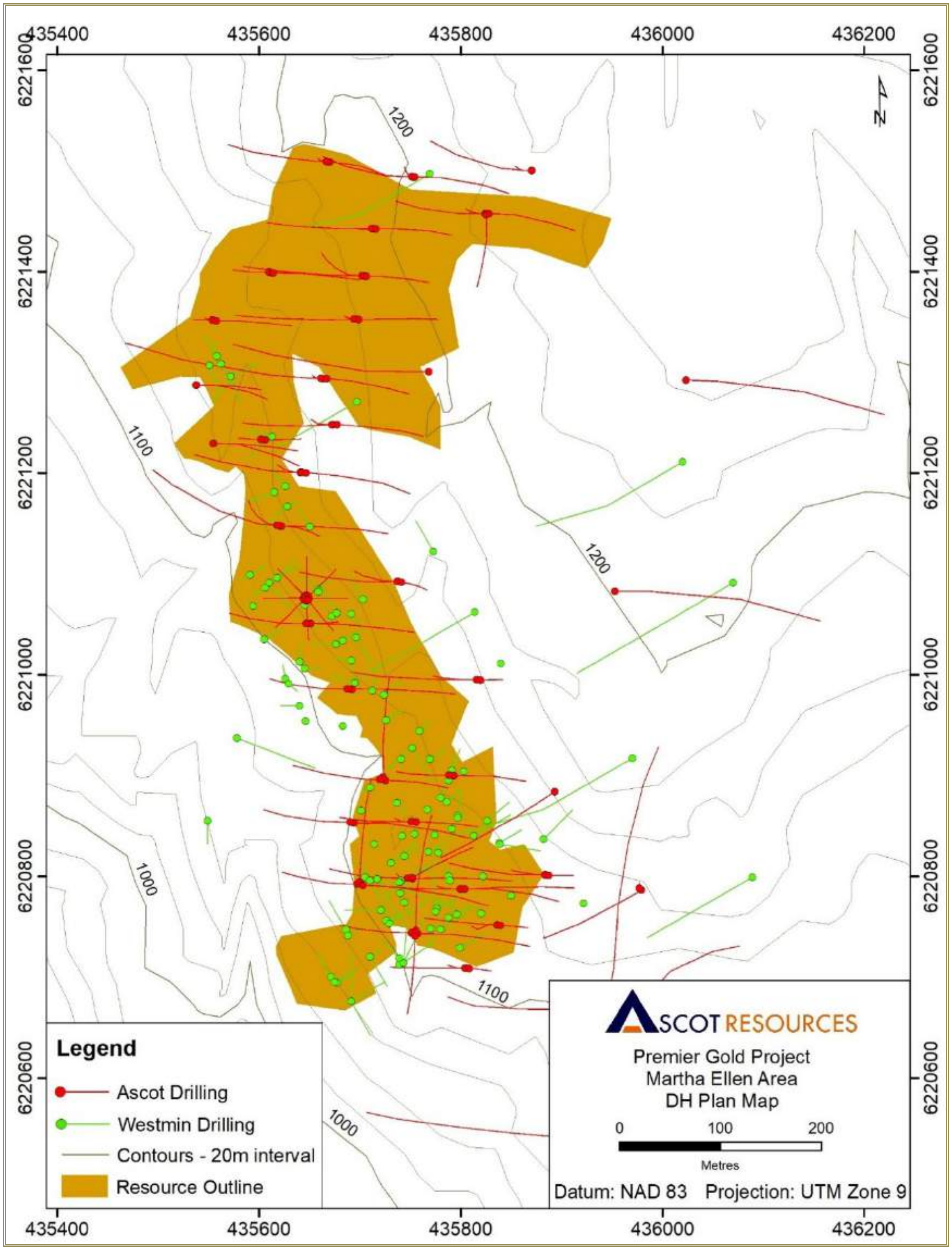


Figure 10-4: Drill Hole Plan—Martha Ellen

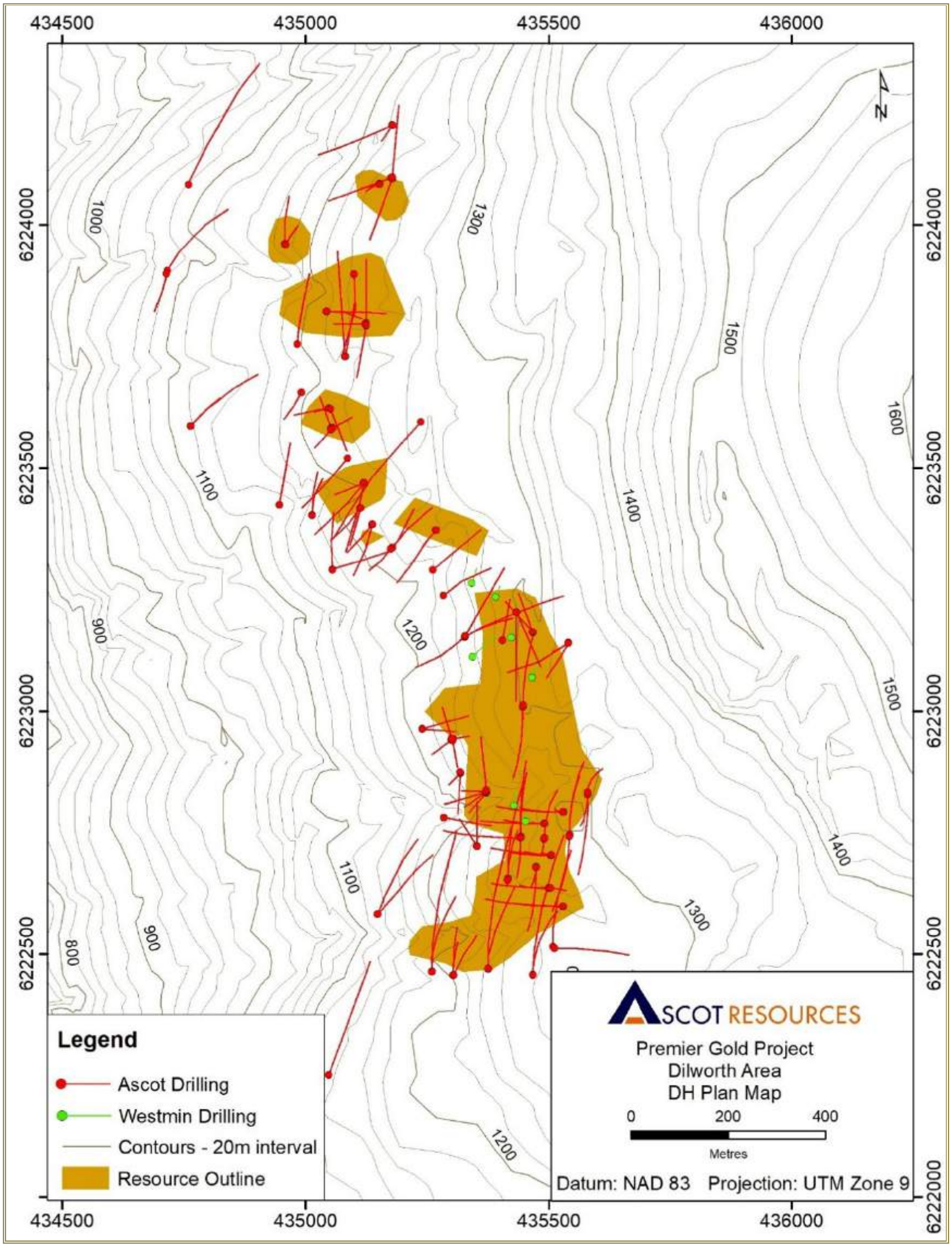


Figure 10-5: Drill Hole Plan—Dilworth

10.1.3 Drilling Methods

From 2009 to 2017 core drilling was carried out with Ascot's own drills which were purchased from Multipower Products Ltd. of Kelowna, BC between 2009 and 2011. There were seven machines, all operated by Ascot personnel, with one drill producing BQ-sized core and the other drills producing NQ-sized core.

The 2018 and 2019 drilling programs were conducted under contract by Discovery Diamond Drilling Ltd. based in Stewart, BC. Four rigs were used all producing NQ-sized core.

10.1.4 Core Handling and Logging

As the drill core was recovered, it was placed in wooden boxes by the drill helper along with a small wooden block placed at the end of every 10 ft drill run (3.048 m) to mark the depth in the hole. Once full, boxes were covered with a wooden lid and secured for transportation. Depending on the drill location, core boxes were either slung by helicopter to a waiting truck or, if the drill was at a road site, core boxes were loaded directly into the truck for transport to Ascot's secure logging facility in Stewart.

Upon delivery to the core shack, core boxes were placed on core logging benches in groups of three where the core examination and logging processes were performed. The box and block labelling were inspected for errors, and once it was assured to be correct the wooden blocks were converted to metres and the ends of the boxes marked with the corresponding metres.

Core logging included recovery and rock quality designation (RQD), geological description, and sample intervals. The geological description included rock type, alteration, structures, mineralization, and any other features the geologist considered relevant. All core was photographed for a permanent record.

Core is stored in stacks at the Premier Mill site.

10.1.5 Recovery

Core recovery for all of the Ascot drilling is very good with no significant statistical differences between the BQ and NQ core recovery. Recovery to the end of August 2019 averages 93.9.

10.1.6 Surveys

Collar Surveys

Predetermined collar locations are initially surveyed using a handheld global positioning system (GPS), typically a Garmin GPS60csx. When the hole is completed, the collars are marked by a large wooden plug with a metal tag listing the drill hole number and orientation. The collar posts are later surveyed by a land surveyor using a differential GPS to provide greater accuracy to the final results. Collar surveys are conducted approximately every four to six weeks. The difference between the handheld and differential GPS is often only few metres in the horizontal direction but sometimes over 10 m in the vertical direction.

Downhole Surveys

Downhole survey readings, measuring azimuth and inclination, were taken near the top of the hole (from 30 m to 50 m), mid-hole (100 m to 150 m), and end of hole (generally within the final 20 m of the hole) by drill personnel using a Single Shot Reflex downhole survey instrument. Magnetic susceptibility measurements are made at each survey point to check for evidence of magnetic interference. Survey readings were generally regarded as accurate and only occasional test readings were considered unreliable when there is a large discrepancy between survey readings, in which case they were removed from the data set.

Collar orientations are not generally surveyed during the exploration drill programs as it would require a surveyor to be on site at all times. During the validation of the database, it had been noted that there were a significant number of holes whose collar orientations as logged differed markedly from the first downhole survey. In some instances, this occurred in places where the holes were collared on dumps and involved a comparatively long interval of tri-cone drilling before reaching bedrock. The drills sometimes shifted when they encountered large boulders in the dump material resulting in abrupt changes in hole direction. In a few holes, there were abrupt changes in surveyed hole orientations that could be attributed to magnetic disturbances. The questionable survey measurements were removed from the database in 2018. This occurred in four holes in the Premier area and one hole at Martha Ellen.

Current drilling by Ascot has average survey intervals of about 30 m. Historical survey intervals were much larger ranging from 50 m to over 100 m. This has resulted in some inaccuracies in drill hole traces and in location of wireframe boundaries in areas dependent on historical drilling. This issue is considered to be non-material from a Resource Estimate point of view since the location of mineralization will be further refined by definition drilling prior to mining.

10.1.7 True Thickness

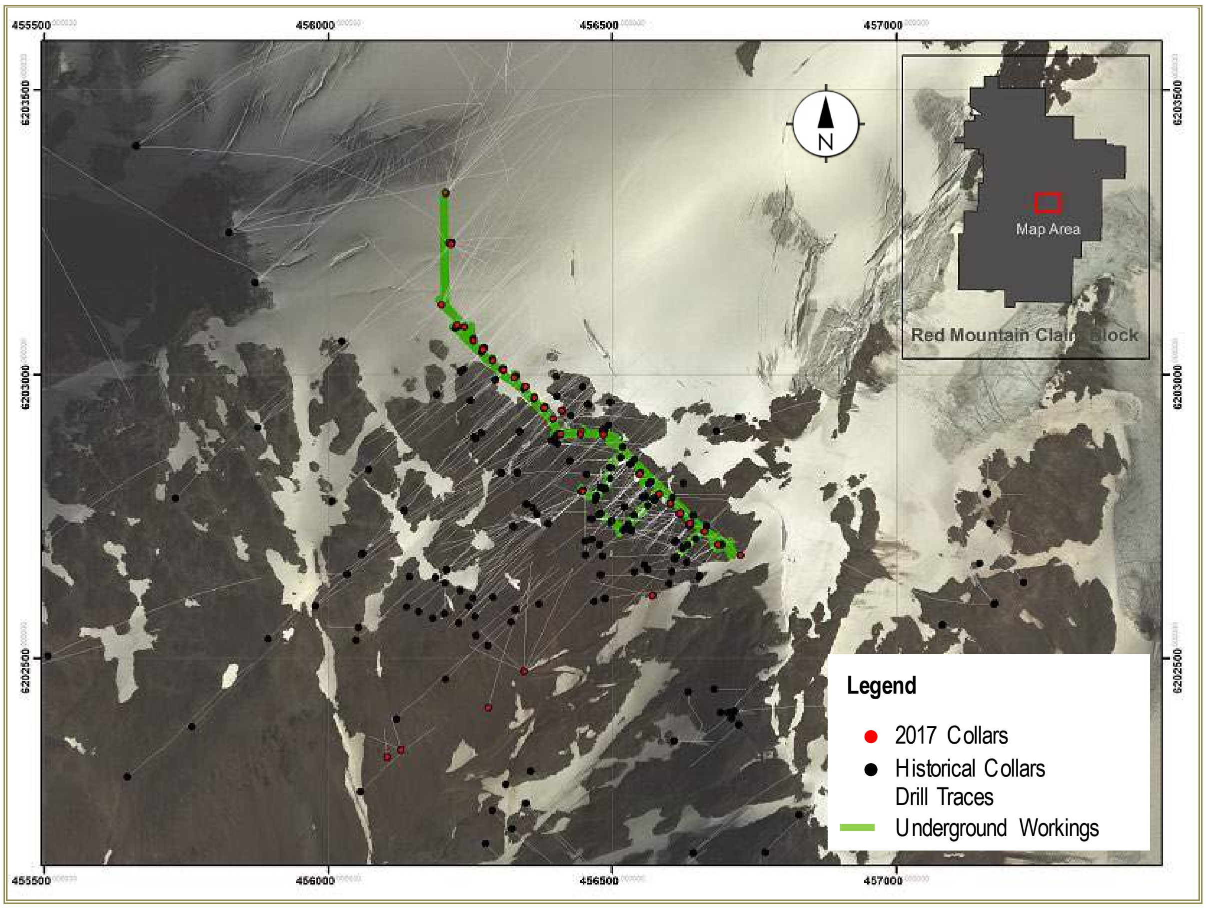
For Big Missouri, Dilworth, and Martha Ellen, most of the mineralized zones are flat to moderately dipping and estimated true widths are generally 70% to 100% of the reported drill intercepts. In the Premier and Silver Coin areas, there is a range of orientations ranging from shallowly dipping to vertical. There are many instances of holes oriented nearly parallel to the zones, which has produced some exaggerated apparent widths. In general, the alteration envelope which encompasses almost all of the mineralized zones ranges up to 20 m to 30 m in thickness. The higher-grade shoots within this envelope tend to be less than five metres thick and commonly two to three metres in true thickness. Holes drilled sub-parallel to the vein orientation are accounted for by calculating a True Thickness item of the zone, based on the strike, dip and intercept thickness. The True Thickness has then been interpolated into the block model with the resulting True thicknesses used as a criterion for resource estimation, with a lower limit of 2.5 m True Thickness.

10.2 Drilling at Red Mountain Project

10.2.1 Introduction

A total of 699 surface and underground diamond-drill holes (180,426 m) have tested a variety of targets on the Red Mountain property. Of these, 406 holes, totalling 100,298 m, were drilled by Bond and LAC between 1989 and 1994, and 60 holes totalling 29,671 m were drilled by Royal Oak in 1996. No drilling was carried out by North American Minerals Inc. (NAMC). During 2012, Banks Island completed three drill holes for 681 m in the Marc zone.

The majority of the historical drilling tested the Marc, AV, and JW zones. A total of 368 drill holes from the Bond and LAC programs, including 207 surface drill holes and 161 underground drill holes, tested these areas. The location of most drill holes on the property is shown on Figure 10-6, which is centred on the resource areas and main prospects.



Source: IDM (2017)

Figure 10-6: Red Mountain Drill Plan Resource Areas and Main Prospects

In 2014 IDM Mining Ltd. (IDM) completed 12 surface drill holes totalling 2,223 m, including two in the AV zone, three in the 141 zone, two in the Marc Extension zone, and five on exploration targets in the Cambria zone.

In 2016 IDM completed 62 holes totalling 8,123 m, including 51 underground drill holes in the Marc, AV, and JW zones, and 11 surface drill holes including five in Lost Valley, one in the Brad zone, and five in the 141 zone.

In 2017 IDM completed 116 holes totalling 29,299.26 m, including 105 underground holes in the Marc, AV, JW, SF, Bray, and Smit zones, and 11 surface drill holes targeting the 141 zone, the Marc zone, and a potential second portal site.

In 2018 IDM completed 40 holes totalling 10,021.81 m in the Marc, AV, JW, SF, NK, and Smit zones, as well as other targets.

10.2.2 Surface Drilling Contractors

The Bond and early LAC surface diamond-drilling programs from 1989 to 1991 were carried out by Falcon Drilling Ltd. of Prince George, BC, and from 1992 to 1994 by J.T. Thomas Diamond Drilling Ltd. of Smithers, BC. Both contractors used equipment suitable for producing BQTK diameter core.

The 1996 Royal Oak surface diamond-drilling program was conducted by Britton Brothers Diamond Drilling Ltd. of Smithers, BC, using equipment suitable for production of NQ and BQTK diameter core.

The Banks Island drilling program on 2013 was conducted by Driftwood Diamond drilling of Smithers, BC, using equipment suitable for production of NQ diameter core.

The 2014, 2016, and 2017 IDM surface drilling programs were conducted by MoreCore Drilling of Stewart, BC, using equipment suitable for production of NQ2 and BTW diameter core.

Nearly half of surface drill holes targeted the Marc, AV, and JW zones. All holes were drilled parallel to the mine grid section lines. The mine grid is a local grid that is rotated 45 degrees clockwise from true north. About a third of the holes were drilled at either 135° or 315° mine grid (090° or 270° true north), which is parallel to the section orientation. The remainder of the surface holes were drilled at off-section orientations. Inclinations for the holes ranged from −45° to −90°.

10.2.3 Underground Drilling Contractors

Underground drilling programs in 1993 and 1994 were carried out by J.T. Thomas Diamond Drilling Ltd. As with the surface drilling, they used equipment suitable for producing thin wall BQ core (BQTW) and NQ.

The 1996 Royal Oak underground diamond drilling program was conducted by Britton Brothers Diamond Drilling Ltd. of Smithers, BC, using equipment suitable for production of NQ and BQTK diameter core.

The 2016, 2017, and 2018, IDM underground drilling programs were carried out by MoreCore Drilling, using equipment suitable for production of HQ and NQ2 diameter core.

A majority of the underground holes were drilled parallel to the section lines, more or less equally at 090° and 270° mine grid. The remaining holes were drilled in off-section orientations. Most of the holes were drilled in fans on section, with the inclination of the holes varying from +87° to −89°.

10.2.4 Field Procedures

For the bulk of the drilling, which was carried out by LAC, field procedures included having a drill geologist who sited-in drill setups, aligned drills, and visited each drill one or more times a day. Continuous monitoring of the drills ensured any drilling problems were noted, and helped to ensure that good core-handling practices were maintained by all drill crews. Royal Oak field procedures are not known. IDM geologists monitored their drilling operations and visited the drill at least once a day.

10.2.5 Core Logging

Bond and LAC Logging

All core was flown down to Stewart for logging and sampling. Most core was logged for geotechnical purposes by a geological technician before it was logged geologically. All logging was done onto a series of paper logging forms:

- Geotech log: Recovery, RQD fracture count, hardness, and fracture filling. Carried out by a geological technician.
- Geological log: Intervals (primary and nested), geological code and description, alteration intensity and character, graphic log. Carried out by a geologist.
- Sample log: Interval, sample number, sample description and mineralization by percent. Samples were marked and tagged by a geologist.

LAC also employed the use of a quick log, completed by the geologist who was monitoring drilling operations before the core was flown to Stewart. The quick log was used for initial interpretation and ongoing drill program planning.

As there were several different people logging core, considerable time was spent trying to standardize logging procedures and data inputs. However, some variance in logging due to different people logging and changes in understanding of the deposit proved apparent when reviewing the various logs.

Royal Oak Logging

Royal Oak logged and sampled their core at the camp on Goldslide Creek. They also used paper logging forms, one for geotech and the same geological logging form that LAC used, but the alteration codes were not used, only written descriptions. There is no written evidence of the sample intervals or sample numbers in their drill hole log files, only computer printouts with intervals, sample numbers, and results. None of the Royal Oak holes are within resource areas.

NAMC Logging

During 2000 and 2001, in preparation for resource estimation, NAMC re-logged all core within the Marc, AV, and JW mineralized zones, including a 20 m envelope outside of the mineralized zones. The purpose of the re-logging was to establish continuity of logging procedures, verify past logging data entry, and determine continuity between sections. If mineralized continuity was not geologically determined between 25 m sections, the mineralization was removed from the geological solids and excluded from resource interpolation.

Banks Island Logging

It is not known how or where Banks Island carried out their core logging and sampling. Detailed logs are presented in the 2013 assessment report, and include a header page with hole information and surveys, and pages with geology, alteration, mineralization, geotechnical, and sampling data.

IDM Logging

IDM logged and sampled their core at the camp on Goldslide creek. Underground and core is delivered by the drillers to the portal area and slung by helicopter to the core shack. Surface drill core was also slung. Only rarely, in inclement weather, was core delivered to the core shack using a side-by-side ATV. Upon receipt from the drill, core boxes were examined to ensure the hole number and box numbers were correct. The drillers' depth markers were checked, and any discrepancies corrected.

Logging was carried out by directly entering data onto computers using a customized Microsoft Access drill-hole database which includes all standard tables. Each table contained drop-down pick list menus that were locked so codes could not be added without administrative consent. Geotechnical information included basic recovery and RQD measurements collected between drill-run marker blocks. Samples were laid out by a geologist, respecting geological boundaries.

10.2.6 Recovery and Rock-Quality Designation

Core recovery and RQD has been measured by all operators. Core recovery is very good, ranging from 96.73% to 98.15% from different operators. RQD percentages do vary, from 61.35% for all pre-IDM operators to 80.26% for the 2016 and 2017 IDM programs. This is most likely a reflection of the core diameter used. Royal Oak and LAC used BQ and BQTK diameter core, respectively, and IDM has used NQII or HQ diameter core. Ground conditions are generally very solid.

10.2.7 Drill Collar and Down Hole Surveys

Drill Collar Surveys

The collar coordinates for all Bond and LAC drill holes were surveyed using a total station. For the 1989 Bond holes and most of the 1993 and 1994 underground holes, collar orientations were determined by surveying while the rods were in the hole or by surveying a rod placed in the drill hole after the rig had moved. As rock conditions underground were good, there was typically a snug fit of the rod within the abandoned hole. Underground surveying was done every one to two weeks.

For most Bond and LAC surface drill holes from 1990 to 1993, the collar orientations appear to be ideal setup orientations as shown in Table 10-1. For 1994 surface drill holes the first down-hole survey orientation was used for collar orientation.

Most, or all, of the pre-1993 collars were re-surveyed with a total station by LAC, and the collar locations from the new surveying were used in the database. Pre-1993 survey coordinates were documented. Surveying in 1993 and 1994 was routinely checked.

The Royal Oak collar locations, both underground and surface, were also surveyed using a total station, although for multiple holes drilled from the same setup the same collar coordinates were entered into the database for each hole. About 25% of the underground collars have surveyed collar coordinates, with the remainder and all of the surface holes using ideal setup orientations.

All three Banks Island drill holes were completed from a single pad. How the pad was located and surveyed is not known.

For the 2014 IDM program drill holes were initially located by hand-held GPS for pad preparation. A second hand-held GPS reading was taken later of the actual collar. Ideal collar orientations were entered for holes with no downhole surveys.

Collar locations for almost all of the IDM 2016, 2017, and 2018 underground and surface drill holes were surveyed using a total station. Where possible, the collar orientation was surveyed, either while the drill was on the hole or afterwards by placing a rod in the hole after the drill rig had moved. Some 2016 drill holes have collar orientations from gyro surveys. A few holes with no surveys of either type had ideal collar orientations entered in the database.

Downhole Surveys

With the exception of the 1989 drill holes and a few of the 1990 drill holes, which had acid dip tests, most holes drilled on the property until 1996 have Sperry Sun surveys, the predominant down-hole survey technique at the time. Banks Island used a Reflex Easy Shot instrument and collected the surveys after the hole was completed. For their 2014 program IDM used a Ranger multi-shot survey instrument, but no surveys were obtained for six of the twelve holes. The IDM 2016 drilling program used a combination of a Reflex multi-shot surveys and Reflex Gyro surveys. Only Reflex multi-shot surveys were used in 2017. Details of the down-hole surveys and collar surveys for all programs given in Table 10-11.

During the LAC programs, the drill geologist generally aided in the Sperry Sun surveying. Sperry Sun photographs were read by the geologist and then checked in the Stewart office. Survey readings that were suspect were not used. Locally, pyrrhotite content is high enough that it could cause a deflection of the Sperry Sun compass. The Sperry Sun photographs were kept, and most from the LAC and Royal Oak programs are available for review.

Table 10-11: Details of Collar and Downhole Surveys

Year	Company	Surface or UG	Collar Location	Collar Orientation	Survey Type	Comments
1989	Bond	S	Y	Y	Acid	Acid dip tests only
1990	Bond	S	Y	N	Sperry	~90 m spacing, ideal collar coordinates
1991	LAC	S	Y	N	Sperry	~90 m spacing, ideal collar coordinates
1992	LAC	S	Y	N	Sperry	~90 m spacing, ideal collar coordinates
1993	LAC	S	Y	N	Sperry	~60 m spacing, ideal collar coordinates
1993	LAC	UG	Y	Y for most	Sperry	Some holes <80 m in length have no surveys. Holes >100 m have surveys every 60 m or at the bottom of the hole.
1994	LAC	S	Y	N	Sperry	First at ~15 m then every 60 m, data from first test used for collar.
1994	LAC	UG	Y	Y for most	Sperry	First at ~15 m depth then every 30 m

Year	Company	Surface or UG	Collar Location	Collar Orientation	Survey Type	Comments
1996	Royal Oak	S	Y	Y for ~25% of holes	Sperry	Variable spacing, 50 m to 100 m or more
1996	Royal Oak	UG	Y	N	Sperry	Variable spacing, 50 m to 100 m or more
2013	Banks Island	S	?	N	Reflex	Every 31 m
2014	IDM	S	Y	Y & N	Ranger MS	Readings taken every 6 m. If surveyed there are collar coordinates otherwise ideal cords were entered.
2016	IDM	S	N	N	Reflex	Reflex every 6 m
2016	IDM	UG	Y	Y & N	Reflex, Gyro	Reflex or Gyro every 3 m
2017	IDM	S	Y	Y & N	Reflex	Every 3 m
2017	IDM	UG	Y	Y & N	Reflex	Every 3 m
2018	IDM	UG	Y	Y & N	Reflex	Every 3 m

10.2.8 Drill Hole Adjustments

During NAMC's preparation of the 2000 Red Mountain geological model it became apparent that a number of drill holes did not fit well with the majority of drill-hole data. After examination of the Gemcom database, diamond-drill hole logs, Sperry Sun readings, cross sections, and level plans, the following problems were encountered, and corrections made. Full details of the drill-hole corrections can be found in NAMC's 2001 Red Mountain resource report by Craig (2001).

- The Sperry Sun surveys for a single 1993 underground hole had been misread. Correct readings were taken, and the values entered into the database.
- For most of the 1989 drill holes and two 1990 drill holes only acid dip tests were taken, and for two 1990 drill holes no down-hole survey information was collected. Average down-hole deviations were calculated by using data from the Sperry Sun tests conducted on a majority of 1990 drill holes, as these holes were drilled in similar orientations to the holes lacking survey data. An average azimuth deviation of +2.2° per 100 m and an average dip deviation of +0.4° per 100 m was calculated. The azimuth deviation was applied to fifteen 1989 holes at depths where the acid tests were taken. Both deviations were applied to one 1989 hole and two 1990 holes that had no downhole survey information, at 100 m intervals.
- Six holes did not fit with known geological data, so the survey data for these holes was adjusted until they corresponded to the known data.

10.2.9 Sample Length / True Thickness

The relationship between sample length, or intersection length, and true width depends upon the angle at which mineralization is intersected. As this varies due to the location from which the drill hole can be completed, on the dip of the drill hole, and on the orientation (strike and dip) of the mineralization, drill intersection lengths at Red Mountain are typically greater than true widths.

10.2.10 Drill Spacing

Drill spacing on the Red Mountain Project (RMP)is variable depending on the stage of exploration or development of a particular zone.

Sectional spacing for the both underground and surface drilling for the Marc, AV, and JW zones is 25 m. On section, drill-hole spacing is typically less than 25 m for the Marc zone, and 25 to 30 m for the AV and JW zones.

Other zones with resource potential, such as the 141 and Smit zones, also have variable drill spacing. The core of the 141 zone has been defined on 25 m centres, with both strike extensions spaced at 50 m, with sectional spacing at 30 m or less. The Smit zone has 25 m sectional spacing and 30 m to 50 m spacing on section.

10.2.11 Drill Intercepts

Table 10-12 shows a selection of intersections through the main resource zones to illustrate typical grades and widths of the deposit.

Table 10-12: Typical Drill Intersections

Zone	Section	Hole ID	From	To	Length	Au g/t	Ag g/t
Marc	1125N	M93123	143.50	151.50	8.00	12.68	32.16
Marc	1175N	931020	74.70	91.50	16.80	9.06	5.83
Marc	1200N	930176	16.00	24.00	8.00	6.02	40.45
Marc	1250N	931070	49.8	65.8	16.00	26.82	195.42
Marc	1300N	M9164	306.00	312.00	8.00	6.39	1.87
AV	1350N	931074	59.00	76.00	17.00	8.16	20.35
AV	1400N	941116	110.00	129.00	19.00	4.50	35.23
AV	1450N	941106	75.00	104.00	29.00	4.66	8.48
AV	1475N	M9278	388.25	392.90	4.65	5.77	13.11
JW	1525N	941141	125.50	129.5	4.00	6.93	51.60
JW	1575N	M93140	487.00	494.00	7.00	2.02	2.11
JW	1600N	941124	172.70	175.70	3.00	6.64	NA
141	1275N	MC14-003	143.50	152.50	9.00	3.52	6.03
141	1325N	M94186	153.00	189.80	36.80	3.32	NA
141	1350N	M93141	168.61	200.00	31.39	4.12	13.94
SF	1725N	U17-1247	241.00	247.61	6.61	5.93	5.06
Bray	1900N	U17-1308	655.30	657.93	2.63	6.34	11.18
Smit	1450N	U17-1287	241.50	280.00	38.50	3.00	0.44
Smit	1475N	U17-1278	220.00	234.50	14.50	2.27	0.91
Cambria	-	CB14-001	41.30	47.00	5.70	5.67	2.80
NK	1075N	U17-1302	105.25	108.00	2.75	12.12	7.44

10.2.12 Comments

In the opinion of the responsible QP, the quantity and quality of the geological, geotechnical, collar, and downhole survey data collected by the past and present operators on the RMP are sufficient to support mineral resource estimation as follows:

- Drilling procedures and core logging meets industry standards
- Recovery data from drill core data are acceptable
- Collar surveys have been performed using industry-standard instrumentation
- Downhole surveys were collected at the time of the programs using industry-standard instrumentation
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the resource areas
- Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths
- Drill spacing has been adequate to first outline, then infill and define mineralized zones. Drill-hole spacing does vary with the stage of exploration and development
- Drill-hole intercepts as summarized in Table 10-11 appropriately reflect the nature of the gold mineralization, and include areas of higher-grade intervals in low-grade drill intercepts
- No factors were identified with the data collection from the drill programs that could materially affect resource estimation accuracy or reliability.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Premier Gold Project

11.1.1 Legacy Drilling

As stated in Section 10, complete documentation has not been found on the drilling, sampling and assaying protocols for the work done prior to Ascot's involvement: in 2007 for Premier, Big Missouri, Dilworth, and Martha Ellen, and in 2018 for Silver Coin. There are some references in Assessment Reports for each project that describe some details of the sampling and assaying. It is also possible to infer from the database what the sampling strategy was.

Due to the lack of information for the legacy drilling at all properties, the data have been verified by an extensive re-assay program of pulps and core. These analyses are presented in Section 12.1.3 for the Westmin drilling, applicable to all areas for drilling from 1978 to 1994. Silver Coin legacy drilling uniquely includes drilling by Tenajon and Jayden and MBM and has been verified by a re-assay program done in 2018, also discussed in Section 12.1.4. Data from Premier from 1928–1941 are not available and has not been used in the resource estimate.

In all cases relevant to legacy drilling the conclusion is that grades within the range applicable to this study have been validated and may be used for Resource Estimation.

Table 11-1 to Table 11-5 provide summaries of the sample widths for the legacy holes compared to the Ascot holes for each of the five PGP deposits.

Table 11-1: Sampling Comparison—Historical and Ascot Assays—Premier

Year	Metres Sampled	Number of Samples	Minimum Sample Length (m)	Maximum Sample Length (m)	Average Sample Length (m)
1980	440	439	0.15	3.54	1.00
1981	1,887	965	0.15	4.87	1.96
1983	771	448	0.80	4.24	1.72
1984	1,170	751	0.30	4.12	1.56
1985	2,095	1,303	0.30	6.40	1.61
1986	5,542	3,414	0.30	6.10	1.62
1987	7,493	4,725	0.15	6.10	1.59
1988	5,382	3,798	0.20	3.60	1.42
1989	1,493	1,133	0.25	3.40	1.32
1990	2,535	1,712	0.30	3.10	1.48
1991	565	561	0.10	2.30	1.01
1992	934	782	0.30	3.20	1.19
1996	8,663	7,550	0.09	6.10	1.15
Subtotal – Historical	38,971	27,581	0.09	6.40	1.95
2009	773	687	0.20	2.36	1.12
2012	36	16	1.00	2.50	2.28

Year	Metres Sampled	Number of Samples	Minimum Sample Length (m)	Maximum Sample Length (m)	Average Sample Length (m)
2013	248	114	0.85	3.29	2.18
2014	10,252	5,904	0.42	3.95	1.74
2015	13,948	8,153	0.47	3.80	1.71
2016	12,087	7,095	0.48	6.41	1.70
2017	25,254	15,033	0.22	8.63	1.68
2018	2,738	1,667	0.87	2.50	1.64
2019	3,463	2,264	0.57	3.10	1.53
Subtotal – Ascot	68,801	40,933	0.20	8.63	1.78

Table 11-2: Sampling Comparison—Historical and Ascot Assays—Big Missouri

Year	Metres Sampled	Number of Samples	Minimum Sample Length (m)	Maximum Sample Length (m)	Average Sample Length (m)
1974	21	9	0.50	6.10	2.35
1976	77	49	0.60	3.50	1.58
1978	383	261	0.31	3.29	1.47
1979	495	336	0.40	2.40	1.47
1980	1,381	854	0.27	3.23	1.62
1981	1,084	590	0.56	4.48	1.84
1982	1,467	800	1.00	4.60	1.83
1984	185	122	0.60	2.60	1.52
1986	826	507	1.00	3.00	1.63
1987	1,929	1,238	0.15	3.66	1.56
1988	3,356	2,320	0.30	8.20	1.45
1989	654	411	0.70	4.57	1.59
Subtotal – Historical	11,859	7,497	0.15	8.20	1.58
2009	3,012	2,526	0.22	3.27	1.19
2010	17,188	11,672	0.18	4.61	1.47
2011	33,026	18,146	0.22	9.83	1.82
2012	20,405	10,546	0.14	8.00	1.93
2013	10,338	5,239	0.40	3.29	1.97
2014	2,514	1,315	0.72	3.85	1.91
2017	781	488	0.65	2.66	1.60
2018	11,661	6,946	0.44	4.48	1.68
2019	11,910	7,459	0.37	8.88	1.60
Subtotal – Ascot	110,835	64,337	0.14	9.83	1.72

Table 11-3: Sampling Comparison—Historical and Ascot Assays—Silver Coin

Year	Metres Sampled	Number of Samples	Minimum Sample Length (m)	Maximum Sample Length (m)	Average Sample Length (m)
1982	850	481	0.15	7.50	1.77
1983	754	356	0.10	4.00	2.12
1986	355	252	0.20	3.40	1.41
1987	1,836	1,446	0.07	7.60	1.27
1988	3,472	2,623	0.06	9.32	1.32
1989	2,349	1,613	0.20	44.60	1.46
1990	6,514	5,723	0.15	6.10	1.14
1993	2,208	1,564	0.29	8.30	1.41
1994	3,496	2,413	0.20	2.50	1.45
2004	2,282	1,428	0.10	4.13	1.60
2005	7,601	3,123	0.30	15.25	2.43
2006	23,669	9,987	0.01	6.10	2.37
Subtotal – Historical	55,385	31,009	0.01	44.60	2.63
2007	2,639	925	0.61	6.50	2.85
2008	12,024	4,437	0.61	11.89	2.71
2009	990	330	1.06	6.10	3.00
2010	3,023	1,862	0.06	12.19	1.62
2011	16,676	12,921	0.04	16.05	1.29
2017	1,981	1,066	0.50	10.46	1.86
2018	1,305	820	0.70	3.05	1.59
2019	7,078	4,267	0.50	6.10	1.66
Subtotal – Ascot	45,717	26,628	0.04	16.05	2.04

Table 11-4: Sampling Comparison—Historical and Ascot Assays—Dilworth

Year	Metres Sampled	Number of Samples	Minimum Sample Length (m)	Maximum Sample Length (m)	Average Sample Length (m)
1981	221	124	0.60	2.99	1.78
sub-total – Historical	221	124	0.60	2.99	1.78
2007	3,466	2,989	0.20	4.77	1.16
2008	8,978	5,669	0.12	7.20	1.58
2010	3,731	2,342	0.28	3.19	1.59
2011	1,253	698	0.42	4.16	1.80
2012	4,346	2,131	0.37	3.79	2.04
2013	3,083	1,578	0.57	3.97	1.95
Subtotal – Ascot	24,857	15,407	0.12	7.20	1.61

Table 11-5: Sampling Comparison—Historical and Ascot Assays—Martha Ellen

Year	Metres Sampled	Number of Samples	Minimum Sample Length (m)	Maximum Sample Length (m)	Average Sample Length (m)
1981	24	13	1.49	2.98	1.87
1982	278	151	0.90	5.18	1.84
1983	331	192	0.70	4.60	1.73
1986	511	324	1.10	2.50	1.58
1987	1,463	933	0.79	2.99	1.57
1988	1,541	1,067	0.40	34.10	1.44
1996	339	415	0.25	2.40	0.82
Subtotal – Historical	4,486	3,095	0.25	34.10	1.45
2009	1,712	1,196	0.14	2.36	1.43
2010	604	316	0.29	5.28	1.91
2012	7,690	3,886	0.33	6.13	1.98
2013	5,048	2,383	0.59	3.70	2.12
2017	1,161	618	0.70	3.34	1.88
2018	271	190	0.65	2.85	1.42
Subtotal – Ascot	20,971	11,684	0.14	34.10	1.79

Table 11-1 to Table 11-5 illustrate that samples lengths of the legacy data are similar to those in practice recently, with the longer assay intervals within unmineralized material.

Two of the Assessment Reports reviewed mention that the legacy core was split but did not state the method used (i.e., splitter or saw). There are also two instances where it was stated that the samples were analyzed at the Premier Mine laboratory. These samples were oven dried, passed through a jaw crusher to -1/4", cone crushed to -1/8", and split with a riffle down to a 250 g subsample that was ground in a ring and puck pulverizer. A half assay ton aliquot was taken from this pulp and subjected to fire assay (FA) for gold with a gravimetric finish. A separate aliquot was taken and analyzed by atomic absorption (AA) for silver, lead, zinc, and copper.

No references are made to an independent assay quality assurance and quality control (QA/QC) program. In one instance it is stated that a selection of duplicate samples was sent to an outside laboratory for checks, Min-En Laboratories Ltd., in Vancouver, BC.

Silver Coin

Esso—1982 to 1983

It is unknown which laboratory or what standards were used by Esso for the 1982–1983 drilling. Due to the lack of QA/QC, the Esso era data have not been used in the Classification of the Silver Coin resource.

Tenajon—1986 to 1990

The Tenajon analyses were completed at several different laboratories over the years, including the NewCana Laboratory in Stewart, BC until 1988 and Eco-Tech Laboratory of Kamloops, BC, which was used for check assays. NewCana was a joint venture between Newhawk Gold Mines Ltd., Lacana Mining Corp., and Granduc Mines Ltd., and was conducting exploration in the Stewart area at the time. The assays in 1989 and 1990 were performed by Eco-Tech, which later became part of the ALS Minerals (ALS) Laboratory group. Due to lack of QA/QC analyses for the Tenajon era drilling, core that has been stored on site was re-assayed in 2019 to ensure that there is no bias to this data. The results are presented in Section 12 of this report.

Westmin—1990 to 1994

The Westmin samples from 1990 to 1994 were managed in the same manner as described above for the other four sites. The data validation done for the Westmin drilling at the other sites and presented in Section 12 of this report is considered to also validate the drilling by Westmin at Silver Coin since the same procedures and the same lab were used.

Jayden and MBM Drilling—2004 to 2017

The Jayden and MBM assaying were completed using certified laboratories and included duplicate sample splits of core as well as pulp splits. The 2004 to 2008 assaying was done at Assayers Canada. Assayers Canada laboratory is described below in the section of this report describing Ascot assay protocols.

From 2009 to 2011, the analyses for Jayden were completed at Inspectorate Laboratories (Inspectorate), now part of the Bureau Veritas group of laboratories (Bureau Veritas). Bureau Veritas has ISO 9001:2008 certification. The specific Inspectorate laboratory codes describing the assay procedures are as follows:

- Au-1AT-AA Au, Ore Grade, 4 Acid, AA – FA (one assay ton) with AA finish
- 30-4A-TR 30 element, 4 Acid, inductively coupled plasma (ICP), Trace Level – Four acid dissolution with ICP detection
- Zn-4A-OR-AA, Zn, Ore Grade, 4 Acid, AA – Four acid dissolution with AA detection of zinc.

The 2017 drilling analyses were done by Activation Laboratories Ltd. (Actlabs) of Kamloops, BC, which is ISO 17025 accredited and/or certified to 9001:2008. The determinations were completed using FA for gold with AA finish. As well, aqua-regia digestion with ICP mass spectrometry (ICP-MS) detection was used for silver and other elements.

11.1.2 Analytical and Test Laboratories for Ascot Drilling at Premier Gold Project—2007 to 2019

Assayers Canada—2007 to 2010

Assayers Canada, located in Vancouver, BC, was used as the primary assay laboratory from 2007 through 2012. In June 2009, Assayers Canada received ISO 9001 certification for Quality Management Systems. Data from the laboratory were provided through email in .csv files and as .pdf certificates.

SGS Minerals Services Canada—2011 to 2012

On July 12, 2010, Assayers Canada became part of SGS Minerals Services (SGS), which was retained as the laboratory for PGP. SGS received ISO 17025 certification for General Requirements for the Competence of Testing and Calibration Laboratories.

ALS Laboratories—2013 to 2019

ALS, Vancouver BC, has been used periodically for analyzing check assays in 2011 as part of the QA/QC procedures. In August 2012, ALS became the principal assay laboratory, with SGS retained to provide check assays as well as specific gravity (SG) determinations. ALS has developed and implemented at each of its locations a quality management system (QMS) designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards.

The QMS operates under global and regional quality control (QC) teams responsible for the execution and monitoring of the quality assurance (QA) and QC programs in each department on a regular basis. Audited both internally and by outside parties, these programs include, but are not limited to, proficiency testing of a variety of parameters, ensuring that all key methods have standard operating procedures (SOP) that are in place and being followed properly, and ensuring that QC standards are producing consistent results.

ALS maintains ISO registrations and accreditations. ISO registration and accreditation provides independent verification that a QMS is in operation at the location in question. Most ALS laboratories are registered or are pending registration to ISO 9001:2008, and a number of analytical facilities have received ISO 17025 accreditations for specific laboratory procedures.

11.1.3 Sampling Methods at Premier Gold Project

The following descriptions of the sampling and analytical work for the Dilworth-Big Missouri-Martha Ellen areas are taken from Simpson (2014). This work spans the period from 2007 to 2013. During that time, only five holes were drilled by Ascot in the Premier area and none in Silver Coin.

Sample coverage was designed to cover all quartz stockwork and surrounding pervasive alteration. The sample intervals could be as small as 20 cm to still provide enough material for the laboratory, or as long as 2.5 m for NQ core and 3.0 m for BQ core. Sample breaks were also inserted by the geologist at changes in the rock type. Once all information was collected, the core was stacked inside the core shack, to await cutting.

The NQ-sized core samples were sawn in half with a gas powered, diamond-bearing saw and BQ-sized core was split in half with a hydraulic splitter. Due to the smaller size of the BQ-sized core, it was decided that too much material was lost with cutting so it was better to process with a mechanical splitter. Also, because the BQ core was often irregular in shape, only the NQ-sized core were used as duplicates in the sampling process. For both methods one half of the sampled core was placed back in the box while the other half was placed in poly sample bags along with the sample tag.

Assayers Canada—2007 to 2010

Drill core samples were dried and crushed to 75% passing 2 mm and pulverized to 75 µm. All gold analyses were performed by conventional FA with AA finish. Over-limit values (generally >10 g/t Au) were analyzed using a gravimetric finish. Metallic gold assays were carried out in cases of identified visual gold.

Silver analyses were by ICP atomic emission spectroscopy (ICP-AES) as part of a 30-element package. Over-limit silver values (>200 g/t Ag) were analyzed by AA with 4-acid digestion.

SGS Canada—2011 to 2012

Drill core samples were dried and crushed to 75% passing 2mm and pulverized to 75 µm. All gold analyses were performed by conventional FA with AA finish. Over-limit values (generally >10 g/t Au) were analyzed using a gravimetric finish. Metallic gold assays were carried out in cases of identified visual gold or for assays exceeding 100 g/t Ag.

Silver analyses were by ICP-AES as part of a 34-element package. Over-limit silver values (>200 g/t Ag) were analyzed by AA with four acid digestion.

ALS Laboratories—2013 to 2019

All gold analyses were performed by conventional FA with AA finish. Over-limit values (>10 g/t Au) were analyzed using a gravimetric finish. Metallic gold assays were carried out in cases of identified visual gold.

Silver analyses were by ICP-AES as part of an ICP-AES 41 element package. Over-limit silver values (>100 g/t Ag) were analyzed using ALS procedure Ag-OG46 (aqua regia digestion, ICP-AES finish).

ALS maintains ISO registrations and accreditation with ISO 9001:2008 and ISO 17025 accreditation for specific laboratory procedures.

11.1.4 Quality Assurance and Quality Control Ascot Drilling at Premier Gold Project—2007 to 2019

This data presented in the following sections (11.5 through11.12) applies to all drilling done by Ascot on the properties it owned at the time, which include Premier, Big Missouri, Martha Ellen, and Dilworth from 2007–2019, and Silver Coin, from 2017–2019.

Ascot has maintained a fairly consistent program of independent assay QA/QC since 2007. The programs include the addition of CRM, blanks, and duplicates to the sample stream, as well as pulps sent from the principal laboratory to a secondary laboratory for checks. Control samples are added at a nominal rate of one for every ten samples, with blanks and standards alternated and the grade range of the CRM continually rotated. Quarter-core field duplicates were nominally taken every 30th sample, always from an obviously mineralized zone. Typically, a group of 100 samples shipped to the laboratory would contain five blanks and five standards, and two or three field duplicates depending on the sequence. Upon receiving the assay QA/QC analyses, a project geologist reviewed them for failures. If more than three control samples from a work order failed, then the batches containing the failures were rerun.

Table 11-6 summarizes the QA/QC by year and presents which areas were drilled with number of drill holes. A discussion of results for these programs follows in Sections 11.5 through 11.16.

Table 11-6: Summary of QA/QC—Ascot Drilling 2007 to 2019

Year	Area	Drill Holes	Blanks	Standard Samples	Field Duplicates
2019	Big Missouri	147	827	835	497
	Silver Coin	81			
	Premier	30			
2018	Big Missouri	194	455	447	189
	Premier	53			
	Silver Coin	13			
	Martha Ellen	10			
2017	Premier	359	88	927	201
	Big Missouri	10			
	Martha Ellen	10			
2016	Premier	279	330	361	23
2015	Premier	198	467	407	48
2014	Premier	149	416	423	133
	Big Missouri	20			
2013	Big Missouri	76		477	
	Martha Ellen	49			
	Dilworth	17			
	Premier	4			
2012	Big Missouri	93	2068	1911	995
	Martha Ellen	54			
	Dilworth	19			
	Premier	1			
2011	Big Missouri	144			
	Dilworth	6			
2010	Big Missouri	52			
	Dilworth	12			
	Martha Ellen	4			
2009	Premier	20			0
	Big Missouri	18			
	Martha Ellen	10			
2008	Dilworth	63			0
2007	Dilworth	36			0

11.1.5 Quality Assurance and Quality Control—Premier, Big Missouri, and Silver Coin—2019

The 2019 drilling campaign of drilling in Premier, Big Missouri and Silver Coin resulted in a total of 17,168 assay samples, of which 4.8% were certified reference materials, 4.8% were blanks, and 5.8% were paired field duplicates, meeting the standard for a sampling program. Analysis of these sample assay results implies acceptability of the 2019 assay database.

Blanks—2019

Figure 11-1 shows the assay results of the blanks for gold. Ten of the 827 samples exceeded five times the detection limit. It was determined that these samples did follow samples of high gold values, for instance the blank with assay value of 0.121 g/t followed a sample with assay value 397 g/t. This indicates there was a minor problem with contamination. There is a minor problem with drift at the end of the stream, with blanks after the 80th, increasing slightly, but not to a significant level. There were no blanks for silver assays exceeding 1 g/t.

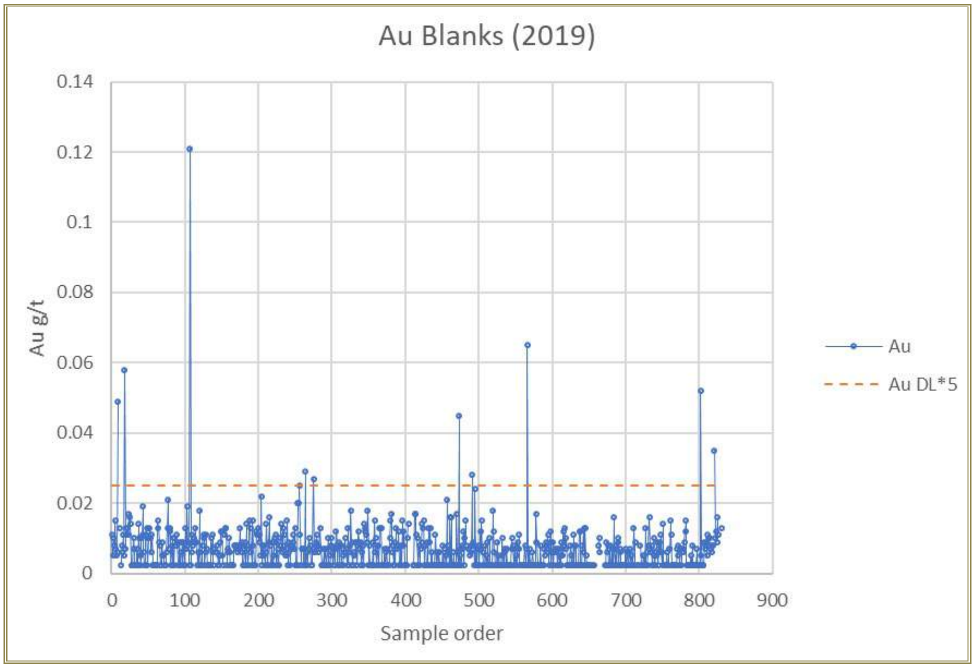


Figure 11-1: Sample Control Chart—Blanks for 2019—Gold

Field Duplicates—2019

Figure 11-2 shows ranked half absolute relative difference (HARD) values of the field duplicates for gold. This data gives only 43% under 10% HARD which indicates highly variable gold mineralization within the deposit.

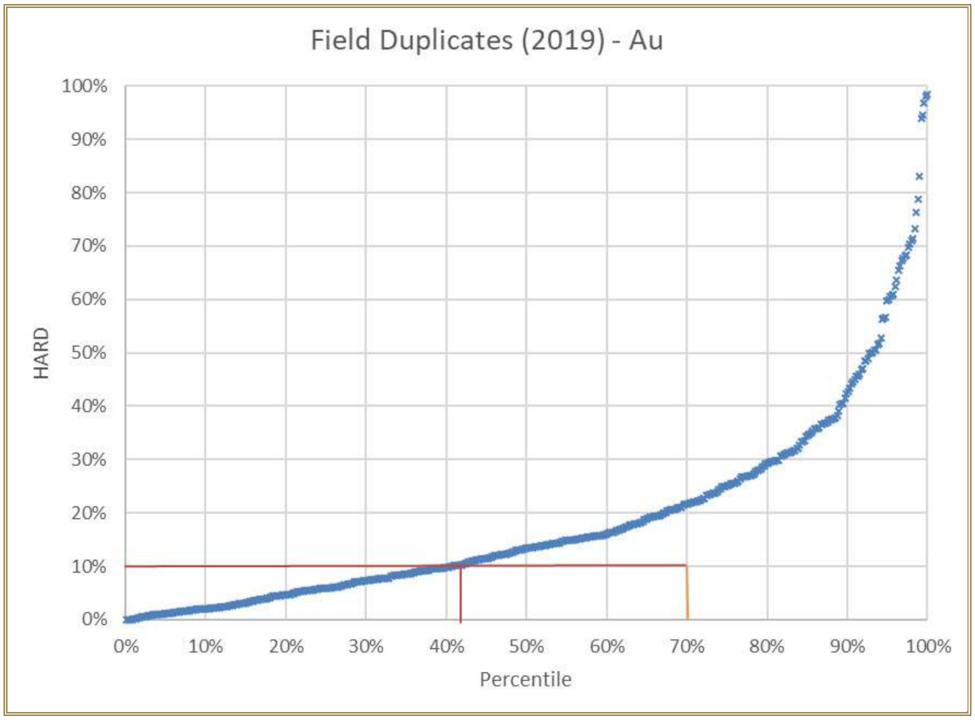


Figure 11-2: 2019 Field Duplicates Ranked—Gold

Certified Reference Materials—2019

Eight hundred and thirty-five instances of eight different certified reference materials (CRM) were inserted blindly into the 2019 sample stream. Process control charts for each of these standard materials are presented in Appendix A; Figures A-24 to A-30 present the gold standards, and A-31 through A-35 present the silver standards.

A summary of the gold CRMs is given in Table 11-7. Of the seven gold CRMs, three performed quite well. CU 190 had no failures. CU 193 had one failure. which, because of its value at 0.689, is likely to be a misidentified CU 190. CU 192 had only one failure and is potentially a misidentified sample, but not likely to be a different CRM.

Three CRMs performed moderately well. PM 933 had two failures and three sets of consecutive samples outside of the warning level. This is not of concern since the mean is in the low direction. PM 1147 had two failures and four sets of two consecutive samples outside of the warning level (WL). One failure, significantly high at 1.47 g/t is likely a misidentified sample, but probably not another reference material. The overall trend for PM 1147 is slightly low. GS1Z had five failures, all of them high, and no consecutive runs outside of the warning level. The overall mean for the assays is close to the expected value.

ME 1807 did not perform very well; it had 16 failures and five runs of two or more outside of the warning level.

In general, the assay results of the CRMs for gold give acceptable results and indicate that the 2019 assay database is of acceptable accuracy.

Table 11-7: Summary of Standard Results for Ascot Drilling at PGP—2019 Gold

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 192	0.675	1	none	59	One result at 0.852 g/t, possibly misidentified, mean is close to expected value
CU 193	0.477	1	none	119	One result at 0.689 g/t, likely misidentified, mean is close to expected value
PM 933	9.59	2	3 sets of 2 – low	62	Error is in low direction
PM 1147	1.12	2	4 sets of two	151	One result at 1.47 g/t possibly misidentified, mean is close to expected value
CU 190	0.68	0	none	124	mean is close to expected value
GS1Z	1.155	5	none	134	mean is close to expected value
ME 1807	7.88	16	5 runs of 2 or more outside wl	180	mean is close to expected value

Process control charts for the silver assay results of the standards are given in Appendix A, Figures A-31 to A-36. Table 11-8 presents a summary of these results. One CRM, CU 190, performed well with no failures, and the mean close to the expected value of 9.4 g/t. Of the remaining five standards, all gave a mean assay result high, closer to the warning level (expected value plus 2 standard deviations [SD]) than the expected value. In general, these results show higher than expected results for silver. If the silver was of primary economic concern to the PGP this would require further investigation. However, the impact of silver is minimal, and this potential error is of little consequence.

Table 11-8: Summary of Standards for Ascot Drilling at PGP—2019 Silver

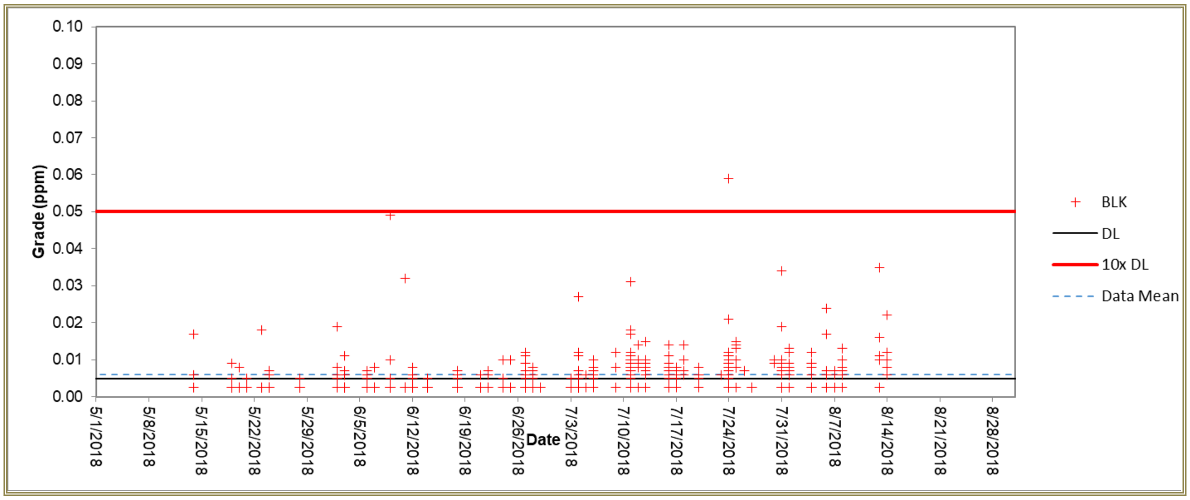
CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
PB 146	81.71	1	None	6	Mean is high, close to wl
PM 933	124.7	7	3 sets of 2 – high	119	Mean is high, close to wl
PM 1147	225.75	4	Multiple runs of multiples, all high	151	Mean is high, close to wl
CU 190	9.4	0	None	124	Mean is close to expected value
GS 1Z	89.5	22	Multiple runs of multiples, all high	134	Mean is high, close to wl
ME 1807	327	19	Multiple runs of multiples, all high	180	Mean is high, close to wl

11.1.6 Quality Assurance and Quality Control—Big Missouri, Premier, Silver Coin, and Martha Ellen—2018

The 2018 PGP drilling results were monitored for QA/QC by Jeremy Vincent, P.Geo. The assay results were reviewed monthly and recommendations were made and subsequently incorporated into the drilling program. These reports and results were obtained and reviewed, and a summary of the results is presented here. In the opinion of the Qualified Person (QP), the assay QA/QC data indicate that the 2018 drilling data are acceptable.

Blanks—2018

As shown in Figure 11-3, of the 455 blanks inserted into the sample stream, only one falls above 10 times the detection limit.

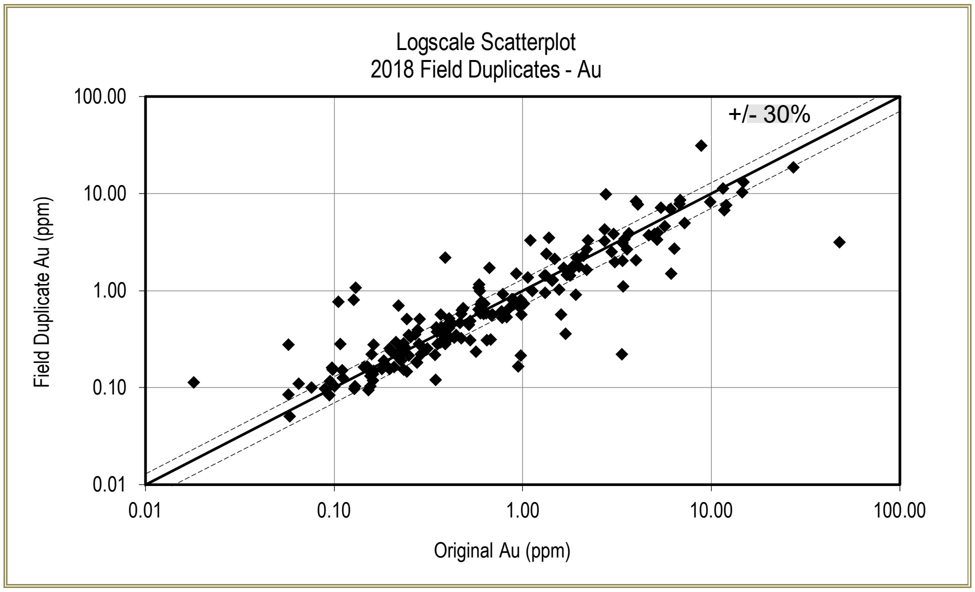


Source: Vincent, 2018

Figure 11-3: Field Blanks—2018—Gold

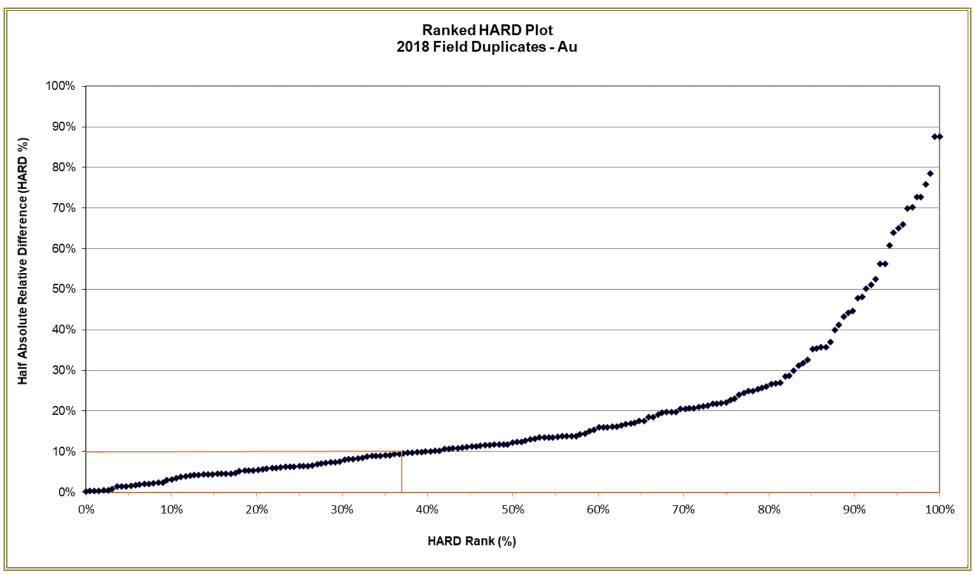
Field Duplicates—2018

A total of 189 pairs of field duplicates were inserted into the sample stream. A scatter plot of these values is seen in Figure 11-4 and shows reasonable correlation along a 1:1 line. The ranked HARD values are given in Figure 11-5 and show results consistent with previous results, as expected due to the heterogeneity of gold mineralization in the deposit.



Source: Vincent, 2018

Figure 11-4: Field Duplicates—2018—Gold



Source: Vincent, 2018

Figure 11-5: Field Duplicates—Ranked HARD—2018—Gold

Certified Reference Materials—2018

Four reference materials certified for Au were inserted into the 2018 sample stream. The comparisons of these assay results to the certified reference values are shown in Appendix A, Section 30.3, Figures A-20 to A-23. These results are summarized in Table 11-9.

For CU 192, the mean is slightly above the expected value, and two assays fall outside of the acceptable range. For CU 193, the mean Au assay is slightly above the expected value, and one value falls outside of the acceptable range. For PM 933, the mean is slightly below the expected value, and one sample falls outside of the acceptable range. For PM 1147, the mean is slightly above the expected value, and no assays fall outside of the acceptable range. The 2018 assay database can be considered to be of acceptable accuracy.

Table 11-9: Summary of Standards for Ascot Drilling at PGP—2018 Gold

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 192	0.67	2	2 pairs	102	Mean is slightly high
CU 193	0.48	2	none	61	Mean is slightly high
PM 933	9.59	1	none	136	Mean is close to expected value
PM 1147	1.12	0	none	148	Mean is close to expected value

11.1.7 Quality Assurance and Quality Control—Premier, Big Missouri, and Martha Ellen—2017

Blanks—2017

There were 882 blanks placed blindly into the 2017 sample stream. Of these, seven 7 returned assay values above five times the detection limit, for a failure rate of less than 1%, as illustrated in Figure 11-6. This indicates a minor problem with contamination during the 2017 drill program.

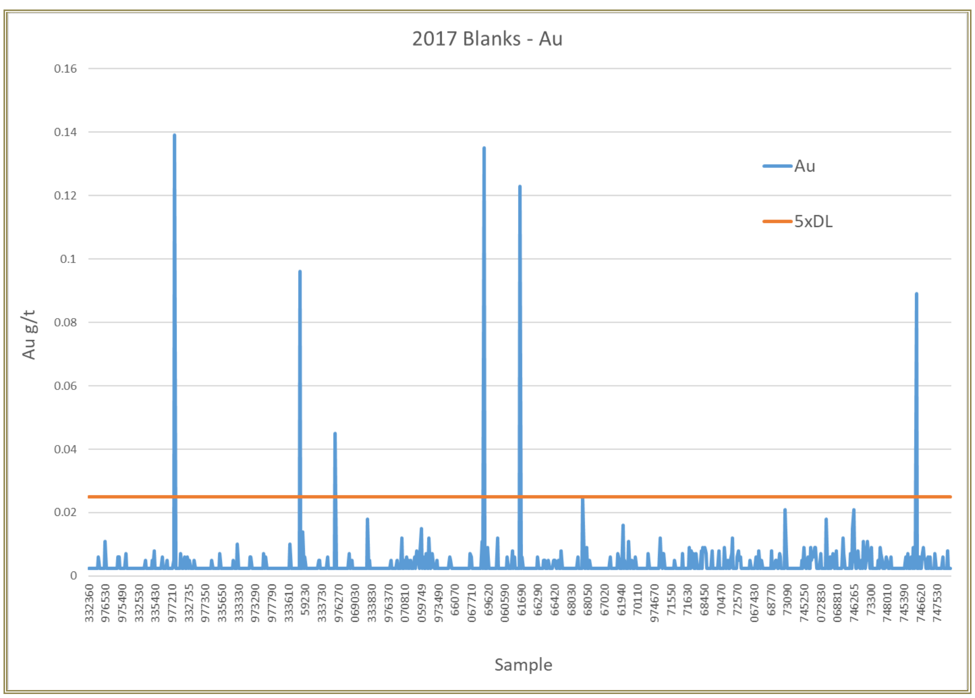


Figure 11-6: 2017 Blanks—Gold

Field Duplicates—2017

A scatter plot of the 201 pairs of field duplicates is presented in Figure 11-7. It is observed that there is some scatter, and the correlation is not good; however, this is to be expected based on the already established heterogenous nature of gold in this deposit. The field duplicates for silver, in Figure 11-8, show less scatter and imply less heterogeneity.

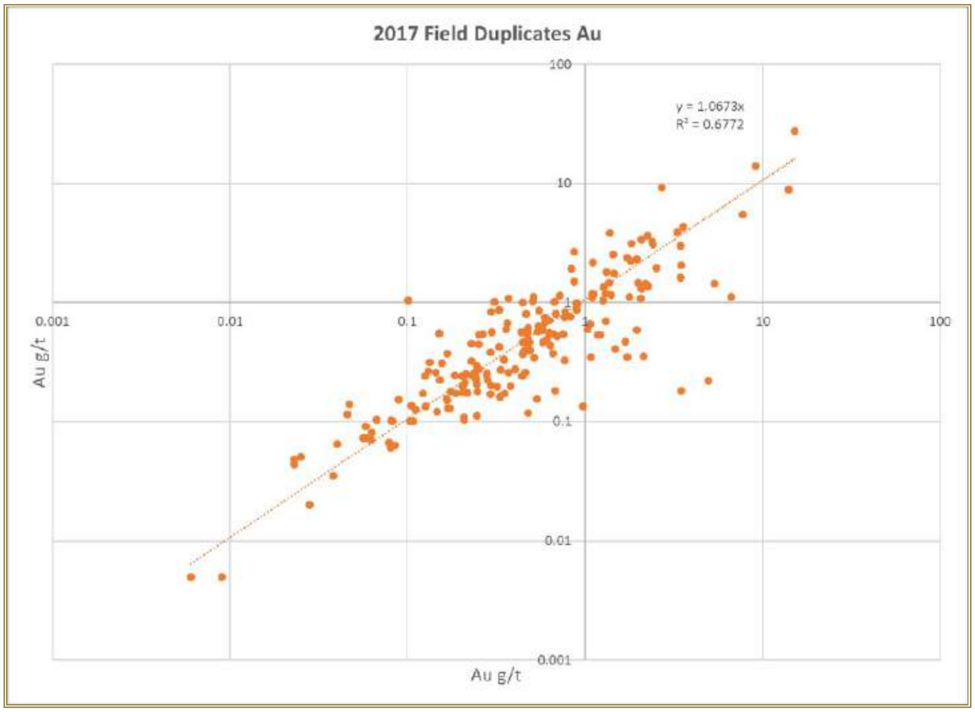


Figure 11-7: 2017 Field Duplicates—Gold

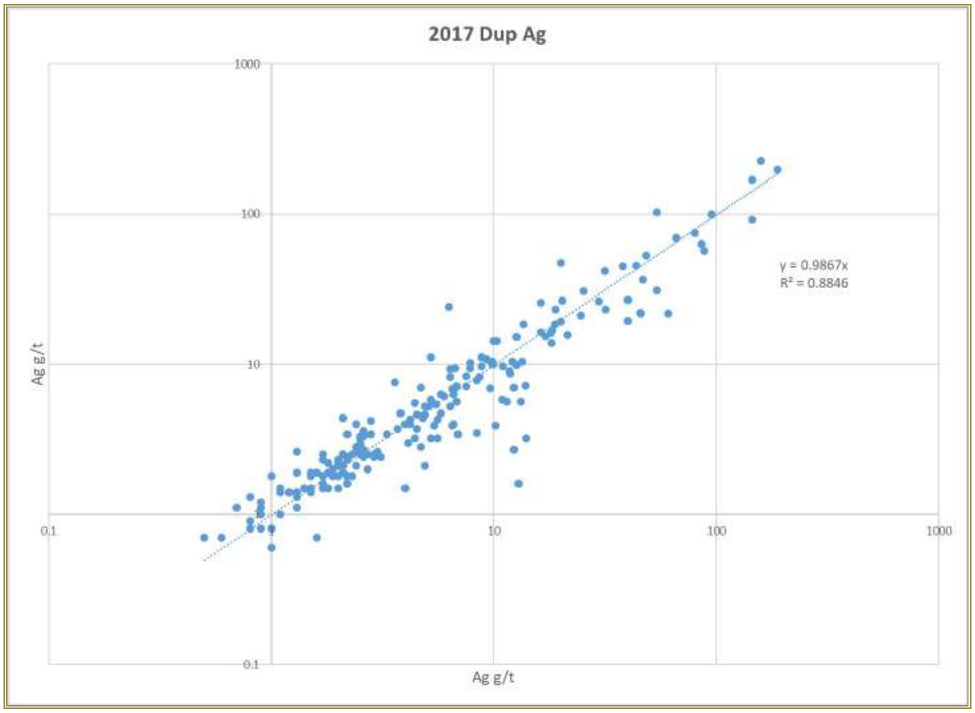


Figure 11-8: 2017 Field Duplicates—Silver

Standards—2017

Five reference materials were inserted into the sample stream for the 379 holes drilled in 2017. The process control charts are given in the Appendix A, Section 30.5, Figures A-37 to A-41.

A summary of the results of standards for gold is given in Table 11-10. For standard CU 193, there were five failures, at least one is likely misidentified. The mean is near the expected value, slightly high. PM 930 has only one failure, with a mean assay close to the expected value. There are no failures for PM 933, with a mean close to the expected value. PM 1147 has one pair of consecutive samples outside of the high warning level; there is significant scatter in both directions, and the mean is close to the expected value. PM 1142 has one failure (low), no consecutive samples outside of the warning level, and the mean is slightly high.

Table 11-10: Summary of Standards for Ascot Drilling at PGP—2017 Gold

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 193	0.48	5	Two instances	280	Mean is slightly high
PM 930	4.02	1	none	173	Mean is close to expected value
PM 933	9.59	0	none	88	Mean is close to expected value
PM 1147	1.12	0	one	261	Mean is close to expected value
PM 1142	1.38	1	none	21	Mean is slightly high

The results for the standard samples for silver are presented in Table 11-11, shown in Appendix A, Figures A-42 to A-46. The silver standards perform very well, with only two failures.

Table 11-11: Summary of Standards for Ascot Drilling—2017 Silver

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
PM 930	52	0	none	173	Mean is close to expected value
PM 933	124.7	1	none	88	Mean is close to expected value
PM 1147	226	1	none	261	Mean is close to expected value
PM 1142	306	0	none	21	Mean is close to expected value
PB 146	82	0	none	106	Mean is close to expected value

In summary, the 2017 QA/QC data indicates acceptable quality for inclusion into the assay database.

11.1.8 Quality Assurance and Quality Control—Premier—2016

The analysis of QA/QC data from 2016 is presented here. During 2016 drilling by Ascot was only done in the Premier deposit area.

Blanks—2016

Assay results of the blanks for gold are presented in Figure 11-9. Five values were above the threshold of five times the detection limit (0.005 g/t). The results for silver are given in Figure 11-10, which shows 10 samples that exceed five times the detection limit. One sample cannot be seen on the graphs as its value is 82 g/t, indicative of contamination or mislabelling. At any rate, these failures are sporadic and do not imply a significant problem with contamination.

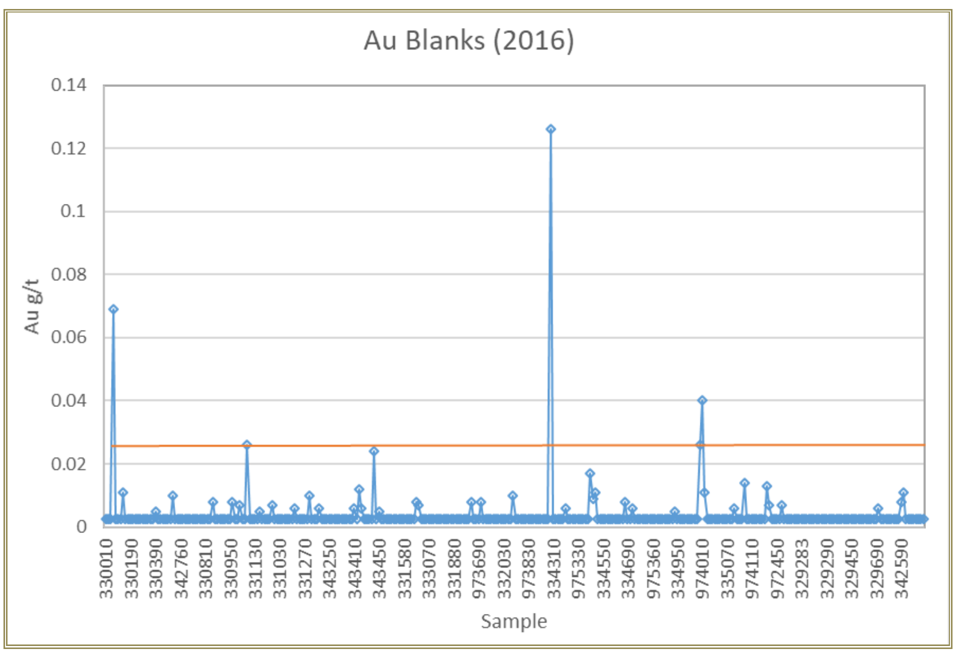


Figure 11-9: 2016 Blanks—Premier—Gold

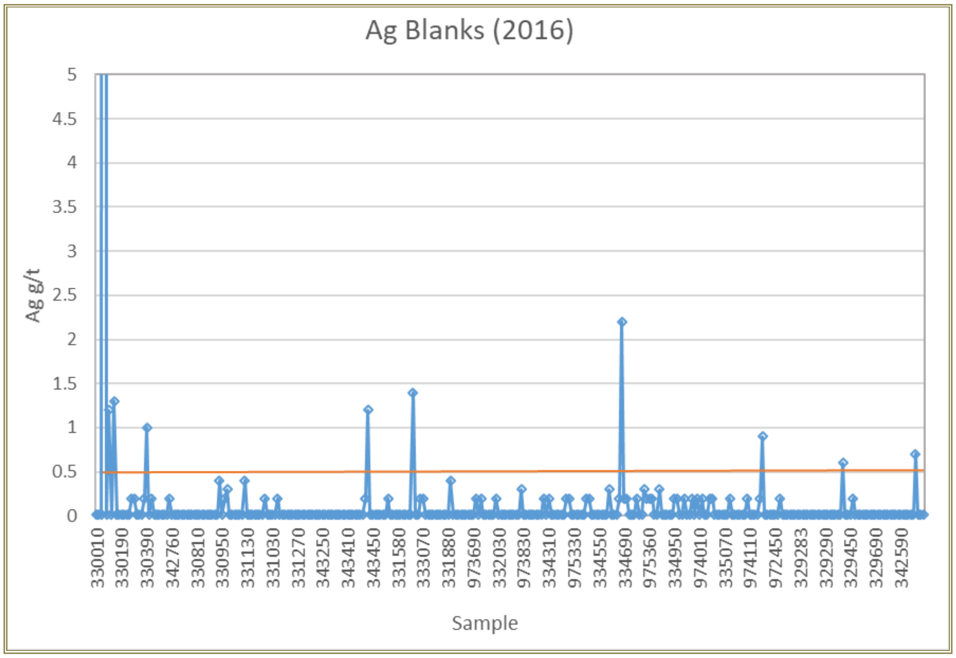


Figure 11-10: 2016 Blanks—Premier—Silver

Field Duplicates—2016

There were only 23 pairs of field duplicates in the 2016 sample stream, not enough to provide a meaningful analysis.

Standards—2016

Results of the CRMs for gold used in 2016 are summarized in Table 11-12. The process control charts are presented in Appendix A, Figures A-47 to A-52. The results show only four failures with means of assay values close to the expected values. In two cases the means were near the +1 SD value. Overall, the gold standards had good performance.

The summary of silver assay results of the standards is given in Table 11-13, and the control charts in Appendix A, Figures A-53 to A-59. The silver results are very good, with no failures outside the acceptable level or consecutive values outside the warning level.

Table 11-12: Summary of Standard Results for Ascot Drilling at PGP—2016 Gold

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 186	1.63	0	None	96	Mean close to EV
CU 193	0.48	2	None	61	Mean close to EV
PM 1123	1.42	0	None	21	Mean higher than EV
PM 1141	0.55	0	None	19	Mean higher than EV
PM 1143	1.38	0	None	30	Mean close to EV
PM 930	4.02	2	None	134	Mean close to EV

Table 11-13: Summary of Standard Results for Ascot Drilling at PGP—2016 Silver

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 186	13.5	0	None	96	Mean close to EV
CU 193	3.43	0	None	61	Mean close to EV
PB 146	81.71	0	None	31	Mean close to EV
PM 1123	31.06	0	None	21	Mean higher than EV
PM 1141	18.55	0	None	19	Mean close than EV
PM 1143	306.48	0	None	30	Mean meets EV
PM 930	52.26	0	None	134	Mean close to EV

Analysis of the blanks and standards shows the 2016 data to be acceptable. The small set of field duplicates is not a hinderance, as the gold has already been shown to be quite heterogenous in nature.

11.1.9 Quality Assurance and Quality Control—Premier—2015

Blanks—2015

The blanks for Au are presented in Figure 11-11. Of the 467 blanks there were failures, four of them after high Au values, indicating a minor problem with contamination. There were no failures in the silver assay values of the blanks.

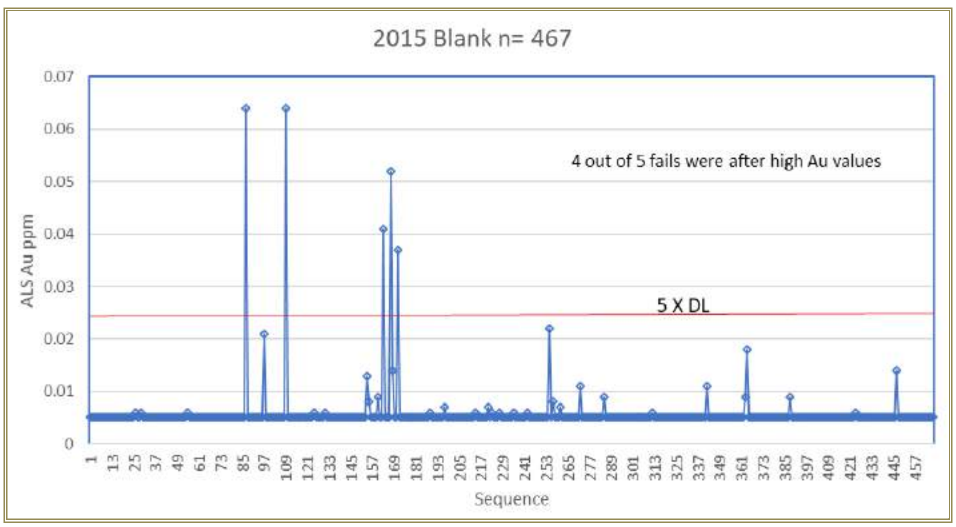


Figure 11-11: 2015 Blanks—Gold

Field Duplicates—2015

A scatter plot of the 48 pairs of field duplicates is given in Figure 11-12. The plot gives a line with near a 1:1 slope and a low correlation which indicates high heterogeneity with respect to gold.

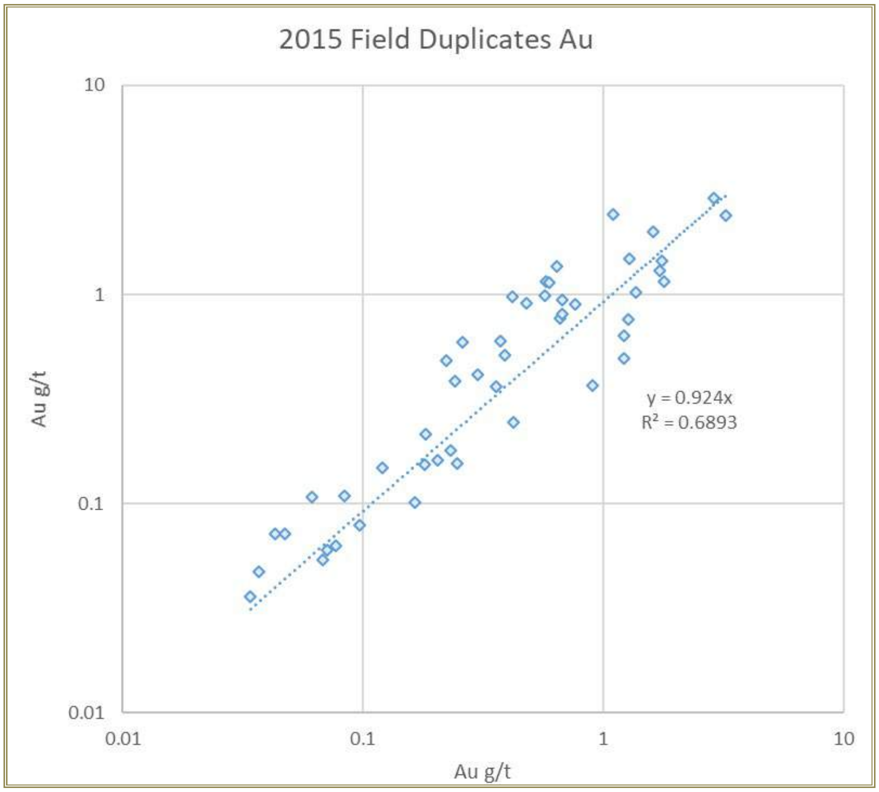


Figure 11-12: Field Duplicates—Gold 2015

Standards—2015

Analysis of the nine standards used in the 2015 drilling is presented here. Appendix A, Figures A-60 to A-68 give process control charts of the standards. Results of CU 165 give no failures for gold; the mean is approximately at +1 SD. CU 192 has two failures; the mean is approximately at +1 SD. The mean of assays of PM 930 is approximately at -1 standard deviation and has no failures. The mean of assays of PM 465 is close to the expected value and it has no failures. Standard sample PM 459 has no failures and the mean is slightly higher than the expected value. Standard PM 928 has no failures and the mean is lower than the expected value. There is only one failure in the PM 1123 series and the mean is higher than the expected value. For PM 1141 there are no failures and the mean is approximately +1 SD higher than expected.

The standards are shown to perform well for gold, implying acceptable accuracy.

Table 11-14: Summary of Standard Results for Ascot Drilling at PGP—2015 Gold

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 165	1.42	0	None	71	Mean higher than EV
CU 192	0.68	2	None	65	Mean higher than EV
PM 930	4.02	0	None	32	Mean lower than EV
PM 465	1.60	0	None	37	Mean close to EV
PM 459	0.37	0	None	46	Mean close to EV

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
PM 928	4.19	0	None	37	Mean lower than EV
PM 1123	1.42	1	None	80	Mean is higher than EV
PM 1141	0.55	0	None	39	Mean is higher than EV

Between Lab Assays—2015

A set of 454 samples was sent to SGS for assay to compare the between lab results. These results for gold are given in Figure 11-13. It is observed that the correlation is very good, giving a slope of nearly one and correlation coefficient of 0.99. The results for silver are similar.

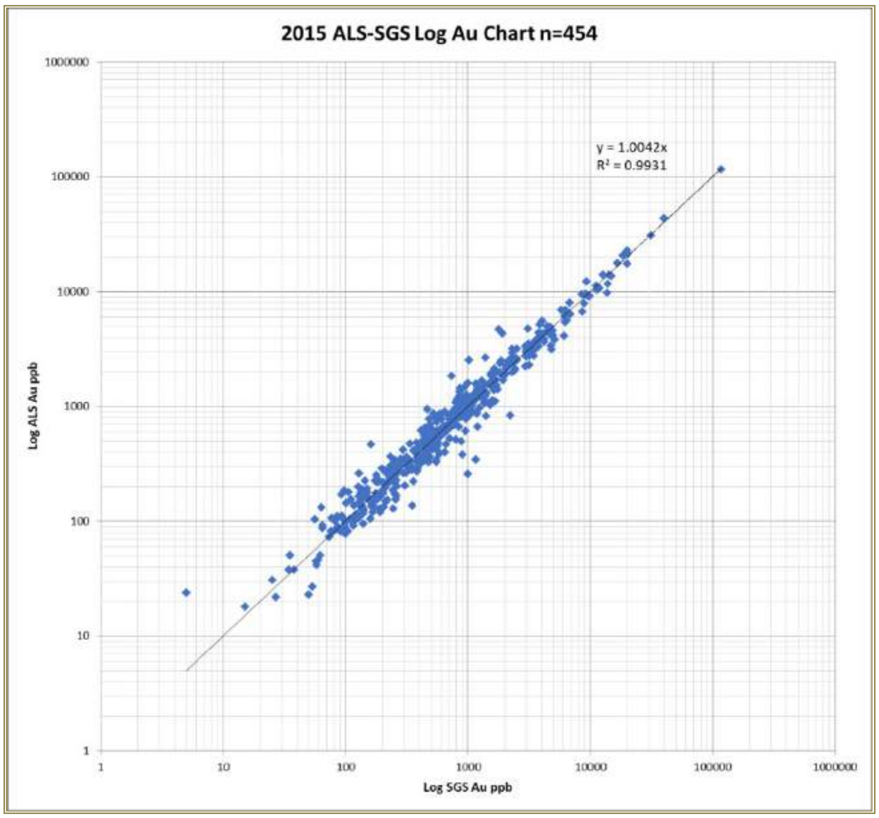


Figure 11-13: Between Lab Assays—2015

11.1.10 Quality Assurance and Quality Control—Premier and Big Missouri—2014

The QA/QC analysis by others was reviewed and accepted. There were 416 blanks assayed, with only six results for each gold and silver above the five times detection limit. The 133 pairs of field duplicates showed typical results for heterogenous gold mineralization.

Four CRMs were used, three of which (CU 165, CU 192, PM 928) are certified as gold standards. The process control charts for these standards are given in Appendix A, Figures A-69 to A-71. Of these, CU 165 has two failures, one so extreme it is likely to be a mislabel; CU 192 has two failures, and PM 928 has none. In all cases, the mean of the assays is close to the expected value.

The 459 check assays are presented in Figure 11-14 and give good correlation between the two labs, lending confidence to the ALS laboratory results.

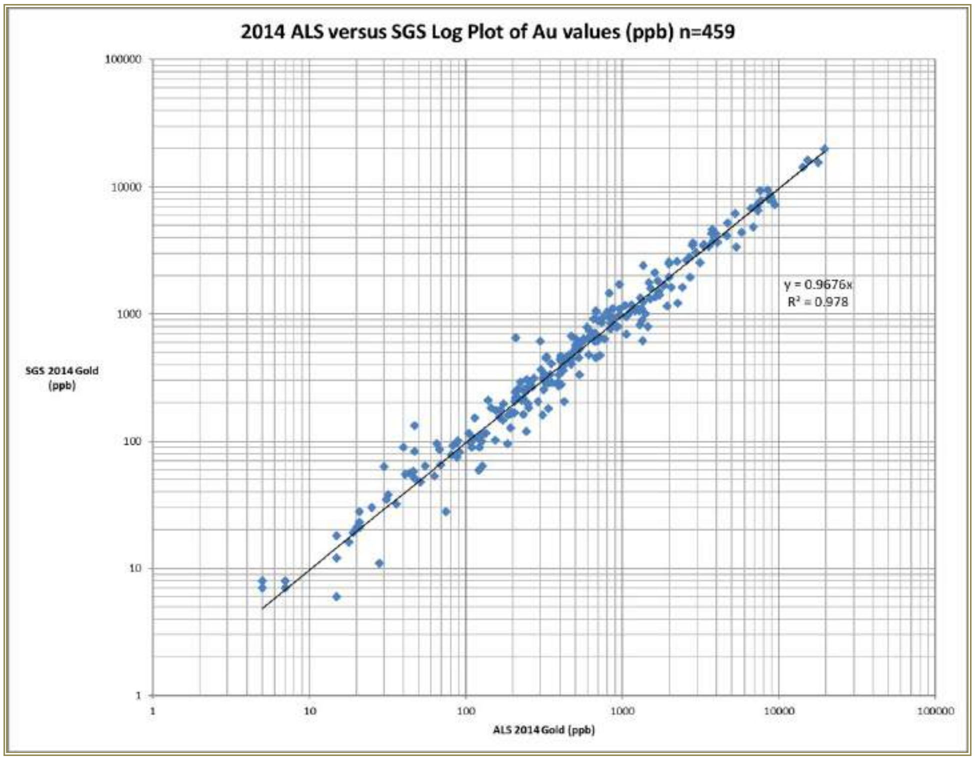


Figure 11-14: Between Lab Assays—2014

11.1.11 Quality Assurance and Quality Control—Big Missouri, Martha Ellen, Dilworth, and Premier—2013

Not all 2013 QA/QC data are available, however, the available data are reviewed and shows that a system of checks was in place and an acceptable level of accuracy is obtained.

Three standards were certified for gold at levels of 0.374 ppm Au (PM 459), 1.6 ppm Au (PM 465), and 4.19 ppm Au (PM 928). Sequence control charts are illustrated in Appendix A, Figures A-16 to A-18. Standard PM 465 shows good results with a mean close to the expected value and only one failure. Standard PM 459 gives a slightly high mean compared to the expected, with one failure and one outlier possibly due to mislabelling. The results for PM 928 give no failures, and a lower mean than expected.

No blanks or field duplicate data are available from 2013.

A total of 628 external laboratory checks were performed on pulps from the 2013 drill program. The external laboratory in this case was SGS. Gold results showed an R^2 value of 0.986, and a nearly 1:1 correlation. A scatterplot of the comparisons for the 2013 data are shown in Figure 11-15.

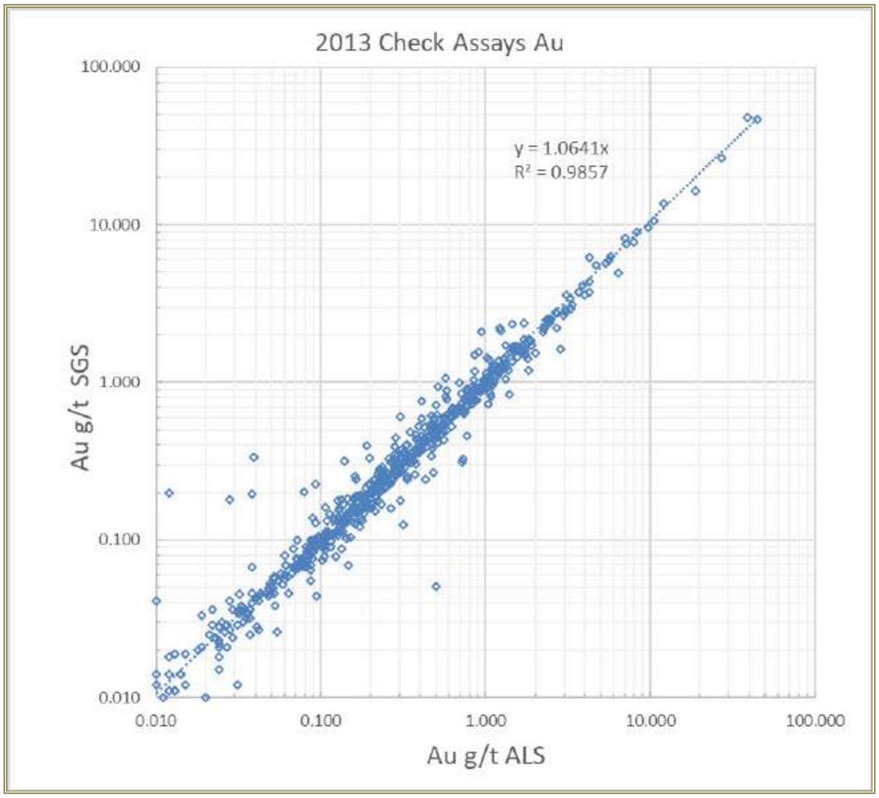


Figure 11-15: Third Party Lab Checks (ALS vs. SGS) for 2013—Gold

11.1.12 Quality Assurance and Quality Control Big Missouri, Martha Ellen, Dilworth, and Premier—2007 to 2012

The data from the years 2007 to 2012 is analyzed together for brevity.

Blanks—2007 to 2012

There were 2,068 blanks inserted in the 2007 to 2012 drilling. A process control chart of these is presented in Figure 11-16. Of these, 10 exceeded five times the detection limit. The assay sample at 0.68 g/t followed a sample with 273 g/t; two other failures also followed high assay values. This is not indicative of a significant contamination issue.

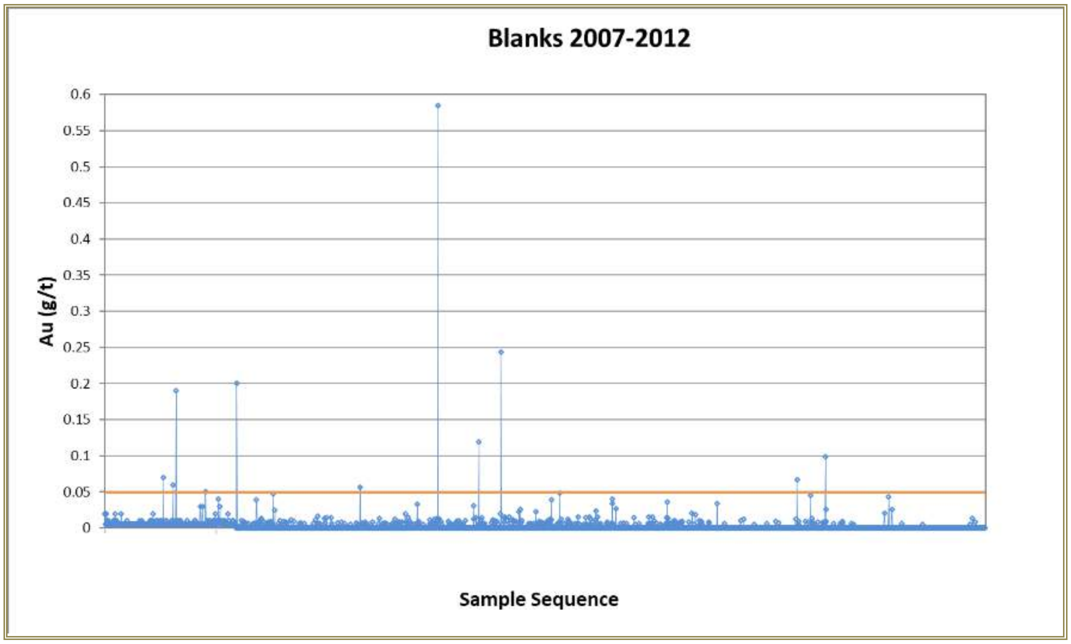


Figure 11-16: Blanks 2007 to 2012

Field Duplicates—2007 to 2012

There were no field duplicate samples identified in the provided database of control samples in years 2007, 2008, and 2009.

The assay results of 995 field duplicates from 2010 to 2012 are presented in scatter plots for Au in Figure 11-17 and silver in Figure 11-18. The ranked plots of the HARD are given in Figure 11-19 and Figure 11-20. The results for gold field duplicate pairs do not meet the desired criteria of 70% less than 10% HARD, but this is more likely to be indicative of the heterogeneity of the deposit, typical for Au, than of a problem with the duplicates. The Ag field duplicates meet the criteria, showing approximately 70% less than 10% HARD.

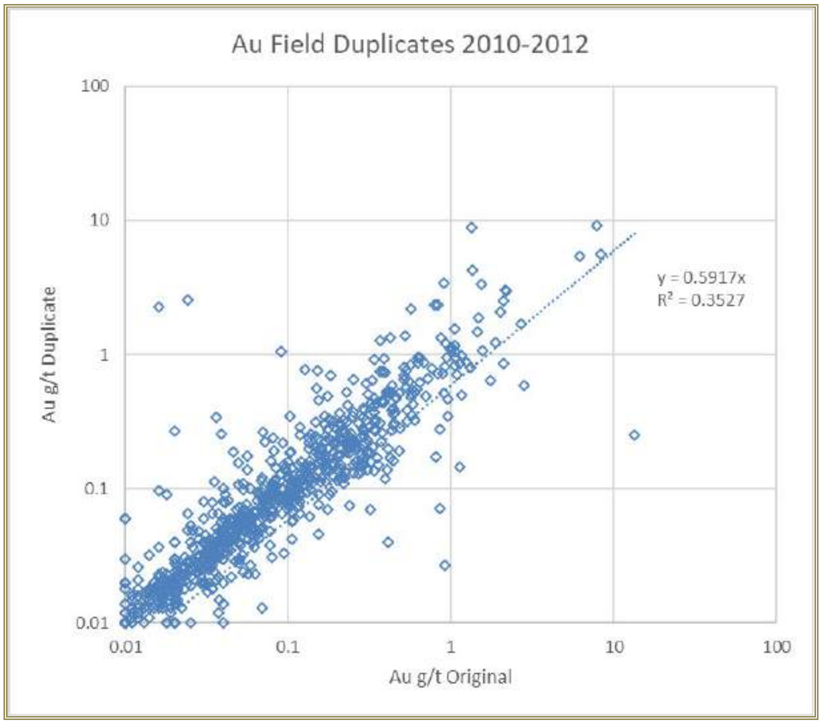


Figure 11-17: Ascot Field Duplicates from 2010 to 2012—Gold

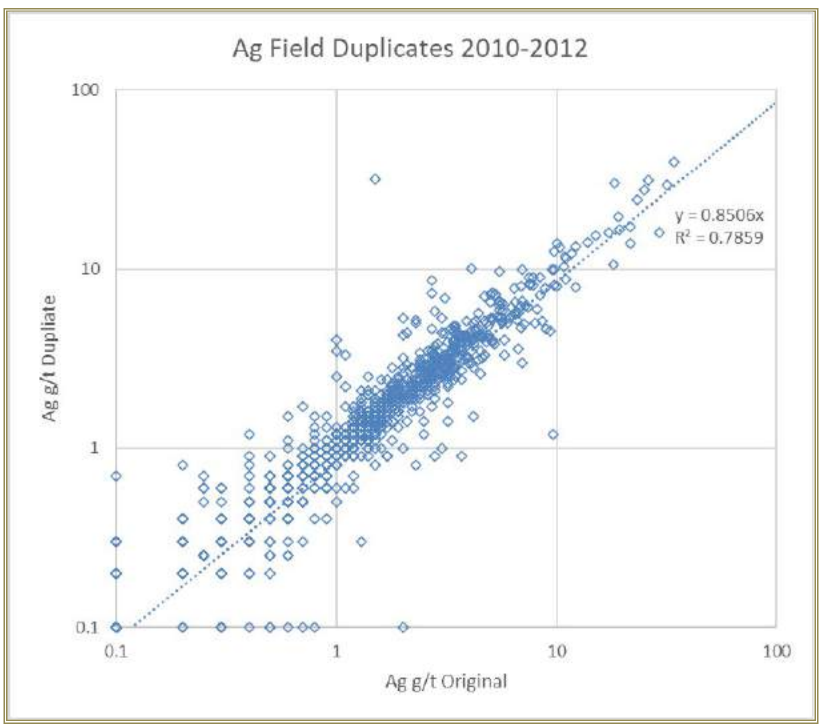


Figure 11-18: Ascot Field Duplicates from 2010 to 2012—Silver

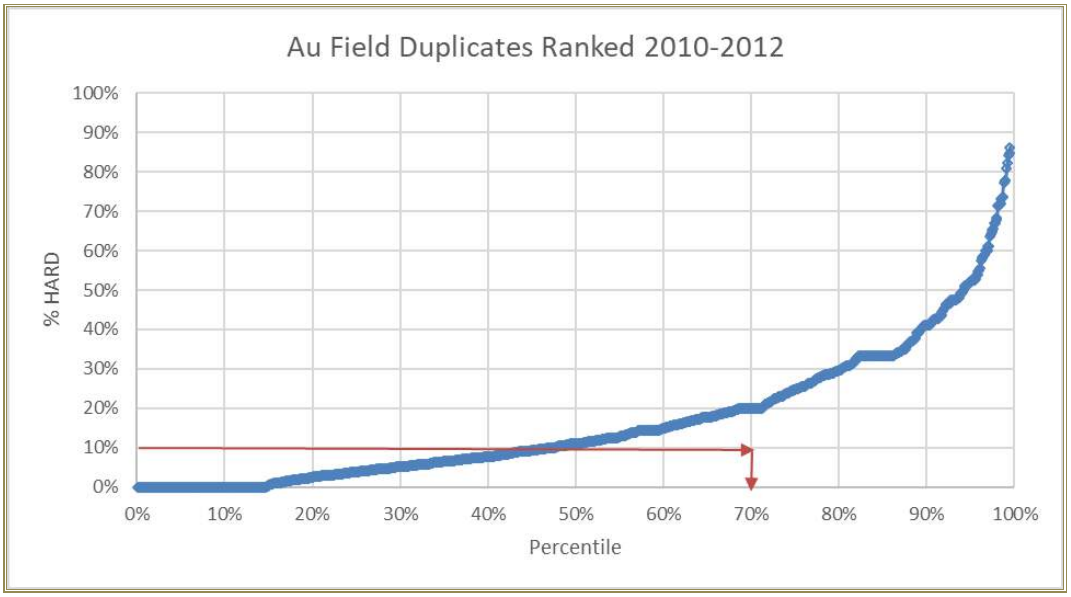


Figure 11-19: Ascot Field Duplicates from 2010 to 2012—Ranked HARD Plot—Gold

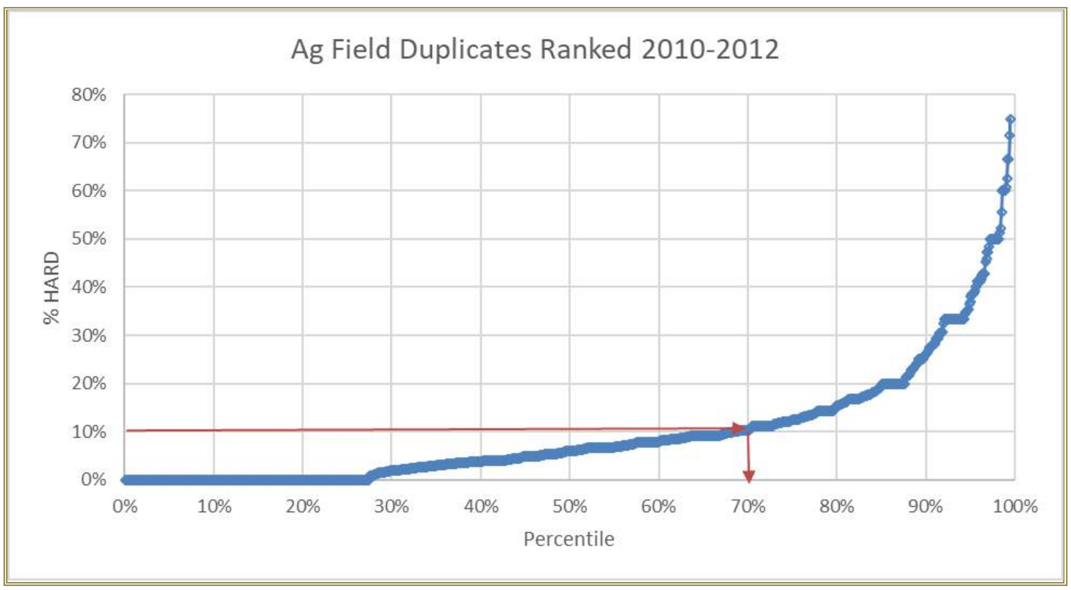


Figure 11-20: Ascot Field Duplicates from 2010 to 2012—Ranked HARD Plot—Silver

Standards—2007 to 2012

Table 11-15 shows standards used from 2007–2012. These years were primarily concerned with drilling in Big Missouri, Dilworth, and Martha Ellen. Examples of the process control charts follow in Figure 11-21 and Figure 11-22. The complete standards for this time period of Ascot drilling are given in Appendix A, Figures A-1 to A-15.

Table 11-15: Standards from 2007 to 2012

Standard Name	Expected Value (g/t Au)	Years Used	Samples
PM 405	0.26	2009	40
PM 459	0.37	2012	276
PM 404	0.41	2010	60
PM 197	0.45	2007-2008	23
CU 178	0.50	2010-2012	217
PM 441	0.53	2011	299
PM 446	1.22	2011	299
PM 1112	1.35	2008	20
PM 454	1.42	2012	278
PM 1110	1.78	2008	20
PM 432	2.03	2010	61
PM 429	2.21	2010-2012	219
PM 427	3.57	2009-2010	99

Figure 11-21 shows that for Standard PM 405 the mean is slightly below the expected value, and two results are outside of the acceptable range, one so high that it is possibly mislabelled.

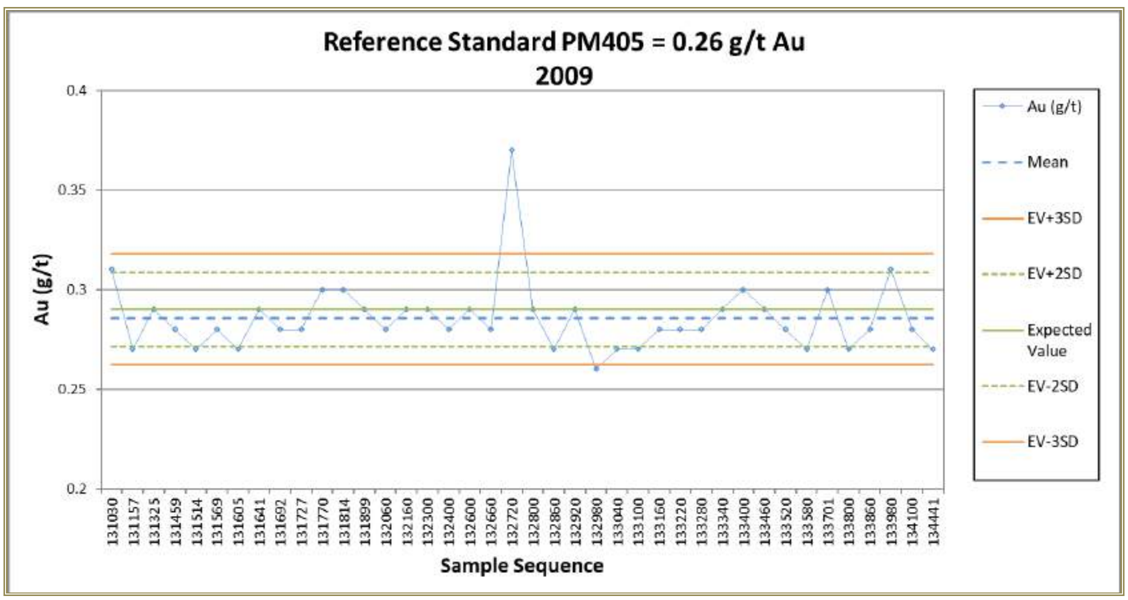


Figure 11-21: Ascot Standard PM405 Control Chart

PM 404 performance gives good results, as shown in Figure 11-22,with most samples within the ±2 SD range and the mean close to the expected value.

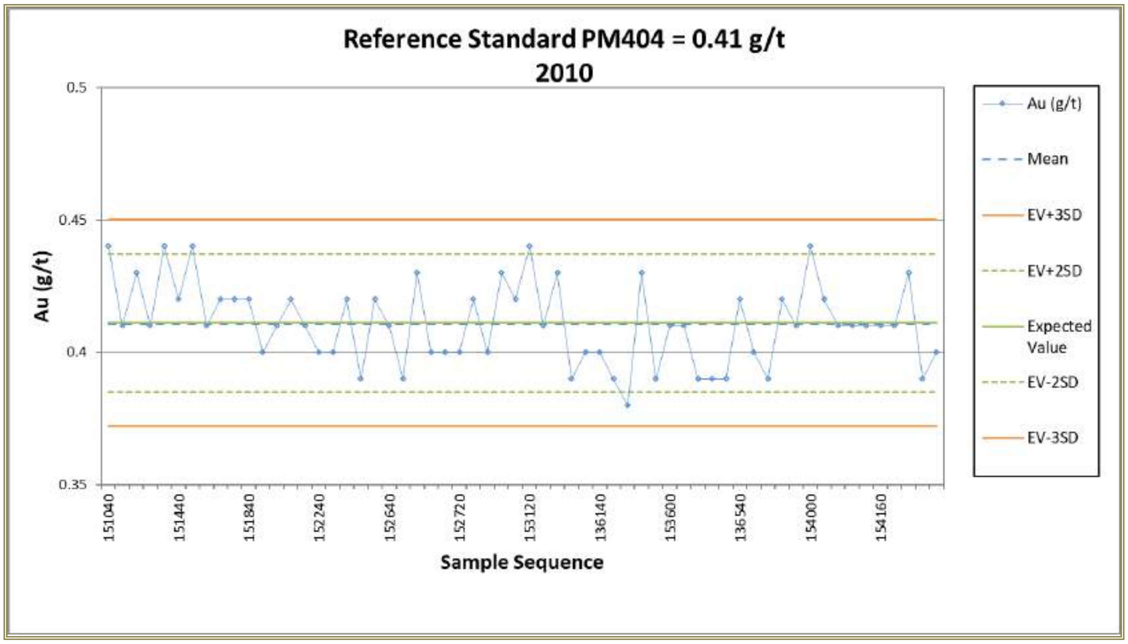


Figure 11-22: Ascot Standard PM 404 Standard Control Chart

It appears that standard PM 432 should be disregarded. Of the remaining sixteen, two, PM 922 and PM 197, indicate problems potentially with labelling or laboratory error. However, the trends of these results are both low, and these samples were inserted in 2007 and 2008 from holes in the Dilworth area, making the impact of these results minimal in the context of the PGP. The remaining fourteen standards show good to reasonable results.

Between Lab Assays—2010 to 2012

Additionally, 1,244 pairs of samples were checked at both ALS and SGS. The results of these assays are given in terms of ranked HARD values in Figure 11-23 and Figure 11-24. The results are to be compared by the same criteria as field duplicates and meet the 70% less than 10% HARD criteria.

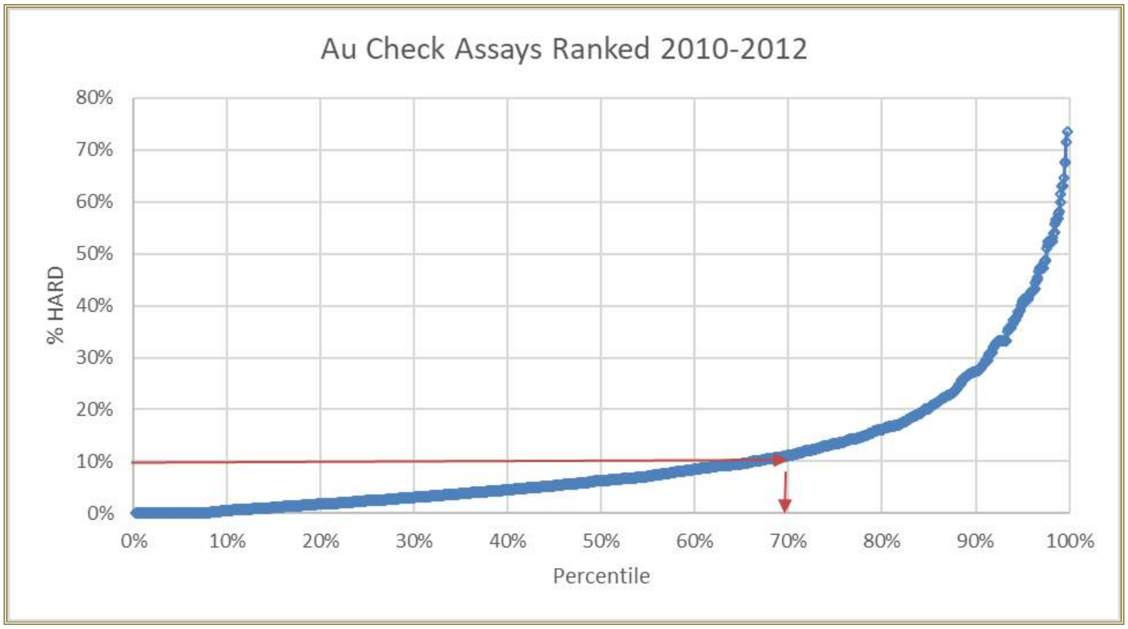


Figure 11-23: Ascot Lab Checks (ALS and SGS) from 2010 to 2012—Ranked HARD Plot—Gold

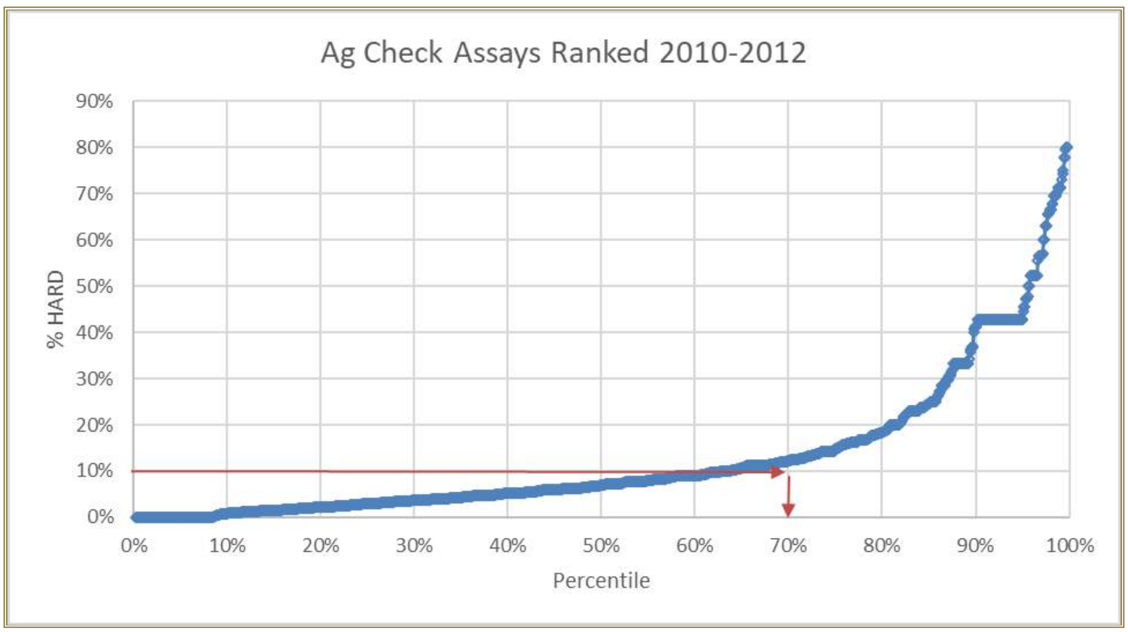


Figure 11-24: Ascot Lab Checks (ALS and SGS) from 2010 to 2012—Ranked HARD Plot—Silver

11.1.13 Silver Coin Quality Assurance and Quality Control—Legacy Drilling

There is no QA/QC data available for the Westmin and Tenajon data in the Silver Coin deposit. Validation of this data by re-assays is presented in Section 12. The QA/QC for the Jayden and MBM data follows.

The QA/QC assaying for the 2004 to 2008 Jayden and MBM programs included duplicates sent to ALS Chemex, where a 30 g FA with an AA finish was used for gold. The assay methods used for the duplicate samples are not known. The QA/QC program records indicate that there was regular insertion of standard and blank samples.

For the 2009 to 2011 programs, the external QA/QC protocols included the insertion of multiple standards, blanks, and duplicates into the sample preparation and assay stream, and continual monitoring of the results.

The available QA/QC for the Jayden and MBM drilling between 2004 and 2011, as summarized in Table 11-16, is reviewed and discussed here. Although the data are not as comprehensive as would be ideal, it is of good quality and indicates that thought and effort was given to a control system. A review of the available data indicates acceptable credibility to the data of this era.

Table 11-16: Summary of Jayden and MBM QA/QC Data

Type of Data	Year
Blanks	2005 – 2008
Standards	2005 – 2008
Between Lab Check Assays	2005 – 2011

Jayden and MBM Blanks—2004 to 2011

The gold assay values from blanks from years 2005 to 2008 are shown in Figure 11-25. It is seen that only four exceed the level of five times the detection limit. Two of these four values follow samples with gold assay above 50 g/t indicative of a minor problem with contamination. The Ag assays of blanks are given in Figure 11-26.

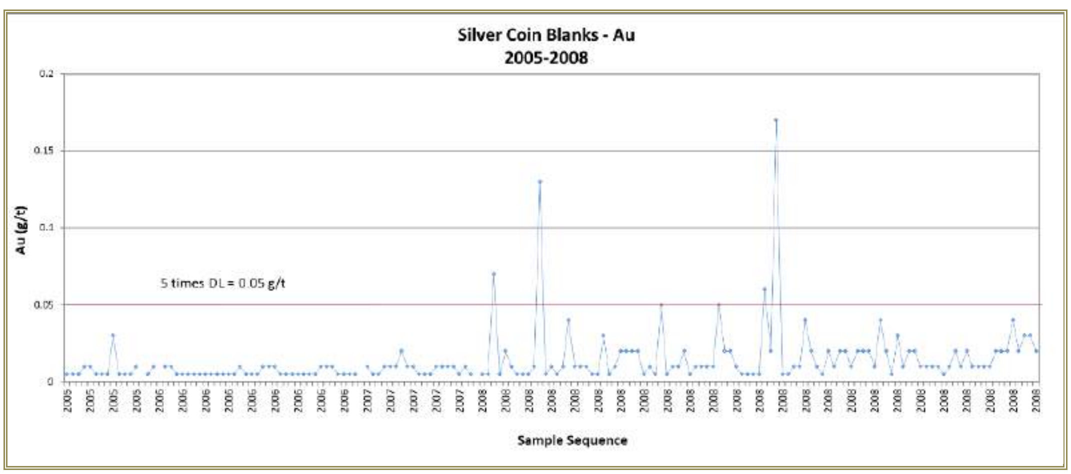


Figure 11-25: Silver Coin Blanks 2005 to 2008—Gold

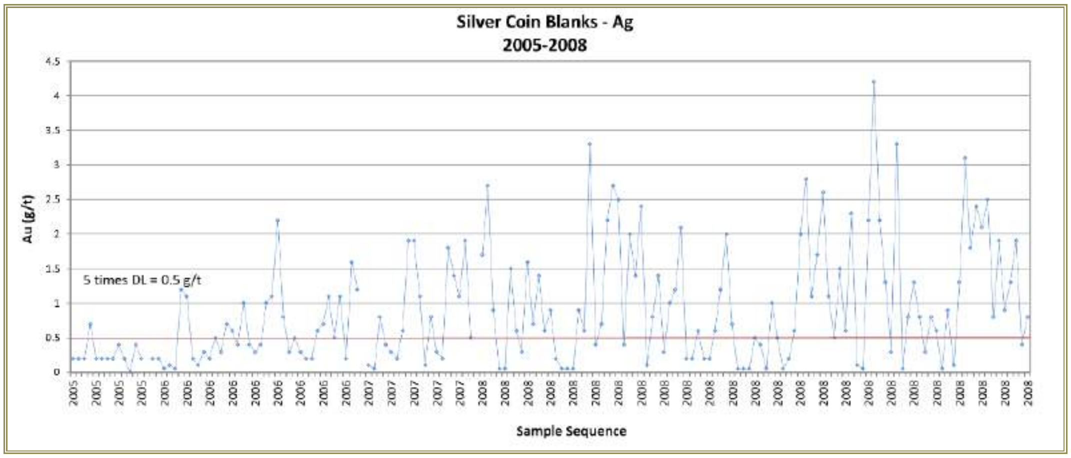


Figure 11-26: Silver Coin Blanks 2005 to 2008—Silver

Jayden Standards—2005 to 2008

A summary of the standard results for this time period at Silver Coin is presented in Table 11-17. Process control charts of the standards used in the Jayden drilling are given in Appendix A, Figures A-72 to A-75. For CU135 there are five failures and multiple sets of two or more outside of the warning limit. There is a definite shift seen of lower than expected assay values in 2007 compared to the other years. Because the mean is close to the expected value and most pronounced trend is low, this issue with accuracy is of little concern. For PM 160, there are multiple failures and the mean is close to the +1 SD value. Because this standard performs so differently from the others, it is likely a problem with the standard itself, and not a problem with the laboratory. PM 911 shows five failures outside the acceptable limit, four are low and one is so high it is likely mislabelled. Overall, the mean is low, close to –1 SD from the expected value. PM 919 has only one failure and one run of two samples outside of the warning level; the mean runs high compared to the expected value. Overall, performance of the standards available in the Jayden data are acceptable.

Table 11-17: Summary of Standard Results for Jayden Drilling at Silver Coin—2005 to 2008—Gold

CRM	Expected Value (g/t)	Failed	Consecutive Outside WL	Samples	Comments
CU 135	5.93	5	4 sets	166	Mean close to EV
CU 160	4.49	23	None	49	Mean close to +1 SD from EV
PM 911	16.2	5	1	49	Mean close to -1 SD from EV
PM 919	2.9	1	1	157	Mean higher than EV

Jayden Check Assays—2005 to 2011

Between 2007 and 2011 check assays were sent to ALS Chemex for comparison. The results of these 904 samples are given in Figure 11-27. It is seen that the data falls nearly on the 1:1 line and the correlation is high, at 0.9884, which implies good inter-lab repeatability.

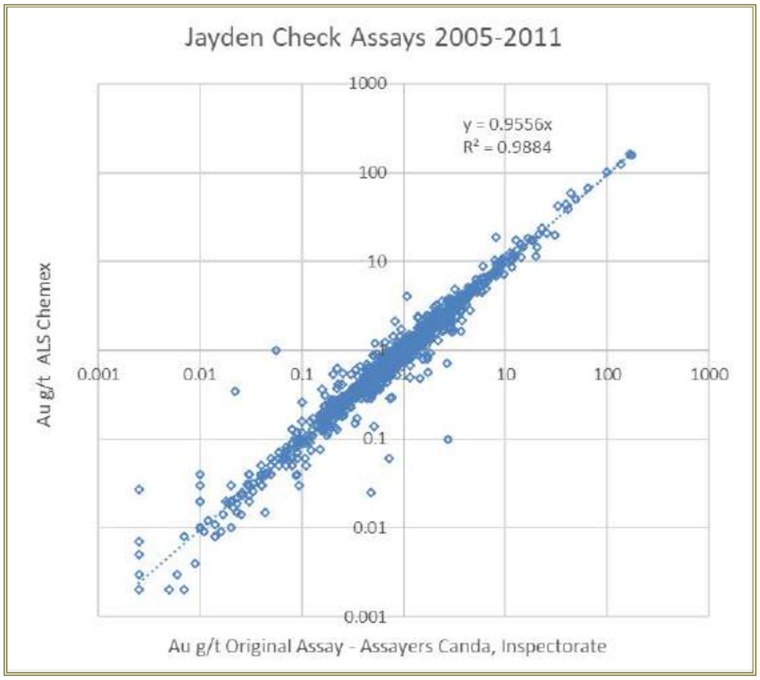


Figure 11-27: Jayden Check Assays

Jayden—2017

For the 2017 drilling, a systematic insertion of blanks and standards was made and followed in a QA/QC program. For the blanks, there were no results greater than five times the detection limits. The certified standards are charted below in Figure 11-28 to Figure 11-30. These standards were prepared by CDN Resource Laboratories Ltd. (CDN) of Langley, BC. The results for gold in Standard CDN-GS-2M were within an expected range on all samples. CDN-ME-1404 had one sample with a very high value, which may be a laboratory detection error or a mislabelled standard. There were also two failures less than 3 SD and one run lower than 2 SD. Results for CDN-ME-1505 were generally below the average suggested by the laboratory, with many more than 3SD below. Although the assayed values of the standards inserted are generally below the expected average value, the impact on the Resource Estimates is expected to be negligible.

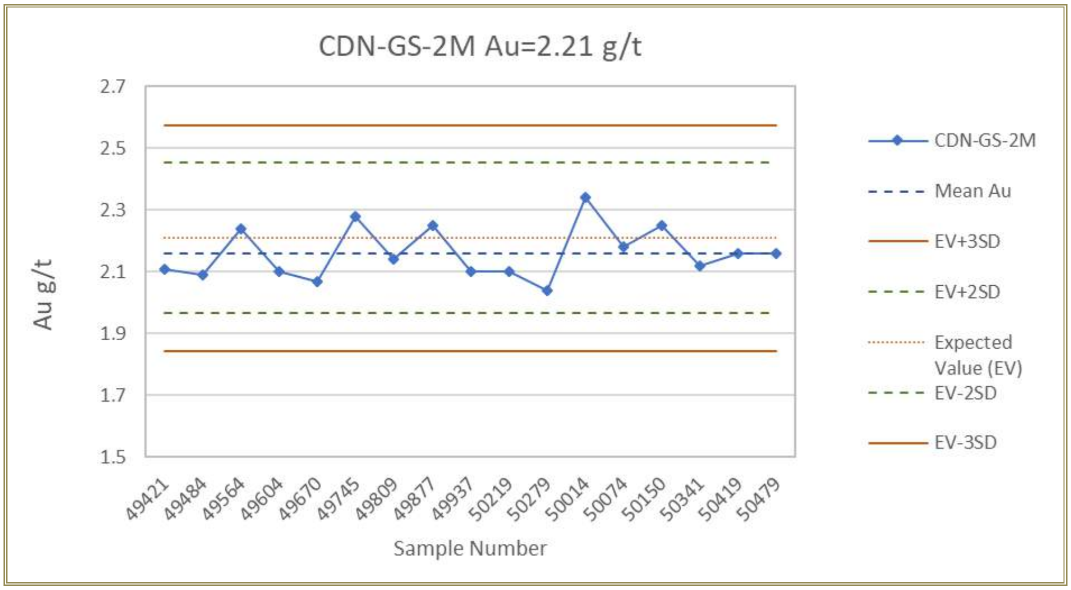


Figure 11-28: CDN-GS-2M Standard Control Chart—Silver Coin

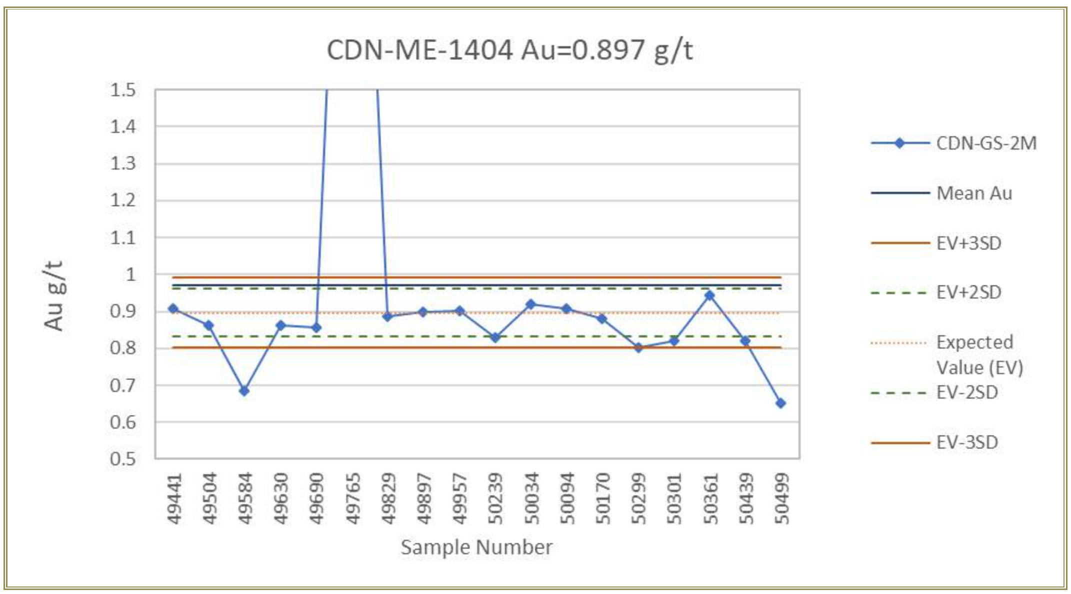


Figure 11-29: CDN-ME-1404 Standard Control Chart—Silver Coin

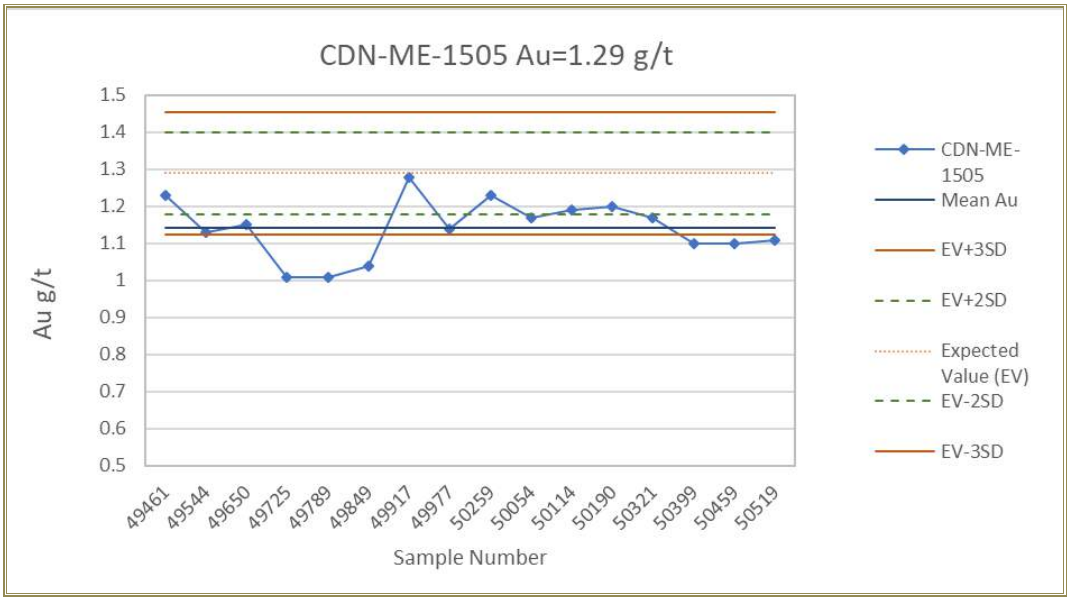


Figure 11-30: CDN-ME-1505 Standard Control Chart—Silver Coin

In the QP's opinion, data availability and analysis of the legacy QA/QC are generally lacking at Silver Coin. Because of this, there have been check assays of the Silver Coin core as well as re-assaying of core and pulps of the Westmin era drilling at Premier which used the same labs and methods as at Silver Coin. This analysis and results are presented in Section 12.

11.1.14 Ascot Sample Security at Premier Gold Project

Ascot maintains a secure logging and storage facility in Stewart, BC. All sample collection and handling are supervised by Ascot personnel. Collected samples are stored in bags sealed with a zap strap, and the samples are combined in large woven rice bags for shipping. The contents of each sealed rice bag are recorded, and full bags are stacked on pallets and shipped to the assay laboratory in Vancouver, BC, in secure transport trucks by commercial carrier Bandstra Transportation Systems Ltd. (with a head office in Smithers, BC).

Premier Gold Project Databases

Analytical and survey data are now organized into one complete relational database for all the PGP deposits. This was a recommendation from RPA (2019) and has since been completed with data by area used for each of the five block models used in the Resource Estimates.

11.1.15 Qualified Persons Discussion regarding Sample Preparation, Analysis, and Security at Premier Gold Project

The QP is of the opinion that the quality of gold and silver analytical data collected during the 2007 to 2019 Ascot drill programs at the PGP project are sufficiently reliable to support Mineral Resource estimation and that sample preparation, analysis, QA/QC, and security were generally in accordance with exploration best practices at the time of collection. The QP is also of the opinion that the legacy Jayden data from Silver Coin was not quite in conformance with best practices at the time of collection, but that no significant problems with the data have been identified; as such, it appears to be reasonable to accept the data as is.

11.1.16 Specific Gravity Estimation at Premier Gold Project

Table 11-18 summarizes the SG values used for Resource Estimation at each deposit. There is an important distinction that should be made between SG and bulk density. Bulk density is the measure of the mass per unit volume of the rock in situ, including both solids and pore spaces. Specific gravity, as determined by a pycnometer, is the mass per unit volume of solids only. Pulverizing the specimen eliminates the pore spaces and can lead to an over-estimate of the bulk density of the original rock mass if it is overly porous or vuggy. However, this is not a concern in the mineralized units at PGP due to the very low porosity.

Table 11-18: Summary of Mean SG Values by Premier Gold Project Deposits

Deposit	Bulk Density used for Resource Estimate
Premier	2.85
Big Missouri	2.80
Silver Coin	2.80
Dilworth	2.80
Martha Ellen	2.80

Specific Gravity Determinations—Premier

Specific gravity determinations were collected by ALS from core sample pulps using a pycnometer. As in earlier programs, ALS utilized a WST-SIM pycnometer instrument with methanol. A total of 2,104 readings were taken between 2014 and 2017. Average SG values, by rock type, are listed in Table 11-19.

Table 11-19: Summary of SG Values by Rock Type

Rock Type	No. of Samples	Mean SG
All Data	1,994	2.85
Andesite	1,009	2.84
Breccias	715	2.87
Porphyry	270	2.82

Specific Gravity—Big Missouri, Martha Ellen, Dilworth

Specific gravity determinations were measured from core samples by SGS and ALS) using a pycnometer.

Between 2011 and 2012 SGS measured SG with a Penta helium gas pycnometer using the concept of inert gas expansion (Boyle's Law) to determine the true volume of a solid sample. In 2013 ALS utilized a WST-SIM pycnometer instrument with methanol.

A total of 2,496 readings were taken between 2011 and 2014 with an average SG of 2.80. The average SG is 2.82 for samples with Au gold above 2.5 g/t. A value for SG of 2.80 has been used in the Resource Estimate for these three deposits.

Specific Gravity—Silver Coin

During the 2011 Silver Coin drill program, density determinations were systematically made using the water submersion method. Rock samples were weighed using wire baskets in water and in air and a value was calculated from these compared values. Bulk density measurements were taken on core samples selected every 2 m to 6 m. A total of 2,852 determinations were made in 2011, and there is also a legacy group of pre-2011 values totaling 266 values recorded using the same water submersion method. The weighted average mean SG of all these measurements is 2.80.

11.2 Red Mountain

11.2.1 Sampling Methods at Red Mountain

Soil Sampling

The methods used by Bond and LAC for collecting soil samples at Red Mountain is not known. IDM collected their 2014 soil samples from the B horizon, or in steeper areas talus fines were collected. In both cases samples were placed in paper soil sample bags.

Rock and Channel Sampling

The methods used by Bond and LAC for collecting rock samples at Red Mountain is not known; however, the MS Access database lists a number of different types including grab, chip, chip-channel, panel, and trench. All of these would be considered standard field rock-sampling techniques.

IDM collected rock samples using geological rock hammers. Channel samples were collected with the use of a portable rock saw. Channel samples were all approximately 1.0 m long and 5 cm wide and deep. The samples were chipped out using a chisel after being cut with the rock saw (McLeod, 2014).

Drill Sampling

Bond and LAC—1989 to 1992

Drill core samples from 1989 to 1992 were collected over 1.50 m intervals regardless of geology. After geological (and some geotechnical) logging of the core was completed, BQTK-sized core was manually split in half. One-half was submitted for sample preparation and analysis, and the other half was kept for future reference at the core storage facility in Stewart, BC.

LAC—1993 to 1994

Drill core samples from the 1993 and 1994 programs were typically collected over 1.0 m intervals, and occasionally over 1.50 m intervals. In some cases, effort was made to break sample intervals at lithological or mineralogical boundaries, resulting in sample intervals shorter than 1.0 m. After detailed geotechnical and geological logging was completed, the core was sawn in half. As in previous programs, half of the core was submitted to the lab for sample preparation and analysis. The second half of the core was stored at the core storage facility in Stewart, BC.

During these large programs, up to four diamond-blade rock saws were running to cut core. A foreman was hired to oversee core sawing, sample tags, and standard insertion to ensure that this process worked efficiently, and to ensure good quality control. A sample sheet, with sample numbers and from–to distances filled in by the logging geologist, was used to assure as best as possible that sample numbers corresponded with the right intervals when samples were collected.

Royal Oak—1996

Royal Oak typically collected samples over 1.0 m (underground and surface) and 1.5 m (surface intervals), and these lengths comprise over 75% of their samples. Minimum and maximum sample lengths are 0.3 m to 6.0 m respectively. Sampling was carried out at the camp in Goldslide Creek where sample intervals were sawn. Multi-part sample tag portions were inserted into the core boxes between each sample interval, with the other part placed in the sample bag.

Banks Island—2013

Banks Island sampled over 0.25 m to 1.5 m intervals that honored geological boundaries. It is known that the core was sawn; however, no other sampling procedures, or the location where sampling was carried out, were documented.

IDM—2014 to 2018

Samples from the 2014 IDM drilling program were collected over 1.0 m intervals for a majority of sampling, and never less than 0.5 m long, and seldom crossed lithological boundaries. Sampling took place at the camp in Goldslide Creek. The core was sawn, and the upper half was placed in a sample bag and sent for assay. Sample tags were placed in the bag and under the second half of the core in the boxes. The core is stored on pallets at the camp on Red Mountain.

Sampling protocols at Red Mountain were the same over the 2016 to 2018 programs, with the exception that in longer sections of suspected barren to low-grade low rock, particularly in some of the surface drill holes, 1.5 m samples were taken. Additionally, in 2017, for 20 HQ diameter underground holes drilled for metallurgical samples, a full half was sent for the test work, ¼ was sent for regular assay and ¼ was retained for future reference.

Whole Rock Samples

During the Bond and LAC drilling programs at Red Mountain, samples were collected from drill core for whole rock analysis. Samples were collected every 20 to 30 m, or with major lithological changes. Proximal to or within the mineralized zones samples were taken every 10 m. Samples were half core and a minimum of 0.5 m long. For samples already selected for conventional assay a portion of sample pulps were submitted for whole rock analysis.

LAC Underground Chip Samples—1993 to 1994

During the 1993 and 1994 programs the ramp and crosscut faces were sampled after every round. Chip samples were collected from fresh faces using a grid with 1.5 x 1.5 m panels, with each face being three panels wide by two panels high. Chips were collected evenly from within the panels.

LAC Bulk Samples—1993 to 1994

A muck sample was collected from every underground round, either from the main decline or from the cross cuts designed to assess the Marc zone mineralization. From crosscut rounds within potential ore, and for several rounds on either side, the muck was stockpiled on surface. A grid was overlain on the stockpile and 20 samples were taken from each round. If the average grade of the resulting assays was less than 2.0 g/t Au the muck was put onto the waste pile. If the average grade was over 2.0 g/t Au, the stockpiled muck was taken through the bulk sampling process. Twenty-three rounds from the underground were treated in this manner.

11.2.2 Analytical Laboratories

Several primary laboratories have been used for Red Mountain samples over the history of the project as shown in Table 11-20. For the majority of drill hole samples, Eco-Tech was the primary laboratory.

Table 11-20: Red Mountain Laboratory Summary

Operator	Laboratory	Time Period	Sample Type Analysed
Bond Gold/LAC	Min-En Labs, North Vancouver, BC	1989–1991	Surface drill hole samples
Bond Gold/LAC	Bondar-Clegg, North Vancouver, BC	1989–1992	Check assays on drill pulps
LAC	Acme Labs, North Vancouver, BC	1989–1991	Whole rock samples
LAC	Acme Labs, North Vancouver, BC	1992	Surface drill hole samples
LAC	Eco-Tech Labs, Stewart, BC	1993–1994	Surface and underground drill hole samples
LAC	Chemex, North Vancouver, BC	1993	Overflow drill samples
LAC	X-RAL, Don Mills, ON	1993	Whole rock samples
LAC	Chemex, North Vancouver, BC	1994	Whole rock samples
LAC	Chemex, North Vancouver, BC	1993–1994	Check assays on drill rejects and pulps
Royal Oak	Eco-Tech Labs, Kamloops, BC	1996	Surface and underground drill hole samples
Royal Oak	Bondar-Clegg, North Vancouver, BC	1996	Check Assays on drill pulps
NAMC	Chemex, North Vancouver, BC	2000	Check assays on drill rejects and pulps
Banks Island	AGAT, Mississauga, ON	2013	Surface drill hole samples
IDM	Acme (BV), Vancouver, BC	2014	Surface drill hole samples
IDM	ALS Global, North Vancouver BC	2016–2017	Surface and underground (UG) drill samples, rock samples
IDM	ActLabs, Kamloops, BC	2016	Check assay on drill pulps
IDM	MS Analytical, Langley, BC	2017	Check assay on drill pulps
IDM	MS analytical, Langley, BC	2018	UG drill samples, surface rock, and channel samples

The ISO accreditations of all labs from 2000 and prior is not known. AGAT Labs, Acme (Bureau Veritas), ALS Global, ActLabs, and MS Analytical are all ISO 9001:2008 accredited laboratories. All laboratories are also ISO/IEC 17025:2005 accredited for some specific tests including FAs with AA and gravimetric finishes.

11.2.3 Sample Preparation and Analysis at Red Mountain

Sample Preparation

Drill core samples were dried as required, crushed, and a sub-sample was then pulverized to produce a pulp sufficient for analytical purposes. Table 11-21 summarizes the sample preparation procedures used by the primary and, where applicable, by the check assay laboratories. Note that crushing and grinding practices for Acme (Bureau Veritas) have changed between work carried out in 1992 and 2014.

Table 11-21: Sample Preparation Procedures at Red Mountain

Laboratory	Procedure
Min-En	Dry, 2 stage crushing to -1/8", 500 g split pulverized to 95% passing -120 mesh
Bondar-Clegg	Dry, crush and pulverize to -150 mesh (on rejects only for checks)
Eco-Tech	Dry, crush to -10 mesh, 250-400 g split pulverized to 85% passing -140 mesh
Acme Labs	Dry, crush to -10 mesh, 250 g split pulverized to 85% passing -150
Chemex	Dry, crush to -10 mesh, 200-300 g split pulverized to 90% passing -150 mesh
AGAT	Dry, crush to 75% passing -10 mesh, 250g split pulverized to 85% passing -200 mesh
Acme (BV)	Dry, crush to 70% passing -10 mesh, 250-gram split pulverized to 85% passing -200 mesh
ALS Global	Dry, crush to 70% passing -10 mesh, 1,000-gram split pulverized to 85% passing -200 mesh
MS Analytical	Dry, crush to 70% passing -10 mesh, 250-gram split pulverized to 85% passing -200 mesh

For the 1993, 1994, and 1996 programs all sample preparation by Eco-Tech was carried out at their facility in Stewart, BC. For the 2013 Banks Island program samples were prepared at the AGAT facility in Terrace, BC. For the 2014 IDM program, samples were prepared at the Acme facility in Smithers, BC before being forwarded to Vancouver, BC, for analysis. The 2016 samples were prepared at ALS Global in Terrace, BC.

Sample Analysis

The analytical methods used on drill core and check assays from Red Mountain are summarized in Table 11-22.

Table 11-22: Analytical Methods

Laboratory	Procedure
Min-En	Fire assay for gold on a 30 g sample with an AA finish. Results over ~17 g/t re-assayed with a gravimetric finish. Multi element ICP package.
Bondar-Clegg	Fire assay for gold and silver on a 30 g sample with an AA finish. Results over ~7 g/t re-assayed with a gravimetric finish.
Acme	Fire Assay for gold on a 30 g sample, Multi element ICP on a 0.5 g sample. Whole rock by lithium borate fusion with an ICP finish.
Eco-Tech	Fire assay for gold on a 30 g sample with an AA finish. Results >10 g/t Au re-assayed with a gravimetric finish and if >30 g/t Au a metallic assay was performed. Silver assayed using an aqua regia digestion and an AA finish on a 2 g sample. 31 element ICP package.
XRAL	Whole rock analyses by XRF.
Chemex	Fire assay for gold on a 30 g sample with an AA finish. Results >10 g/t Au re-assayed with a gravimetric finish and if >30 g/t Au a metallic assay was performed. Silver assayed using an aqua regia digestion and an AA finish. Also, multi element ICP on 1993 over flow samples. Whole rock analyses by XRF.

Laboratory	Procedure
AGAT	Fire assay for gold on a 30 g sample with an ICP-OES finish, results >10 g/t re-assayed using a gravimetric finish. 45 element ICP-OES package with aqua regia digestion.
Acme (BV)	Fire assay for gold on a 30 g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. 36 element ICP-ES on a 0.25 g sample.
ALS Global	Fire assay for gold on a 30 g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. Silver by Acid digestions with AA finish, repeated if >100 g/t Ag, 48 element 4 acid, ICP-MS package.
MS Analytical	Fire assay for gold on a 30 g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. Silver over-limits from ICP by fire assay with gravimetric finish, multi-element 4 acid ICP package

For the 1993, 1994, and 1996 programs, most gold and silver analyses were performed at Eco-Tech's Stewart facility, while the ICP analyses were carried out at Eco-Tech's Kamloops facility. The exception for this is for late 1994, starting in November, when the Eco-Tech's Stewart analytical facility closed and both FA and ICP work was done at the Kamloops facility. For the 1996 Royal Oak samples, all analytical work was carried out at Kamloops.

11.2.4 Quality Assurance and Quality Control at Red Mountain

The QA/QC for the Red Mountain drilling programs has previously been presented by Anderson (2000) and reported in Craig (2001) and Smee (1993). All historical QA/QC data were recompiled and assessed in early 2016.

Bond and LAC Quality Assurance and Quality Control—1989 to 1992

There is little, if any, information regarding the insertion of QA/QC materials (standards, blanks, duplicates) into the sample stream by Bond or LAC prior to 1993.

A significant amount of check assaying was carried out on samples from the 1989 to 1992 drill holes with 1,243 (1,121 pulps and 122 rejects) of 13,256 samples (9.48%) submitted to Bondar-Clegg.

The compiled data show small to modest high biases for the Bondar-Clegg check assay analyses. For gold, Bondar-Clegg results were 2.8% and 4.73% higher than the original Min-En results for pulps and rejects, respectively. For silver, Bondar-Clegg results were 1.02% and 2.3% higher than Min-En for pulps and rejects, respectively. Four samples, two pulp and two reject, were removed from the analysis due to outlier results in the Bondar-Clegg dataset.

The results indicate good assay accuracy between the two labs. The higher bias in the rejects results may be due to the preparation of a second pulp from a second split.

LAC Quality Assurance and Quality Control—1993 to 1994

Standards

LAC initiated the use of standards in 1993, but the number was very limited, at only 53 in total. The standards used were Canmet standards as shown in Table 11-23. Note that in 2000, ±2 SD were used as the failure limits for all standards. Current industry standards are to use ±2 SD as a warning limit and ±3 SD as failure limit; this has been followed here.

Table 11-23: Red Mountain Canmet Standards

Standard Name	Value Au (g/t)	+3 SD	-3 SD
MA-1b	17.00	16.55	17.45
MA-2b	2.39	2.47	2.31
MA-3	7.49	7.78	7.2

When drilling recommenced in April 1994, a more stringent standard insertion program was instituted with an insertion approximately every 20 samples. While some of the 1993 Canmet standards were used, for this program four site specific standards were created by CDN, using material from the Marc zone bulk samples (Sanderson, 1994). Material was crushed, pulverized to –200 mesh and then homogenized. Splits were taken for round-robin analysis and sent to six assay laboratories: Bondar-Clegg, Chemex, CDN Resource, Acme Analytical, Min-En, and Eco-Tech. Each lab received five splits of each standard, and two assays were performed on each split. Standard values and ±3 SD failure limits, based on the round-robin results and analysis, are presented in Table 11-24.

Table 11-24: Red Mountain LAC Site Specific Standards

Standard Name	Value (Au g/t)	+3 SD	-3 SD
LAC #1	1.90	2.35	1.45
LAC #2	3.19	3.70	2.68
LAC #3	6.35	7.34	5.36
LAC #4	14.15	16.07	12.23

The results of the standard insertions from the 1993 and 194 LAC drilling programs are summarized in Table 11-25.

Table 11-25: Summary of Standard Insertions at Red Mountain

Standard	Number of Analyses	Mean of Analyses	Expected Value	Percentage Difference	No. High Fails	No. Low Fails
MA-1b	22	17.10	17.00	+0.6	5	3
MA-2b	39	2.11	2.39	-11.7	1	29
MA-3	37	7.26	7.49	-3.1	2	13
LAC 1	235	1.91	1.90	+0.5	6	1
LAC 2	242	3.18	3.19	-0.3	3	2
LAC 3	281	6.57	6.35	+3.5	2	2
LAC 4	124	14.50	14.15	+2.4	0	0

In general, the Canmet standards did not perform well relative to their ±3 SD failure limits. Many failures may be attributable to quite tight failure limits relative to standards of similar grades from other commercial suppliers, as the ranges for a majority of results for each standard appear to indicate reasonable accuracy. The majority of failures were low relative to the expected values, suggesting that assay data may underestimate gold values.

The LAC standards performed well, indicating good assay accuracy. Standards LAC 1 and LAC 2 show no biases relative to the expected values. Standards LAC 3 and LAC 4 do show small positive biases, but most values still fall within the ±3 SD failure limit. Examples of timeline plots are shown in Figure 11-31 (LAC 2) and Figure 11-32 (LAC 3). Note that a tightening of results relative to the expected values is evident in both plots at approximately samples 185 and 210, corresponding to the moving of all analytical work from the Stewart Eco-Tech facility to the Kamloops facility.

During the LAC programs analytical results for standards were tracked and if results were out of acceptable limits, the lab was asked to re-assay all samples that were analyzed in the same batch as the standard.

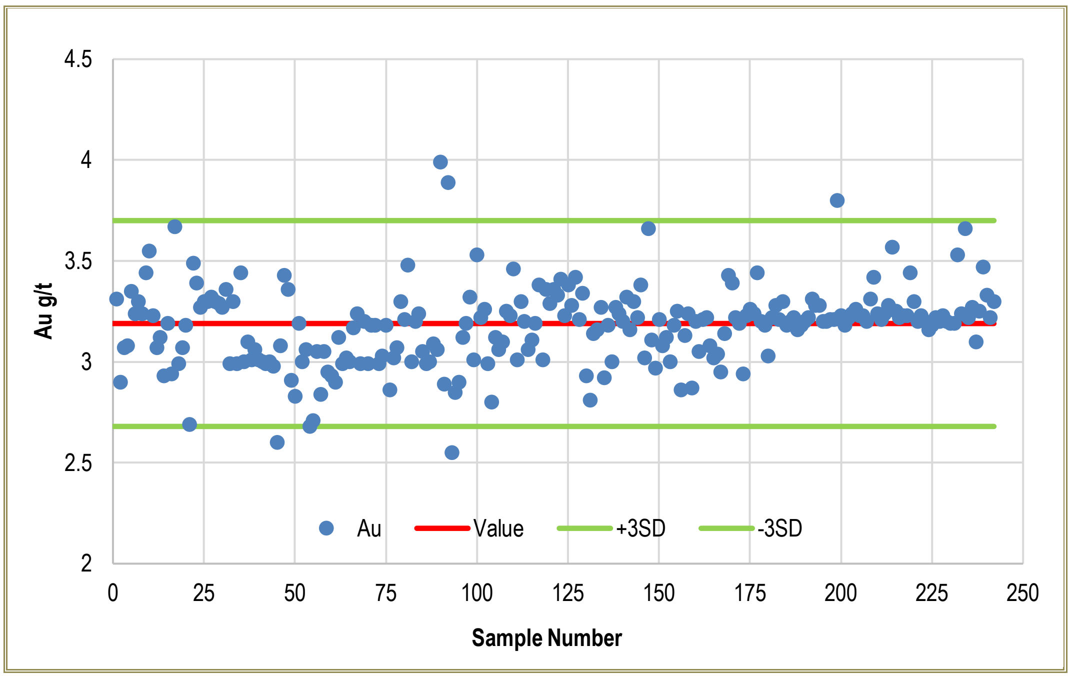


Figure 11-31: Timeline Plot for Standard LAC 2

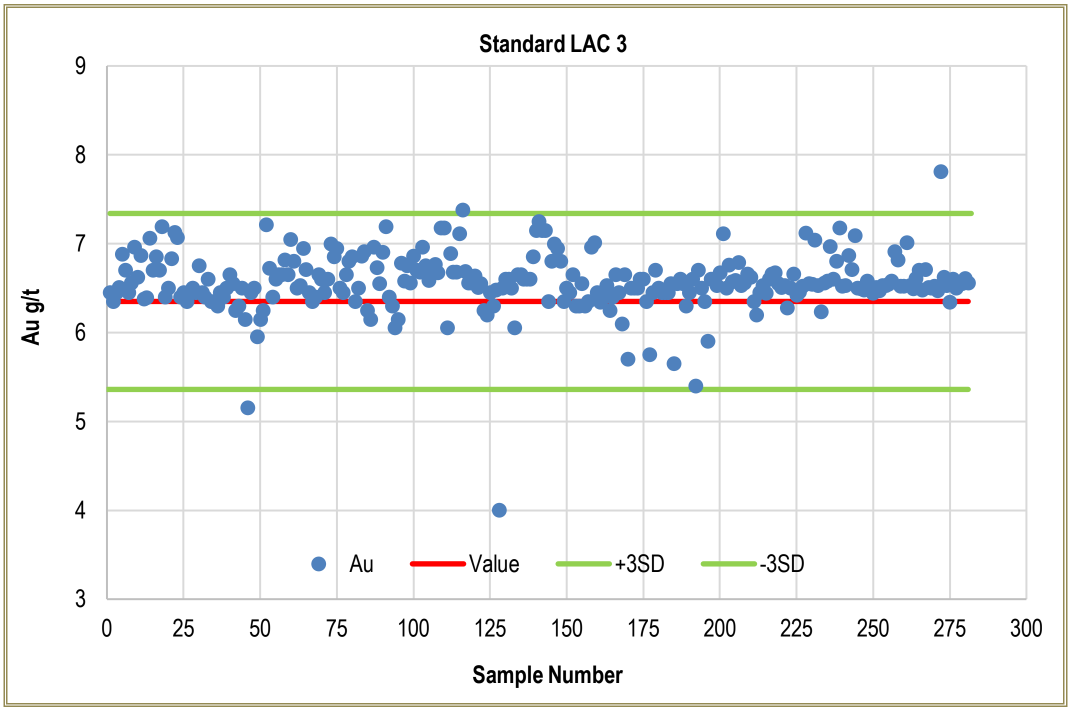


Figure 11-32: Timeline Plot for Standard LAC 3

Check Assays at Red Mountain

A rigorous check assay program was implemented by LAC in 1993 with a protocol whereby, for 1 in every 10 pulps and 1 in every 20 duplicates, half were to be submitted to Chemex for check assay. This protocol was not used in 1994 and instead two cross sections, both in the AV zone, were chosen for check assays. From Section 1400N rejects were sent to Chemex, and from Section 1500N pulps were sent to Chemex.

In total, check assays were submitted from 168 of 301 surface and underground LAC drill holes, totalling 3,060 checks from 31,064 original samples (9.9%). The samples actually submitted did not end up in the proportions suggested by the protocols with 371 pulp submittals and 2,689 reject submittals. In all, 925 check assays in the historical MS Access database were not included in the compilation, as the material, whether pulp or reject, could not be determined. Results are summarized in Table 11-26.

Table 11-26: Summary of Check Assays at Red Mountain—1993 to 1994

Material	Number	Eco-Tech Au	Chemex Au	% Diff	Eco-Tech Ag	Chemex Ag	% Diff
Pulp	371	1.71	1.83	+7.0	7.31	7.37	+0.8
Reject	2,689	3.02	2.84	-6.0	13.44	13.04	-3.0

The pulp check assay results show a modest to strong high bias by Chemex for gold, and a very small high bias for silver. The high bias for gold occurs in samples with values of over 3.0 g/t. With the reject checks there is a consistent low bias by Chemex at all grade levels compared to Eco-Tech. This bias has not been resolved, although it is possible that fine gold could have settled during transport of the rejects, resulting in lower values.

The influence of a different level of sample support (original pulp versus new pulp from a second split of rejects) is also not known.

No standards or blanks were included with check assay shipments to Chemex.

Duplicates

Anderson (2000) reported a LAC 1993 to 94 duplicate database consisting of 369 samples. From twenty-one 1994 underground drill holes, a high and low-grade sample was collected within the mineralized zones for each hole. The first half of core was assigned a sample number and the resulting pulps were analyzed twice. The second half was assigned a new sample number and also analyzed twice. If needed, gravimetric and metallic assays were carried out. Additionally, four holes (U94-1155, 94-1156, U94-1157, and U94-1158) were drilled in the Marc zone, on Section 1275N, in a 1.0 m box spacing to test variance. The first three of these and hole U94-1160 were sampled from top to bottom and original and duplicate halves were analyzed (no extra pulp splits).

The comparison of results from the first pulp from both original and duplicate halves of the core (n=369) for the global dataset show extremely good assay precision with the originals having a mean of 8.02 g/t Au and the duplicates having a mean of 8.05 g/t Au. On an individual assay basis, there is some modest variability, probably reflecting differing proportions of the sulfide veins in opposing halves of core.

Duplicate Holes

During 1994, four short drill holes were drilled on section 1275 N from collar points 1 m apart in a square pattern. As well as to serve as individual assay duplicates, the purpose was to evaluate the variance within the stockwork zone over full intersection distances. Table 11-27 summarizes the weighted assay averages for the higher-grade intervals in the four drill holes from 13 to 29 m.

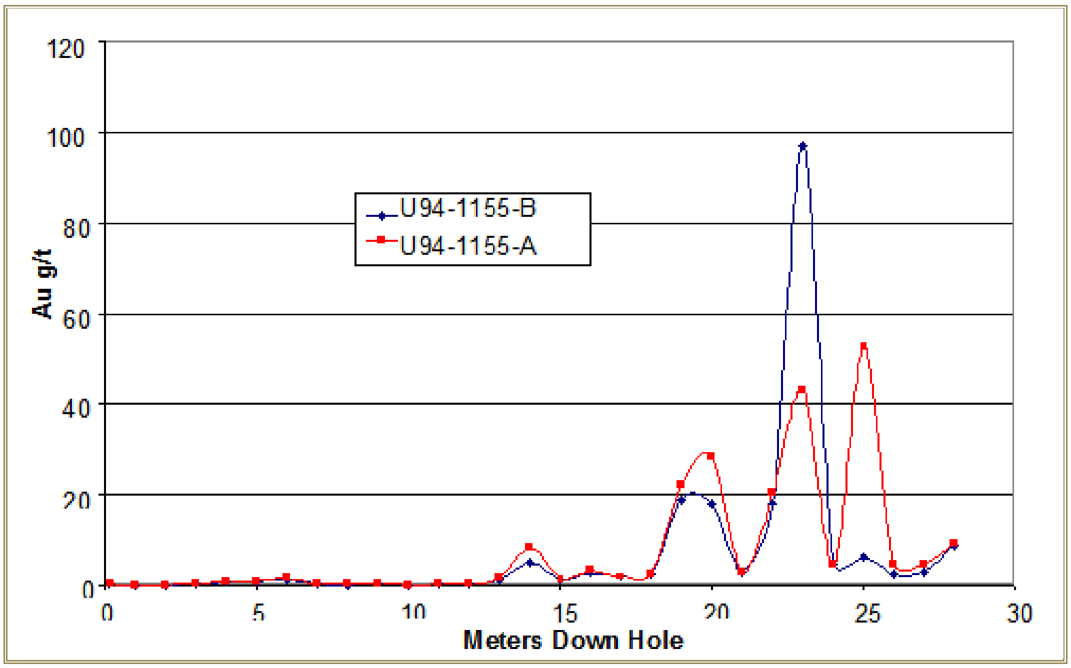
Table 11-27: Weighted Assay Averages

Drill Hole	From 13 to 29 m (Au g/t)
U94-1155	18.21
U94-1155, second half	12.11
U94-1156	16.43
U94-1156, second half	17.48
U94-1157	19.96
U94-1157, second half	18.32
U94-1158	16.31

Figure 11-33 to Figure 11-35 display the down hole assay comparisons for each half of the core for holes U94-1155 to U94-1157. Figure 11-36 displays the variance of holes U94-1155 to U94-1158 for the first ½ split of core in each hole.

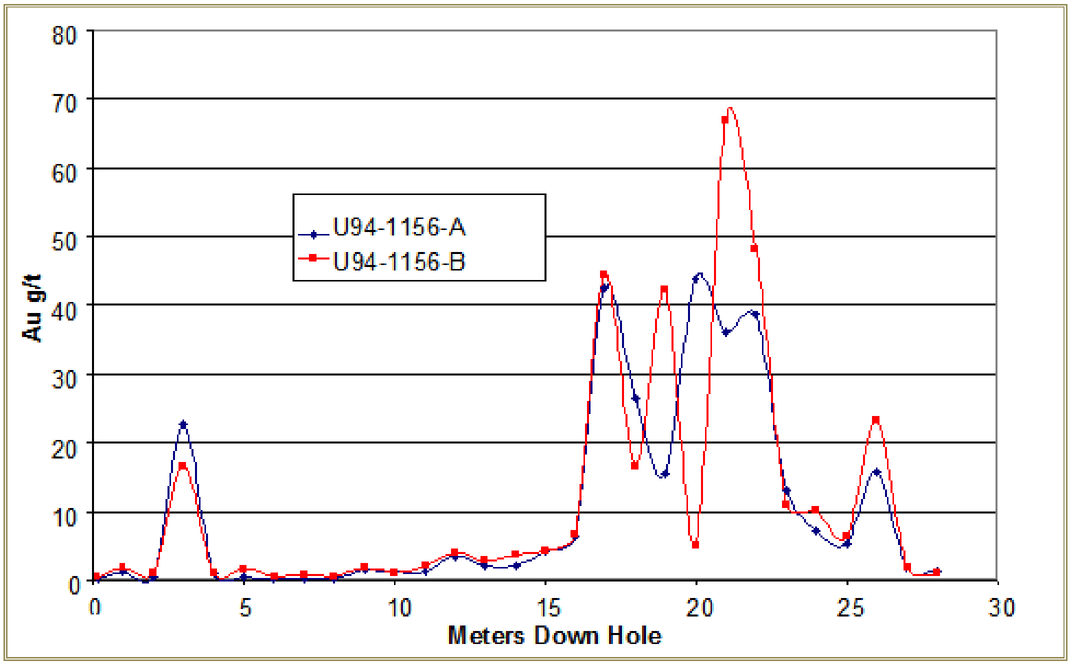
Variance on an assay by assay in the two half-split comparison is relatively normal for a gold deposit and affects almost all ranges of assays. This would be expected in the Red Mountain style of stockwork. Stereonet analysis of the stockwork veining show that only 20% of the veins have a consistent trend within the stockwork envelope (Barclay, 2000) with the balance being relatively random. This randomness and rapid thickening and thinning over sub-metre and sub-centimetre distances was observed in both core and cross-cuts, and is an explanation for variance in grade as gold grade is associated with the percentage of coarse pyrite in a given interval.

This variance is evident in the four individual drill holes (Figure 11-33 to Figure 11-35). When these plots are considered in conjunction with the mean results for the 369 duplicates presented above (which suggest extremely good global precision), it is evident that variability on an individual sample basis can be considerable, particularly at higher grades, as can be seen in Figure 11-37.



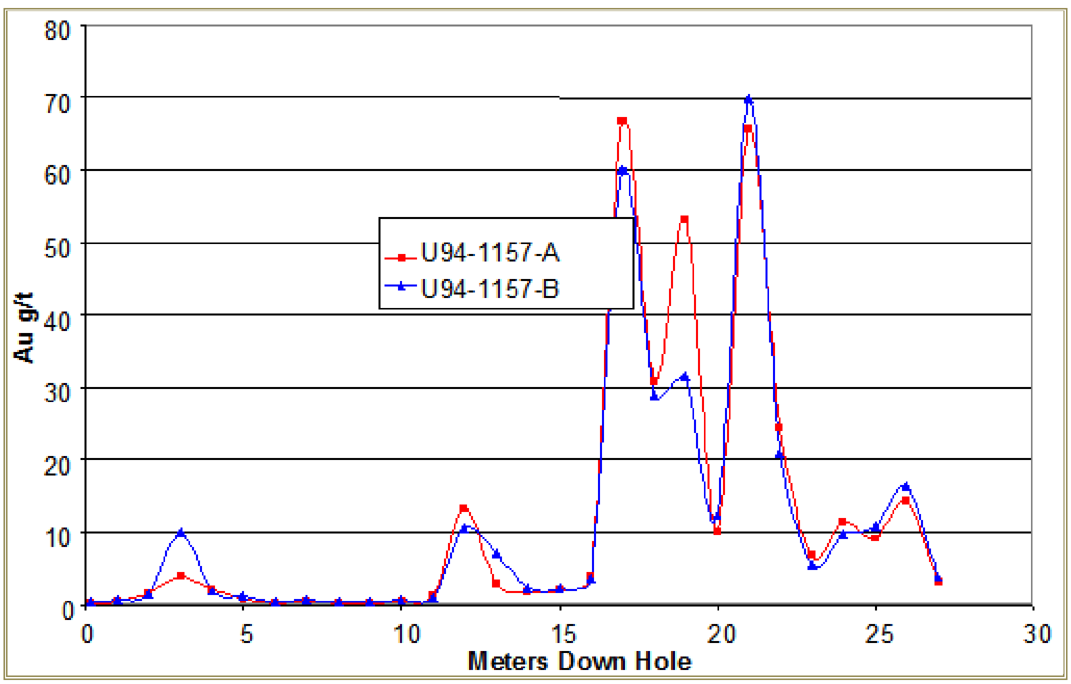
Source: NAMC (2001)

Figure 11-33: U94-1155 Gold Assay on Both Halves of Core



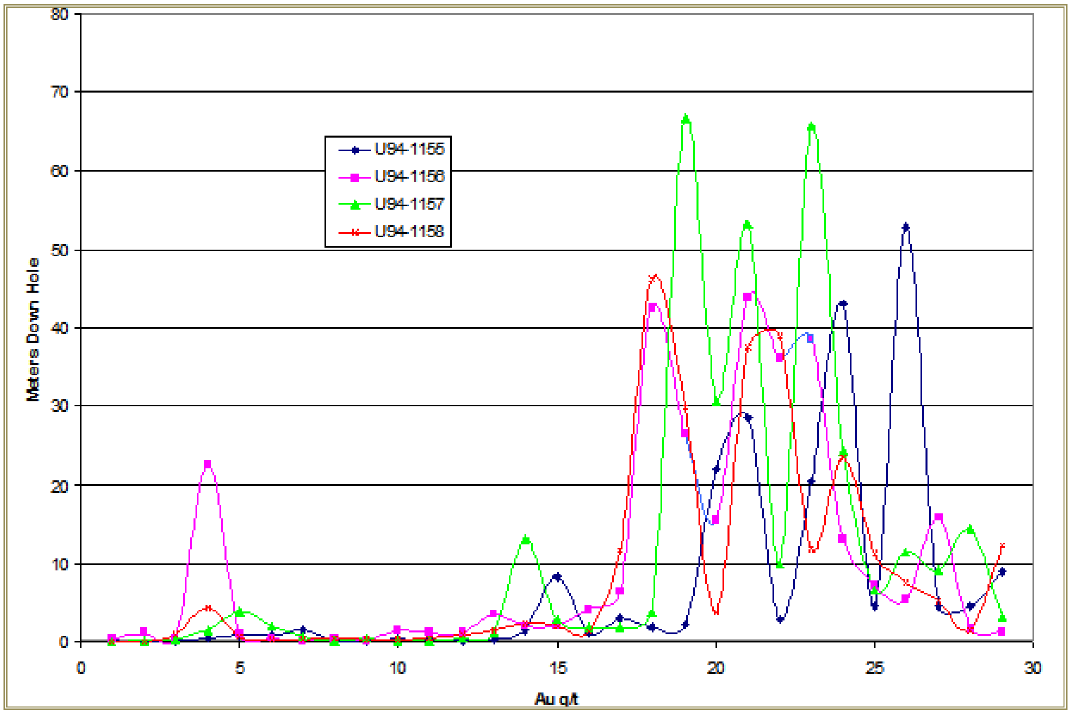
Source: NAMC (2001)

Figure 11-34: U94-1156 Gold Assays on Both Halves of Core



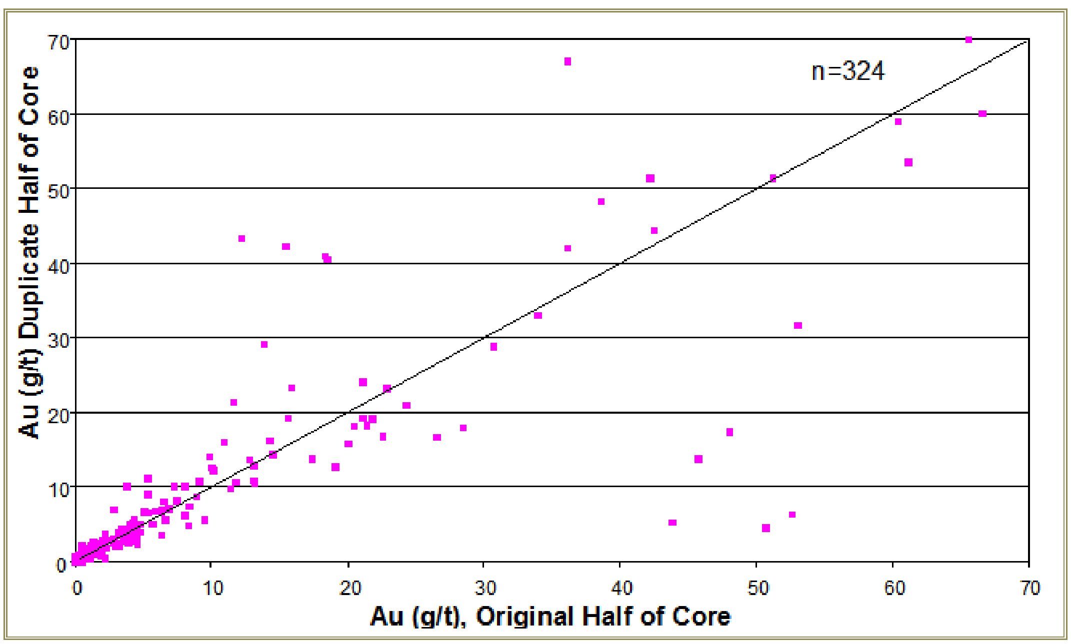
Source: NAMC (2001)

Figure 11-35: U94-1157 Gold Assays on Both Halves of Core



Source: NAMC (2001)

Figure 11-36: Gold Assay Comparison for DDH U94-1155, -1156, -1157, and -1158



Source: NAMC (2001)

Figure 11-37: Comparison of Original Gold Assays vs. Duplicate Halves of Core

Lab Audits/Visits

An important part of LAC's QA/QC program were routine visits to the Eco-Tech laboratory facilities in Stewart. This was done by a LAC geologist on a regular basis during the 1993 and 1994 programs.

Early in 1993 Eco-Tech had a small facility in Stewart which could not cope with the large volume of samples, and the quality of some results were suspect. In order to resolve this Eco-Tech built a separate sample preparation facility in July 1993, which was inspected by a sampling consultant form Vancouver who considered the updated facilities adequate.

In 1994 a second consultant, Jack Stanley, was contracted to visit the Eco-Tech lab and audit sample preparation, assaying procedures, and internal lab QA/QC. He made two visits and, on each occasion noted some issues that were subsequently addressed (Stanley, 1994a, 1994b, and 1994c).

Extra Sample Splits

In 1994 at least 1 in 40 samples had two assay splits from the coarse (-10 mesh) sample taken and 1 in 40 samples had a duplicate assay done on the assay pulp. When a duplicate assay was carried out by Eco-Tech on the same pulp, the average was given on the analytical certificate for the sample result, with the two individual results given at the end of the certificate with other QA/QC data. With samples with a second pulp (re-split), the assay from the original pulp was given as the sample result with the re-split result at the end of the certificate. As noted by Smit (2000) the individual assays were never compiled, but would be useful if done, as an additional assessment of sample variance.

Royal Oak—1996

Royal Oak did not include QA/QC materials in their drill-hole sample shipments, but they did submit 221 pulps to Bondar-Clegg for check assay. For both gold and silver Bondar-Clegg results exhibited small low biases relative to the original Eco-Tech results. None of the Royal Oak holes are currently within resource areas.

NAMC Quality Assurance and Quality Control 2000

NAMC submitted 197 samples, 167 of pulp and 30 of reject from mainly 1993 and 1994 drill holes in the Marc and AV mineralized zones for check assay. The results for this modest program indicated that Chemex was biased low relative to the original results by ~4.5% for gold, for both pulps and rejects.

Results for nine LAC standards (three different) included with these check samples indicate good assay accuracy.

Banks Island—2013

Banks Island inserted standards, coarse field blanks, and pulp duplicates in their sample stream, at a rate of one for every 20 samples. In addition, they randomly inserted a few pulp blanks.

Details of the standards and pulp blank purchased from WCM Minerals of Burnaby, BC, are given in Table 11-28. The coarse field blank used came from a local quarry along the highway near the mouth of Bear Creek (55° 57' 26" N, 129°58' 52" W). The rock was from a barren Bitter Creek pluton of quartz monzonite composition.

Table 11-28: Banks Island Standard Reference Material

Standard	Au g/t	+3 SD	-3 SD	Ag g/t	+3 SD	-3 SD
PM 929	5.1	5.81	4.39	65.0	72.5	57.5
PM 451	1.77	1.95	1.59	NA	NA	NA
BL 118	<0.005	NA	NA	<0.3	NA	NA

A total of six standard insertions were made, with all returning values within the ±3 SD limits, but both having average values 7% to 8% below the expected values. Two of nine coarse blanks failed, one after a 13.8 g/t Au sample suggesting contamination, the other unexplained. Visual inspection of the pulp duplicate results for gold indicate good assay precision.

IDM Quality Assurance and Quality Control—2014

IDM inserted one standard every 20 samples and one blank every 20 samples into its 2014 drill sample shipments from Red Mountain. No duplicates were inserted, and no check assays were done.

The standards used were from CDN Labs in Vancouver, with values and limits as shown in Table 11-29. Timeline plots show good accuracy for gold. For silver, Acme is biased high relative to the expected value by about 6%, but most results still fall within failure limits. This bias may be related to the relatively high grade of the standard for an ICP analysis.

Table 11-29: 2014 CDN Labs Standards used at Red Mountain

Standard	Au g/t	+3 SD	-3 SD	Ag g/t	+3 SD	-3 SD
GS13A	13.2	14.28	12.12	NA	NA	NA
GS3M	3.10	3.45	2.85	95.4	103.8	87.0

The field blank used came from the same local quarry as used by Banks Island. All results were within the failure limits of three times detection limit for gold (DL was 5 part per billion (ppb), so failure limit is 15 ppb).

IDM Quality Assurance and Quality Control—2016 to 2018

IDM started using a stronger QA/QC program at Red Mountain in 2016, consisting of a QC material once every 10 samples, rotating between standards, blanks, and field duplicates.

Over three years, eight standards have been used—four from CDN Labs and four OREAS. Expected values and limits are shown in Table 11-30. Timeline plots shows good assay accuracy for both gold and silver. Figure 11-38 and Figure 11-39 show the results for Standard CDN-GS-5Q for gold and silver respectively.

Table 11-30: 2016 Red Mountain Standards

Standard	Au Value	+3 SD	-3 SD	Ag Value	+3 SD	-3 SD
CDN-GS-1Q	1.24	1.36	1.12	40.7	44	37.4
CDN-GS-5Q	5.59	6.12	5.06	60.3	66.2	54.4
CDN-GS-5T	4.76	5.08	4.44	126.0	141.0	111.0
CDN-GS-1V	1.02	1.17	0.87	71.7	79.2	64.2
OREAS 60C	2.47	2.22	2.71	4.87	4.2	5.54
OREAS 62E	9.13	10.36	7.9	9.86	10.88	8.83
OREAS 60D	2.47	2.71	2.23	4.57	5.10	4.04
OREAS 62F	9.71	10.43	8.99	5.47	6.26	4.68

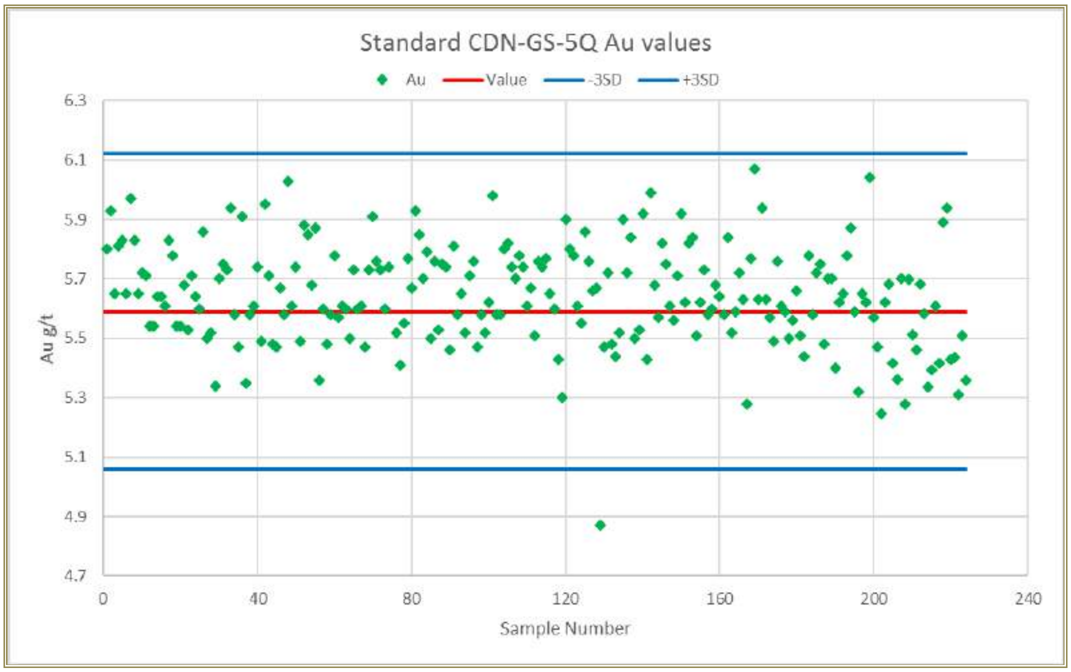


Figure 11-38: Standard CDN-GS-5Q Gold Values

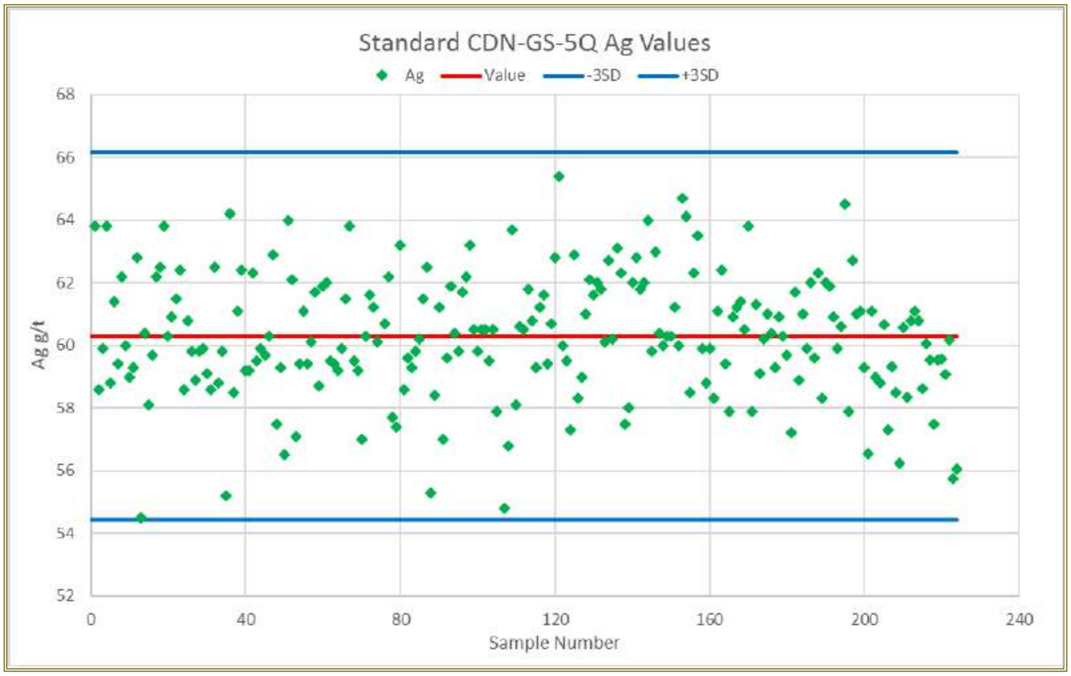


Figure 11-39: Standard CDN-GS-5A Silver Values

The same field blank as used in 2014 has been used in the 2016 to 2018 Red Mountain programs. For gold, there were three failures in the 65–110 ppb range, and a few milder failures in the 20–30 ppb range. The stronger failures were found to follow high-grade gold samples (after 63.8 and 1,400 g/t values) and represent cases of mild contamination. A silver failure of 2.3 g/t Ag could not be explained. The number and tenor of both gold and silver failures are not considered serious and will have negligible effect on resource estimation.

For field duplicates, the full second half of NQ core was submitted for the duplicate sample, and in the case of HQ core the last ¼ core was submitted as the duplicate to match the ¼ core submitted as the original. Both gold and silver show moderate variability at all grade ranges reflecting the variable distribution of coarse stockwork pyrite in original and duplicate pairs.

IDM submitted 98 pulps from 2016 drill holes for check assay to ActLabs in Kamloops. The samples were selected mainly from within mineralized intersections but also included a few samples selected from low-grade to unmineralized sample intervals. Correlations for both gold and silver are good indicating good accuracy between laboratories. A plot for gold is shown in Figure 11-39.

In 2017, ninety-five pulps from 2017 Red Mountain drill holes were submitted for check assay to MS analytical in Langley BC. As with the check assays from 2016, correlations for both gold and silver are good, indicating good accuracy between laboratories.

No pulps from the 2018 drill holes have yet been submitted for check assay.

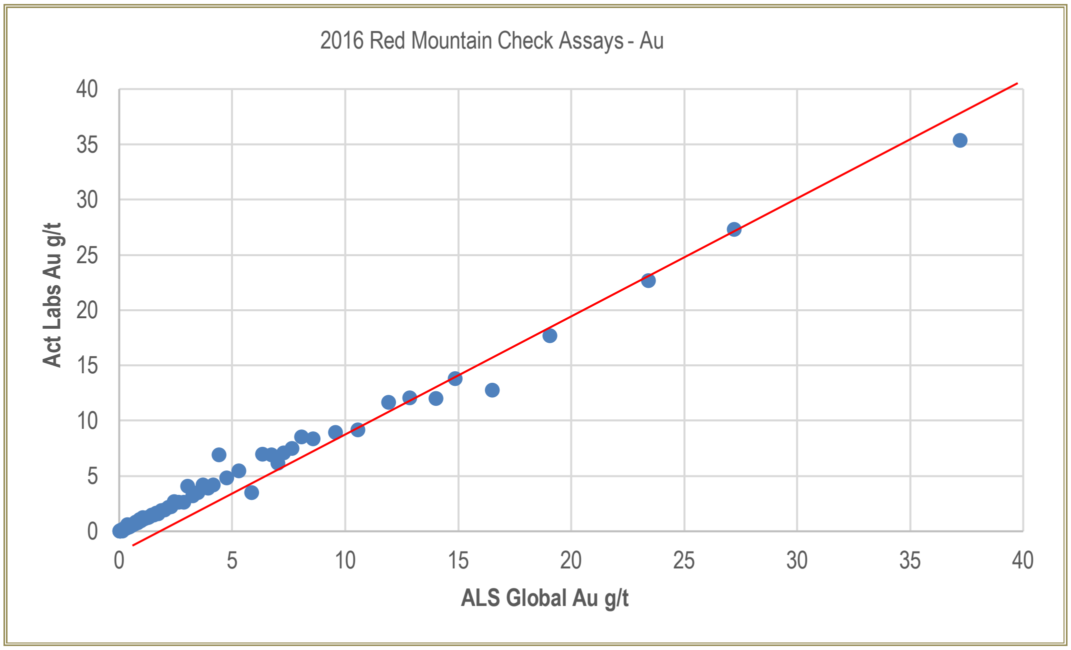


Figure 11-40: 2016 Check Assay Comparison

11.2.5 Qualified Person Comments on Quality Assurance and Quality Control at Red Mountain

The historical QA/QC for Red Mountain is not as robust as current QA/QC programs. Standard and duplicate coverage is weak for some programs, and no blanks were run to test for contamination issues associated with sample preparation on all but the recent IDM drilling programs. However, considering the 1993 and 1994 dates over which most of the historical was carried out, the program was quite strong and extensive for the time. Additionally, strong check assay programs from some of the earlier years mitigate other weaknesses.

Standard results indicate no issues with assay accuracy, as do the check assays that compare pulps to pulps as a measure of inter-lab accuracy. Similarly, true duplicate comparisons indicate good assay precision, although the data set is quite small.

Historical comparisons for some sets of data between original pulp results, and the results of rejects sent as checks or comparisons between differing analyses on the same pulps (e.g., AA vs. gravimetric), or a combination of both, are problematic as the sample support and analytical ranges of the different methodologies are not the same.

Current QA/QC protocols follow standard industry practices and are deemed adequate for inclusion of the assay data in resource estimation.

11.2.6 Red Mountain Databases

Bray (2000) indicates that in 1993 and 1994 all Bond and LAC data were in a series of FoxPro databases. In 2000, these were combined into a smaller number of "master" FoxPro databases and then into a single master MS Access database. This MS Access database contained much of the Project data, and was used in 2000 to populate a Gemcom Red Mountain drill database that formed the basis for the current mineral resource estimate.

Since 2016 IDM has maintained seasonal MS Access databases for drilling and surface samples (soils, rocks).

11.2.7 Sample Security and Storage at Red Mountain

Security

For all Red Mountain drilling programs, samples were under the control of drill contractors and project staff until they have left the immediate area, as it has helicopter access only.

Bond security measures were not recorded at the time, and normal security processes for the period are assumed.

LAC followed a diligent process of flying the core directly to the core storage facility in Stewart, where logging and sampling was carried out under LAC supervision. Samples were delivered directly to the Eco-Tech laboratory in Stewart, accompanied by sample submittal forms.

Royal Oak samples were collected in the Goldslide Creek camp and subsequently delivered from the project area to the Eco-Tech sample preparation facility in Stewart.

NAMC samples were collected by a staff professional geologist and delivered to the Chemex laboratory under the direct supervision of the geologist.

In 2014, samples were shipped in rice bags and delivered from the project to a commercial trucking company based in Stewart. The samples were then delivered to Acme lab's sample preparation facility in Smithers. The same procedure was used in 2016 and 2017 except that sample shipments were delivered to the ALS Global sample preparation facility in Terrace, and in 2018 sample shipments were sent to the MS Analytical sample preparation facility, also in Terrace.

Storage

All drill core from 1989 to 1996 (Bond, LAC, and Royal Oak) is stored in a fenced compound immediately next to the Stewart airstrip. The bulk samples and rejects are also stored in this location, but have deteriorated to a point whereby they are no longer usable.

The Banks Island core was initially stored in the Banks Island warehouse in Smithers. The authors are unaware of the current location of the Banks Island core or if it still exists.

Core from the 2014, 2016, 2017, and 2018 IDM drilling programs is stacked on pallets at the Goldslide Creek camp. Sample rejects have not been maintained. Pulps are currently stored in Seacans in the Red Mountain core storage facility in Stewart.

11.2.8 Qualified Person Comments on Sample Preparation, Analyses, and Security at Red Mountain

In the opinion of the QP the quality of the analytical data is sufficiently reliable to support mineral resource estimation. Sample collection, preparation, analysis, and security were generally performed in accordance with exploration best practices and industry standards as follows:

- Sample collection and preparation for samples that support mineral resource estimation has been in line with industry-standard methods for the pyritic, stockwork hosted gold mineralization that occurs at Red Mountain.

- Drill core samples were analyzed by independent laboratories using industry-standard methods for gold and silver analyses.
- Drill programs have included the insertion of an adequate number of QA/QC materials.
- The QA/QC program results do not indicate any problems with the analytical programs, and demonstrate that the results are accurate and precise.
- Sample security has relied upon the fact that the samples were always attended to by drill crews or company staff while at the site or logging facilities, and delivered to the lab either directly by project staff or commercial trucking companies.
- The data that were collected were entered in databases and validated through visual checks prior to being imported into the master drill database(s).
- Current sample storage procedures and storage areas are consistent with industry standards.

12 DATA VERIFICATION

12.1 Premier Gold Project

12.1.1 Site Visits

Several site visits have been conducted in the past by independent Qualified Persons (QP) as detailed in the previous Premier Gold Project (PGP) report (Rennie and Bird, 2019). The site visits carried out by the current QP are summarized below:

Sue Bird, P.Eng., visited all five deposits at PGP from September 4 to 6, 2018 and from June 17 to June 20, 2019. The site visits included:

- Inspection of the current drilling and drill hole collar locations and survey methods
- Verification of historical drill holes
- Flyover to obtain the general site geology for all five deposits, as well as examination of outcrops and adits
- Discussion of geology and updated structural interpretations including examination of the core for several mineralized intervals
- Discussion with the site geologists of sample preparation, handling, storage, and transportation
- Picking of core samples at Silver Coin for re-assay validation of legacy drilling.

12.1.2 Premier Gold Project Database Checks

The drill hole database for each of PGP's five areas have been supplied by Ascot from their master database in the form of Excel files.

Premier Gold Project Collar Elevation Corrections

It had been noted in 2018 that the Westmin collar elevations were generally higher than the updated LiDAR topography. To correct this, the collar elevations were adjusted to the topography elevation by draping the collar to the current topography. Where there had been previous open pit mining (i.e., the Dago and S1 pit areas), this was not possible because the original topography was not available. Therefore, the adjustment of 4.1 m has been used to adjust the collars in these areas, based on the average correction made where the original topography remained.

Validation of survey data for legacy data was completed for the previous NI 43-101 report (Ascot, 2019). Validation was by visual inspection, cross-reference to other digital files, and checks against hard-copy records. Some field verification using handheld GPS was also conducted. Printouts from GEOLOG records were used to compare to and validate digital files for 836 holes. Some of the holes could not be validated, or were clearly incorrect, and were excluded from the database.

The grid system varied depending on the location within the property area and collar locations had to be manually reconciled by overlaying the plotted information with orthophotos. In the Premier area, the old mine grid was converted to UTM NAD 83 in this manner, and also by translating the elevations by 18.72 m.

Premier Gold Project Collar, Survey, and Assay Database Checks

All drill hole data, when imported to MineSight®, is checked for survey and assay interval errors such as duplicates or overlaps. Assay values are checked for adherence to value limits, missing data, and duplicate entries. Minor errors when data was initially imported have been corrected in the master database and imported files.

Premier Gold Project Assay Certificate Checks

Ascot has provided the assay certificates in pdf format for all areas. Ten percent of the gold assay values and about 2% of the silver values have been checked within areas of mineralization that have been used to inform the block model. Only minor errors were found in this check, giving no cause for concern regarding the integrity of the database.

At Premier up to 6,778 historical assays were checked from the years 1981, 1984, 1987 to 1989, 1990, 1996, 2009, and 2013 to 2019. At Silver Coin up to 5,826 historical assays were checked from 1982, 1983, 1986 to 1990, 1993, 1994, 2004, 2005 to 2008, 2010, 2011, and 2017 to 2019. At Big Missouri, up to 722 historical assays were checked from the year 2019.

Only minor errors were found in these checks, giving no cause for concern regarding the integrity of the database.

All of the core and coarse reject re-assays done in 2016 and 2017 to validate the historical data were added to the Ascot master database and are now used for resource modelling. Therefore, ten percent of these re-assay certificates have also been checked.

Premier Gold Project Validation of Historical Assays—Pre-1999

The coarse rejects and core sample duplicates were re-assayed and compared to the pre-1999 historical data, with the analyses summarized below. The conclusion from this analysis is that above about 0.3 g/t Au the historical data compares well to the re-assayed data and therefore can be used.

12.1.3 Ascot Validation of Westmin Sampling at Premier Gold Project

Beginning in 2016 and carrying on into 2017, Ascot collected rejects from the 1996 Westmin drill holes and had them re-assayed. A total of 6,761 rejects were sent to SGS for analysis. Ascot estimates that approximately 90% of the drill samples Westmin collected at Premier in 1996 have been re-assayed.

In 2017, Ascot conducted a program of reassembling and resampling core from Westmin's drilling programs spanning the period from 1980 to 1995. A total of 1,970 samples were sent to SGS and analyzed for gold by fire assay (FA) with atomic absorption (AA) finish (gravimetric finish for over-limit values) and silver by ICP-AES as part of a 41-element package. The samples were from holes that spanned the period 1980 to 1990, but were mostly from 1987, 1988, and 1990. Ascot personnel were able to salvage parts of 78 holes.

The core had been cross-stacked on pallets and had been left out in the open for some time. As a result, many of the piles had collapsed, rendering much of the core unusable. Most of the core was NQ size with some BQ, and all but approximately 5% of the samples had been taken with a blade splitter as opposed to a saw. The boxes had been marked with Dymo labels which had largely survived, as had most of the footage blocks and some of the sample tags. Where a sample interval could be reliably identified, all remaining core in that interval was collected, bagged, and sent for assay. The analysis is presented in more detail in the January 2019 NI 43-101 report (Ascot, 2019), with a summary analysis of combined results presented here.

Figure 12-1 and Figure 12-2 show ranked scatter plots for gold and silver grams per tonne (g/t), respectively. Both plots indicate slightly higher grades for the re-assay values, and therefore no overall bias in the historical data.

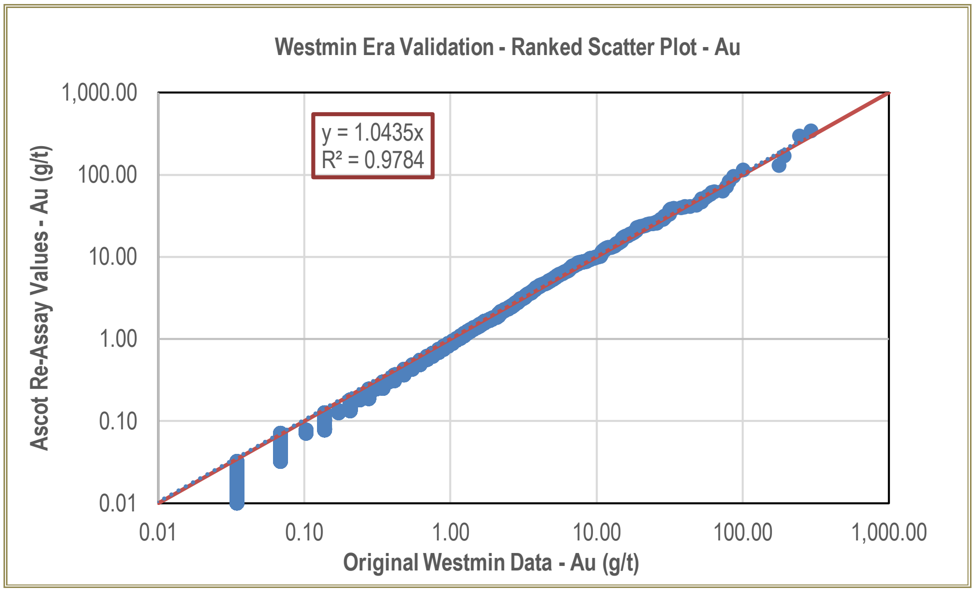


Figure 12-1: Ranked Scatter Plot Comparing Historical Westmin Data to Re-assay Values – Gold

The difference in grade distribution for Au below about 0.3 g/t (0.01 oz/t) is concluded to be due to the higher detection limit the historical Westmin lab used FA with gravimetric finish compared to SGS's AA finish. This value corresponds to 0.01 oz/t, which seems to be a likely lower detection limit for the time period of Westmin drilling. Since 0.3 g/t Au is well below the cutoff grade of 1.0 g/t AuEq used for the wireframe building and of 3.5 g/t gold equivalent (AuEq) used for reporting the Resource Estimate, these differences are considered immaterial.

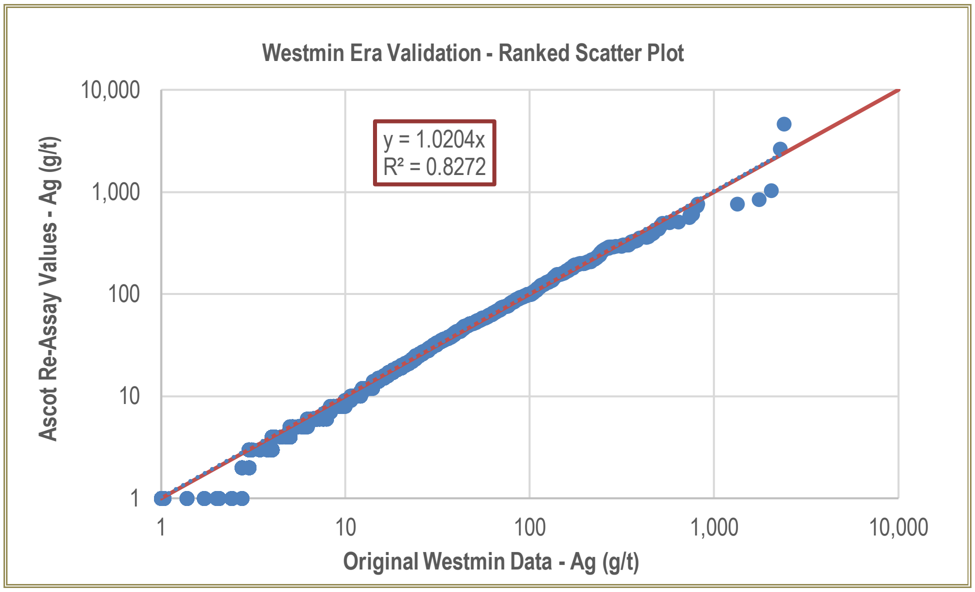


Figure 12-2: Ranked Scatter Plot Comparing Historical Westmin Data to Re-assay Values – Silver

The difference in grade distribution for silver below about 2 g/t is due to the fact that the detection limit of the re-assays is 2 g/t (SGS lab), whereas for the Westmin data the detection limit was 1 g/t Ag.

As stated above, the results obtained in the rejects re-assay program do not indicate any issues in the Westmin laboratory. Similarly, Ascot's external assay quality assurance / quality control (QA/QC) protocols indicate that the SGS laboratory is producing reasonable results.

12.1.4 Ascot Validation of Tenajon Data—Silver Coin

Due to the lack of knowledge about Tenajon-era drilling and assay protocols, a re-assay program was undertaken in 2019 to check the Tenajon data. Finding good samples proved difficult due to the age of the core and the fact that the core boxes had been stored outside. In many instances these boxes had broken and the samples were no longer viable for re-assay. A total of 42 core samples in the areas of SC used for wireframing were selected and sent to SGS for re-assay of gold and silver. The comparison results are presented below. The plot for gold required that an outlier for both the original and the Tenajon had to be removed because of inconsistent results. The remaining data provided a very good correlation with the re-assay values, slightly higher than the original assay for both gold and silver. The conclusion from this analysis is that the Tenajon data are of good enough quality to be used in the interpolations for the Resource Estimate of Silver Coin.

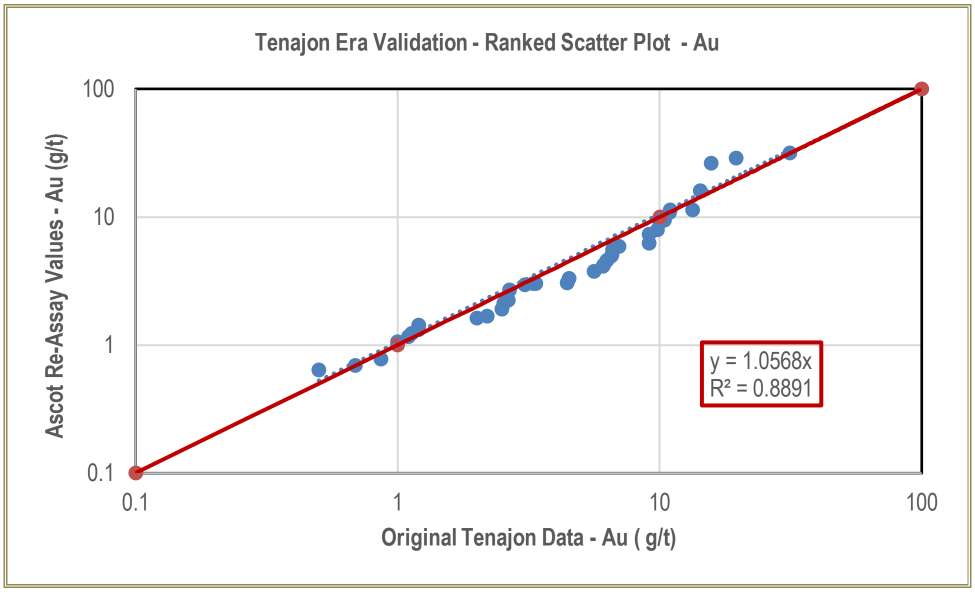


Figure 12-3: Ranked Scatter Plot Comparing Historical Tenajon Data to Re-assay Values – Gold

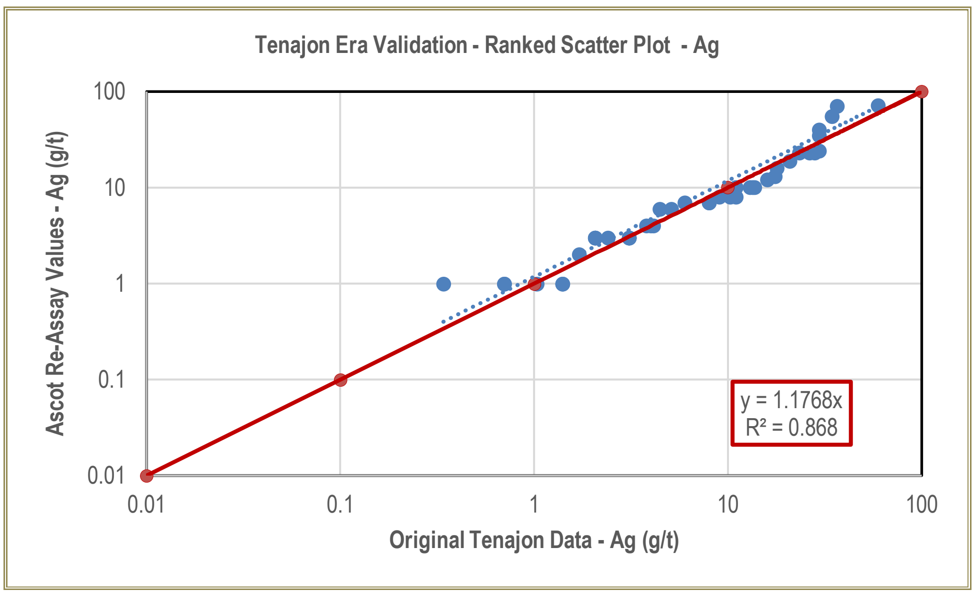


Figure 12-4: Ranked Scatter Plot Comparing Historical Tenajon Data to Re-assay Values – Silver

12.1.5 Premier Gold Project Underground Surveys

The wireframes of the underground workings at PGP could not be fully recovered, so they remain as invalid solids, with missing triangles and overlapping segments. The overall accuracy of their location is also somewhat in doubt. Comparison with the intercepts of void spaces in the drill holes shows good agreement in some areas and poorer agreement in others.

Underground surveying conducted by Ascot indicated that there was a small translation error (i.e., no rotation error) between the underground and surface surveys. This error was determined to be 3.14 m in easting, 0.96 m in northing, and 1.73 m in elevation, for a total 3D translation error of 3.71 m. This error was applied to pre-Ascot drill holes and wireframes that had been tied to the old mine grid.

12.1.6 Premier Gold Project Data Verification Discussion

In the QP's opinion, the Ascot drill data have generally been collected in a manner consistent with industry best practice. The assaying used for the Resource Estimate has been carried out at accredited commercial laboratories using conventional industry-standard methods. Ascot has implemented an assay QA/QC program that is also consistent with best practice guidelines.

The database verification procedures applied by Ascot comply with industry standards and are adequate for the purposes of Mineral Resource estimation. This includes the validation for use of the legacy drill results, for values above 0.3 g/t Au.

12.2 Red Mountain

12.2.1 LAC Database Verification

Data verification has been carried out by previous operators of the Project including Bond, LAC and NAMC. In 2000, NAMC cross-referenced and catalogued all data from previous operators.

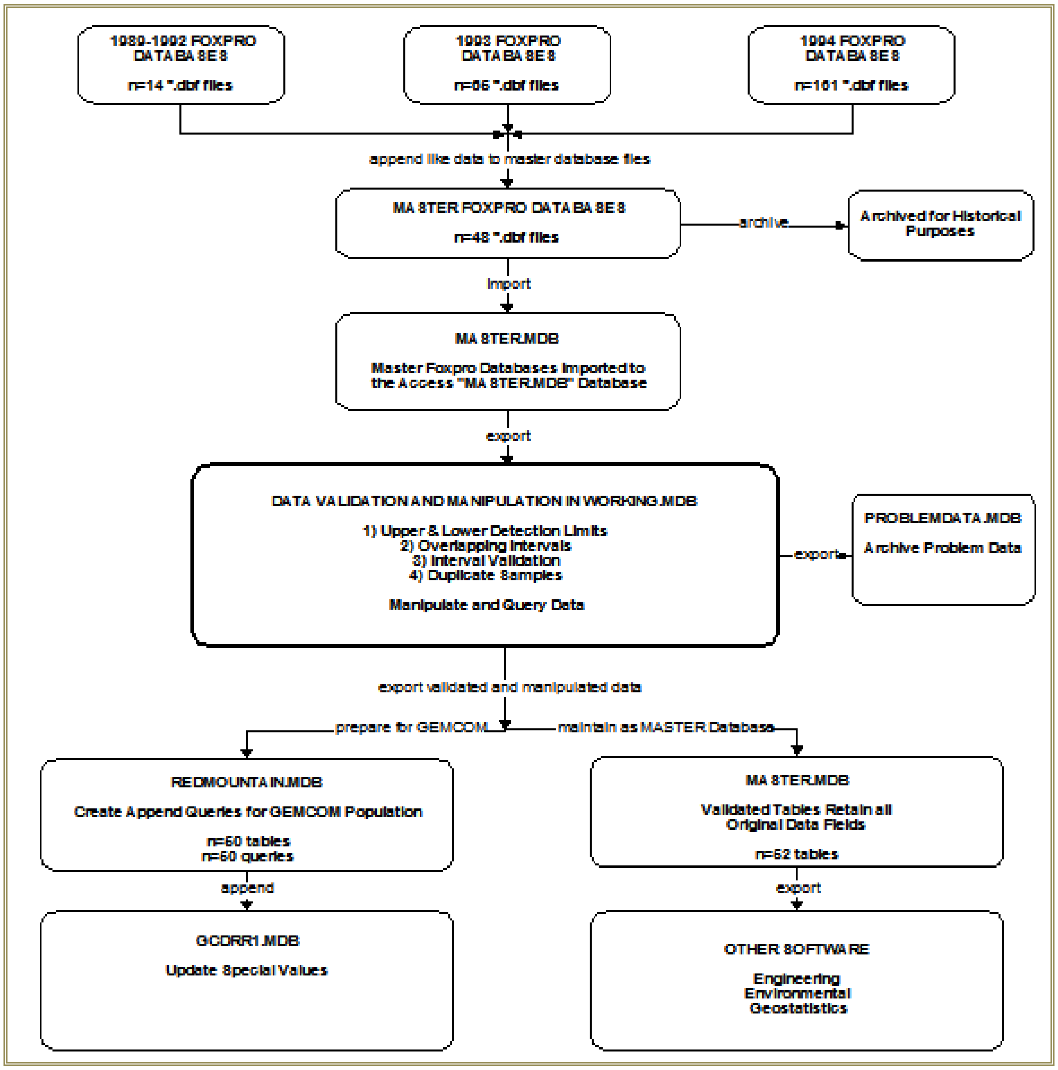
For all but the 2014 IDM program, data have been transferred from paper format to electronic format. Data were entered into the computer by data entry personnel. All 1993 LAC data were checked in January and February 1994. In 1994, LAC instituted a system where all drill hole data were entered and checked by different people as soon as possible after logging. The geologist who logged a hole was responsible for ensuring all data was entered and checked, and that data printouts were with completed logs in the files. The system manager merged new data into the master drill databases.

12.2.2 Red Mountain Electronic Data Verification

LAC collected and organized over one gigabyte of electronic information during its work on the Red Mountain property during 1993 and 1994. As PGP was under fast track conditions by LAC management, the programs were never compiled into a cohesive database that was accessible by a single program. NAMC, upon receiving PGP data, undertook to create and validate a Microsoft Access database that held all the site exploration and environmental work.

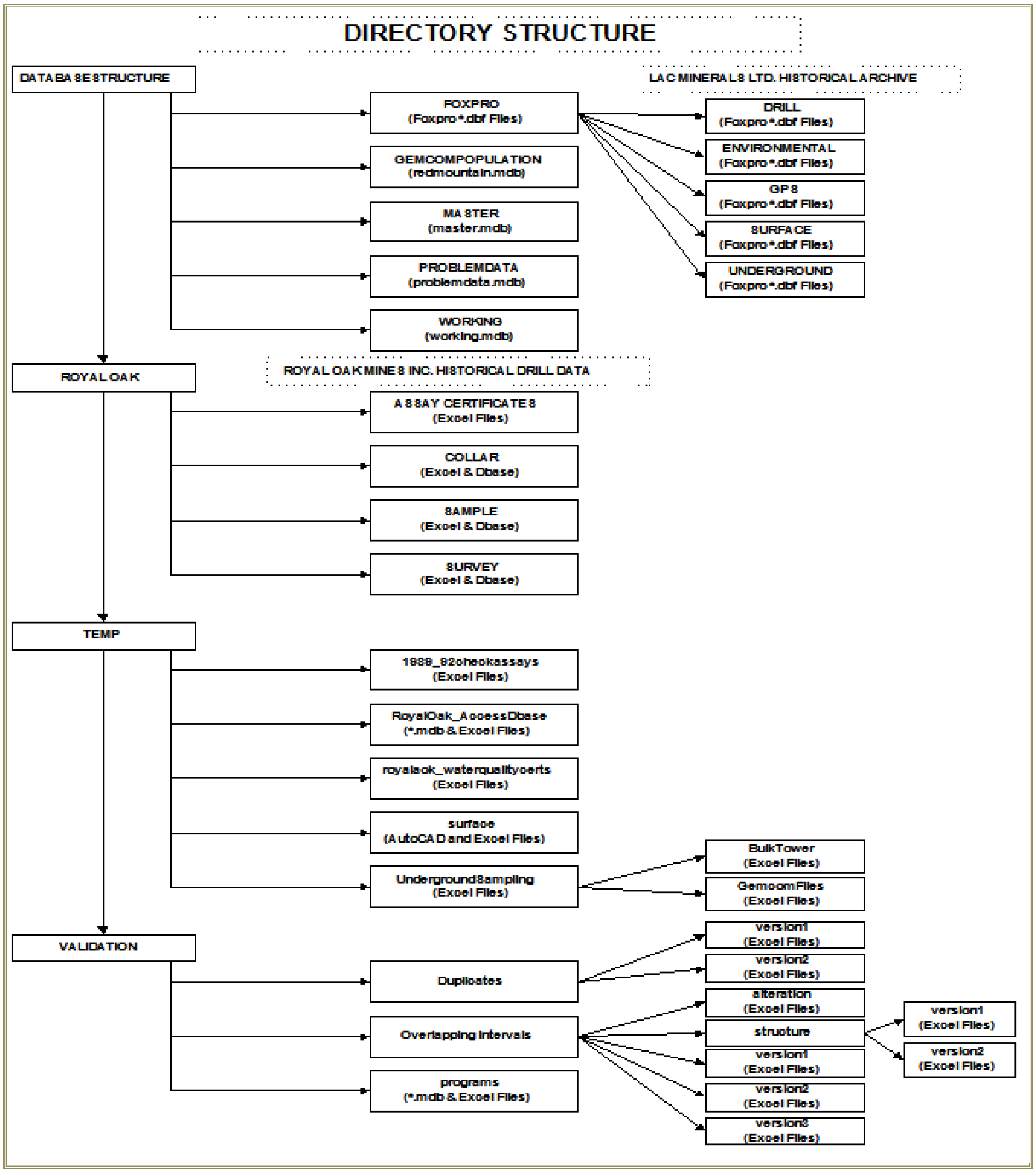
During 2000, NAMC cross-referenced and catalogued all data from previous operators. Data that could not be verified were removed from the database (Craig et al., 2014).

Flowsheets illustrating the database compilation procedures and resulting directory structure as shown in Figure 12-5 and Figure 12-6, respectively.



Source: NAMC (2001)

Figure 12-5: Red Mountain Data Validation Flowchart



Source: NAMC (2001)

Figure 12-6: Red Mountain Database Directory Structure

12.2.3 NAMC Metallurgical Composites at Red Mountain

NAMC compiled five metallurgical composite suites from drill core. Samples were taken from intervals in the Marc and AV zones and were selected to give an average gold grade and distribution similar to the estimated milled head grade of 5 to 15 g/t Au. These composites were taken from the remaining half of drill core in the boxes, sawn to a ¼ sample and individually bagged in the original sample interval length. The samples were sent to Process Research Associates Ltd., where they were dried, weighed, and pulverised to >90% -150 mesh. The pulps were then sent to IPL Laboratories in Vancouver, BC for FA/AAS for gold and FA/Gravimetric in silver analysis. NAMC standards were included in the assay stream for quality control. These standards remained within acceptable limits.

Table 12-1 augments the quality control discussion. The composite assay comparison acts as a gold and silver assay verification, and as a large-scale quality control device.

Table 12-1: Red Mountain Diamond Drill Hole (DDH) Composite Assays vs. Metallurgical Composites

Metallurgical Composite	DDH Comp. Average (Au g/t)	Met Comp. Average (Au g/t)	DDH Comp. Average (Ag g/t)	Met Comp. Average (Ag g/t)
Composite 1 – Section 1220	9.03	8.60	26.17	28.0
Composite 2 – Section 1200	7.77	8.14	52.8	62.3
Composite 3 – Section 1100	8.99	8.31	44.6	45.7
Stage 2 – Marc Zone	13.51	12.87	24.0	51.4
Stage 2 – AV Zone	16.8	14.84	16.0	22.0

Source: NAMC (2001)

12.2.4 ACS Data Verification

For the resource update, some of the key tables in the GEMs database were audited for holes affecting the Red Mountain resource solids.

Red Mountain Collar Table

Drill collar locations were audited through examination in 3-D GEMs software to ensure that collars were properly located in underground drill stations, and in the case of surface holes coincident, within reasonable limits, with the topographic surface. No anomalies were noted.

Red Mountain Survey Table

The down hole survey table from the GEMs database was checked by examining the changes for both azimuth and dip from one survey to the next in all holes. A total of six holes from the 1993 and 1994 surface drilling programs have anomalous azimuth or dip deviations that should be checked through a combination of re-examining the Sperry Sun photos and looking at the mineralization data for the presence of pyrrhotite. One of these holes, M93157, pierces the 141 Zone solid, while the rest do not intersect resource solids.

Red Mountain Assay

Most original historical assay certificates are available in the Red Mountain files. A check was made between the gold and silver values in the GEMs database and values on the historical assay certificates for assays from within the resource solids. A selection of drill holes from all resource zones was made that tried to cover different years of drilling and assayers, as well as being spatially representative. The number of historical assays checked for each zone is given in Table 12-2.

Table 12-2: Red Mountain Assay Validation Summary

Zone	No. Holes Checked	No. of Assays in Solid	No. of Assays Checked	% Checked
Marc	10	1,978	202	10.2
Marc Footwall	3	53	11	20.7
AV	5	442	116	26.2
AV Lower	2	21	5	23.8
JW	3	104	20	19.2
JW Lower	1	36	6	16.7
132 Zone	2	95	11	11.6
141 Zone	7	328	76	23.2
Total	**33**	**3,057**	**447**	**14.6**

Overall, the database was found to be very clean. Two instances of errors in the second decimal place for gold were found and are most likely data entry errors. A third discrepancy was found whereby a gold value of 4.33 was entered instead of the 3.98 listed on the certificate. No discrepancies were noted in silver values.

The 2016, 2017, and 2018 assays were validated by comparing the database values to certificates obtained directly from ALS Global and MS Analytical. Assays from certificates representing between 10% and 15% of samples from each year were evaluated. No discrepancies were found.

ACS Site Visit and Check Samples

ACS carried out a site visit to the Red Mountain Gold Project on October 23 and 24, 2018. During the site visit, ACS verified the property access and logistics, audited logging procedures, and visited the underground.

During a similar site visit in 2016 the underground workings were examined, and seven check samples were collected for validation. Three samples were collected from the Marc Zone from the underground cross cuts and four samples were collected from drill core stored in Stewart. Table 12-3 summarizes the results of the resampling program carried out by Arseneau Consulting Services Inc. (ACS).

Overall, the ACS sample results agree well with the previous results. The sampling program was not intended to be a robust validation program, instead the samples were collected only to verify that the Red Mountain Gold Project did host gold and silver mineralization in the range of grades that have been reported for PGP in the past.

Table 12-3: Results of Red Mountain 2016 Resampling Program

Sample Number	Sample Location	Original Au Value (g/t)	Re-Assay Au Value (g/t)	Original Ag Value (g/t)	Re-Assay Ag Value (g/t)
I95066	1100 crosscut	4.95	7.43	26	16
I95067	1200 crosscut	1.3	0.1	1.7	<5
I95068	1295 crosscut	6.5	1.25	48	<5
I95069	DH941148	1.26	1.69	0.8	<5
I95070	DHM93154	3.95	6.62	3.8	<5
I90571	DHM9054	4.78	7.03	38	42
I95072	DH941122	5.7	2.35	0.05	<5

Comments on Red Mountain Data Verification

The QP has reviewed the appropriate reports and data and is of the opinion that the data verification programs undertaken on the data collected adequately support the geological interpretations and the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Mineral processing and metallurgical testing is split between the two separate ore types that will be feeding the mill over the life-of-mine (LOM); Premier ore which will represent the majority of feed in the first two years of the mine life, and Red Mountain ore which will be phased in as a feed in latter part of Year 2 and represent approximately half of the yearly processed ore from Q4 of Year 2 onwards. Premier ore mineral processing and metallurgical testing is discussed in Sections 13.1 (2018 program) and 13.2 (2019 program), and Red Mountain mineral processing and metallurgical testing is discussed in Section 13.3.

A previous metallurgical test program for the Premier resource was conducted in 2015, at ALS's facilities in Kamloops, BC, on behalf of Ascot Resources Ltd. (Ascot). These tests determined the chemical composition, amenability to cyanide (CN) leaching and gravity concentration of the ore. In 2018, a more comprehensive metallurgical test program was assigned to Base Metallurgical Laboratories Ltd. (BML) in Kamloops, British Columbia, Canada, by Ascot Resources.

This section mainly describes the BML 2018 testwork (which was used as a basis of design for the processing facility) as well as the subsequent testwork performed in 2019/20, by SGS, Kemetco, and BML.

13.1 Premier Metallurgical Testwork (BML, 2018)

Approximately 590 kg of Premier ore samples as half-core were received at BML in three shipments, on August 15, October 19, and September 25, 2018. An additional 46 kg of samples were also received for comminution testing. The testwork consisted of chemical evaluation, comminution tests, and gravity concentration, followed by cyanidation leach testwork, gravity recovery gold (GRG) testing, and CN detoxification testing.

Table 13-1 shows sample composites selected for testwork identified as Ben Prew, 602, Lunchroom, (all from the Premier deposit) with samples from Silver Coin; Northstar, and Big Missouri (both from Big Missouri) deposits.

Table 13-1: Metallurgical Testwork Summary

Sample ID	Head Assay	E-GRG	Leach Test	Cyanide Destruction	BBWi Test	Grind Calibration
Ben Prew	×	×	×	×	×	×
602	×	×	×	×	×	×
Lunchroom	×	×	×	×	×	×
Silvercoin	×		×	×	×	×
Northstar	×		×	×	×	×
Big Missouri	×		×	×	×	×

13.1.1 Samples and Head Assays

Each sample was assayed for gold, silver, sulfur, and carbon. An inductively coupled plasma (ICP) mass spectrometry scan was also conducted on the samples. The average assays results are summarized in Table 13-2.

Table 13-2: Head Assays

Sample ID	Au (g/t)	Ag (g/t)	S (%)	C (%)	TOC (%)
Ben Prew	4.51	12	3.79	0.97	0.04
Lunchroom	9.08	14	3.74	1.26	0.04
602	7.57	69	7.75	1.51	0.02
Northstar	4.33	20	7.56	0.12	0.03
Big Missouri	2.87	7	2.80	1.10	0.03
Silver Coin	8.01	17	6.16	1.43	0.06

The samples were relatively high in gold, ranging from 2.87 to 9.08 g/t across the composites. Silver grades varied from 7 to 69 g/t. Sulfur in the samples was present between 2.8% and 7.8%, indicating a component of sulfide mineralization. The total organic carbon (TOC) in the samples was low, measuring between 0.02% and 0.06%. The TOC can be problematic regarding cyanidation of gold ores; however, at these levels it would likely have a negligible impact on gold dissolution.

13.1.2 Comminution

SMC® Tests

Semi-autogenous grinding (SAG) mill comminution (SMC) tests were conducted on the andesite-rich composite (AXXZ) and silica/breccia-rich composite samples (CBXX). The two composites were hand selected to represent material types that might present the highest degree of resistance to comminution processes. The selection of these composites was not derived from any spatial or geological basis. The data were interpreted by JKTech Pty. Ltd. in 2018. The JK Drop-Weight index (DWT) results are used to determine the JK DWi, which is a measure of the strength of the rock when broken under impact conditions and has the unit of kilowatt hours per cubic metre (kWh/m^3). Table 13-3 presents the summary of results.

Table 13-3: SMC Test Results Summary

Sample ID	SG	DWi (kWh/m^3)	Calculated Axb	Morrell Parameters (kWh/t)		
				Mia	Mih	Mic
AXXZ Composite	2.84	9.4	30.6	24.2	19.1	9.9
CBXX Composite	2.87	8.1	35.9	21.2	16.3	8.4

The SMC-derived A x b values of the two samples was determined to be 30.6 and 35.9, respectively, which rank at the 83rd and 70th percentiles of all the samples in the JKTech database. Based on these values, the samples would be considered moderate-to-hard with respect to breakage in a SAG Mill.

Bond Ball Mill Work Index

Bond Ball Mill Work Index (BWi) tests were conducted on the six composites and Abrasion index (Ai) tests were conducted on three of the composites with the test results summarized in Table 13-4.

Table 13-4: Bond Ball Mill Test Results Summary

Sample ID	F_{80} (µm)	P_{80} (µm)	BWi (KWh/t)	Ai
Ben Prew	1,872	76	15.6	-
Lunchroom	1,985	75	15.6	-
602	2,218	76	17.5	-
Northstar	2,036	80	17.7	0.38
Big Missouri	2,140	81	14.7	0.13
Silver Coin	1,941	82	14.8	0.53

The BWi test was conducted using a closing screen sizing of 106 µm, resulting in a P_{80} product sizing of approximately 75 µm. At this closing screen size, the BWi of the samples ranged between 14.8 and 17.7 kWh/t, indicating the mineralization to be of moderate-to-high hardness from a ball milling perspective.

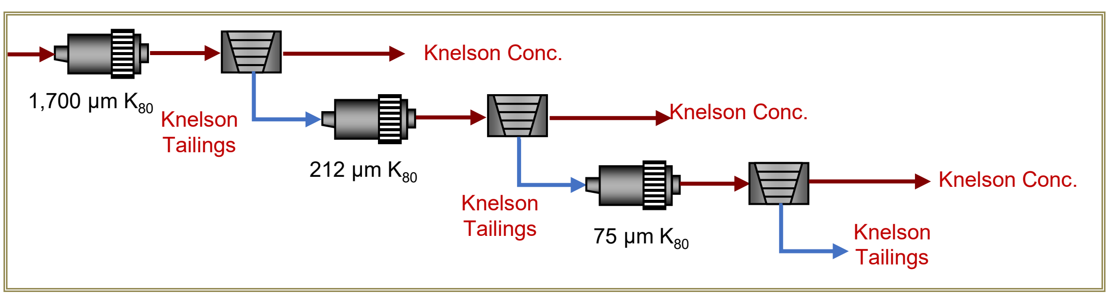
The Ai of the three initial samples tested ranged between 0.13 and 0.53, classifying the samples as mild to moderately abrasive.

13.1.3 Gravity Testwork

A series of Knelson gravity concentration tests, followed by standard CN bottle roll leach tests, were conducted on the six samples. Details and a summary of the results will be discussed in subsequent sections.

Extended-Gravity Recoverable Gold (E-GRG) Testwork

The extended gravity recoverable gold (E-GRG) test consists of three sequential liberation and recovery stages using a 29 kg sample. It utilizes the Knelson concentrator after each stage of grinding to concentrate the GRG. Each gravity separation concentrate, and a subsample of the final stage tailings, are analyzed for size distribution and gold assay. E-GRG testwork flowsheet is presented in Figure 13-1.



Source: Base Met Lab Report BL0366, November 2018

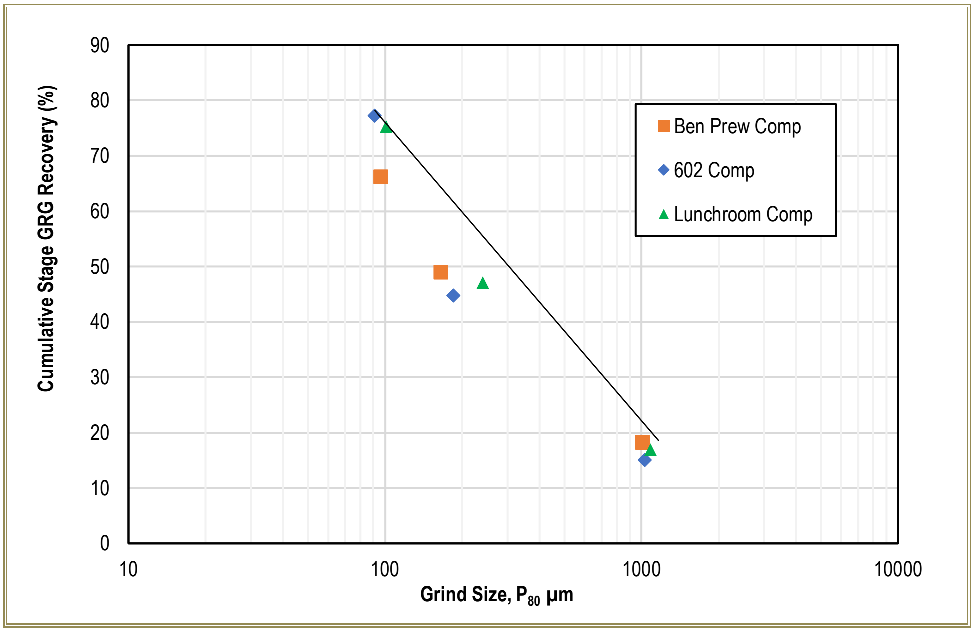
Figure 13-1: Gravity Recoverable Gold Test Flowsheet

Three samples covering a range of head grades (Ben Prew, Lunchroom, and 602) were tested for E-GRG. The test results are summarized in Table 13-5 and Figure 13-2.

The total gold recovery for the Ben Prew composite was 66.2% with the calculated head gold grade of 4.78 g/t. The total gold recovery for the 602 composite was 77.3% with the calculated head gold grade of 6.65 g/t. The total gold recovery for the Lunchroom composite was 75.3% with the calculated head gold grade of 8.99 g/t. The total gold recoveries were higher for the higher head grade gold samples.

Table 13-5: E-GRG Test Results Summary

Sample ID	Concentrate P_{80} (µm)			Cumulative Au Recovery (%)			Conc. Au Grade (g/t)	Head Grade Au (g/t)	
	Stage 1	Stage 2	Stage 3	Stage 1	Stage 2	Stage 3		Direct	Calculated
Ben Prew	1,023	179	102	18.3	49.1	66.2	222	4.31	4.78
Lunchroom	1,092	260	122	16.9	47.1	75.3	596	8.11	8.99
602	1,042	194	117	15.1	44.8	77.3	395	7.33	6.65

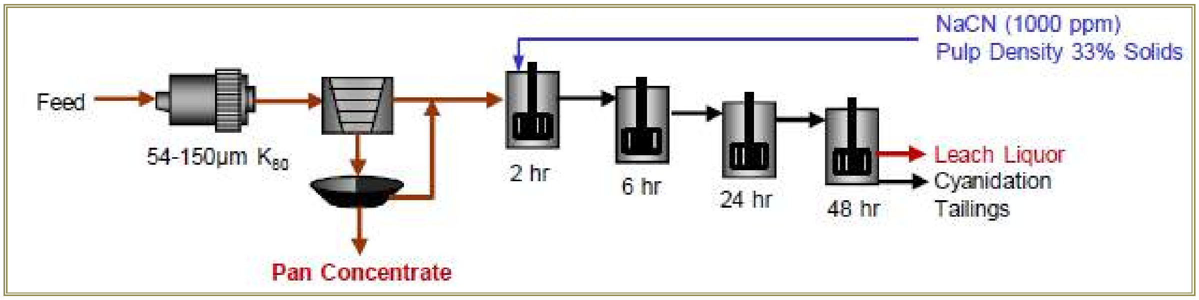


Source: Base Met Lab Report BL0366, November 2018

Figure 13-2: E-GRG Test Results

13.1.4 Gravity and Bottle Roll Leach Tests

Sequential gravity and bottle roll leach tests were carried out with a Knelson Concentrator followed by bottle roll leaching tests. The ground sample (1,000 g) was passed through the Knelson Concentrator and the concentrate obtained was further upgraded by panning, and assayed for gold and silver. The Knelson tailings and pan tailings were combined and subjected to a 48-h continuous bottle rolling leach tests. During the leaching test, the CN leach solution was sampled on 0, 2, 6, 24, and 48 h, and assayed for gold and silver. The remaining tailings from the leaching test was sampled and assayed for gold and silver for extraction analysis. The test procedure is shown in Figure 13-3.



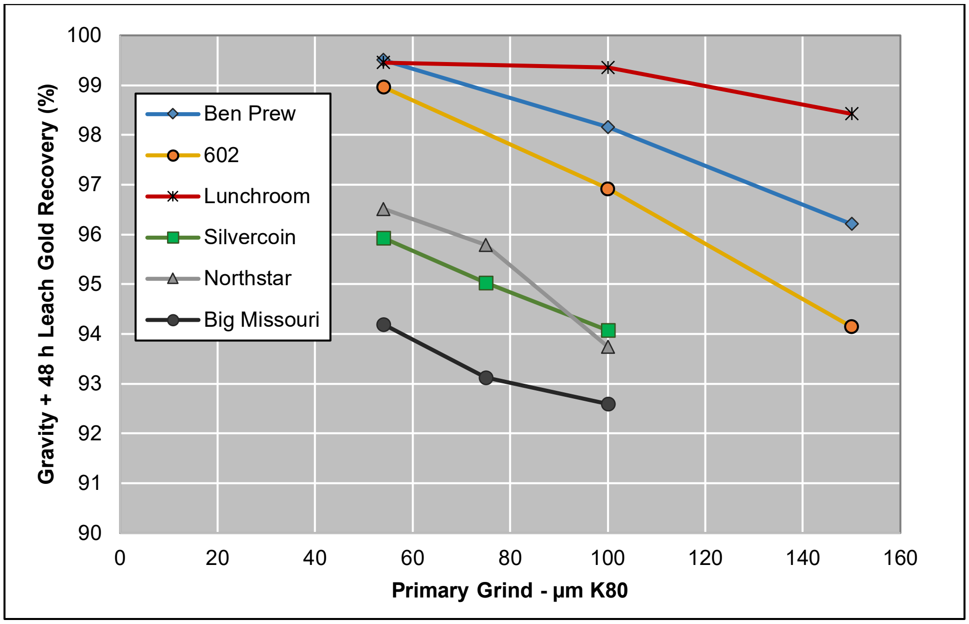
Source: Base Met Lab Report BL0366, November 2018

Figure 13-3: Gravity Recoverable Gold and Leaching Flowsheet

The gravity and leach tests were conducted on all six samples with each sample subjected to three different grind sizes ranging between 54 and 150 µm P_{80}. The test results are summarized in Table 13-6. The gold recoveries ranged from 92.6% to 99.5%, while the silver recoveries ranged from 63.5% to 83.1%, depending on the ore type and grind size. Generally, at the finer primary grind size, higher gold recovery was obtained. This is shown in Figure 13-4.

Table 13-6: Gravity Recoverable Gold and Leaching Test Results

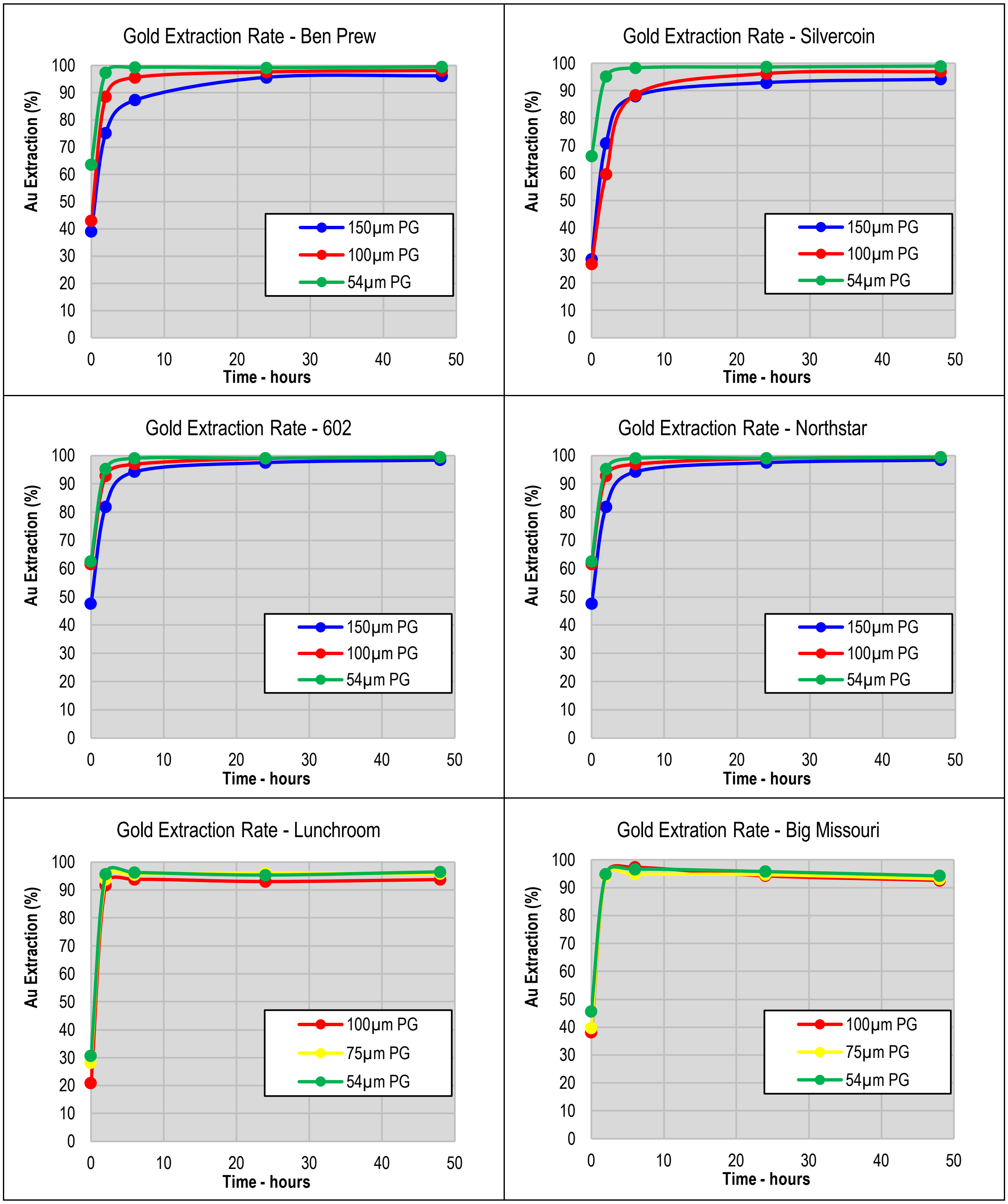
	Grind Size	Gold Recovery (%)			Consumption (kg/t)	
Sample ID	P_{80} (µm)	Pan Conc.	48-h Leach	Total Recovery	NaCN	Lime
Ben Prew	150	39.1	57.1	96.2	0.98	0.43
	100	43.0	55.2	98.2	1.24	0.44
	54	63.7	35.8	99.5	1.56	0.43
602	150	28.7	65.4	94.1	0.92	0.42
	100	26.9	70.0	96.9	1.48	0.41
	54	66.1	32.8	99.0	2.00	0.43
Lunchroom	150	47.6	50.8	98.4	0.70	0.42
	100	61.5	37.8	99.4	0.72	0.44
	54	62.5	37.0	99.5	1.40	0.42
Silvercoin	100	27.6	66.5	94.1	1.30	0.60
	75	26.2	68.9	95.0	1.88	0.53
	54	26.9	69.0	95.9	2.06	0.40
Northstar	100	21.0	72.8	93.7	1.44	1.24
	75	28.2	67.6	95.8	1.52	1.45
	54	30.6	65.9	96.5	1.68	1.30
Big Missouri	100	38.1	54.5	92.6	1.24	0.41
	75	39.6	53.5	93.1	1.46	0.45
	54	45.6	48.6	94.2	1.74	0.55



Source: Base Met Lab Report BL0366, November 2018

Figure 13-4: Gold Recovery against Primary Grind Size

Sodium cyanide (NaCN) consumption for the samples ranged from 0.7 and 2.1 kg/t. The tests conducted at the finer primary grind sizes resulted in higher NaCN consumptions. Further optimization could focus on reducing the NaCN consumption by evaluating lower NaCN additions, given the rapid leach kinetics for these samples. The kinetic curves shown in Figure 13-5 indicate most of the gold is leached after 6 h. A reduction in NaCN consumption and reagent consumptions in downstream CN detox processes may be achieved by evaluating pre-oxidation.



Source: Base Met Lab Report BL0366, November 2018

Figure 13-5: Individual Gold Recoveries against Primary Grind Sizes

13.1.5 Cyanide Detoxification Testwork

A LOM composite was prepared to generate a tailings sample for external vendor testing. A 30-kg sample of this composite was subjected to gravity separation, cyanidation, and detoxification. The tailings slurry was subjected to several small-scale continuous detoxification tests, while evaluating basic reagent conditions. At selected conditions, the residual leached tailings were processed to generate samples for external vendors. A summary of the CN detoxification results is displayed in Table 13-7.

Table 13-7: Cyanide Detoxification Tests Results

Detox. Test	Test Parameters				Test Length		Feed/Detox Solution Assay (ppm)				
	pH	Ret. Time (min.)	Reagent SO_2 g/CN_{MP} (g)	Used Cu (mg/L)	min.	No. of Tests	CN_{MP}	Cu	Fe	Ni	Zn
Feed	-	-	-	-	-	-	444.4	38.8	7.9	0.23	62.4
C1	8.4	60	6	15	180	3	0.84	0.89	< 0.01	< 0.01	7.87
C2	8.5	61	5	15	180	3	3.14	7.07	0.01	0.02	9.12
C3	8.4	61	4	15	180	3	3.25	8.68	1.12	0.02	7.55
C4	8.4	62	6	15	180	3	1.31	0.53	< 0.01	< 0.01	1.66

The results indicate that the LOM Composite can achieve a final weakly acidic dissociable CN (CN_{WAD}) levels of below 5 parts per million (ppm) when a sulfur dioxide (SO_2):CN_{WAD} ratio of between 4:1 and 6:1 is used and 15 ppm copper (Cu) is added as a catalyst with a reactor retention time of 60 minutes (min). Further optimization may be achieved through testing of various reagent conditions.

13.2 Premier Metallurgical Testwork (SGS, BML, and Kemetco, 2019)

13.2.1 Sample Preparation

One hundred half core samples from the Silver Coin, Big Missouri, and Premier Northern Lights ore deposits were submitted to SGS's Burnaby facility for characterization and metallurgical testing. The objective of the program was to determine the gold recovery of composite and variability samples through a conventional gravity-leach flowsheet.

The Qualified Person (QP) has reviewed sample locations and sampling procedures for samples collected by Ascot Resources Ltd. from the Silver Coin, Big Missouri, and Premier respective deposits for 2019 metallurgical confirmation testwork campaigns at the SGS and BML facilities, and concludes that they meet industry standards for representative samples from the aforementioned deposits, their styles of mineralogy, lithology, and mineral deposits as a whole.

The compositing and variability sample generation procedures meet industry standards and are consistent with requirements and best practices for preparing representative metallurgical testwork samples.

From the available core samples, a total of 19 composite samples were prepared for testwork as per directions provided by Ascot Resources Ltd. metallurgy representatives. Sixteen variability samples were prepared: six from the Silver Coin ore zone; five from the Big Missouri ore zone, and five from the Premier ore zone. From the selected variability samples, three master composite samples, one from each of the respective ore zones, were generated.

Table 13-8 presents a summary of the generated samples that were utilized in this metallurgical testwork campaign.

Table 13-8: Composite and Variability Sample Inventory

Ore Zone	Sample ID	Received Weight (kg)	Weight Split for Composite (kg)	Sample Weight for Testwork (kg)
Silver Coin	SC687-1	63	25.2	37.8
	SC897-1	45.8	18.3	27.5
	SC100	70.8	28.3	42.5
	SC687-2	15.6	0	15.6
	SC897-2	29.0	0	29.0
	SC302	6.9	0	6.9
	SC Master Comp.	-	-	71.8
Big Missouri	BM1	61.1	17.1	44.0
	BM2	60.7	17	43.7
	BM3	48.1	13.5	34.6
	BM4	93.3	26.1	67.2
	BM5	163.7	45.8	117.9
	BM Master Comp.	-	-	119.5
Premier	PM609	81.8	20.5	61.3
	PM602	86.1	21.5	64.6
	PM NL	87.8	22	65.8
	PM PREW	80.0	20	60.0
	PM BEN	99.6	24.9	74.7
	PM Master Comp.	-	-	108.9

13.2.2 Head Assays

A 1 kg charge from each of the samples was submitted for head analysis including gold by fire assay (AuFA) in triplicate, silver assay in triplicate, total sulfur (S), sulfide sulfur ($S^{=}$), and ICP-Scan.

All gold and silver assay values for the composite and variability samples are presented Table 13-9.

Table 13-9: Gold and Silver Head Assays for Composite and Variability Samples

Sample name	Single Suite Au Assay (g/t)	Triplicate Au Assay (g/t)			Single Suite Ag Assay (g/t)	Triplicate Ag Assay (g/t)		
SC Master Comp.	4.3	6.0	4.8	7.1	12.2	16.0	16.5	15.8
SC687-1	-	4.7	4.3	4.3	-	11.3	11.5	11.1
SC897-1	-	5.0	5.3	9.1	-	6.3	5.7	5.3
SC100	-	4.5	3.8	4.1	-	11.1	10.7	11.7
SC687-2	-	4.9	5.1	5.0	-	30.9	29.4	27.9
SC897-2	-	9.5	6.9	8.2	-	22.9	20.1	20.9

Sample name	Single Suite Au Assay (g/t)	Triplicate Au Assay (g/t)			Single Suite Ag Assay (g/t)	Triplicate Ag Assay (g/t)		
SC302	-	3.5	3.6	3.5	-	24.6	24.2	23.7
BM Master Comp.	7.2	3.8	3.5	3.7	12.1	9.2	9.4	12.1
BM1	-	3.5	2.8	2.7	-	9.8	9.1	9.0
BM2	-	2.8	2.8	3.3	-	8.3	8.3	7.7
BM3	-	7.5	6.8	7.8	-	11.1	10.7	10.6
BM4	-	2.6	2.4	2.8	-	12.3	12.9	12.4
BM5	-	2.8	3.2	3.1	-	10.5	11.7	10.6
PM Master Comp.	10.1	8.1	10.7	6.6	27.4	45.7	20.5	18.5
PM609	-	6.9	7.5	6.6	-	7.1	6.7	8.0
PM602	-	15.1	10.9	14.2	-	29.6	29.9	29.6
PM NL	-	4.0	4.7	4.4	-	35.1	21.9	27.5
PM PREW	-	7.1	6.4	5.8	-	10.2	7.4	7.1
PM BEN	-	19.4	19.1	31.7	-	20.3	12.1	22.8

As shown in Table 13-10, the variability in the duplicate assays for the composites reveal that there is some presence of coarse gold and silver in the samples.

Table 13-10: Composite and Variability Samples Head Assays

Sample Name	Au Assay (g/t)	Ag Assay (g/t)	S (%)	$S^=$ (%)
SC Master Comp.	5.6	15.1	3.3	3.1
SC687-1	4.4	11.3	3.0	2.9
SC897-1	6.5	5.8	2.1	2
SC100	4.1	11.2	2.0	1.7
SC687-2	5.0	29.4	10.9	10.4
SC897-2	8.2	21.3	2.2	2.1
SC302	3.5	24.2	3.2	3
BM Master Comp.	4.5	10.7	4.1	3.6
BM1	3.0	9.3	3.2	3.2
BM2	2.9	8.1	4.0	3.7
BM3	7.4	10.6	2.8	2.6
BM4	2.6	12.5	4.4	3.9
BM5	3.1	10.9	2.9	2.8
PM Master Comp.	8.9	28	4.1	4.1
PM609	7.0	7.3	2.7	2.5
PM602	13.4	29.7	5.1	5.1
PM NL	4.4	28.2	3.5	3.4
PM PREW	6.4	8.2	2.2	1.9
PM BEN	23.4	18.4	2.5	2.4

The assay values shown for gold and silver for the composite samples are an average of the four values obtained from the single suite and triplicate assay numbers, and the gold and silver assay values presented for the variability samples are an average of the three values obtained from the triplicate assays. The average of the duplicate values was used as the direct head assay for the gravity circuit metallurgical balance.

13.2.3 QEMSCAN Mineralogy

A 1 kg charge from each of the composite and variability samples (nineteen samples in total) was submitted for mineralogical assessment. The purpose of analyzing the samples was to determine the modal mineralogy and the exposure and association to support the metallurgical testwork.

Table 13-11 presents the modal mineralogy for the composite samples.

Table 13-11: Modal Mineralogy of Composite Samples

Mineral Mass	Silver Coin Master Comp.	Big Missouri Master Comp.	Premier Master Comp.
Gold	0.00	0.00	0.00
Ag Minerals	0.00	0.00	0.00
Pyrite	4.35	5.90	5.73
Sphalerite	0.96	2.23	0.97
Other Sulfides	0.28	0.40	0.26
Quartz	39.8	39.1	45.1
Plagioclase	4.36	0.35	1.56
K-Feldspar	18.1	17.0	15.6
Mica	14.7	17.3	15.9
Clay	0.91	0.57	0.50
Chlorite	4.35	7.21	4.93
Pyroxene	0.15	0.24	0.21
Amphibole	0.79	0.86	0.89
Other Silicates	0.07	0.21	0.08
Fe-Oxides	0.02	0.01	0.02
Other Oxides	0.94	0.83	1.16
Carbonates	9.64	7.31	6.75
Apatite	0.56	0.33	0.31
Other	0.03	0.15	0.02
Total	**100.00**	**100.00**	**100.00**

For each composite, pyrite is well liberated (pure, free, and liberated combined) at 84%, 90.9%, and 89.3% for Silver Coin, Premier, and Big Missouri, respectively, and non-liberated grains are mostly as complex particles (pyrite plus multiple mineral phases).

The modal mineralogy of all 19 samples indicates that they are predominantly composed of quartz (35% to 58%); potassium-feldspar (8% to 22%); mica (9% to 21%), moderate amounts of chlorite (<11%) and carbonates (<13%). Minor phases such as plagioclase (<5%) are also present. Trace minerals of note are clays (<2.2%) and other oxides (<2%). The minerals of interest across all samples are pyrite and sphalerite which occur in moderate to trace amounts (4% to 12% and 0.2% to 2.2%, respectively).

Table 13-12 presents the modal mineralogy of the variability samples.

Table 13-12: Modal Mineralogy of Variability Samples

Mineral Mass	BM1	BM2	BM3	BM4	BM5	PM Ben	PM NL	PM Prew	PM602	PM609	SC100	SC302	SC687-1	SC687-2	SC897-1	SC897-2
Gold	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ag Minerals	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01
Pyrite	6.11	7.26	5.35	8.17	6.05	5.39	5.05	4.06	9.93	5.80	4.82	4.99	5.20	11.6	4.79	4.64
Sphalerite	1.38	0.88	0.88	2.20	0.91	0.69	4.13	0.76	2.14	0.79	0.50	2.42	1.12	12.0	0.21	2.00
Chalcopyrite	0.16	0.16	0.07	0.22	0.10	0.08	0.39	0.02	0.16	0.01	0.07	0.10	0.10	0.67	0.03	0.18
Galena	0.20	0.04	0.23	0.57	0.29	0.20	0.33	0.23	0.90	0.06	0.06	0.72	0.55	0.39	0.04	0.58
Other Sulfides	0.04	0.01	0.02	0.06	0.02	0.03	0.04	0.03	0.04	0.01	0.02	0.09	0.05	0.04	0.00	0.06
Quartz	44.1	37.1	45.7	51.5	52.2	34.5	44.7	45.4	46.3	43.4	43.8	57.5	47.0	35.3	38.4	44.7
Plagioclase	1.09	2.18	3.19	0.54	0.48	0.16	0.47	0.86	0.08	0.63	1.99	0.47	4.90	0.50	4.76	3.07
K-Feldspar	12.6	13.9	13.0	11.8	15.0	21.2	15.3	17.7	11.7	16.7	20.9	10.5	10.7	8.10	21.5	11.7
Mica	15.2	15.3	11.1	9.47	11.1	20.8	13.2	12.1	11.1	15.9	8.85	12.0	9.62	11.8	16.3	10.5
Clay	1.61	1.07	0.92	0.56	0.96	0.85	0.79	0.79	0.48	1.41	0.71	0.81	1.15	0.44	1.45	2.22
Chlorite	8.07	10.8	5.91	3.92	3.73	7.04	7.49	5.42	6.10	6.30	2.77	0.28	5.33	8.45	6.64	4.41
Pyroxene	0.14	0.09	0.18	0.19	0.22	0.13	0.08	0.69	0.07	0.09	0.16	0.23	0.13	0.07	0.10	0.19
Amphibole	0.68	0.67	1.00	0.78	1.11	0.31	0.36	0.59	0.33	0.39	0.37	1.25	0.71	0.39	0.31	0.41
Other Silicates	0.06	0.08	0.10	0.02	0.06	0.14	0.06	0.95	0.04	0.07	0.05	0.06	0.06	0.05	0.06	0.07
Fe-Oxides	0.00	0.01	0.00	0.03	0.00	0.00	0.03	0.00	0.01	0.03	0.00	0.02	0.00	0.01	0.06	0.02
Other Oxides	0.78	1.33	0.82	0.47	0.80	0.83	0.63	0.99	0.62	0.85	0.52	0.49	0.83	0.54	0.99	1.94
Carbonates	7.00	8.59	10.9	8.94	6.31	6.99	6.33	8.82	9.43	6.89	13.8	7.55	12.0	9.21	3.63	12.6
Apatite	0.62	0.45	0.29	0.37	0.51	0.48	0.37	0.36	0.24	0.54	0.49	0.28	0.40	0.32	0.65	0.60
Other	0.18	0.12	0.33	0.24	0.21	0.13	0.23	0.20	0.26	0.11	0.12	0.21	0.15	0.10	0.12	0.17
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Of all the sixteen variability samples, pyrite is well liberated (pure, free, and liberated combined) and ranges from 75.3% in SC 897-2 to 96.4% in PM Prew.

13.2.4 Comminution Testwork

SMC Tests

The SMC test is an abbreviated version of the standard JK drop-weight test and is performed on rocks from a single size fraction (-31.5 mm + 26.5 mm). The test provides a cost-effective means of obtaining ore-specific parameters for use in the JKSimMet Mineral Processing simulator software. In JKSimMet, these parameters are combined with equipment details and operating conditions to analyze and predict SAG/autogenous mill performance. The results can also be used to select comminution equipment.

The SMC test was performed on each of the three master composites, as well as their respective variability samples (except for variability sample SC 687-2 and SC 302 from the Silver Coin deposit). A summary of the SMC test results is presented in Table 13-13. The average specific gravity values for all 17 samples tested ranged from 2.72 to 2.94 t/m^3.

Table 13-13: Summary of the SMC Test Results

	Deposit Name	Sample ID	Specific Gravity SMC	JK Parameters SMC				
				A	b	A x b	t_a	DWi (kWh/m^3)
1	Premier	602	2.94	71.8	0.60	43.1	0.38	6.81
2	Premier	609	2.80	63.4	0.67	42.5	0.39	6.63
3	Premier	Ben	2.81	70.2	0.54	37.9	0.35	7.37
4	Premier	NL	2.79	66.9	0.64	42.8	0.40	6.50
5	Premier	Prew	2.78	66.2	0.65	43.0	0.40	6.43
6	Premier	PM Master Comp.	2.81	77.0	0.49	37.7	0.35	7.45
7	Big Missouri	BM1	2.81	71.3	0.55	39.2	0.36	7.22
8	Big Missouri	BM2	2.87	74.5	0.51	38.0	0.34	7.50
9	Big Missouri	BM3	2.77	68.6	0.60	41.2	0.38	6.70
10	Big Missouri	BM4	2.81	79.1	0.49	38.8	0.36	7.27
11	Big Missouri	BM5	2.78	73.6	0.55	40.5	0.38	6.86
12	Big Missouri	BM Master Comp.	2.84	76.9	0.50	38.5	0.35	7.38
13	Silver Coin	SC100	2.72	71.7	0.60	43.0	0.41	6.32
14	Silver Coin	SC687-1	2.81	76.3	0.51	38.9	0.36	7.16
15	Silver Coin	SC897-1	2.77	75.8	0.50	37.9	0.35	7.34
16	Silver Coin	SC897-2	2.76	71.6	0.55	39.4	0.37	7.00
17	Silver Coin	SC Master Comp.	2.75	76.3	0.51	38.9	0.37	7.01

The derived A x b parameter values ranged between 37.7 and 43.1, which indicates ore from the Premier deposits can be classified as moderately hard to hard, aligning with the earlier SMC testwork data from 2018. Based on all available SMC results it can be concluded that a SAG/Ball (SAB) milling circuit is suitable and is recommended for processing of ore from the Premier deposits.

Bond Abrasion Tests

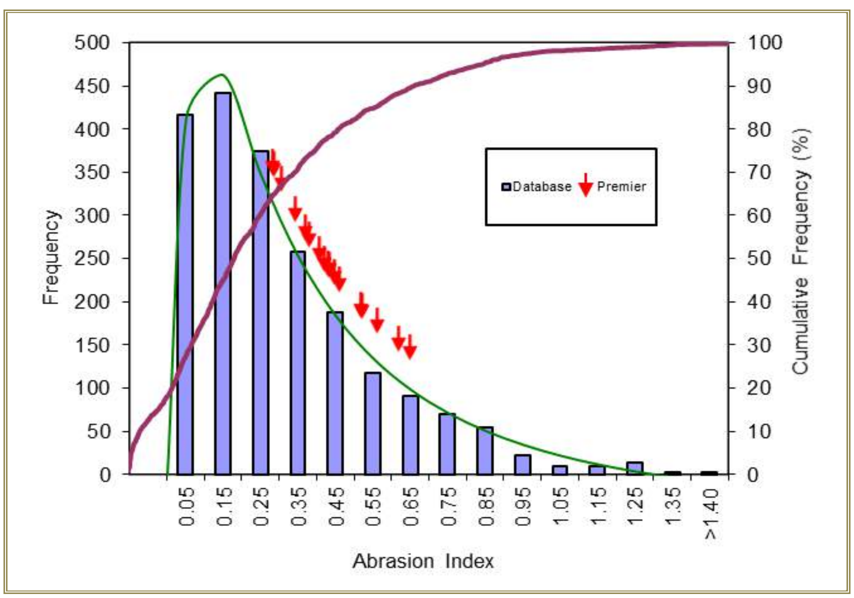
The Bond abrasion test determines the Ai, which is used to determine steel media and liner wear in crushers, SAG, and ball mills.

The abrasion test was performed on each of the three master composites, as well as their respective variability samples (except for variability sample SC 687-2 and SC 302 from the Silver Coin deposit). A summary of the abrasion test results is presented in Table 13-14.

Table 13-14: Summary of the Abrasion Test Results

Sample ID	Ai, g	Percentile of Abrasivity	Category
602	0.560	85	Abrasive
609	0.285	57	Medium
Ben	0.431	75	Moderately Abrasive
NL	0.518	82	Abrasive
Prew	0.521	82	Abrasive
PM Master Comp.	0.379	69	Moderately Abrasive
BM1	0.303	60	Medium
BM2	0.281	56	Medium
BM3	0.429	75	Moderately Abrasive
BM4	0.619	88	Abrasive
BM5	0.647	89	Abrasive
BM Master Comp.	0.420	74	Moderately Abrasive
SC100	0.460	78	Abrasive
SC687-1	0.369	69	Moderately Abrasive
SC897-1	0.343	66	Moderately Abrasive
SC897-2	0.404	73	Moderately Abrasive
SC Master Comp.	0.446	77	Abrasive

The abrasion indices ranged from 0.281 g to 0.647 g with an average of 0.436 g, characterizing the Premier ore samples as moderately abrasive to abrasive, which aligns with the historical testwork data from 2018. A comparison with SGS's database is depicted in Figure 13-6.



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-6: Comparison of Ai Results to SGS Database

Bond Rod Mill Work Index

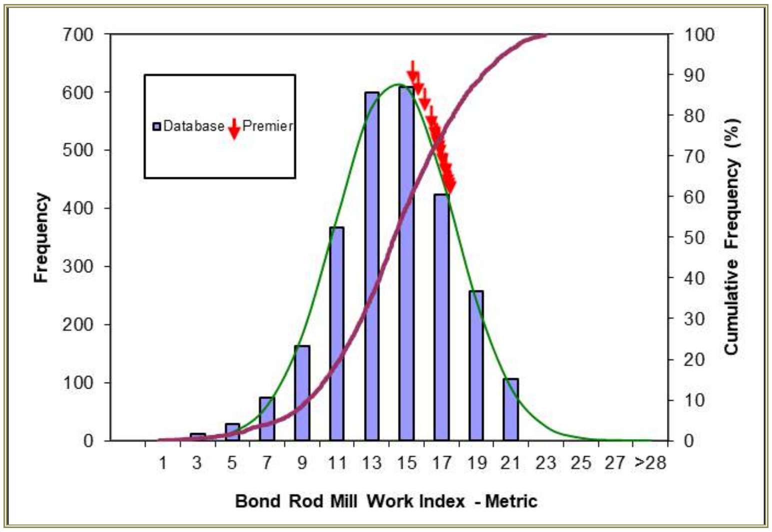
The Bond Rod Mill grindability tests determines the Bond Rod Mill Work index (RWi), which can be used with Bond's Third theory of comminution to calculate the net power requirements for sizing rod mills or primary ball mills. It can be also used in conjunction with other Bond tests (BWi and CWi) for SAG mill selection using semi-empirical relationships.

The Bond rod mill grindability test was performed at a 14 mesh of grind (1,180 µm) on all three composites and 14 variability samples. The test results are summarized in Table 13-15.

Table 13-15: Summary of RWi Test Results

Sample ID	Mesh of Grind	f_{80} (µm)	(µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile	Category
602	14	10,945	942	8.47	15.4	63	Moderately Hard
609	14	11,305	977	8.15	16.0	70	Moderately Hard
Ben	14	11,062	973	7.61	16.8	76	Hard
NL	14	10,732	986	7.37	17.4	82	Hard
Prew	14	10,702	942	7.37	16.8	77	Hard
PM Master Comp.	14	9,851	956	7.76	16.8	77	Hard
BM1	14	10,793	917	7.29	16.6	75	Moderately Hard
BM2	14	10,549	934	6.94	17.4	82	Hard
BM3	14	10,426	964	7.19	17.5	82	Hard
BM4	14	10,963	939	7.37	16.7	76	Hard
BM5	14	9,881	960	7.70	16.9	78	Hard
BM Master Comp.	14	9,437	953	7.34	17.5	83	Hard
SC100	14	10,027	951	8.53	15.7	67	Moderately Hard
SC687-1	14	8,895	944	7.74	17.0	79	Hard
SC897-1	14	9,425	970	7.69	17.2	81	Hard
SC897-2	14	9,900	967	7.54	17.2	80	Hard
SC Master Comp.	14	9,999	947	7.91	16.4	73	Moderately Hard

Generally, the composites and variability samples are moderately hard to hard, with their RWi values ranging from 15.4 to 17.5 kWh/t. A comparison to the SGS database is depicted in Figure 13-7.



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-7: Comparison of RWi Results to SGS Database

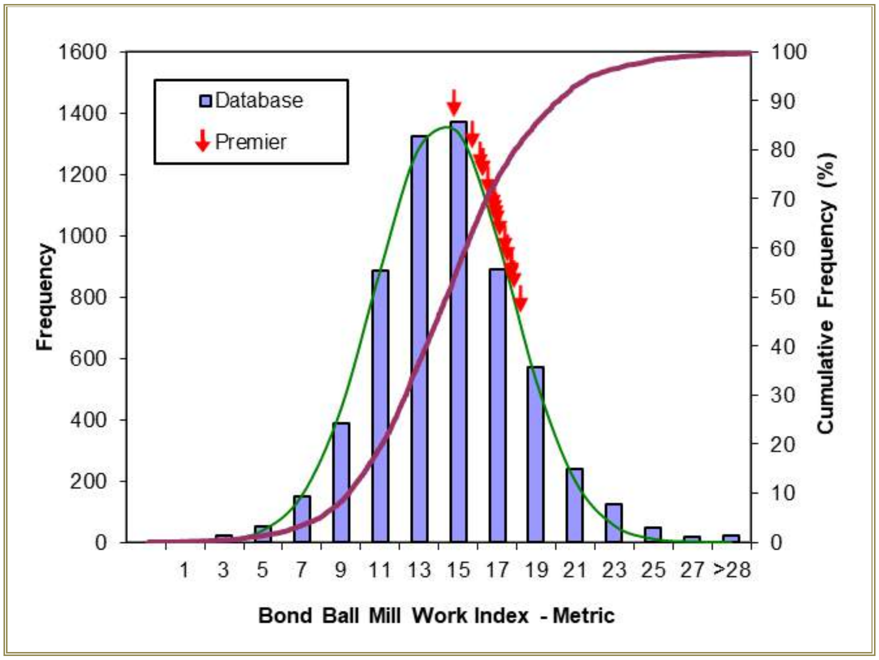
Bond Ball Mill Work Index

Bond ball mill grindability tests were performed according to the original Bond procedure. The test was performed at a 150 mesh of grind (106 µm), on all three master-composite, and fourteen variability samples. The test results are summarized in Table 13-16.

Table 13-16: Summary of BWi Test Results

Sample ID	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Grams per Revolution	Work Index (kWh/t)	Hardness Percentile	Category
602	150	2,499	88	1.40	14.8	55	Medium
609	150	2,540	89	1.30	15.7	65	Moderately Hard
Ben	150	2,500	87	1.23	16.2	71	Moderately Hard
NL	150	2,292	88	1.27	16.1	70	Moderately Hard
Prew	150	2,385	91	1.20	17.1	78	Hard
PM Master Comp.	150	2,316	86	1.21	16.5	73	Moderately Hard
BM1	150	2,291	89	1.14	17.7	81	Hard
BM2	150	2,315	89	1.13	17.8	82	Hard
BM3	150	2,355	89	1.16	17.3	80	Hard
BM4	150	2,496	88	1.20	16.7	75	Moderately Hard
BM5	150	2,340	89	1.20	16.9	76	Hard
BM Master Comp.	150	2,546	86	1.10	17.7	82	Hard
SC100	150	2,529	85	1.16	16.8	76	Hard
SC687-1	150	2,369	86	1.07	18.2	84	Hard
SC897-1	150	2,576	85	1.10	17.5	80	Hard
SC897-2	150	2,483	87	1.23	16.2	71	Moderately Hard
SC Master Comp.	150	2,649	83	1.12	17.0	77	Hard

The samples were characterized as moderately hard to hard from the ball milling perspective, with their values ranging from 14.8 to 17.8 kWh/t. These results align with the historical tests suggesting once again that a SAB (SAG/Ball milling circuit) will be suitable for processing of the Premier ores. In Figure 13-8, the test results are shown relative to the SGS database.



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-8: Comparison of Bond Ball Mill Work Index Results to SGS Database

13.2.5 Hydrometallurgical Testwork—Master Composites

Extended Gravity Recoverable Gold Tests

It is worth noting that the previous testwork campaign failed to produce E-GRG tests for Big Missouri, Northstar, and Silver Coin samples. The purpose of the 2019 testwork was to address these gaps and provide estimates of gold recoveries for equipment sizing, based on a complete set of E-GRG results that are able to support a feasibility study level of project definition.

An E-GRG test was performed on all three master composite samples. The GRG value obtained from the E-GRG test is an indication of the ore's amenability to gravity concentration as a function of particle size distribution.

The E-GRG test utilizes a Knelson gravity concentrator to perform gravity separation after each stage of grinding, and consists of three sequential liberation points (at approximate (P_{80}s of 840, 250, and 75 µm, respectively) and recovery stages. The assays from the all three stages were used to construct a metallurgical balance from which the GRG value is calculated. A summary of the results from the E-GRG tests for the each of the master composites is presented in Table 13-17, Table 13-18, and Table 13-19, respectively.

Table 13-17: E-GRG Results for the Silver Coin Master Composite Sample

Grind Size		Product	Mass (g)	Mass (%)	Assay Au (g/t)	Unit Au	Distribution (%)
P_{80} =	780 µm	Stage 1 Conc.	85.3	0.90	130.5	11,128	19.2
		Sampled Tailings	224	2.36	3.98	891	1.5
P_{80} =	174 µm	Stage 2 Conc.	102	1.08	201.5	20,574	35.5
		Sampled Tailings	481	5.07	2.54	1,224	2.1
P_{80} =	50 µm	Stage 3 Conc.	86	0.91	144.1	12,395	21.4
		Final Tailings	8,513	89.7	1.39	11,820	20.4
		Totals (Head)	9,491	100	6.11	58,034	100
		Knelson Conc.	273.4	2.88	161.3	44,098	76.0
		Direct Head (3x)	-	-	5.56	-	-

Table 13-18: E-GRG Results for the Big Missouri Master Composite Sample

Grind Size		Product	Mass (g)	Mass (%)	Assay Au (g/t)	Unit Au	Distribution (%)
P_{80} =	667 µm	Stage 1 Conc.	83.5	0.88	68.4	5,714	11.2
		Sampled Tailings	276	2.92	4.16	1,148	2.2
P_{80} =	146 µm	Stage 2 Conc.	107	1.13	183.3	19,527	38.2
		Sampled Tailings	466	4.93	2.45	1,140	2.2
P_{80} =	44 µm	Stage 3 Conc.	81	0.86	117.2	9,466	18.5
		Final Tailings	8,427	89.3	1.68	14,168	27.7
		Totals (Head)	9,439	100	5.42	51,163	100
		Knelson Conc.	270.8	2.87	128.2	34,707	67.8
		Direct Head (3x)	-	-	4.55	-	-

Table 13-19: E-GRG Results for the Premier Master Composite Sample

Grind Size		Product	Mass (g)	Mass (%)	Assay Au (g/t)	Unit Au	Distribution (%)
P_{80} =	708 µm	Stage 1 Conc	83.4	0.89	243.5	20,305	23.3
		Sampled Tailings	380	4.07	4.90	1,861	2.1
P_{80} =	176 µm	Stage 2 Conc	104	1.11	409.4	42,411	48.7
		Sampled Tailings	285	3.05	2.24	637	0.7
P_{80} =	45 µm	Stage 3 Conc	85	0.91	143.3	12,125	13.9
		Final Tailings	8,388	90.0	1.16	9,755	11.2
		Totals (Head)	9,324	100	9.34	87,095	100
		Knelson Conc	271.6	2.91	275.6	74,841	85.9
		Direct Head (3x)	-	-	8.88	-	-

The GRG values for the SC, BM, and PM Master composites after three stages were 76.0%, 67.8%, and 85.9%. respectively. Each composite responded very well to gravity separation, with the majority of the gold being recovered within the first two passes (combined mass pull of approximately 2%), at measured P_{80} sizes between 146 and 176 µm.

Optimization Testing—Gravity Separation and Standard Whole Ore Leach Tests

Optimization tests were conducted on three master composite samples with the purpose of determining the ideal gravity and leach conditions.

For each of the three master composites (SC, BM, and PM) a total of six 2 kg charges were used for four gravity separation tests for each master composite. The first three 2 kg charges were ground to three different particle sizes (P_{80}s of 75, 95, and 115 µm) to investigate the effect of particle size on the overall gold recovery by gravity and leaching of the gravity tailings. The fourth test sample was composed of the final three 2 kg charges which were ground separately, then blended together to form a homogenous 6 kg sample for gravity, and split into three charges for leach testing under various NaCN concentrations (0.5, 0.75, and 1.0 g/L). The target grind size for the bulk gravity separation test was 75 µm. A summary of the composite samples gravity separation results is presented in Table 13-20.

For the Silver Coin composite (SC Comp), the gravity gold recovery ranged from 41.1% to 55.2% for the calculated head grades of 5.77 to 6.57 g/t Au. The Big Missouri composite (BM Comp) was less responsive to gravity separation than SC Comp, with gold recoveries ranging from 29.9% to 42.3% (calculated head grades ranged from 4.91 to 6.78 g/t Au). Finally, the most encouraging gravity separation gold recoveries (48.8% to 63.9%) were observed with the Premier composite (PM Comp), with the gold head grades ranging from 7.7 to 9.8 g/t Au. There was no clear relationship between particle size and gold recovery.

As described above, all composite gravity tailings samples were submitted to a whole ore leach (WOL) under the following conditions:

- A pH between 10 and 10.5
- A leach slurry density of 40% solids wieght by weight (w/w)
- A NaCN concentration between 0.5 and 1 g/L
- Sampling at 2, 4 ,6 ,8 ,12, 24, 36, and 48 h increments.

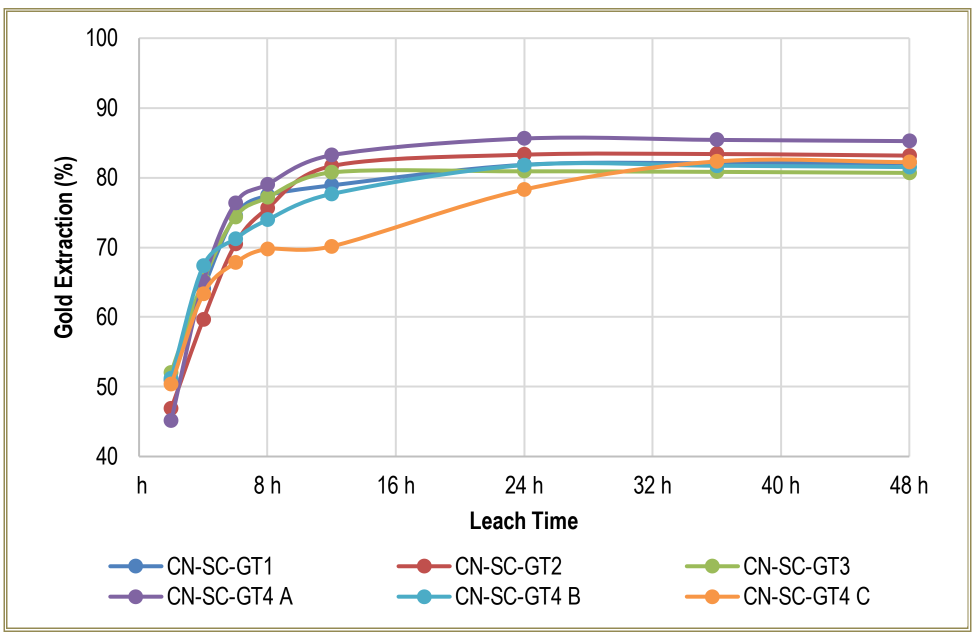
A summary of the WOL results on composite tailings samples is presented in Table 13-21. Gold extraction kinetics for the SC Comp, BM Comp, and PM Comp tailings are presented in Figure 13-9, Figure 13-10, and Figure 13-11, respectively.

Table 13-20: Composite Gravity Separation Results

Test ID	Composite ID	Grind Size (K_{80}, µm)	Mass Processed (kg)	Stream	Weight (%)	Assay (g/t Au)	Distribution (% Au)
SC Comp G1	SC Composite	82	2	Knel Conc.	1.37	219	50.2
				Gravity Tailings	98.6	3.01	49.8
				Calc. head	100.0	5.96	
SC Comp G2	SC Composite	96	2	Knel Conc.	1.49	203	45.7
				Gravity Tailings	98.5	3.62	54.3
				Calc. head	100.0	6.57	
SC Comp G3	SC Composite	114	2	Knel Conc.	1.74	191	55.2
				Gravity Tailings	98.3	2.74	44.8
				Calc. head	100.0	6.01	
SC Comp G4	SC Composite	82	6	Knel Conc.	0.52	461	41.1
				Gravity Tailings	99.5	3.42	58.9
				Calc. head	100.0	5.77	
BM Comp G1	BM Composite	72	2	Knel Conc.	0.95	213	29.9
				Gravity Tailings	99.1	4.80	70.1
				Calc. head	100.0	6.78	
BM Comp G2	BM Composite	92	2	Knel Conc.	1.23	167	41.8
				Gravity Tailings	98.8	2.89	58.2
				Calc. head	100.0	4.91	
BM Comp G3	BM Composite	112	2	Knel Conc.	1.56	103	32.8
				Gravity Tailings	98.4	3.35	67.2
				Calc. head	100.0	4.91	
BM Comp G4	BM Composite	71	6	Knel Conc.	0.78	280	42.3
				Gravity Tailings	99.2	3.01	57.7
				Calc. head	100.0	5.17	
PM Comp G1	PM Composite	76	2	Knel Conc.	1.64	307	61.9
				Gravity Tailings	98.4	3.14	38.1
				Calc. head	100.0	8.10	
PM Comp G2	PM Composite	92	2	Knel Conc.	1.45	432	63.9
				Gravity Tailings	98.6	3.59	36.1
				Calc. head	100.0	9.80	
PM Comp G3	PM Composite	119	2	Knel Conc.	1.58	289	58.6
				Gravity Tailings	98.4	3.28	41.4
				Calc. head	100.0	7.80	
PM Comp G4	PM Composite	74	6	Knel Con	0.58	650	48.8
				Gravity Tailings	99.4	3.97	51.2
				Calc. head	100.0	7.70	

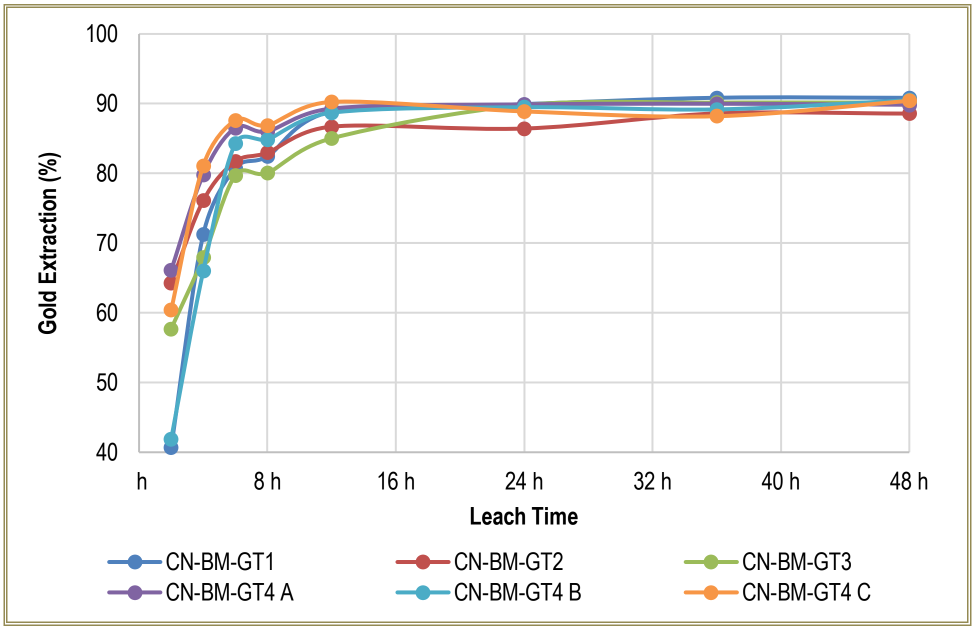
Table 13-21: Composite Gravity Tailings WOL Test Results

Test ID	Actual (K_{80} µm)	NaCN Concentration (g/L)	Head Assays Calc Au (g/t)	Residue Au (g/t)	Addition (kg/t) NaCN	CaO	Consumption (kg/t) NaCN	CaO	Final Gold Extraction Calc. (%)
CN-SC-GT1	82	1.00	3.01	0.54	1.94	0.98	0.45	0.85	82.1
CN-SC-GT2	96	1.00	3.62	0.61	1.89	0.96	0.40	0.81	83.2
CN-SC-GT3	114	1.00	2.74	0.53	1.82	0.96	0.33	0.80	80.7
CN-SC-GT4 A	82	0.50	4.13	0.61	1.14	0.90	0.32	0.78	85.2
CN-SC-GT4 B	82	0.75	2.97	0.55	1.35	0.79	0.35	0.71	81.5
CN-SC-GT4 C	82	1.00	3.15	0.56	1.46	0.83	0.43	0.73	82.2
CN-BM-GT1	72	1.00	4.80	0.44	1.91	0.74	0.42	0.69	90.8
CN-BM-GT2	92	1.00	2.89	0.33	1.89	0.76	0.44	0.70	88.6
CN-BM-GT3	112	1.00	3.35	0.33	1.88	0.76	0.39	0.69	90.1
CN-BM-GT4 A	71	0.50	2.76	0.28	1.01	0.77	0.31	0.71	89.8
CN-BM-GT4 B	71	0.75	2.94	0.28	1.46	0.75	0.40	0.70	90.5
CN-BM-GT4 C	71	1.00	3.33	0.32	1.48	0.74	0.43	0.69	90.4
CN-PM-GT1	76	1.00	3.14	0.11	2.12	0.86	0.52	0.77	96.5
CN-PM-GT2	92	1.00	3.59	0.18	1.93	0.80	0.68	0.72	95.0
CN-PM-GT3	119	1.00	3.28	0.12	1.90	0.78	0.42	0.65	96.3
CN-PM-GT4 A	74	0.50	3.60	0.23	1.01	0.76	0.33	0.67	93.6
CN-PM-GT4 B	74	0.75	3.64	0.12	1.49	0.74	0.44	0.66	96.7
CN-PM-GT4 C	74	1.00	4.66	0.14	1.53	0.77	0.47	0.67	97.0



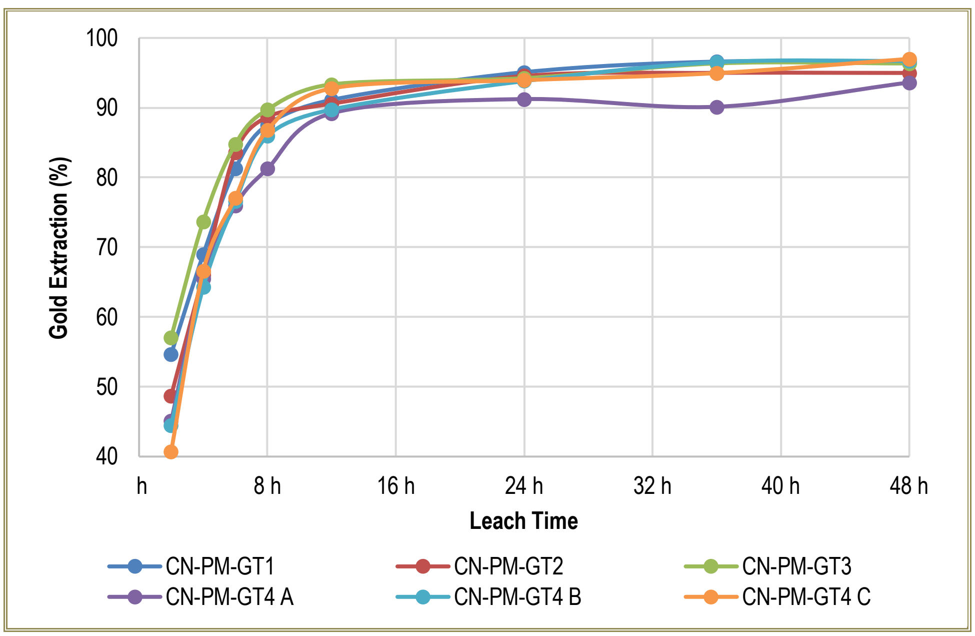
Source: SGS Canada, Project 17709-011, February 2020

Figure 13-9: WOL Gold Extraction Kinetics—SC Comp



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-10: WOL Gold Extraction Kinetics—BM Comp



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-11: WOL Gold Extraction Kinetics—PM Comp

The highest gold extraction values were obtained from the Premier Composite (PM Comp), with the final gold extractions ranging from 93.6% to 97.0%. The gold extraction values ranged from 80.7% to 85.2% for the SC Comp and from 88.6% to 90.8% for the BM Comp. Overall, the leach kinetics were very fast with the majority of gold extraction occurring within the first 12 h of leaching.

Gold extraction was slightly higher at the finest grind size. With an accommodation of the wide variability in calculated head grade for the grind series, the effect is not clearly defined. However, it can be concluded that residual gold grades were lowest at the finer grind size.

Bulk Gravity and Carbon-In-Leach Testing

Each of the three master composites were subjected to a bulk gravity test followed by the carbon-in-leach (CIL) testing aiming to establish a response that is more likely to align with a full-scale gravity-leach flowsheet in an operating plant. Optimal conditions selected for the bulk gravity and CIL tests included a particle grind size of 75 µm, a slurry pH of 10 to 10.5, and a CN concentration of 0.5 g/L of NaCN.

A gravity separation metallurgical balance was generated from the assays of the gravity concentrates, and the calculated head assay of the gravity Tailings submitted for the CIL tests. The results from the bulk gravity tests are tabulated in Table 13-22.

Table 13-22: Composite Bulk Gravity Separation Results

Test ID	Composite ID	Grind Size (K_{80} µm)	Stream	Weight (%)	Assay (g/t) Au	Ag	Distribution (%) Au	Ag
SC Comp Bulk	SC Composite	82	Knel Conc.	0.53	617	572	54.5	20.7
			Gravity Tailings	99.5	2.73	11.60	45.5	79.3
			Calc. head	100.0	5.96	14.60		
BM Comp Bulk	BM Composite	73	Knel Conc.	0.51	475	358	51.0	20.1
			Gravity Tailings	99.5	2.33	7.30	49.0	79.9
			Calc. head	100.0	4.73	9.05		
PM Comp Bulk	PM Composite	69	Knel Conc.	0.53	1364	934	75.2	28.3
			Gravity Tailings	99.5	2.40	12.70	24.8	71.7
			Calc. head	100.0	9.64	17.60		

For the SC Comp, the gold recovery was 54.5% with a calculated head grade of 5.96 g/t Au. The BM Comp was slightly less responsive to gravity separation than the SC Comp, with a gold recovery of 51% and calculated head grade of 4.73 g/t Au. Finally, the Premier composite (PM Comp) test demonstrated the highest recovery result from gravity separation, with the gold recovery of 75.2% and a calculated head grade of 9.64 g/t Au. The gold recovery results align with the optimization test results (Table 13-20).

The gravity tailings were re-pulped to form a 40% w/w solids slurry and subjected to CIL tests under optimized conditions; targeting a grind size of 75 µm, slurry pH of 10 to 10.5, and CN concentration of 0.5 g/L NaCN, which was maintained throughout the 48-h leach period.

Table 13-23 shows a summary of the bulk gravity tailings CIL test results.

Table 13-23: Composite Bulk CIL Test Results

Test #	Target (P_{80} µm)	Actual (P_{80} µm)	Head Assays Calculated Au (g/t)	Head Assays Calculated Ag (g/t)	Residue Assays Au (g/t)	Residue Assays Ag (g/t)	Consumption NaCN (kg/t)	Consumption CaO (kg/t)	Extraction Calculations Au (%)	Extraction Calculations Ag (%)
SC Comp Bulk CIL	75	82	2.78	11.61	0.37	5.00	0.94	0.46	86.7	56.9
BM Comp Bulk CIL	75	73	2.33	7.27	0.32	3.00	0.95	0.57	86.3	58.7
PM Comp Bulk CIL	75	69	2.40	12.67	0.20	6.00	0.96	0.42	91.7	52.6

The final gold extractions were 86.7% with a calculated head grade of 2.78 g/t Au for the SC Comp test; 86.3% with a calculated head grade of 2.33 g/t Au for the BM Comp and 91.7% with a calculated head grade of 2.40 g/t Au for the PM Comp test. A considerable amount of silver was also extracted in the bulk CIL tests: 56.9%, 58.7%, and 52.6% for the SC, BM, and PM Comp samples, respectively.

The bulk CIL tests performed very well under the test conditions. Although the recoveries for the BM and PM Composites appeared slightly lower than the optimization tests, the residual gold grades of the leach tailings are considered to be generally similar.

The remaining gravity tailings quantities were dispatched to Kemetco's laboratory in Richmond, BC, for the CN destruction/detoxification testwork.

Carbon Adsorption and Isotherm Testing

Carbon Adsorption

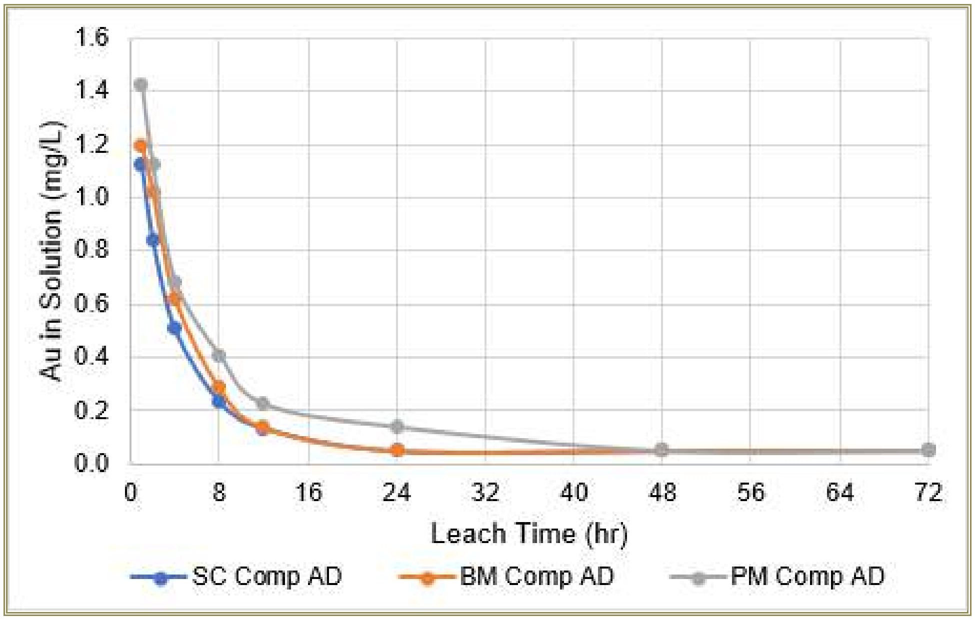
A blend of the Pregnant Leach Solution (PLS) from the composite WOL bottle roll tests was submitted for carbon adsorption and isotherm testing.

The carbon adsorption tests (AD) were completed; one for each composite, identified as SC Comp AD, BM Comp AD, and PM Comp AD. Each test charge was contacted with a carbon concentration of 1.5 g/L. Throughout each test, the pH level was maintained at 10.5 using lime addition. The kinetic barren solution and a subsample of the final loaded carbon was also submitted for gold and silver assay.

A summary of the results for the carbon adsorption tests is presented in Table 13-24. The kinetics of gold in solution are shown in Figure 13-12.

Table 13-24: Carbon Adsorption Test Results

Test ID	Sample Type	Carbon Conc. (g/L)	Head Assays Calc. Au (g/t)	Head Assays Calc. Ag (g/t)	Final Barren Solution Assays Au (g/t)	Final Barren Solution Assays Ag (g/t)	Loaded Carbon Assays Au (g/t)	Loaded Carbon Assays Ag (g/t)	Final Extraction Calc. Au (%)	Final Extraction Calc. Ag (%)
SC Comp. AD	CN-SC-GT1 to GT4 PLS	1.5	1.47	5.14	0.05	0.778	879	2,649	90.5	78.1
BM Comp. AD	CN-BM-GT1 to GT4 PLS	1.5	1.65	2.49	0.05	0.510	978	1,153	90.6	70.6
PM Comp. AD	CN-PM-GT1 to GT4 PLS	1.5	1.85	4.08	0.05	0.865	1089	1,867	90.3	70.1



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-12: Gold in Solution for Carbon Adsorption Tests

The final gold extraction for each composite was essentially the same, with values of 90.5%, 90.6%, and 90.3% for the SC Comp, BM Comp, and PM Comp, respectively. The final gold extraction value represents the gold recovery using carbon, therefore suggesting that the carbon was effective at adsorbing the valuable gold in solution. From Figure 13-12 it can be concluded that the gold in solution decreases as the carbon adsorbs the gold and becomes loaded.

Isotherm Testing

A total of 15 carbon isotherms were completed, five for each composite, identified as ISO-1 to ISO-5. Approximately 1-L charges of PLS were contacted with five different carbon concentrations (0.5 g/L, 1.0 g/L, 2.0 g/L, 3.0 g/L, and 8.0 g/L). Lime was used to maintain the pH levels at 10.5 throughout each test. Subsamples of the final barren solution and loaded carbon were submitted for gold and silver assays to complete the metallurgical balance. Table 13-25 presents a summary of results for the carbon adsorption isotherm tests.

The final gold extractions ranged from 88.6% to 97.0% for the SC Comp; 91.1% to 97.1% for the BM Comp, and 86.1% to 97.5% for the PM Comp. The first test (ISO-1) for each composite (using the lowest carbon concentration of 0.5 g/L), demonstrated the lowest extraction value. With an increase in carbon concentration to 1.0 g/L, the gold extraction increased by 6% to 13%. Further increases in carbon concentration to 2.0, 3.0, and 8.0 g/L resulted in a modest increase in gold extraction, by only 0.1%.

A silver extraction of 95% was achieved only at the highest carbon concentration of 8 g/L, suggesting that more than 3 g/L carbon is required to achieve 90% plus silver adsorption onto the carbon.

Table 13-25: Carbon Isotherm Test Results

Test ID	Sample Type	Carbon Conc. (g/L)	Head Assays Calc. (g/t) Au	Ag	Loaded Carbon Assays (g/t) Au	Ag	Final Extraction Calc. (%) Au	Ag
SC Comp ISO-1	CN-SC-GT1 to GT4 PLS	0.5	1.53	4.68	2,575	4,377	88.6	49.3
SC Comp ISO-2	CN-SC-GT1 to GT4 PLS	1.0	1.54	4.60	1,419	3,091	96.9	70.6
SC Comp ISO-3	CN-SC-GT1 to GT4 PLS	2.0	1.47	4.67	691	1,974	96.8	87.1
SC Comp ISO-4	CN-SC-GT1 to GT4 PLS	3.0	1.53	2.64	478	742	96.9	86.8
SC Comp ISO-5	CN-SC-GT1 to GT4 PLS	8.0	1.57	4.25	186	490	97.0	94.6
BM Comp ISO-1	CN-BM-GT1 to GT4 PLS	0.5	1.66	2.93	2,917	2,771	91.1	49.0
BM Comp ISO-2	CN-BM-GT1 to GT4 PLS	1.0	1.61	2.74	1,503	1,941	97.0	73.4
BM Comp ISO-3	CN-BM-GT1 to GT4 PLS	2.0	1.66	2.73	781	1,158	97.1	87.5
BM Comp ISO-4	CN-BM-GT1 to GT4 PLS	3.0	1.66	1.77	527	512	97.1	88.6
BM Comp ISO-5	CN-BM-GT1 to GT4 PLS	8.0	1.61	4.45	191	522	97.0	96.2
PM Comp ISO-1	CN-PM-GT1 to GT4 PLS	0.5	1.97	4.72	3,179	3,807	86.1	43.0
PM Comp ISO-2	CN-PM-GT1 to GT4 PLS	1.0	1.93	4.55	1,805	2,894	97.5	66.2
PM Comp ISO-3	CN-PM-GT1 to GT4 PLS	2.0	1.90	4.52	892	1,878	97.5	86.2
PM Comp ISO-4	CN-PM-GT1 to GT4 PLS	3.0	1.88	4.65	594	1,383	97.4	91.9
PM Comp ISO-5	CN-PM-GT1 to GT4 PLS	8.0	1.92	4.60	229	534	97.5	95.1

13.2.6 Hydrometallurgical Testwork—Variability Samples

A 2 kg charge of each of the variability samples was used for the gravity separation and CIL testing circuit to determine the gold recovery, leaching kinetics, and to generate gold-bearing process slurry for potential further characterization. Sixteen tests were conducted at the optimal particle grind size and CN dosage conditions as identified during the composite samples testwork phase.

Gravity Separation

A 2 kg charge for each variability samples was ground to a target P_{80} of 75 µm and fed to the Knelson concentrator as a pulp. The concentrate obtained was further upgraded by a Mozley separator, and the final concentrate was submitted for gold assaying. The remaining gravity tailings were subjected to CIL bottle roll tests. Table 13-26 presents the results from the gravity separation testwork.

The recovery of gravity gold ranged from 17.3% to 71.7% for the Silver Coin variability samples. Recovery ranges were16.3% to 52.9% for the Big Missouri variability samples and 30.8% to 71.7% for the Premier variability samples.

Table 13-26: Variability Gravity Separation Test Results

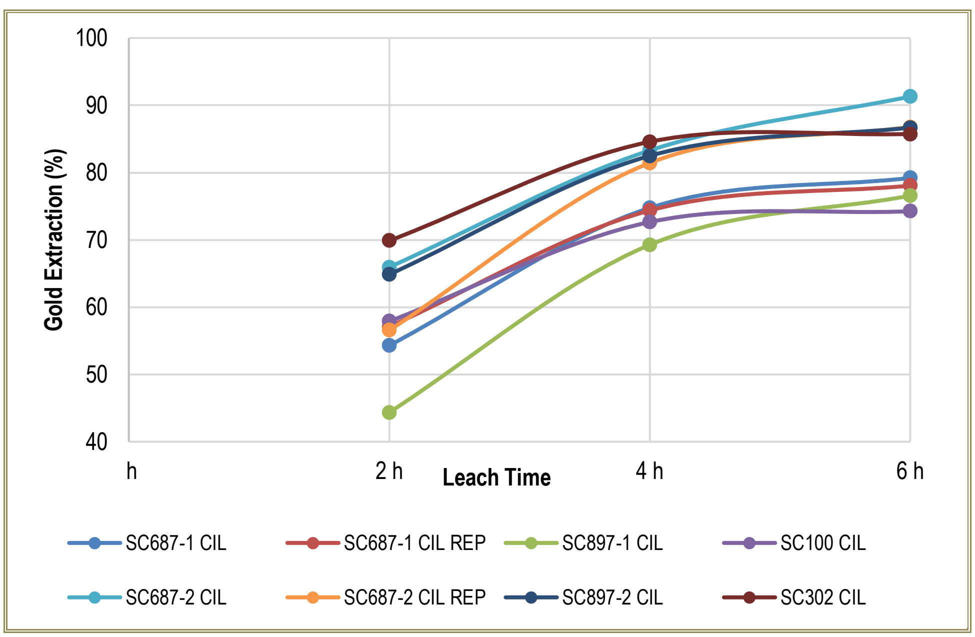
Test ID	Composite ID	Grind Size (P_{80}, µm)	Stream	Weight (%)	Assay (g/t Au)	Distribution (% Au)
SC687-1 Gravity REP	SC687-1	76	Knel Conc.	1.16	176	44.0
			Gravity Tailings	98.8	2.63	56.0
			Calc. head	100.0	4.65	
SC687-2 Gravity REP	SC687-2	95	Knel Conc.	1.21	79	17.3
			Gravity Tailings	98.8	4.65	82.7
			Calc. head	100.0	5.55	
SC897-1 Gravity	SC897-1	76	Knel Conc.	1.16	330	64.6
			Gravity Tailings	98.8	2.13	35.4
			Calc. head	100.0	5.95	
SC897-2 Gravity	SC897-2	83	Knel Conc.	1.03	574	71.7
			Gravity Tailings	99.0	2.35	28.3
			Calc. head	100.0	8.22	
SC100 Gravity	SC100 Gr	74	Knel Conc.	1.12	241	64.4
			Gravity Tailings	98.9	1.51	35.6
			Calc. head	100.0	4.19	
SC302 Gravity	SC302 Gr	82	Knel Conc.	1.71	82	37.2
			Gravity Tailings	98.3	2.41	62.8
			Calc. head	100.0	3.77	
BM1 Gravity	BM1 Grav	78	Knel Conc.	1.22	209	52.9
			Gravity Tailings	98.8	2.30	47.1
			Calc. head	100.0	4.82	
BM2 Gravity	BM2 Grav	71	Knel Conc.	1.06	102	33.3
			Gravity Tailings	98.9	2.19	66.7
			Calc. head	100.0	3.25	
BM3 Gravity	BM3 Grav	71	Knel Conc.	1.21	320	44.5
			Gravity Tailings	98.8	4.88	55.5
			Calc. head	100.0	8.69	
BM4 Gravity	BM4 Grav	80	Knel Conc.	0.53	90	16.5
			Gravity Tailings	99.5	2.44	83.5
			Calc. head	100.0	2.91	
BM5 Gravity	BM5 Grav	76	Knel Conc.	0.43	92	16.3
			Gravity Tailings	99.6	2.05	83.7
			Calc. head	100.0	2.44	
PM609 Gravity	PM609 Gr	76	Knel Conc.	1.24	159	30.8
			Gravity Tailings	98.8	4.48	69.2
			Calc. head	100.0	6.40	
PM602 Gravity	PM602 Gr	81	Knel Conc.	1.47	628	65.4
			Gravity Tailings	98.5	4.96	34.6
			Calc. head	100.0	14.11	

Test ID	Composite ID	Grind Size (P_{80}, µm)	Stream	Weight (%)	Assay (g/t Au)	Distribution (% Au)
PM NL Gravity	PM NL Gr	74	Knel Conc.	0.73	402	57.9
			Gravity Tailings	99.3	2.14	42.1
			Calc. head	100.0	5.04	
PM Prew Gravity	PM Prew	73	Knel Conc.	1.00	320	47.2
			Gravity Tailings	99.0	3.62	52.8
			Calc. head	100.0	6.79	
PM Ben Gravity	PM Ben G	81	Knel Conc.	1.11	1,492	71.7
			Gravity Tailings	98.9	6.59	28.3
			Calc. head	100.0	23.01	

Carbon-in-Leach Bottle Roll Testing

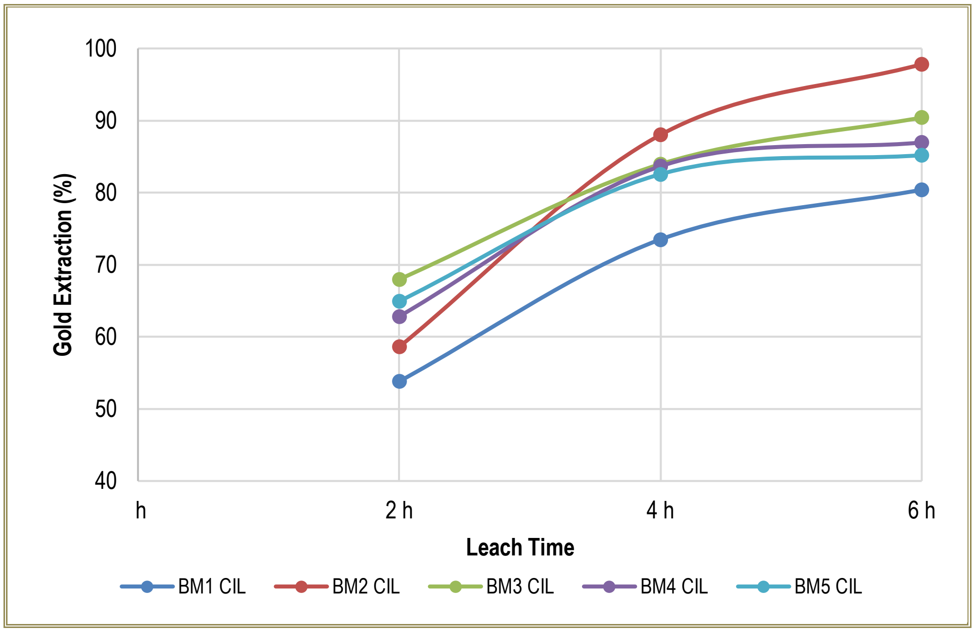
The tailings from the variability gravity separation tests were re-pulped to 40% w/w solids slurry in bottles and placed on the rollers for a CIL test. Lime was added to maintain a slurry pH between 10 and 10.5 and the NaCN concentration was maintained at 0.5 g/L throughout the entire 48 h leach period. The sixteen CIL tests were monitored at 2 h, 4 h, and 6 h, recording pH measurements and DO levels with kinetic PLS subsamples being removed for gold and silver assays. Activated carbon was added 6 h into the leach, at a concentration of 10 g/L. After 48 h, the tests were terminated. The loaded carbon was screened out and submitted for gold and ICP assay (to extinction) and a subsample of the leach residue was submitted for gold, silver, and ICP assay.

The leach kinetics from 2 to 6 h are presented in Figure 13-13, Figure 13-14, and Figure 13-15, respectively.



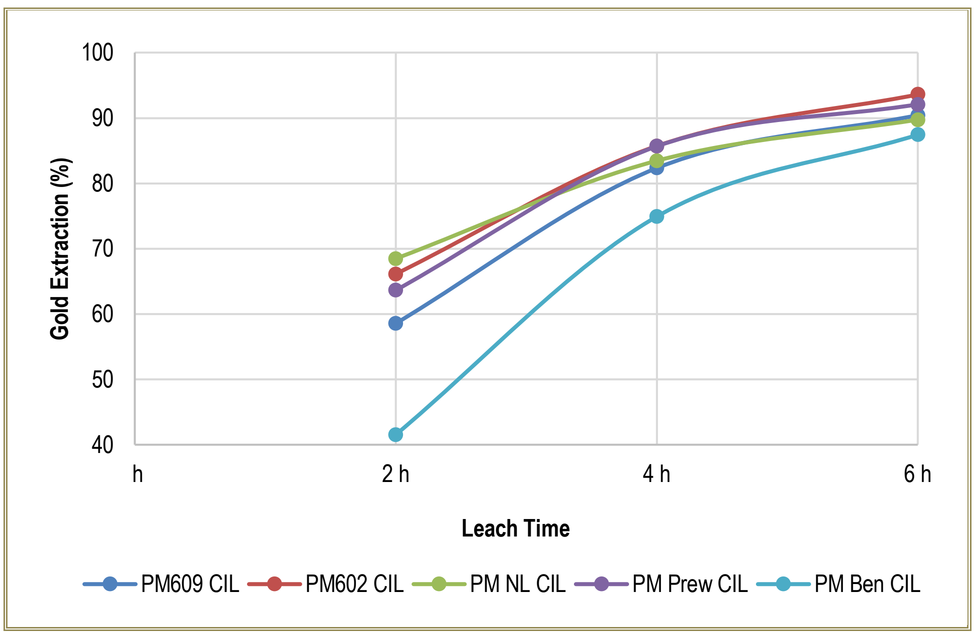
Source: SGS Canada, Project 17709-011, February 2020

Figure 13-13: Silver Coin Variability CIL Kinetics



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-14: Big Missouri Variability CIL Kinetics



Source: SGS Canada, Project 17709-011, February 2020

Figure 13-15: Premier Variability CIL Kinetics

During the first 6 h period, the extent of gold leaching in the solution ranged from 74% to 91% for the Silver Coin variability samples; from 80% to 98% for the Big Missouri variability samples and from 87% to 94% for the Premier variability samples. After 6 h of leaching, activated carbon was added in solution and gold was recovered on the carbon. Table 13-27 presents the test results from the variability CIL tests.

Table 13-27: Variability CIL Test Results

Test #	Actual (P_{80} µm)	Head Assays Calc. (Au g/t)	Residue Assays (Au g/t)	Consumption (kg/t) NaCN	Consumption (kg/t) CaO	Au Extraction 6 h PLS (%)	Final Au Extraction Calc. (%)
SC687-1 CIL REP	76	2.71	0.34	1.02	0.46	78.1	87.5
SC897-1 CIL	76	2.20	0.25	0.96	0.55	76.6	88.7
SC100 CIL	74	1.59	0.16	0.97	0.52	74.3	89.9
SC687-2 CIL REP	95	4.72	0.36	1.29	0.27	86.8	92.4
SC897-2 CIL	83	2.43	0.23	1.03	0.55	86.7	90.5
SC302 CIL	82	2.49	0.30	1.04	0.56	85.7	88.0
BM1 CIL	78	2.38	0.29	0.97	0.45	80.4	87.8
BM2 CIL	71	2.27	0.18	1.10	0.45	97.8	92.1
BM3 CIL	71	4.96	0.29	0.98	0.50	90.4	94.2
BM4 CIL	80	2.52	0.20	1.03	0.45	87.0	92.1
BM5 CIL	76	2.12	0.25	0.99	0.51	85.2	88.2
PM609 CIL	76	4.55	0.16	0.95	0.31	90.4	96.5
PM602 CIL	81	5.04	0.16	1.03	0.32	93.6	96.8
PM NL CIL	74	2.22	0.14	1.08	0.28	89.7	93.7
PM Prew CIL	73	3.70	0.12	0.95	0.33	92.1	96.8
PM Ben CIL	81	6.67	0.07	0.98	0.32	87.4	98.9

The Silver Coin variability samples leached well, generating residues of 0.16 to 0.36 g/t Au, with the final gold extraction values between 87.5% and 92.4%.

The Big Missouri samples also leached effectively, achieving slightly higher extraction values (87.8% to 94.2%), while generating leach residues that contained between 0.18 and 0.29 g/t Au.

The most encouraging results were achieved while leaching the Premier variability samples, with the final gold extractions ranging from 93.7% to 98.9%. The gold content in the leach residue was between 0.07 and 0.16 g/t.

The average NaCN and lime consumptions were 1.04 and 0.44 kg/t, respectively.

The bottle roll leach extractions of the variability samples were generally similar to, or higher than, the results achieved with the composites, with some Silver Coin samples exhibiting up to a 10% increase in the gold extraction. The difference can be attributed to the fact that the variability bottle roll tests were operated as a CIL sequence, whereas the composite bottle roll tests were operated as WOL.

13.2.7 Cyanide Detoxification Testwork (Kemetco 2019)

Three composite residue samples from the bulk CIL testwork labelled SC (Silver Coin); BM (Big Missouri and PM (Premier) were delivered to Kemetco's research facility in December 2019. The samples were delivered promptly (within 4 h), such that the CN destruction tests could be completed without the impact of oxidative processes taking effect.

All three slurry samples were subjected to CN destruction testing using the SO_2/Oxygen (O_2) and Caro's acid processes, targeting a residual CN concentration of 1 ppm in the detox effluent.

The feed composition of the residue samples received at Kemetco Research is presented in Table 13-28.

Table 13-28: Detox Feed Composition

Element	Detox Feed BM ½ (mg/L)	Detox Feed BM 2/2 (mg/L)	Detox Feed PM ½ (mg/L)	Detox Feed PM 2/2 (mg/L)	Detox Feed SC ½ (mg/L)	Detox Feed SC 2/2 (mg/L)
CN_{WAD} (picric)	258	250	259	268	256	255
CN_{WAD} (distillation)		271		281		266
Ag silver	< 0.1	< 0.1	0.15	0.156	0.14	0.15
Al Aluminum	0.94	1.02	27.87	2.57	1.77	1.53
As Arsenic	< 0.4	< 0.4	< 0.4	< 0.4	< 0.4	< 0.4
Au Gold	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
B Boron	< 1.0	< 1.0	< 1.0	< 1.0	< 1.0	< 1.0
Ba Barium	0.07	0.09	0.12	0.11	0.12	0.12
Be Beryllium	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02
Bi Bismuth	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
Ca Calcium	16.0	18.3	10.4	9.3	17.6	16.8
Cd Cadmium	0.16	0.16	0.29	0.29	0.17	0.17
Co Cobalt	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Cr Chromium	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Cu Copper	6.41	6.34	4.96	5.01	6.30	6.52
Fe Iron	0.49	0.5	1.09	1.27	0.8	0.58
K Potassium	45.6	47.0	46.2	45.8	46.9	48.3
Li Lithium	<0.2	0.21	0.28	0.2	0.52	0.45
Mg Magnesium	<0.2	0.81	1.08	0.67	2.00	1.39
Mn Manganese	<0.02	<0.02	0.03	0.05	<0.02	<0.02
Mo Molybdenum	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Na Sodium	270	271	277	279	280	286
Ni Nickel	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
P Phosphorus	<0.6	<0.6	<0.6	<0.6	<0.6	<0.6
Pb Lead	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
S Sulfur	73.6	81.4	88.2	85.4	80.6	81.3
Sb Antimony	0.42	<0.4	<0.4	<0.4	0.59	0.64
Se Selenium	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
Si Silica	11.9	11.8	11.4	11.0	13.5	13.2
Sn Tin	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
Sr Strontium	0.33	0.37	0.18	0.16	0.17	0.16
Ti Titanium	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Tl Thalium	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
U Uranium	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0
V Vanadium	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Zn Zinc	24.5	24	45.5	46.5	22.4	23.1

Slurry SO_2/O_2 Testwork

The SO_2/O_2 process can be applied to both the treatments of CN solution and pulp. The main advantage of this process is that the reduction of the total CN (CN_{total}) to levels of less than 1 mg/L in a continuous mode can be achieved at a low operating cost. The technology uses sulfur dioxide (SO_2) in various reagent forms, most commonly sodium metabisulfite in combination with air or pure oxygen.

For each of the three samples, continuous flow CN destruction tests were started after an initial batch treatment. All the tests were conducted in a single stage. During the treatment of two of the three slurry samples (BM and SC), process variables were adjusted to minimize the CN_{WAD}. The third sample (PM) was treated using the conditions established during previous tests.

Eight continuous CN destruction tests were conducted in total, with the test conditions and results tabulated below in Table 13-29. Table 13-30 presents the CN detox effluent analysis.

Table 13-29: SO_2/O_2 Cyanide Destruction Testing Conditions and Results

					Solution						Treatment Conditions				
Feed	Test No.	Stream	CN ISE (mV)	D.O. ppm	CN_{WAD} (ppm)	CNt (ppm)	Cu (ppm)	Ni (ppm)	Fe (ppm)	Zn (ppm)	Return Time (min)	pH	SO_2 Ratio gSO_2/gCN	Cu (mg/L)	CaO $gCaO/gSO_2$
SC	T-4	Feed	40	11.0	250	252	7.2	0.1	0.7	28.4	52	10.9	5.1	19.0	0.66
		Effluent			1.0		7.2	0.1	0.7	28.4		8.5			
	T-5	Feed	-40	11.1	250		7.2	0.1	0.7	28.4	53	10.0	5.1	9.0	0.53
		Effluent			2.0		7.2	0.1	0.7	28.4		8.5			
	T-6	Feed	-42	12.1	250		8.0	0.1	1.2	34.5	76	11.0	4.7	9.0	0.77
		Effluent			0.5		0.3	0.1	0.1	0.3		8.5			
	T-7	Feed	-10	11.9	250		8.0	0.1	1.2	34.5	75	11.0	4.5	18.0	0.58
		Effluent			0.2		0.4	0.1	0.1	0.5		8.5			
BM	T-1	Feed	-25	10.5	228	228	5.0	0.1	1.1	45.5	52	10.9	5.0	20.0	0.77
		Effluent			1.4		0.5	0.1	0.2	0.4		8.0			
	T-2	Feed	-68	10.7	200		8.4	0.1	1.2	38.4	54	10.9	4.5	9.0	0.80
		Effluent			1.1		0.5	0.1	0.2	0.4		8.0			
	T-3	Feed	-2	10.5	200		8.4	0.1	1.2	38.4	54	10.9	4.6	19.0	0.55
		Effluent			1.0		0.5	0.1	0.2	0.4		8.0			
PM	T-8	Feed	-76	9.8	270	273	5.6	0.1	0.9	53.6	52	10.9	5.0	19.0	0.67
		Effluent			0.5		0.6	0.1	0.1	0.5		8.5			

The SO_2/Oxygen CN destruction process was successful in reducing the CN_{WAD} to 1 ppm (or below) in all three Premier slurry samples.

Positive results were observed while treating the PM slurry sample by using the 19 ppm Cu^{2+} catalyzing agent, with an effluent pH of 8.5 and a retention time of 52 min. The SO_2 to CN_{WAD} ratio of 5:1 and CN_{WAD} levels of 0.5 mg/L were achieved. The copper and zinc concentration in the detox effluent were 0.6 and 0.5 mg/L, respectively.

For the BM sample, 1.0 ppm CN_{WAD} was achieved using 19 ppm Cu^{2+}, with a retention time of 54 min, a pH of 8 and an SO_2 to CN_{WAD} ratio of 4.6:1. The copper and zinc were also reduced to 0.5 and 0.4 mg/L, respectively.

For the SC sample, 18 ppm Cu^{2+} was applied, producing an SO_2 to CN_{WAD} ratio of 4.5:1 at the slurry pH of 8.5 and a retention time of 75 min. A CN_{WAD} concentration of 0.2 mg/L in the detox effluent was achieved. The metal concentrations were also reduced 0.4 mg/L Cu and 0.5 mg/L Zn, respectively.

Table 13-30: Cyanide Detox Effluent Analysis

Collection Time (h)	BM			SC			PM		
Element	(mg/L)	24 (mg/L)	48 (mg/L)	(mg/L)	24 (mg/L)	48 (mg/L)	(mg/L)	24 (mg/L)	48 (mg/L)
CN_{wad}	0.8			<0.2	<0.1	<0.1	0.5	<0.1	<0.1
CN_{total}	1			<0.2			0.5		
SCN	77.1			94.8			85.4		
CNO	264			259			260		
Ag Silver	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Al Aluminum	<0.4	<0.4	<0.4	<0.2	<0.4	<0.4	<0.2	<0.4	<0.4
As Arsenic	<0.2	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
Au Gold	<0.4			<0.4			<0.4		
B Boron	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0
Ba Barium	0.14	0.11	0.11	0.31	0.06	0.06	0.35	0.1	0.13
Be Beryillium	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Bi Bismuth	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Ca Calcium	283	222	224	428	435	445	404	500	500
Cd Cadmium	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Co Cobalt	<0.1	0.1	0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Cr Chromium	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Cu Copper	0.51	0.24	<0.1	0.31	<0.1	<0.1	0.36	0.14	<0.1
Fe Iron	<0.2	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
K Potassium	70	70	70	78	72	73	77	71	71
Li Lithium	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Mg Magnesium	5.7	5.4	5.4	5.7	6.0	6.0	7.3	4.0	4.3
Mn Manganese	0.17	0.1	0.2	0.16	0.1	0.1	0.16	0.3	0.3
Mo Molybdenum	<0.2	0.2	0.2	0.3	0.2	0.2	0.29	<0.2	<0.2
Na Sodium	624	597	597	934	850	851	904	904	904
Ni Nickel	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Pb Lead	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
Sb Antimony	<0.4	<0.4	<0.4	0.53	<0.4	<0.4	0.5	<0.4	<0.4
Se Selenium	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
Si Silica	6.86	6.6	6.6	5.14	4.4	4.4	5.4	4.3	4.3
Sn Tin	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4	<0.4
Sr Strontium	2.1	1.9	1.9	1.1	1.1	1.1	1.15	1.6	1.6
Ti Titanium	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Tl Thalium	<0.4	<0.4	<0.4	<0.4	<0.4	<0.2	<0.4	<0.4	<0.4
U Uranium	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0
V Vanadium	<1.0	<0.2	<1.0	<1.0	<0.2	<0.2	<1.0	<0.2	<0.2
Zn Zinc	0.44	0.2	<0.2	0.42	<0.2	<0.2	0.45	0.2	<0.2

The treated slurries from the tests T-3; T-7 and T-8 were left open to air in the presence of the solids for 48 hours. Supernatant aliquots were collected at the completion of the test and after 24 and 48 h, filtered and re-analysed for the CN_{WAD} and base metals. The results in Table 13-30 suggest that some post-reaction of the treated slurry occurred within the first 24 h. All aged samples assayed below 0.1 mg/L CN_{WAD}, while the copper and zinc concentration also dropped below the detection limit within 48 h.

Caro's Acid Testwork

Three scoping tests using Caro's acid for CN destruction tests were conducted on the PM slurry sample. Prior to starting the tests, the feed solution was assayed again. The analysis of the feed (presented in Table 13-31), revealed a slightly lower CN_{WAD} concentration of 250 mg/L. Over time, the CN solution in contact with the solids continued to leach zinc, while the copper and iron concentrations remained low.

Table 13-31: Feed and Effluent Analysis During Cyanide Destruction with Caro's Acid Testing

Reagent to CN_{WAD} Molar Ratio	Feed (mg/L)	Effluent		
		1	2	3
Element		(mg/L)		
Ag Silver	0.38	<0.1	<0.1	<0.1
Al Aluminum	1.52	1.58	<0.4	<0.4
As Arsenic	<0.4	<0.4	<0.4	<0.4
B Boron	<1.0	<1.0	<1.0	<1.0
Ba Barium	0.02	0.22	0.3	0.43
Be Beryillium	<0.02	<0.02	<0.02	<0.02
Bi Bismuth	<0.5	<0.5	<0.5	<0.5
Ca Calcium	4.98	283	222	224
Cd Cadmium	0.25	<0.02	<0.02	<0.02
Co Cobalt	<0.1	<0.1	<0.1	<0.1
Cr Chromium	<0.1	<0.1	<0.1	<0.1
Cu Copper	5.92	14.37	<0.1	0.56
Fe Iron	1.11	0.64	<0.1	<0.1
K Potassium	37	1021	2020	2814
Li Lithium	<0.2	<0.2	<0.2	<0.2
Mg Magnesium	0.85	2.95	23.6	31.4
Mn Manganese	<0.02	0.03	0.9	0.03
Mo Molybdenum	<0.2	<0.2	0.2	0.2
Na Sodium	252	304	302	303
Ni Nickel	<0.1	<0.1	<0.1	<0.1
Pb Lead	<0.4	<0.4	<0.4	<0.4
Sb Antimony	0.41	<0.4	<0.4	<0.4
Se Selenium	<0.4	<0.4	<0.4	<0.4
Si Silica	6.95	4.39	4.73	4.49
Sn Tin	<0.4	<0.4	<0.4	<0.4
Sr Strontium	0.06	1.59	2.48	2.59
Ti Titanium	<0.2	<0.2	<0.2	<0.2

Reagent to CN_{WAD} Molar Ratio	Feed	Effluent		
		1	2	3
Element	(mg/L)	(mg/L)		
Tl Thalium	<0.4	<0.4	<0.4	<0.4
U Uranium	<1.0	<1.0	<1.0	<1.0
V Vanadium	<0.2	<0.2	<0.2	<0.2
Zn Zinc	62.04	<0.1	2.71	3.24

Increasing the oxidant-to-CN_{WAD} ratio was successful in reducing the copper concentration in the effluent below the detection limit, although the residual zinc levels in solution remained higher than 1 mg/L.

13.2.8 Liquid/Solid Separation Testwork (BML 2019)

Approximately 25 kg of each sample, labelled PM, BM, and SC Composites were delivered to BML Kamloops facility in December 2019, for the subsequent dewatering and tailings slurry characterization testwork. The samples had been previously subjected to the CN detoxification process at the Kemetco Research facility in Richmond.

The dewatering/tailings characterization testwork consisted of following:

- Flocculant screening tests to select the appropriate polymer for the application
- Dynamic settling tests to optimize the flocculant dosage and establish the solids settling rates needed for thickener selection and design
- Tailings slurry characterization:
  - Specific gravity determination
  - Yield stress and sheared viscosity to establish the thickener underflow slurry properties required for the downstream tailings slurry deposition.

Specific Gravity Determination

Upon sample receipt, the samples were homogenized, and the specific gravity of each sample was measured. A summary of the results is presented in Table 13-32.

Table 13-32: Tailings Samples Specific Gravity

Sample	Specific Gravity (t/m^3)
SC Composite	2.58
BM Composite	2.62
PM Composite	2.57

Flocculant Screening Tests

Scoping settling tests for all three final tailings samples were conducted in a 1 L graduated cylinder. Different flocculants and their dosages were tested, and the free settling rate and final solids bed densities were recorded.

For the purposes of the Premier static settling testwork, a total of six different flocculants were tested: anionic flocculants, MF10, MF 156, MF 1011, and MF 336; cationic flocculant, MF 380; and non-ionic flocculant, MF 351. Table 13-33 presents the results from the static settling test.

Table 13-33: Flocculant Screening Tests

Sample	Test	pH	Initial Density (% solids)	Flocculent Name	Flocculent Type	Floc Dosage (g/t)	Settling Rate (mm/sec)	Final Density (% solids)
SC Composite	F2	8.5	13.3	MF 10	Anionic	20	2.83	50.6
		8.5	13.3	MF 380	Cationic	20	0.06	15.6
		8.5	13.3	MF 351	Non-Ionic	20	0.08	16.1
		8.5	13.3	MF 156	Anionic	20	4.27	52.0
		8.5	13.3	MF 1011	Anionic	20	4.86	53.5
		8.5	13.3	MF 336	Anionic	20	4.78	53.5
BM Composite	F1	8.6	13.3	MF 10	Anionic	20	4.86	56.4
		8.6	13.3	MF 380	Cationic	20	0.08	16.1
		8.6	13.3	MF 351	Non-Ionic	20	0.55	42.4
		8.6	13.3	MF 156	Anionic	20	4.56	51.7
		8.6	13.3	MF 1011	Anionic	20	2.58	52.5
		8.6	13.3	MF 336	Anionic	20	3.68	50.3
PM Composite	F3	8.5	13.3	MF 10	Anionic	20	3.53	49.3
		8.5	13.3	MF 380	Cationic	20	0.66	36.0
		8.5	13.3	MF 351	Non-Ionic	20	4.42	49.3
		8.5	13.3	MF 156	Anionic	20	3.61	47.4
		8.5	13.3	MF 1011	Anionic	20	4.64	48.0
		8.5	13.3	MF 336	Anionic	20	4.20	48.0

The performance of the cationic polymer MF380 is considered poor relative to the other flocculants tested. As a result, this polymer was not considered for future use.

The results from the use of the non-ionic MF 351 on the PM sample were exhibiting the highest solids densities that came at the expense of water turbidity, which was considered poor. The results from the remaining samples were not conclusive, thus eliminating use of MF 351 in the subsequent testwork.

Among the remaining flocculants, M10 has produced slightly better densities for the PM and BM composite samples, when compared to flocculants MF 1011 and MF 336.

Dynamic Settling Tests

Following the static settling tests, dynamic settling tests were conducted using a bench scale thickener. Slurry and flocculant were continuously added to the thickener feed well, testing each flocculant at different dosages. The thickener overflow was collected, and the settled bed of solids was built up to a predefined height. The thickener underflow was sampled to determine the slurry density, while the thickener overflow was checked for turbidity (amount of total suspended solids [TSS]). Table 13-34 presents the dynamic settling test results.

Two different flocculants, MF 1011 and MF 336, with dosages of 20, 40, 60, and 80 g/t, respectively were tested to determine the optimum settling rates for the BM, SC, and PM composite samples of the detoxified tailings slurry. All of the tests were performed by using the same flocculant concentration while maintaining the slurry pH at 8.5. Flocculant MF 336 was used for testing the SC and PM composites, while flocculant MF1011 was used for the BM composite testwork.

It is not clear why the MF 10 flocculant was excluded, even though it clearly demonstrated acceptable performance in the previous testing stage.

For the Silver Coin Composite sample, flocculant MF 336 was selected, with a dosage of 40 g/t which yielded the highest solids underflow density and lowest TSS values, resulting in a solids loading-rate of 0.42 t/h/m^2. For the Premier sample, an MF 336 dosage of 40 g/t also yielded the highest solids underflow density and lowest TSS values, resulting in a solids loading-rate of 0.42 t/h/m^2.

For the BM Composite sample, flocculant MF 1011 was used, and settling rates were observed while applying the aforementioned flocculant dosages. The lowest TSS values and highest solids underflow densities were achieved with an MF 1011 dosage of 40 g/t resulting in the solids loading rate of 0.45 t/h/m^2.

It can be concluded that thickener underflow densities of approximately 63% solids w/w are achievable with a flocculant dosage of 20 g/t. At this dosage, the clarity of the thickener overflow is considered poor relative to a 40 g/t flocculant dosage. However, it is possible that a trade-off between addition rate and pH can be used to improve clarity in the full-scale operation.

Based on the results presented in Table 13-34, a polymer dosage of 25 g/t and solids loading rate of 0.45 t/h/m^2 is recommended for the Premier tailings thickener sizing and design. Based on the solids settling rate and plant throughput, it is estimated that an 18 m-diameter tailings thickener will be suitable for thickening duty. In the next phase of the project, it is recommended that optimization testwork be conducted to improve the thickener overflow clarity.

Yield Stress and Sheared Viscosity

The unsheared yield stress of the samples was measured at the various densities as shown in Table 13-35. The sheared viscosity of the three samples was also measured at various shear rates; Table 13-36 presents the results. Slurry viscosities of less than 3,500 centipoise (cPs) at a shear rate of approximately 5.3 s^{-1} are considered acceptable for mixing and screening applications.

Table 13-34: Dynamic Settling Test Results

Sample	Test	Loading Rate ($t/m^2/h$)	Floc Dosage (g/t)	Final U/F Density (% solids)	Overflow TSS (mg/L)
SC Composite	DST – 2A	0.42	20	63.5	318
	DST – 2B	0.42	40	65.8	98
	DST – 2C	0.42	60	64.8	102
	DST – 2D	0.42	80	63.6	130
	DST – 2E	0.25	40	63.5	111
	DST – 2F	0.59	40	60.8	203
BM Composite	DST – 1A	0.45	20	62.4	214
	DST – 1B	0.45	40	62.7	127
	DST – 1C	0.45	60	56.9	179
	DST – 1D	0.45	80	58.3	225
	DST – 1E	0.27	40	66.2	49
	DST – 1F	0.63	40	61.7	150
PM Composite	DST – 3A	0.45	20	62.6	241
	DST – 3B	0.45	40	64.3	91
	DST – 3C	0.45	60	64.2	188
	DST – 3D	0.45	80	61.4	232
	DST – 3E	0.27	40	66.8	108
	DST – 3F	0.63	40	60.3	140

Table 13-35: Yield Stress Results

Sample	Test	Initial Density (% solids)	Yield Stress (Pa)
BM Composite	V1C	56.9	84
	V1D	58.3	84
	V1E	61.7	125
	V1A	62.4	183
	V1B	62.7	124
	V1E	66.2	61
SC Composite	V2F	60.8	99
	V2A	63.5	64
	V2E	63.5	133
	V2D	63.6	184
	V2C	64.8	127
	V2B	65.8	99
PM Composite	V3F	60.3	60
	V3D	61.4	85
	V3A	62.6	59
	V3C	64.2	107
	V3B	64.3	104
	V3E	66.8	102

Table 13-36: Sheared Viscosity

Sample	Test	Density (% solids)	Shear Rate – s^{-1}							
			43.8	28.5	15.3	9.2	5.3	3.1	1.8	1.0
			Viscosity – cPs							
BM Composite U/F	V4B	55	169	229	376	589	976	1,610	2,789	4,933
	V4A	59	415	621	1,085	1,745	2,920	4,891	7,824	13,553
	V4C	66	910	1,370	2,201	3,503	6,041	10,089	16,986	28,177
SC Composite U/F	V5B	58	211	299	506	788	1,318	2,172	3,802	6,635
	V5A	61	532	761	1,292	2,076	3,433	5,388	9,162	15,931
	V5C	67	1,650	2,305	3,974	6,012	11,213	18,037	3,0294	52,311
PM Composite U/F	V6A	59	376	552	929	1,340	2,276	3,544	5,730	10,557
	V6C	61	463	683	1,169	1,783	2,965	4,891	7,958	13,910
	V6B	64	1,231	1,749	2,988	4,917	7,691	12,955	21,734	36,261

13.3 Red Mountain Metallurgical Testwork—Historical

Historical metallurgical testing was performed on Red Mountain samples by Lakefield Research (1991), Brenda Process Technology (1994), and International Metallurgical and Environmental (1997) (a derivative of Brenda Process Technology). The majority of the testwork conducted between 1991 and 1997 focused on CN leaching as the primary process for extracting gold and silver form the deposit.

In spring 2000, a metallurgical testwork program was conducted at Process Research Associates (PRA), under the direction of Dr. Morris Beattie, P.Eng. This testwork focused on producing a saleable gold-and-silver-rich flotation concentrate.

In 2015, testwork was completed by Gekko Systems (Gekko). Gravity, flotation, and comminution testwork was completed to test the amenability of the Red Mountain deposit to Gekko's Python modular plant. The results from that study are also applicable for generic flotation plants.

Results from 1991 to 2015 are documented in A.R. MacPherson (1994), Beattie (2001), Brenda (1994), Gekko (2015), International (1997), and Lakefield (1991), and have been summarized in JDS (2016).

The following section of this study will focus exclusively on the 2016–2017 testwork program—BL0084, BL0184—completed by BML in Kamloops, BC (BML 2017a, 2017b). The recovery method and process design criteria outlined in Section 17 are based primarily on the results from this program.

13.3.1 Red Mountain Metallurgical Testwork (BML 2016–2017)

The 2016–2017 testwork program was completed on variability and composite samples for Marc, AV, JW, and 141 zones. Samples for this program were taken from drill core in late 2016 and delivered to the BML facility by the end of November 2016. The metallurgical testwork program was executed from December 2016 until March 2017.

The QP for this section was not involved in the 2016/17 testwork campaign, but believes that the test samples are generally representative of the various deposits and styles of mineralization and the mineral deposit as a whole, and that there are no indications of any processing factors or deleterious elements that could have a significant effect on potential economic extraction.

The objectives of this metallurgical test program were as follows:

- Define the metallurgical response of the two process options available
- Identify the parameters affecting process response for optimization
- Assess the metallurgical availability of the deposit by assessing discrete subsamples of the various geographical zones
- Generate metallurgical estimates relating to the mine sequence of block modelling
- Generate advanced process engineering data for equipment selection
- Generate tailings samples for environmental testing.

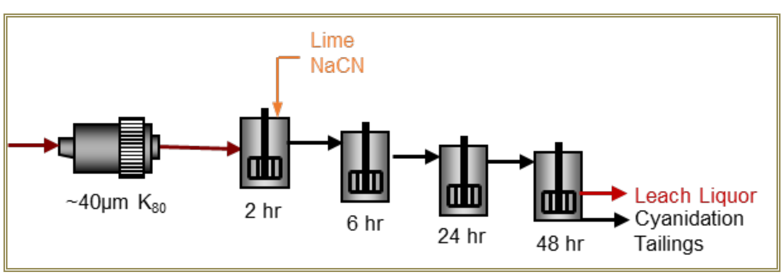
Initially, the testwork focused on the flowsheet from the JDS (2016) Preliminary Economic Assessment (PEA) report, which considered rougher flotation followed by a concentrate leaching stage. The pyrrhotite levels varied significantly in the deposit, and due to the increased reactivity and oxidation of the material, were found to dramatically affect flotation performance. As a result, a WOL process became the focus of the program. Further testwork continued, primarily on the Marc zone master composite, and was confirmed with the AV, JW, and 141 zone samples. The final flowsheet recommended a primary grind (P_{80}) of 25 µm followed by CIL recovery of gold and silver. The metallurgical test procedures and results for the 2016–2017 test program are documented in BML (2017a, 2017b).

Metallurgical Testing—Variability Samples

A total of 36 variability samples were generated from three zones: eighteen from Marc designated as MV; and nine from AV and JW each. Variability composites were selected to test a range of feed grades and geological lithologies in each zone, including various sulfur levels.

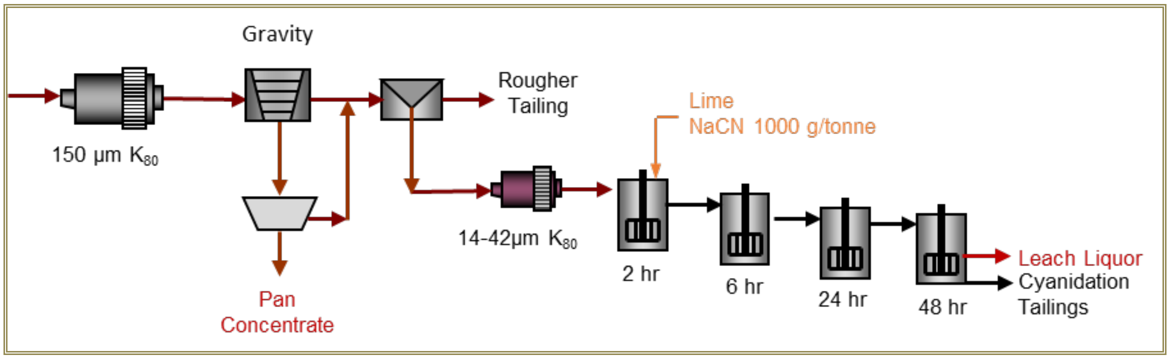
Historically, the Red Mountain project has attracted many detailed metallurgical test campaigns in the past. As a result, two process flowsheets were advanced:

- WOL – presented in Figure 13-16
- Gravity concentration followed by flotation to produce gold bearing sulfide concentrate, regrind and CN leach of the same (GFL) – presented in Figure 13-17.



Source: Base Met Lab Report BL00084, 2017

Figure 13-16: Schematic for the WOL Process



Source: Base Met Lab Report BL00084, 2017

Figure 13-17: Schematic for the GFL Test Flowsheet

The testing indicated that metallurgical performances for both circuits were comparable; however, early estimates of capital and operating costs gave a clear advantage to a GFL circuit. In consideration that there had been previous development of the GFL flowsheet, the focus of the testing program was to apply this flowsheet to the 36 variability samples of varying grade, spatial distribution, and geological lithology.

Head Assays

Table 13-37 presents the head assays showing chemical content of key elements in the variability samples.

Table 13-37: Variability Samples Head Assays

Sample	Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
MV1	5.79	42.0	7.50	6.66	0.03
MV2	9.06	46.0	9.00	9.85	0.04
MV3	3.52	7.40	6.70	4.96	0.03
MV4	3.96	41.0	6.10	5.50	0.02
MV5	12.3	68.0	11.2	13.2	0.02
MV6	5.10	24.0	9.80	10.4	0.03
MV7	3.73	5.90	8.00	5.39	0.03
MV8	5.81	7.40	7.00	4.68	0.01
MV9	8.53	42.0	7.50	5.38	0.11
MV10	7.14	18.0	7.70	7.09	0.19
MV11	0.87	0.90	7.40	5.37	0.05
MV12	22.4	12.0	11.20	11.7	0.04
MV13	5.33	50.0	8.50	9.83	0.02
MV14	3.94	2.40	9.30	5.98	0.01
MV15	14.7	16.0	10.80	12.2	0.02
MV16	4.84	2.70	7.70	4.81	0.02
MV17	16.2	71.0	13.0	17.4	0.02
MV18	2.35	38.0	8.40	8.94	0.01
JW1	0.86	0.70	6.40	3.80	0.02
JW2	5.70	26.0	10.4	13.0	0.02

Sample	Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
JW3	6.77	130.0	9.20	10.2	0.02
JW4	7.31	11.0	8.90	10.6	0.02
JW5	5.35	3.80	15.1	11.4	0.04
JW6	2.82	15.0	12.2	14.4	0.02
JW7	2.75	1.10	9.80	6.33	0.02
JW8	6.10	6.20	9.20	6.00	0.01
JW9	1.83	1.50	7.20	4.37	0.01
AV1	6.96	22.0	11.9	14.1	0.03
AV2	3.90	11.0	9.90	11.7	0.02
AV3	4.76	15.0	9.70	10.9	0.05
AV4	6.41	1.40	7.40	4.65	0.01
AV5	0.38	1.70	7.60	4.13	0.02
AV6	5.15	28.0	10.0	11.5	0.02
AV7	31.0	49.0	14.1	18.8	0.02
AV8	29.5	42.0	11.20	11.9	0.01
AV9	0.21	3.00	5.80	3.42	0.01

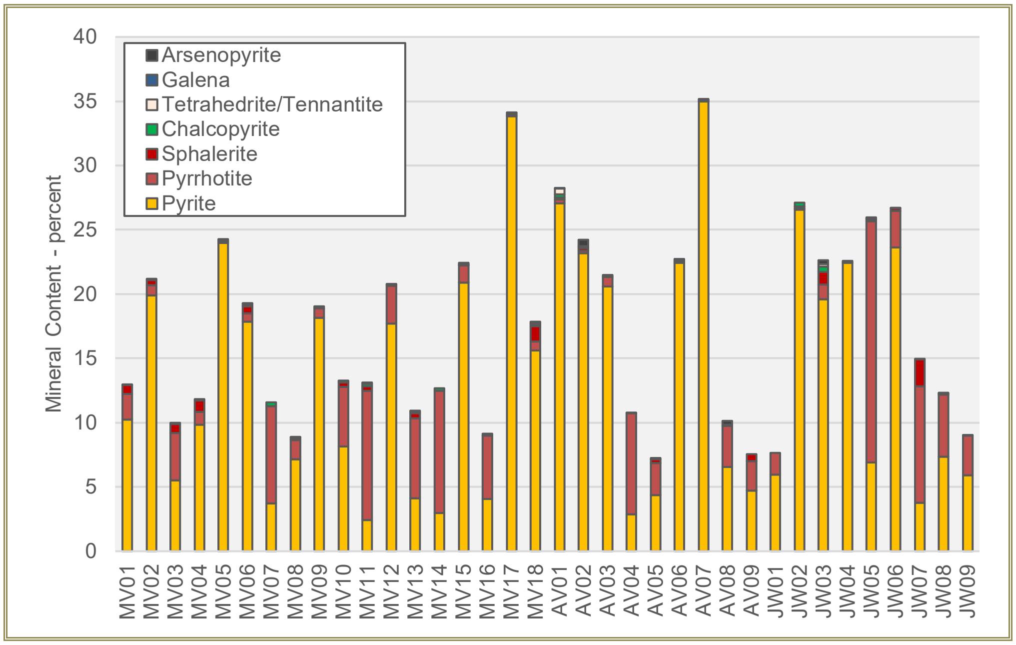
The gold and silver content were highly variable throughout the samples, ranging from 0.2 to 31 g/t Au and approximately 1 to 130 g/t Ag. The wide range of values allows for metallurgical performance analysis on both sides of the anticipated mine feed grades spectrum.

The sulfur and iron values also indicate a high degree of variability, with sulfur grades ranging between 3.4% and 18.8% and iron grades between 5.5% and 15.1%. On occasion, higher sulfur grades can dictate the process selection for the precious metals extraction, as presence of sulfides can result in higher CN and oxygen consumption as well as an association of additional metals in solution which can ultimately increase the detoxification complexity and associated costs. In addition, if the GFL flowsheet is selected as a processing option, the mass recovery of the flotation concentrate will vary proportionally with the sulfur levels, influencing concentrate leach circuit design (which is based on a concentrate mass).

The total TOC was detected in minor amounts (between 0.01% and 0.19%), which generally suggests that a flotation or CN leaching process will be unaffected.

Mineralogy

The mineral composition of each variability sample was established using QEMSCAN—Bulk Mineral Analysis (BMA) determinations. The main non-sulfide minerals include muscovite, quartz, feldspars, and chlorite. The sulfide minerals, which are of particular interest, are shown in detail in Figure 13-18.



Source: Base Met Lab Report BL00084, 2017

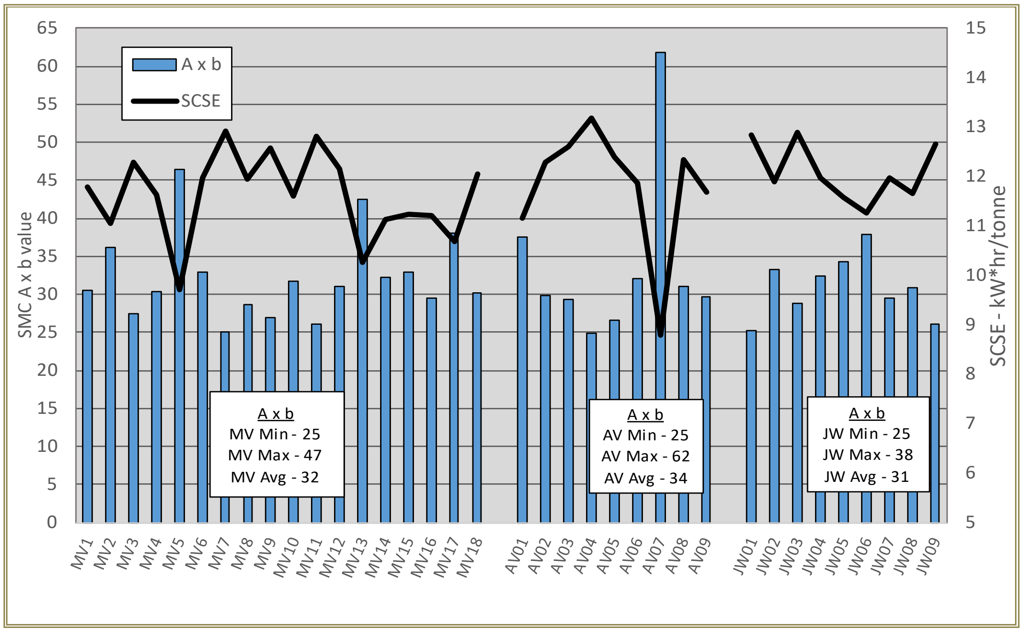
Figure 13-18: Sulfide Mineral Content

In the samples examined, pyrite and pyrrhotite represent the majority of sulfide minerals with a peak content of 35%. In most cases, pyrrhotite is susceptible to oxidation and, as a highly reactive mineral, will affect both the process flowsheet selection and associated metal recoveries.

Sphalerite was present in the majority of variability samples; however, levels of up to approximately 2% do not suggest that an economical extraction can be successful. It is noted that other sulfides such as chalcopyrite, arsenopyrite, galena, and Tetrahedrite/Tennantite were present in amounts of less than 0.5% in some samples. At these levels, the presence of these minerals can suggest additional increases in sodium cyanide consumption as they are all partially or completely leachable.

Comminution Testwork

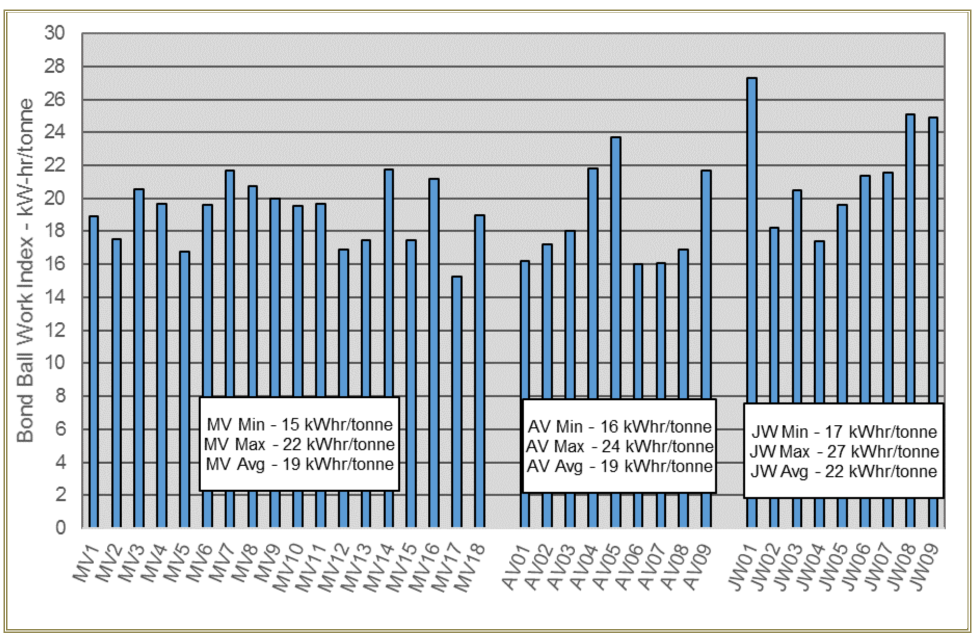
All 36 variability samples were subjected to SMC and BWi testwork. The results are presented in Figure 13-19 and Figure 13-20.



Source: Base Met Lab Report BL00084, 2017

Figure 13-19: SMC Test Results

The Red Mountain SMC test results sit between the 80th and 95th percentiles of hardness in the JK database, suggesting that the Red Mountain zones are considered hard from a coarse particle comminution perspective.



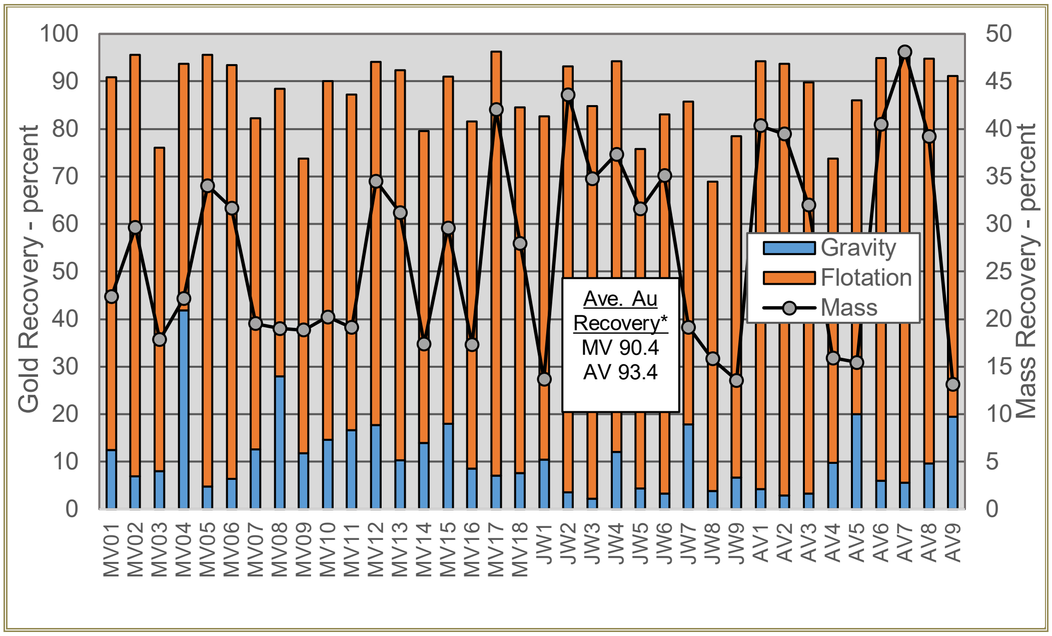
Source: Base Met Lab Report BL00084, 2017

Figure 13-20: Bond Work index (BWi) Test Results

Based on the average BWi test results for the three deposits, 19.0, 19.0, and 22.2 kWh/t, respectively, most of the samples would be considered to be hard or very hard from a ball milling perspective.

Gravity and Rougher Flotation Testwork

Gravity concentration tests were followed by bulk sulfide flotation tests and were conducted on all variability samples at a primary grind particle size (P_{80}) of approximately 150 µm. The tests were performed at a natural pH using potassium amyl xanthate (PAX) as a primary sulfide mineral collector. A summary of the combined gravity–flotation test results is presented in Figure 13-21.



Source: Base Met Lab Report BL00084, 2017

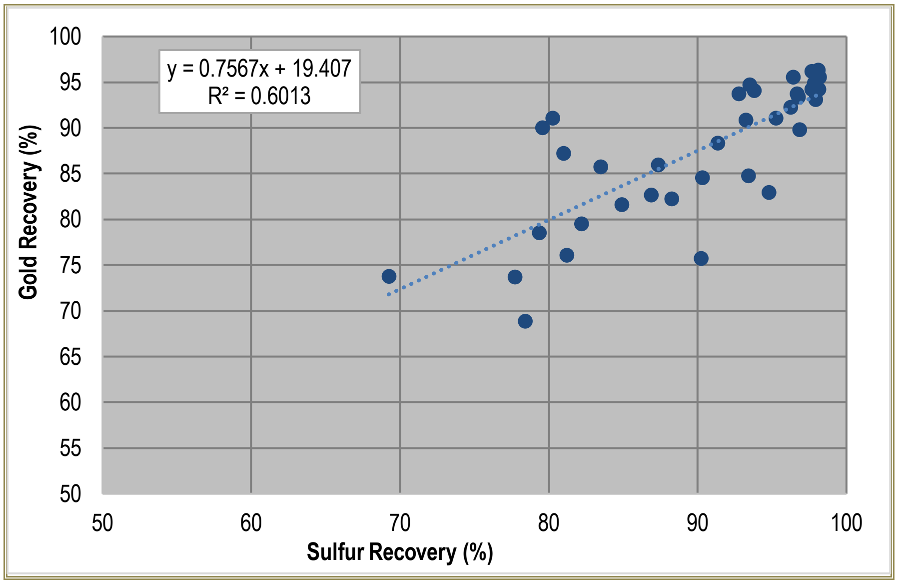
Figure 13-21: Gravity–Flotation Test Results

The mass recovery of a gravity concentrate was quite poor for most of the samples, with only two samples exhibiting recoveries over 20%; the average recovery for all the samples was 11%. Based on these outcomes it can be concluded that minor amounts of GRG are present in the RM deposits, implying that the use of the gravity concentration circuit is unlikely to benefit the process metallurgically.

Overall, tests performed using the GFL flowsheet recovered between 69% and 96% of gold, 41% to 98% of silver, and 69% to 98% of the sulfur.

It is noted that due to a high sulfide content in the samples, the mass recovery of the rougher concentrate was quite high, peaking at 48% with an average 27%. It is likely that this reduces the obvious benefits of a flotation pre-concentration stage prior to the rougher concentrate regrind step, and the subsequent leaching as a significant portion of the feed mass will still require fine grinding, ultimately imposing an energy requirement penalty for the fine grinding circuit. The flotation tests were not optimized using finer P_{80}'s than the initial 150 µm particle grind size, so it is not known what the effect of a finer grind size on gold flotation recovery might have been.

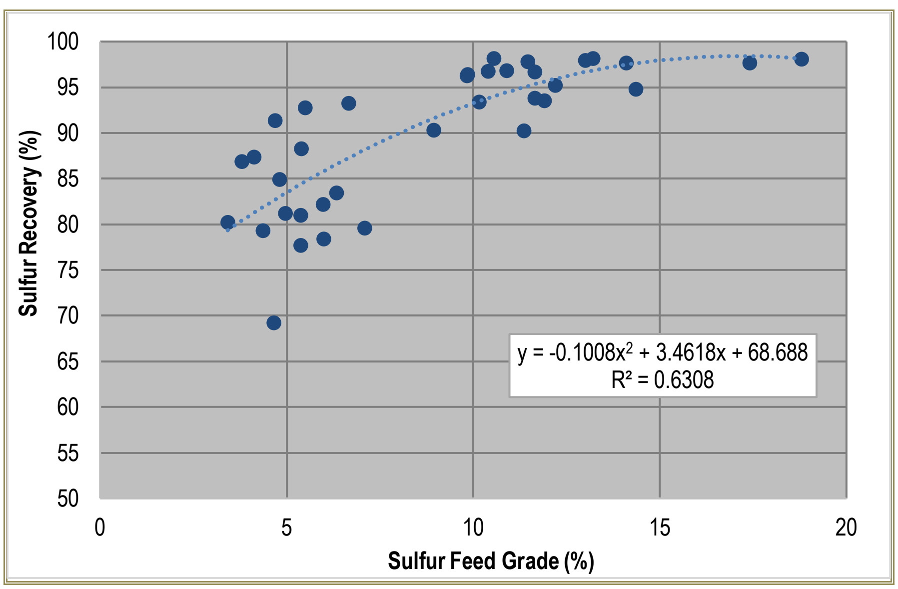
In Figure 13-22, it can be observed that the gold recovery trended relatively well with the sulfur recovery for some samples, indicating that the gold present is associated with, or had similar flotation performances to the sulfide minerals. However, it is also apparent that the majority of the samples did not achieve a high sulfur recovery.



Source: Base Met Lab Report BL00084, 2017

Figure 13-22: Gold vs. Sulfur Recovery

Upon further investigation, it is apparent that the sulfur feed grade is associated with an effect on the sulfur recovery in a GFL process. Figure 13-23 demonstrates the relationship between the sulfur feed grade and sulfur recovery.



Source: Base Met Lab Report BL00084, 2017

Figure 13-23: Sulfur Feed Grade vs. Sulfur Recovery

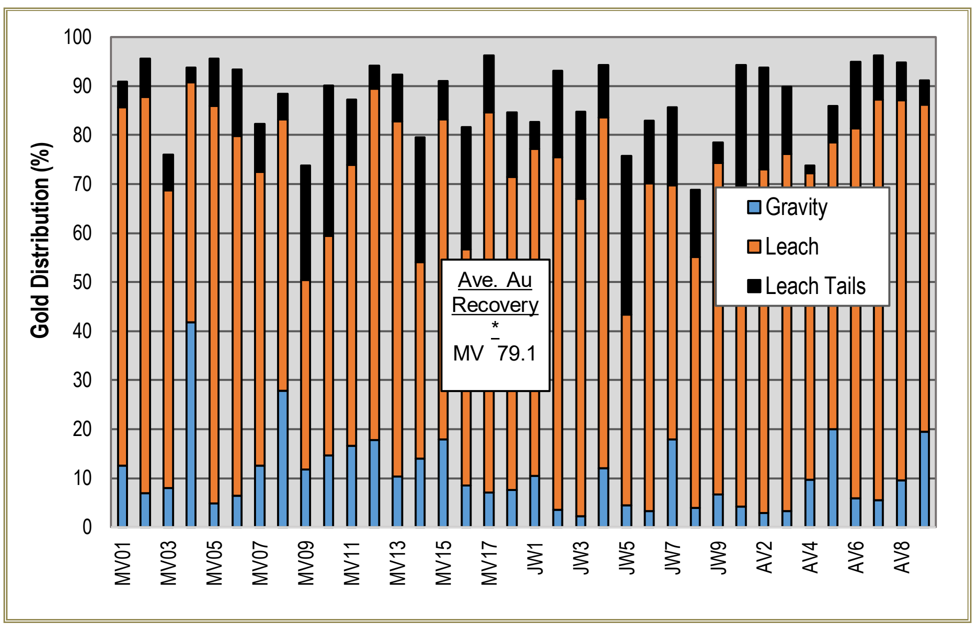
From the presented data it can be concluded that for samples with sulfur grades in excess of 10%, the sulfur recovery was predictably over 90%. For samples with the sulfur grade of less than 10%, the associated sulfur recoveries were less predictable and often lower than 85%.

This discovery of the poor gold flotation performance was unexpected and could have a negative influence on the metallurgical performance of the GFL process flowsheet. The cause for the poor metallurgical performance was not attributed to a particular geographical zone in the deposit, or to a geological lithology. The real cause for the poor metallurgical performance could not be established, as optimization tests with the variability samples were not conducted.

Rougher Concentrate Cyanidation Tests

The rougher concentrates were subsequently reground and leached with CN to determine the resultant gold and silver extractions. The average rougher concentrate regrind size was 27 µm (P_{80}), and the tests were conducted at a pH of 11, with a 1,000 ppm NaCN concentration. There was no pre-oxidation stage; however, the leached samples were sparged with oxygen at the sampling intervals.

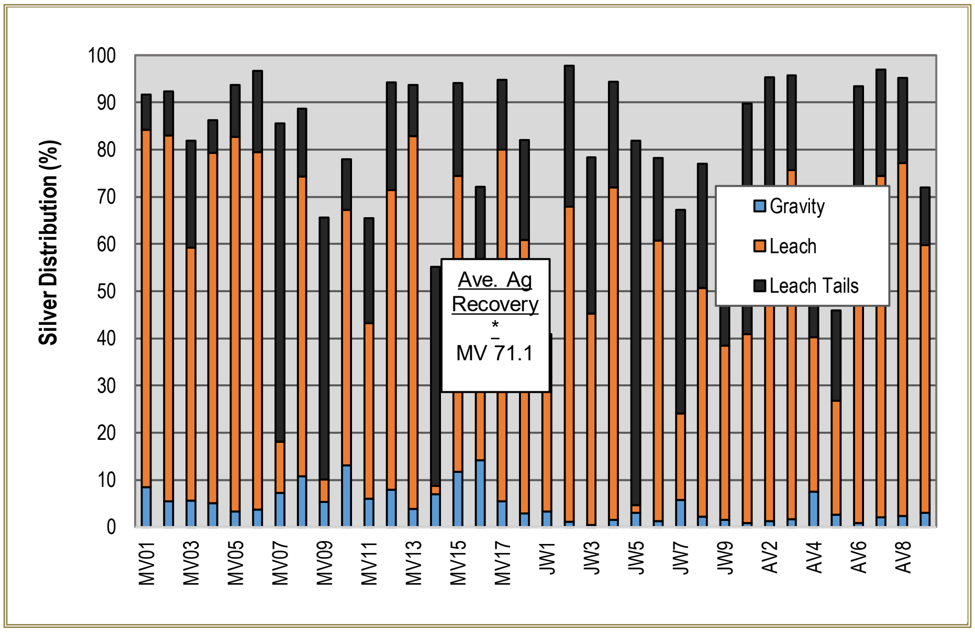
Summaries of gold and silver distributions, including in the CN leach residues, are shown in Figure 13-24 and Figure 13-25.



Source: Base Met Lab Report BL00084, 2017

Figure 13-24: Rougher Concentrate Leach—Gold Results

Within Figure 13-24 and Figure 13-25, the cumulative sums of the blue and orange bars represent the final recovery of the combined GFL process flowsheet. This final recovery is presented as an average for each zone in the inset table. The black portion of the bar indicates the metal lost to the leach residue.



Source: Base Met Lab Report BL00084, 2017

Figure 13-25: Rougher Concentrate Leach—Silver Results

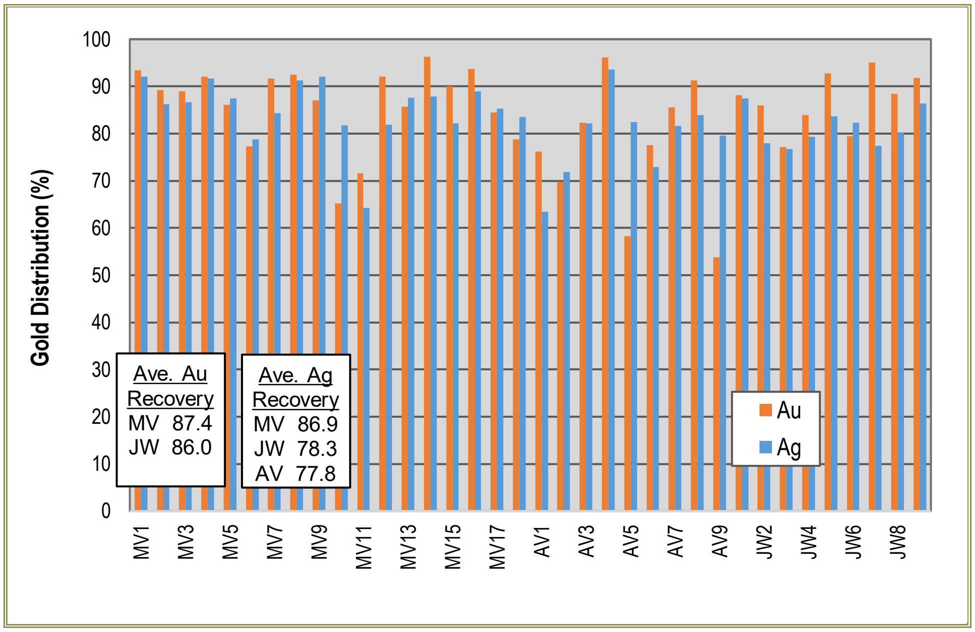
Many of the samples have exhibited significant gold and silver losses to the leach residue. The overall recoveries, which included gravity concentration and leach extractions, ranged from 43% to 91% for gold and 5% to 84% for silver.

The overall performance of the GFL process flowsheet was highly variable, which was caused by several factors:

- Low initial flotation recovery
- Poor leach performance due to high oxygen demand of the sulfides, resulting in insufficient oxygen in the leach (samples that have exhibited this behaviour often had high CN consumption, which was caused by high pyrrhotite levels)
- Variability in rougher concentrate regrind size distribution, as coarser samples often had poorer performance
- Two samples that had higher TOC levels have also exhibited poor performance
- Presence of tellurium, arsenic, copper, and zinc were investigated as a reason for the poor performance, but no conclusive trends were observed
- No conclusive trend was developed between geological zone, gold and silver feed grade, and geological lithology.

Carbon-in-Leach Whole Ore Leach Tests

Whole ore CIL tests were conducted on all variability samples, at a targeted primary particle grind size of 40 µm, while the pH was maintained at 11 and the NaCN concentration at 1,000 ppm. Lead nitrate and carbon additions were 250 g/t and 50 g/L, respectively. Oxygen sparging was performed at 2, 6, 24, and 48 h time intervals, with an addition of 250 g/t. Gold and silver recoveries are presented in Figure 13-26 with the average figures displayed in the inserts.



Source: Base Met Lab Report BL00084, 2017

Figure 13-26: Whole Ore Leach CIL Test Results

With consideration given to the fact that tests were done on whole ore, rather than an upgraded rougher concentrate, the ratio of sulfides and gold to both CN and oxygen is significantly lower. As a result of this observation, the dissolved oxygen levels and high CN consumptions were generally observed not to represent issues for the WOL process when compared to the GFL outcomes. The tests were conducted as CIL to mitigate the effects of TOC, which is believed to be responsible for the poor leaching performance of the Marc composites 10 and 11.

Composite Samples Testwork

Two sets of master composites were generated for testing. One set of composite samples was allocated for the metallurgical testwork, while the other was reserved for comminution testwork. This set of samples was dedicated for the crushing and Ai testing from each of the deposit zones. The nature of the testwork demanded a special sample size requirement and were composed from smaller intervals of whole, unsplit drill core.

Comminution

Table 13-38 presents the master composite comminution test results.

Table 13-38: Master Composite Comminution Testwork Results

Sample ID	Bond Crushing Work Index (CWi) (kWh/t)	Bond Ai (g)
Marc Composite	11.7	0.24
AV Composite	12.1	0.30
JW Composite	9.5	0.29

Based on the above CWi results it can be concluded that the JW sample is considered soft, while the Marc and AV samples are considered of averages hardness (with respect to rock breakage in a crusher). With regards to abrasiveness, the Ai values ranged from 0.24 to 0.30, which suggests the samples tested can be considered abrasive.

Master Composite Head Assays

The master composites for the metallurgical testwork were generated using the variability composite samples from the Marc; AV and JW ore zones. The composites were generated to represent the average feed grades for the respective deposits.

Table 13-39 presents the head assays, as well as average recalculated average head assays, from the AV and Marc composite tests.

Table 13-39: Master Composite Head Assays

Analyte Symbol (Unit)	Au (g/t)	Ag (g/t)	S (%)	Fe (%)	TOC (%)	Cu (g/t)	Pb (g/t)	Zn (g/t)	As (g/t)	Sb (g/t)	Te (g/t)
Marc Master Hd1	9.1	33	8.98	8.5	0.05	355	165	2,290	471	75	52
Marc Master Hd2	11.0	32	8.77	8.7	-	341	181	2,530	502	786	53
Average	10.1	33	8.88	8.6	0.05	348	173	2,410	487	76	53
Recalc. average	8.13	34	8.95	9.1	-	-	-	-	-	-	-
AV Master Hd1	6.45	16	10.9	9.2	0.01	657	58	1,130	481	13	35
AV Master Hd1	6.16	16	10.9	9.4	-	737	57	1,210	497	352	30
Average	6.31	16	10.9	9.3	0.01	697	58	1,170	489	333	33
Recalc. average	6.09	17	10.7	10.4	-	-	-	-	-	-	-
JW Master Hd1	5.63	13	12.3	10.2	0.02	467	36	606	302	94	29
JW Master Hd1	5.55	16	12.2	10.2	-	479	39	518	327	102	32
Average	5.59	14	12.3	10.2	0.02	473	38	562	317	98	31

The gold values are considered consistent, with the exception of the Marc master composite which suggests a degree of variability. The TOC was low but measurable, particularly for the Marc composite, where it was 0.05% due to inclusion of the MV10 and MV11 variability samples when the master sample composite was generated.

The JW zone samples were only able to provide a limited mass of material for generating the master composite, and as such the overall composite was relatively low in weight when compared to the other two composites. As a result, the subsequent optimization tests focused on the Marc and AV master composites. The final test conditions derived from the Marc and AV zone tests were then applied to the JW master composite.

Gold Mineralogical Assessment

All three master composites were subjected to gold trace mineral examination to investigate gold occurrences in the samples. A summary of the gold deportment data is presented in Table 13-40, Table 13-41, and Table 13-42, respectively.

Table 13-40: Gold Deportment by Gold Mineral Species and Mass Percentage

Gold Minerals Mineral Formula	Native Gold (Au)	Electrum (Au/Ag)	Petzite (Ag_3AuTe_2)	Sylvanite ($(Au,Ag)_2Te_4$)	Hessite ($(Ag,Au)_2Te$)	Aurostibite ($Au(TeSb)_2$)	Calaverite ($AuTe_2$)
BL84 Marc master	94.8	0.0	2.7	1.2	1.2	0.0	0.0
BL 84 AV Master	81.8	0.2	2.7	6.1	1.9	7.4	0.0
BL 84 JW Master	91.6	0.0	3.2	0.8	1.8	0.0	2.6

Although the presence of some tellurides was observed, most of the gold weight by mass was observed as native gold.

Table 13-41: Gold Distribution by Gold Grain Size and Mass Percentage

Sample	Particle Size Range (µm)				
	<= 2	2 – 5	5 – 10	10 – 20	> 20
BL84 Marc master	1	3	96	0	0
BL 84 AV Master	70	30	0	0	0
BL 84 JW Master	9	91	0	0	0

The majority of the gold particles were considered small, with typical sizes less than 10 µm in diameter.

Table 13-42: Gold Locking Characteristics by Mineral Mass Percentage

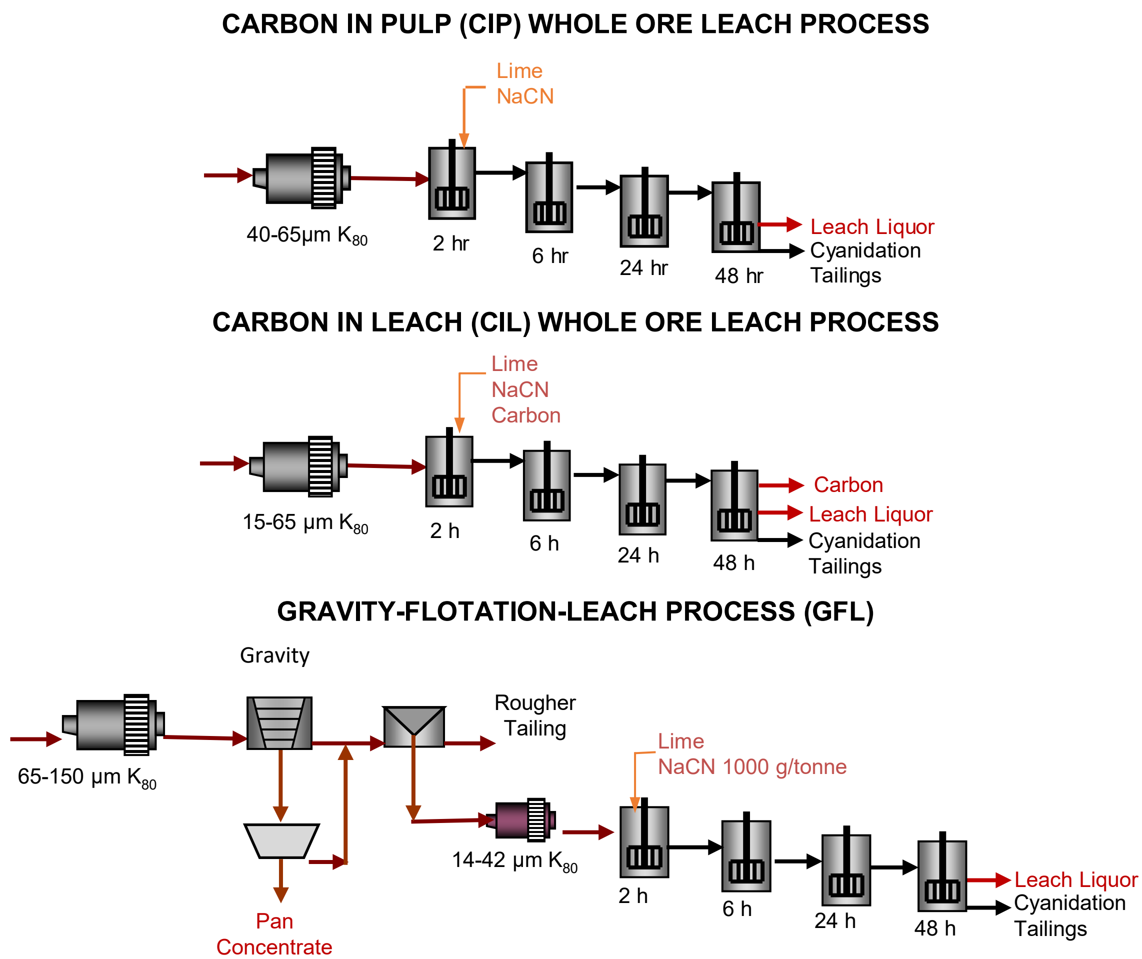
Sample	Liberated (Au)	Au (Adhesions)	Au Inclusions Associated with:				
			Hessite	Galena	Pyrite	Gangue	Multiphase
BL84 Marc M.	2.6	96.8	0.0	0.0	0.5	0.0	0.1
BL 84 AV M.	7.4	21.7	0.0	0.0	10.0	0.5	60.5
BL 84 JW M.	0.6	93.0	0.0	0.0	4.9	0.1	1.3

Much of the gold was unliberated, locked with gangue minerals, most of which is pyrite. The gold deportment data, such a fine gold particle size, and the observed degree of liberation were an early indication that:

- Very fine grind (tertiary grinding) will be needed to liberate gold particles and achieve satisfactory metal extraction
- The use of gravity concentration on the Red Mountain ores are unlikely to improve overall gold recovery.

Composite Samples Metallurgical Testwork

Both the direct CN leaching process and the GFL flowsheet option were evaluated on the master composite samples. Due to the observed poor performance of some of the variability samples, a new focus was placed on the development of a WOL CN leach process. Process flowsheets that were tested during the metallurgical testwork campaign are depicted in Figure 13-27.



Source: Base Met Lab Report BL00084, 2017

Figure 13-27: Process Flowsheets Evaluated on Master Composite Samples

CIL and CIP Metallurgical Testing

A series of leach tests were conducted on the whole ore feed of the AV and Marc master composites. Table 13-43 presents a summary of the text conditions.

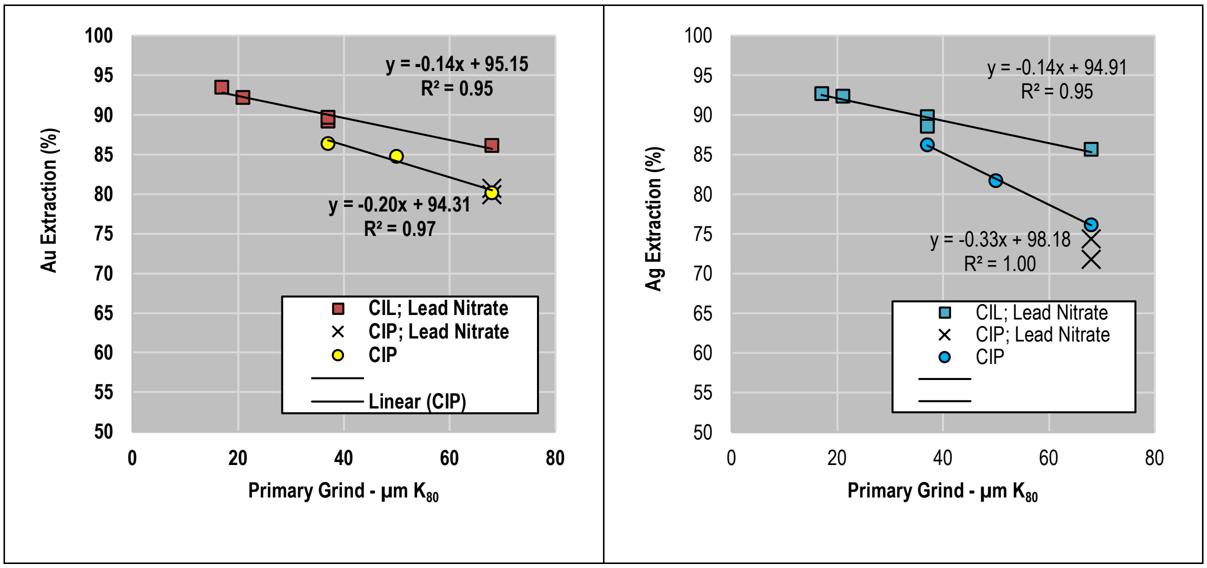
Table 13-43: Summary of Master Composite Leach Tests

Test Number	Grind (P_{80})	Gravity Stage	pH	Pb $(No_3)_2$ (g/t)	Carbon (g/L)	Oxygen Sparged	Pre-Ox. (24 h)	Extraction Au	Ag %
Marc Master Comp.									
43	68	No	11.0	0	0	Yes	none	80	76
44	68	Yes	11.0	0	0	Yes	none	82	84
45	50	No	11.0	0	0	Yes	none	85	82
45	37	No	11.0	0	0	Yes	none	86	86
47	68	No	11.0	0	0	Yes	none	81	79
48	68	No	11.0	0	0	Yes	none	81	77
49	68	No	11.0	0	0	Yes	none	81	81
50	68	No	10.0	0	0	Yes	none	81	78
51	68	No	11.5	0	0	Yes	none	81	79
52	68	No	12.0	0	0	Yes	none	79	79
53	68	No	11.0	250	0	Yes	none	81	74
54	68	No	11.0	500	0	Yes	none	80	72
55	68	No	11.0	0	0	No	Air	82	75
56	68	No	11.0	0	0	Yes	O_2	82	76
57	37	No	11.0	250	50	Yes	none	89	90
58	68	No	11.0	250	50	Yes	none	86	86
117	37	No	11.0	250	50	Yes	none	90	89
118	21	No	11.0	250	50	Yes	none	92	92
119	17	No	11.0	250	50	Yes	none	93	93
AV Master Comp.									
43	68	No	11.0	0	0	Yes	none	80	76
44	68	Yes	11.0	0	0	Yes	none	82	84
45	50	No	11.0	0	0	Yes	none	85	82
46	37	No	11.0	0	0	Yes	none	86	86
47	68	No	11.0	0	0	Yes	none	81	79
48	68	No	11.0	0	0	Yes	none	81	77
49	68	No	11.0	0	0	Yes	none	81	81
50	68	No	10.0	0	0	Yes	none	81	78
51	68	No	11.5	0	0	Yes	none	81	79
52	68	No	12.0	0	0	Yes	none	79	79
53	68	No	11.0	250	0	Yes	none	81	74
54	68	No	11.0	500	0	Yes	none	80	72
55	68	No	11.0	0	0	No	Air	82	75
56	68	No	11.0	0	0	Yes	O_2	82	76
57	37	No	11.0	250	50	Yes	none	89	90
58	68	No	11.0	250	50	Yes	none	86	86
117	37	No	11.0	250	50	Yes	none	90	89
118	21	No	11.0	250	50	Yes	none	92	92
119	17	No	11.0	250	50	Yes	none	93	93

The effect of certain parameters on the leaching performance is explained further in the text below.

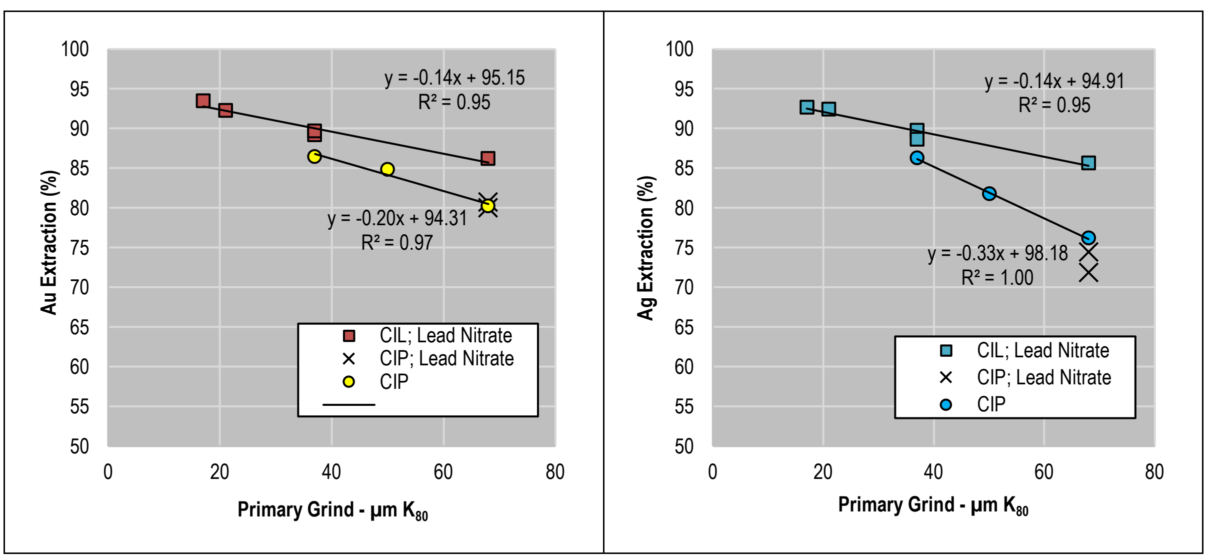
Effect of Primary Grind Size and Carbon-In-Leach

The primary grind size and CIL process have demonstrated a dominant influence on gold and silver extraction. Tests were conducted on two primary grind size targets for both composite samples. The first test conducted consisted of a CIP configuration, without carbon or lead nitrate additions, while the second one was performed as a CIL test with lead nitrate. The second test also aimed to investigate a finer grind at P_{80} of approximately 16 µm. The resultant extractions follow a linear trend in general, which is graphically depicted in Figure 13-28 and Figure 13-29, respectively.



Source: Base Met Lab Report BL00084, 2017

Figure 13-28: Effect of Primary Grind Size and CIL—AV Master Composite



Source: Base Met Lab Report BL00084, 2017

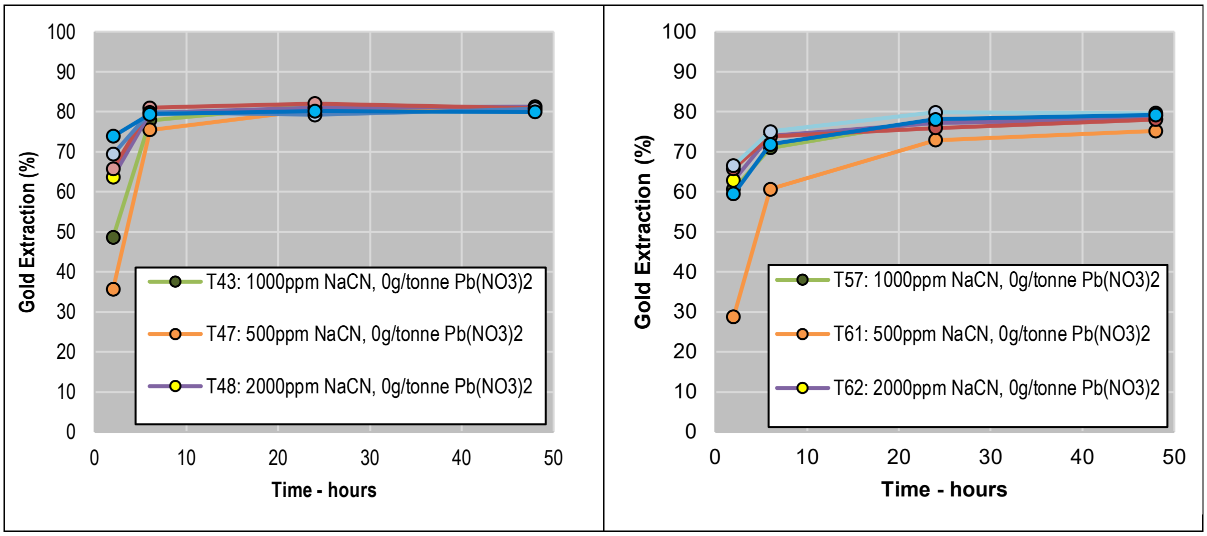
Figure 13-29: Effect of Primary Grind Size and CIL—Marc Master Composite

The highest extractions were achieved at the finest primary grind size, with 93% of gold and silver extractions for the Marc master composite at the P_{80} of 17 µm and 89% and 85% extractions for gold and silver, respectively, for the AV master composite, at the P_{80} of 16 µm.

The effect of lead nitrate on the overall extraction was considered negligible, while the CIL tests outperformed the CIP tests at the same primary grind size, which was confirmation of the trend that was observed during the variability testwork. This observation was particularly visible for the AV master composite test which yielded a 3% higher extraction at the grind size P_{80} of 37 µm.

Effect of Lead Nitrate and Cyanide Concentration

The effect of lead nitrate addition and CN concentration on both composite samples is presented in Figure 13-30.



Source: Base Met Lab Report BL00084, 2017

Figure 13-30: Effect of Lead Nitrate Addition and Cyanide Concentration

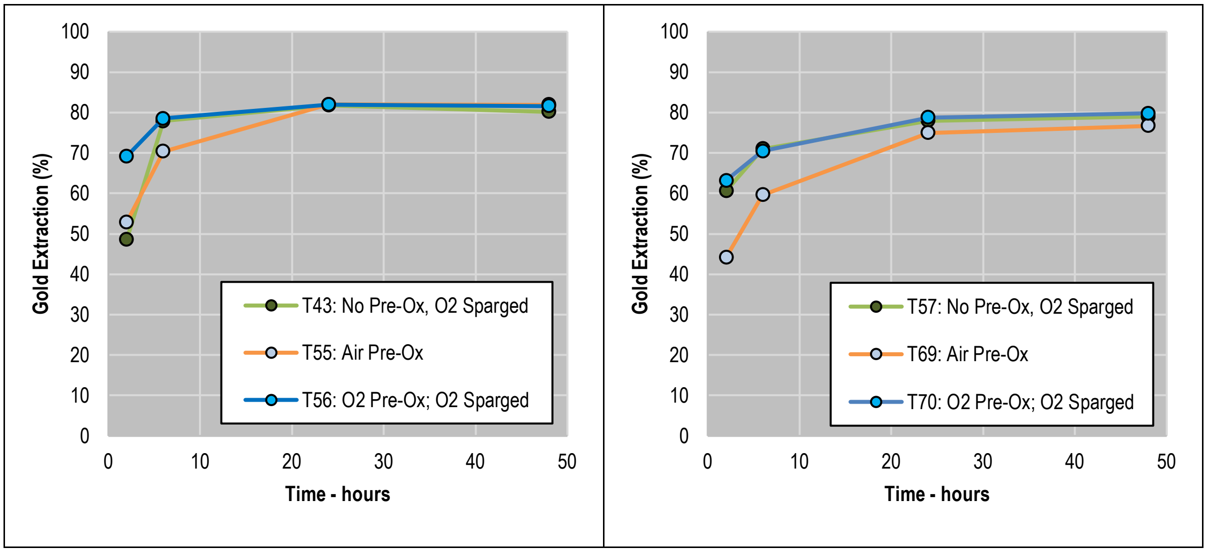
The final precious metals extractions were fairly constant for both samples; however, a slight increase in extractions is evident at the start of the Marc master composite tests, potentially demonstrating an association with the lead nitrate concentration. The gold extraction rate was also improved with the increase of CN concentration to 2,000 ppm, while an additional increase to 3,000 ppm did not extend the relationship. Conversely, a decrease in the NaCN concentration to 500 ppm demonstrated a reduction in overall leach kinetics.

Effect of pH

The adjustment of pH did not produce a noticeable effect on the Marc and AV master composite leach performance. It is commonly observed that a pH adjustment can assist with the dissolution of the resilient gold alloys such as tellurium and antimony containing gold minerals. If extraction of some gold minerals was poor, the overall effect was not noticeable for the leach conditions that were used for the constructed master composites.

Effect of Pre-oxygenation

As was noticeable with the variability testing, some of the samples and their precious metals extractions demonstrated susceptibility to oxygen consumption. Two tests were conducted on both composites with 24 hours of pre-oxygenation; one with oxygen and one with air. Neither pre-oxygenation mode greatly affected leaching performance, as shown in Figure 13-31.



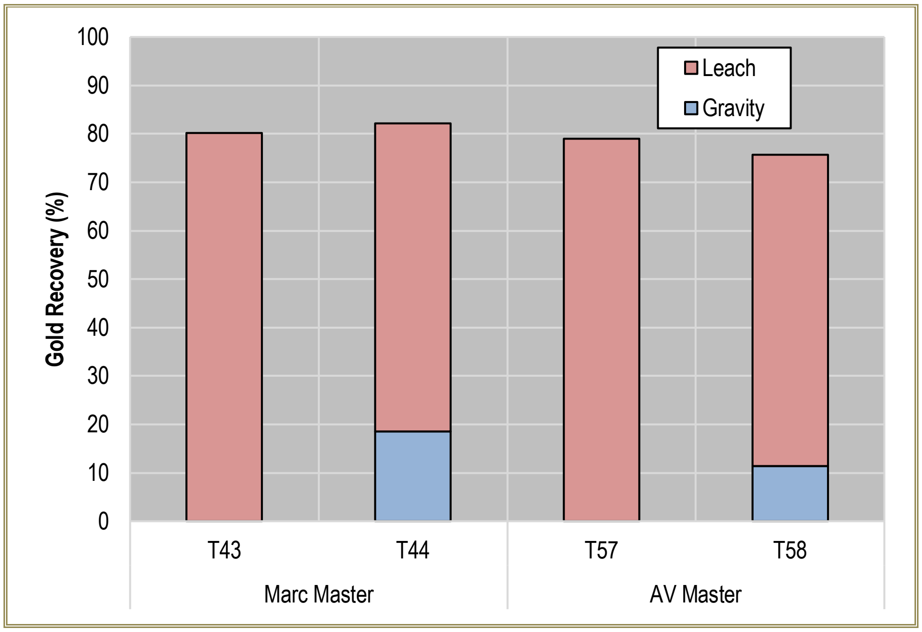
Source: Base Met Lab Report BL00084, 2017

Figure 13-31: Effect of Pre-Oxygenation on Leaching Performance

Following a 2-h observation, a small increase in initial gold extraction (52% vs. 44%) was recorded for the Marc master composite sample; however, all subsequent readings for the test purged with oxygen appeared to align.

Gravity Concentration

Gravity concentration tests prior to leaching were conducted on both master composite samples. Presented in Figure 13-32, the gold recovery from gravity concentration is not considered sufficient to warrant additional gravity testwork efforts.



Source: Base Met Lab Report BL00084, 2017

Figure 13-32: Gravity Concentration on Master Composite Samples

Bulk Sulfide Flotation

Flotation tests using a bulk sulfide flotation flowsheet were conducted on the Marc and AV master composites to establish the metallurgical response. Previous testing on the variability samples indicated that gold losses in the flotation stage outweighed the additional process costs of the WOL. Table 13-44 presents the results from the master composite bulk flotation tests.

Table 13-44: Master Composite Bulk Sulfide Flotation Tests

Test	Composite	Primary Grind (P_{80} µm)	PAX (g/t)	$CuSO_4$ (g/t)	Recovery (%) Mass	Au	Ag	S
71	Marc Master	150	35	0	24	91	87	91
72	Marc Master	68	50	0	22	92	85	89
73	Marc Master	150	35	150	27	93	87	95
74	AV Master	150	35	0	30	92	91	93
75	AV Master	70	50	0	27	92	90	93
76	AV Master	50	35	150	30	91	92	94

On average, the gold recovery of the rougher flotation concentrate was 92% for both samples. A finer primary grind and addition of $CuSO_4$ have only marginally improved recovery for the Marc master composite; however, it was unchanged for the AV master composite sample.

JDS concluded that subsequent regrinding and leaching of the rougher concentrate would likely incur further gold losses, which would confirm conclusions from the flotation variability testing that recoveries from the WOL would likely be higher than of those achieved by implementation of the GFL process. Also, the fact that RM ores will be stockpiled and processed on a campaign basis, could increase the extent of the pyrrhotite oxidation, which could be detrimental to bulk sulfide flotation.

13.3.2 Red Mountain Metallurgical Testwork (BML, 2019–2020)

This testwork, conducted on behalf of Ascot, in 2019–2020 was confirmatory in nature, and its objective was to fill the gaps from the previous testwork campaigns, establish fine grinding parameters, assess gold and silver metallurgical recoveries, and efficiency of the suggested CN detoxification methods, as well as generate liquid/solid separation data needed for the disposal of tailings.

The samples used for this testwork were initially collected in April 2017 for the metallurgical testwork campaign conducted by BML (BL0084) (BML 2017a) and have been stored according to common industry standards. As per Ascot's Resources' instructions, samples were combined to generate JW, MV, and AV composite samples.

The testwork consisted of:

- Isa Mill signature plots for tertiary grinding testwork, performed at ALS's Kamloops facility
- Six CIL tests
- Settling tests and tailings slurry characterizations
- Cyanide detoxification of simulated tailings slurry samples.

Head Assays

The composite samples were generated to reflect gold and sulfide grades representative of the estimated resource grade of the represented zones. Duplicate, representative head cuts were removed from the three composites and assayed for elements of interest. Table 13-45 presents the results.

Table 13-45: Head Assays

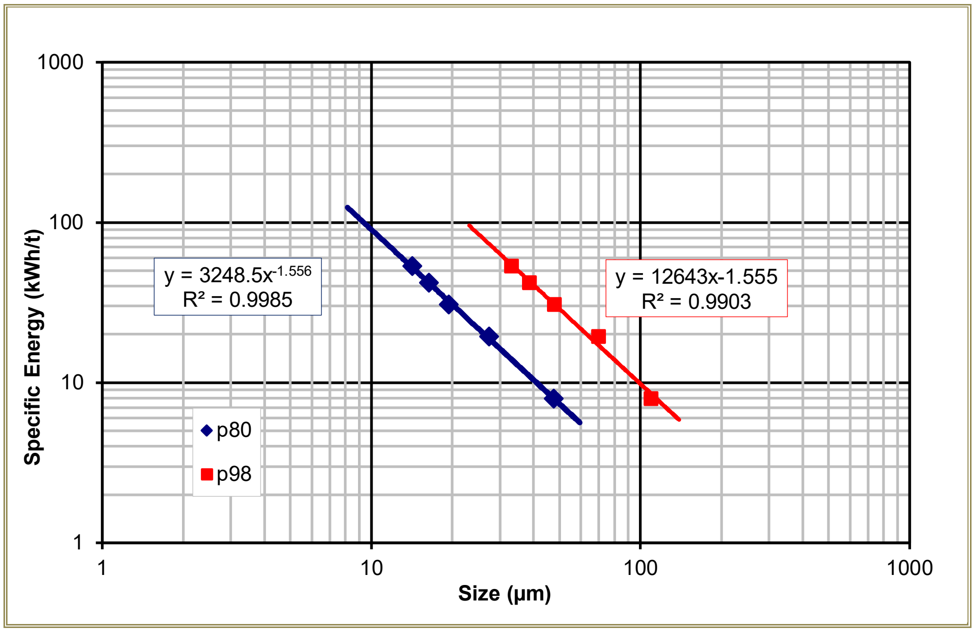
Sample ID	Au (g/t)	Ag (g/t)	S (%)	Cu (%)
AV Composite 3	12.6	20.0	12.2	0.05
JW Composite 3	7.29	39.0	9.76	0.06
MV Composite 3	10.3	31.0	9.54	0.03

The gold content in the composite samples ranged from 7.3 to 12.6 g/t and the silver content between 20 and 39 g/t. There was some presence of sulfide minerals, ranging from 9.5% to 12.2%, while copper was present in trace amounts.

Isa Mill Grinding Tests

All three composite samples were submitted to ALS's Laboratory in Kamloops, BC, which is certified by Glencore to conduct fine grinding testing and complete the development of the IsaMill signature plot.

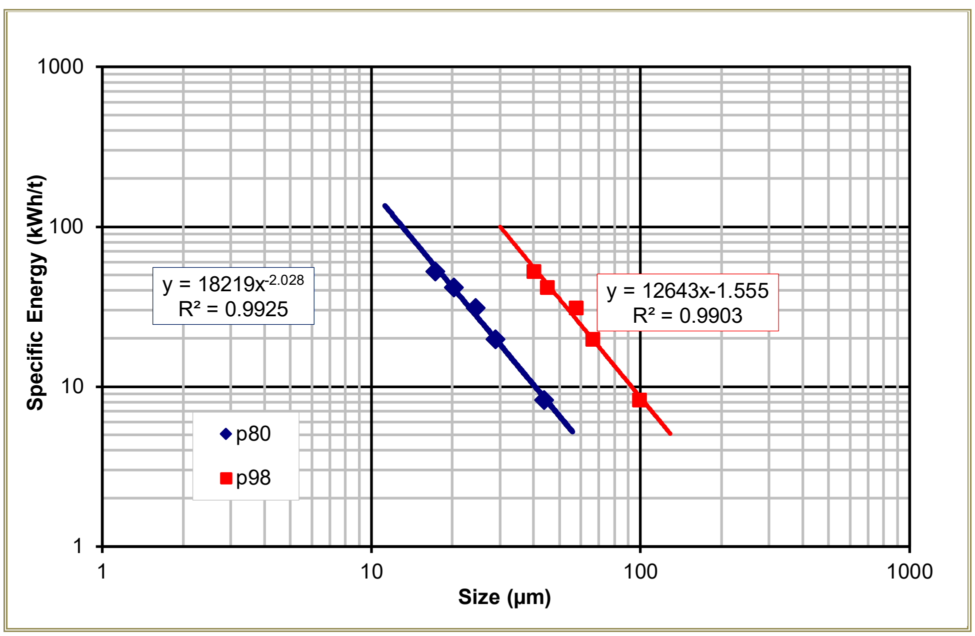
All three provided samples had an initial feed sizing, particle grind size F_{80} of around 80 µm. Samples were subjected to an IsaMill signature plot test, with the discharge sizing target particle grind size P_{80} of 25 µm. Results from the tests are shown in Figure 13-33, Figure 13-34, and Figure 13-35.



Source: ALS Kamloops KM 680 Report, 2019

Figure 13-33: JW Composite 3 Sample Signature Plot

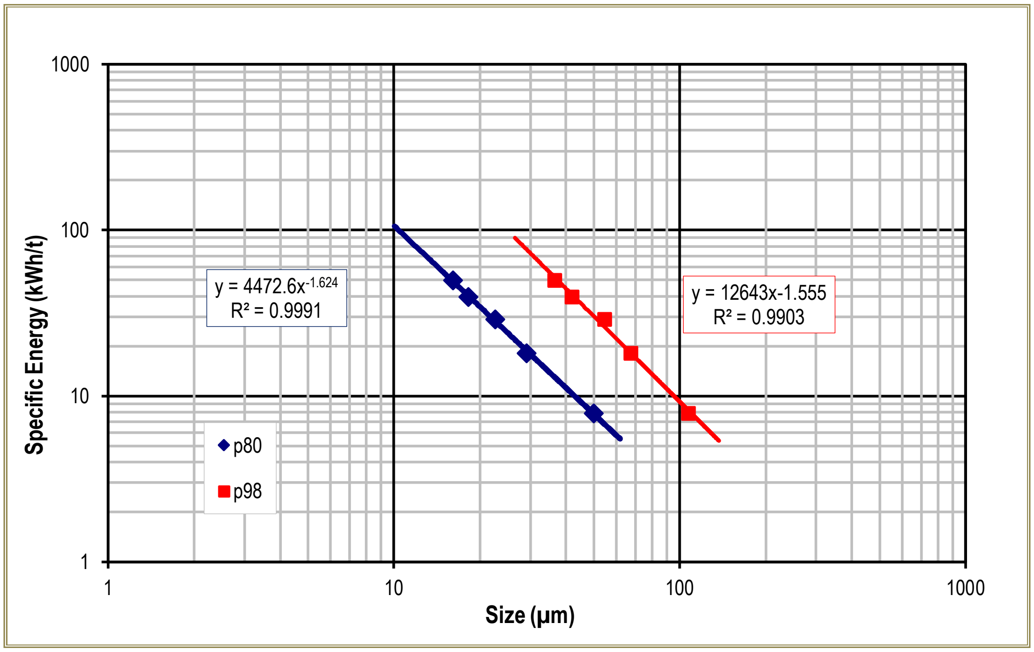
To achieve a targeted particle grind size (P_{80}) of 25 µm for the JW zone ores, an estimated required specific energy application is 21.7 kWh/t.



Source: ALS Kamloops KM 680 Report, 2019

Figure 13-34 : AV Composite 3 Sample Signature Plot

To achieve a targeted particle grind size (P_{80}) of 25 µm for the AV deposit ores, the estimated specific energy requirement is 26.7 kWh/t.



Source: ALS Kamloops KM 680 Report, 2019

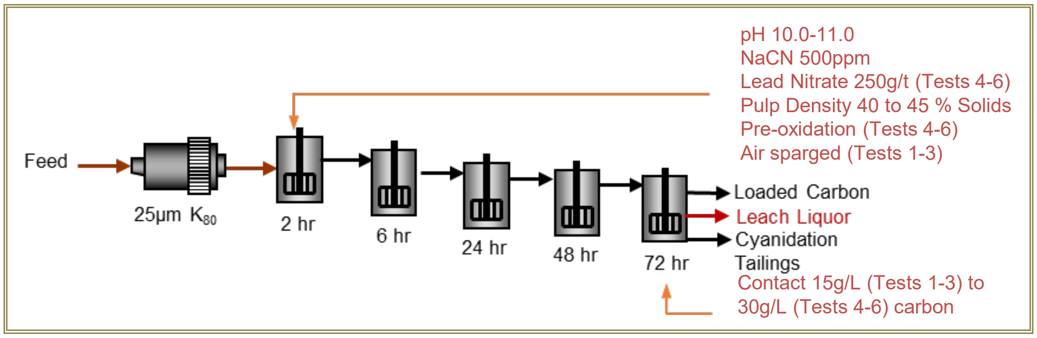
Figure 13-35: MV Composite 3 Sample Signature Plot

To achieve a targeted particle grind size (P_{80}) of 25 µm for the MV deposit ores, the estimated specific energy application requirement is 24.0 kWh/t.

With consideration of all of the available signature plot data, the estimated design specific energy for the fine grinding duty of the Red Mountain ores is 25.6 kWh/t. Based on the fine grinding mill throughput and presented signature plot data, it is estimated that high-speed stirred mill with an installed power of 3,000 kW will be needed for the tertiary milling application.

Metallurgical Testwork

Metallurgical testwork was conducted using the testwork basis for each of the Premier and Red Mountain flowsheets on the samples that had a particle grind size (P_{80}) of 25 µm. The testwork flowsheet used is presented in Figure 13-36. The testwork conditions associated with both of the Premier or Red Mountain flowsheets is displayed in the figure and further details are outlined in subsequent description.



Source: Base Met Lab Report BL0482, 2020

Figure 13-36: Metallurgical Testwork Flowsheet Schematics

The Premier ore CIL tests (1–3) flowsheet used by BML consisted of air being used for sparging, a NaCN concentration of 500 ppm, a pH being maintained at 10 to 10.5 using lime, with 5 g/L carbon addition after 6 h, through to the test being terminated at 72 h. This differed from the SGS test flowsheet which used 10 g/L carbon and was terminated after 48 h.

The Red Mountain ore CIL tests (4–6) flowsheet consisted of oxygen being used for sparging, a NaCN concentration of 500 ppm, a pH being maintained at 11 using lime, with 30 g/L carbon addition, 2 h pre-oxidation, and a lead nitrate addition of 250 g/L.

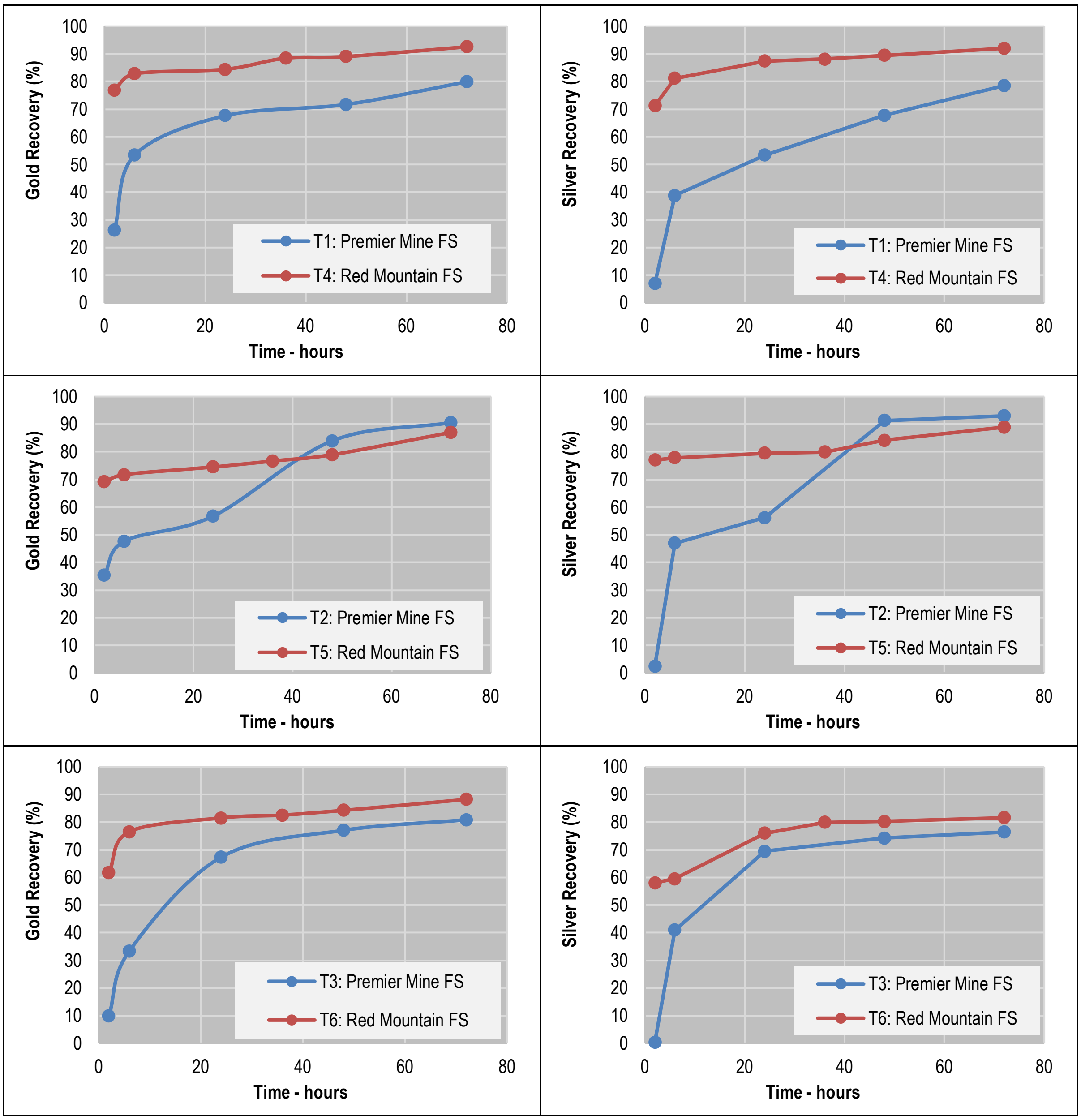
Table 13-46 shows a summary of the results.

The use of the Red Mountain ore testwork flowsheet yielded gold extraction rates for the JW, MV and AV Composite 3 samples of 93%, 87%, and 88%, respectively. With the use of the Premier ore testwork flowsheet, the achieved gold extraction rates for the JW; MV and AV Composite 3 samples were 80%, 90%, and 91%, respectively. It was noted that when the Red Mountain ore testwork flowsheet was applied, the lime consumption was more than double when compared to the Premier ore testwork flowsheet. The NaCN consumption varied for all three composites, ranging from 1.9 and 3.1 kg/t, without an observed trend between the two flowsheets. Even though the leaching kinetics were faster for the JW and AV samples, an additional 24 hrs leach retention time has resulted in higher final extraction rates for the Red Mountain ore testwork flow sheet. The exception was the MV sample where PGP ore testwork flow sheet exhibited faster leach kinetics and higher final extraction, which could be attributed to lowest sodium cyanide consumption rate for that particular test.

Table 13-46: Carbon-in-Leach Test Summaries

Test Program No.	Composite	Test No.	Flowsheet	Consumption (kg/t) NaCN	Consumption (kg/t) CaO	72-h Recovery (%) Au	72-h Recovery (%) Ag
BL 482	JW Comp 3	1	Premier	2.31	0.78	79.9	78.5
		4	Red Mountain	2.76	1.77	92.6	92.0
BL 184	JW Comp 2	47	Red Mountain	2.22	1.39	91.7	86.5
BL 084	JW Comp	134	Red Mountain	3.94	1.71	87.8	84.5
BL 482	MV Comp 3	2	Premier	2.40	0.76	90.6	93.0
		5	Red Mountain	1.88	2.30	87.0	89.0
BL 184	Marc Comp	41	Red Mountain	1.73	1.45	92.1	89.9
BL 084	Marc Comp	118	Red Mountain	4.06	1.20	92.2	92.4

Test Program No.	Composite	Test No.	Flowsheet	Consumption NaCN (kg/t)	Consumption CaO (kg/t)	72-h Recovery Au (%)	72-h Recovery Ag (%)
BL 482	AV Comp 3	3	Premier	2.75	1.12	80.8	76.4
		6	Red Mountain	3.11	2.53	88.2	81.6
Bl 184	AV Comp 2	44	Red Mountain	2.30	1.52	86.1	72.0
BL 084	AV Comp	121	Red Mountain	5.22	1.03	87.2	83.7



Source: Base Met Lab Report BL0482, 2020

Figure 13-37: Carbon-in-Leach Test Summaries

Cyanide Detoxification Testwork

The CN leach tailings from all 2019 tests were submitted to several small-scale continuous detoxification tests where the SO_2/Air process was applied. The results are summarized in Table 13-47. The tailings cyanide detoxification tests from all six CN leach tests, targeting CN_{WAD} values of less than 5 ppm, were very successful. It's worth noting that testwork on samples T01; T02 and T03 were conducted using Premier testwork flow sheet, whilst the testwork on samples T04; T05 and T06 were conducted using the Red Mountain testwork flow sheet.

Overall, the conditions were more favourable for the tests conducted using the Red Mountain ore flowsheet. When the Premier ore testwork flowsheet was employed, the copper addition and g SO_2/ g CN_{WAD}g–CN_{WAD} ratio were particularly high for the JW Composite 3. It is evident from the results that the main benefit of the Red Mountain flowsheet regime is the applied pre-oxygenation stage, which reduced the total iron in solution, which would otherwise cause increased reagent consumption in the detoxification process.

Table 13-47: Summary of Detoxification Test Results

			Solution Assays					Treatment Conditions			
Comp.	Sample	Detox Test	CN_{MP}* (ppm)	Cu (ppm)	Fe (ppm)	Ni (ppm)	Zn (ppm)	Return Time (min)	pH	SO_2 Ratio $gSO_2/gCNMP$	Cu (mg/L)
JW 3	T01	Feed	93.0	61.9	30.2	0.68	0.30	-	9.4	-	-
		C7	2.4	5.54	< 1	0.07	0.01	90	8.5	10.0	160
	T04	Feed	361.5	158.0	3.80	0.05	29.5	-	10.7	-	-
		C19	2.7	2.48	0.50	0.05	0.02	97	8.6	4.5	0
MV 3	T02	Feed	14.0	11.6	21.5	0.68	0.30	-	10.5	-	-
		C10	1.2					90	8.5	6.0	15
	T05	Feed	223.0	71.0	2.34	0.05	0.06	-	10.5	-	-
		C24	0.3	1.2	0.50	0.05	0.01	93	8.4	4.5	0
AV 3	T03	Feed	191.8	170.0	24.3	-	-	-	9.3	-	-
		C12	3.3					90	8.6	6.0	15
	T06	Feed	243.5	165.0	5.16	0.05	0.77	-	10.1	-	-
		C25	1.8	1.00	0.50	0.10	0.15	93	8.7	5.5	15

Note: CN_{MP} is calculated equivalent of CN_{WAD}.

Liquid/Solid Separation Testwork

A series of dewatering tests was conducted on the three composite feed samples and detoxified tailings to characterize the tailings properties and determine design data needed for thickening equipment selection and design.

Flocculant Screening Tests

Scoping settling tests for all three final tailings samples were conducted in a 1 L graduated cylinder to determine the most suitable flocculant for the subsequent static and dynamic settling tests. Different flocculants and their dosages were tested, and the free settling rate and final bed solids densities were recorded.

For the purposes of the Red Mountain static settling testwork, a total of four different flocculants were tested: two anionic flocculants, MF10 and MF 1011; one cationic, MF 380; and one non-ionic, MF 351. Table 13-48 presents the results from the flocculant screening tests.

The performance of the flocculants MF 380 and MF 351 was poorest relative to the use of the other flocculants. As a result, these two flocculants were not considered for future use.

Among the remaining flocculants, two anionic compounds—MF1011 and MF10—produced comparable results. However, the clarity of the water component (turbidity) was improved with the use of MF 10, which was selected for the subsequent dewatering testwork.

Table 13-48: Flocculant Screening Tests

Sample	Test	pH	Initial Density (% solids)	Flocculant Name	Flocculant Type	Floc Dosage (g/t)	Settling Rate (mm/sec)	Final Density (% solids)
MV Composite 3	FS1	8.2	15	MF 1011	Anionic	17	3.68	42.9
		8.2	15	MF 10	Anionic	17	n/a	n/a
		8.2	15	MF 351	Non-ionic	17	n/a	n/a
		8.2	15	MF 380	Cationic	17	n/a	n/a
		10.5	15	MF 1011	Anionic	17	2.94	40.1
		8.2	15	MF 1011	Anionic	41	0.33	30.1
		8.2	15	MF 10	Anionic	41	0.82	33.5
		11.0	15	MF 10	Anionic	41	0.22	27.7
		8.2	15	MF 10	Anionic	59	3.05	35.9
AV Composite 3	FS2	7.9	15	MF 1011	Anionic	17	0.21	28.5
		7.9	15	MF 10	Anionic	17	2.65	35.2
		7.9	15	MF 351	Non-ionic	17	0.43	29.3
		7.9	15	MF 380	Cationic	17	3.09	33.6
		10.5	15	MF 10	Anionic	17	0.63	31.5
		11.0	15	MF 10	Anionic	17	0.37	34.1
		7.9	15	MF 10	Anionic	24	2.83	35.5
JW Composite 3	F3	Natural	15	MF 1011	Anionic	40	0.12	20.7
		Natural	15	MF 351	Non-ionic	40	0.09	19.3

Static Settling Tests

Static settling tests were conducted on the feed samples as well as the detoxified tailings from each composite. Tests were conducted using the flocculant MF10 with dosages ranging from 40 to 80 g/t, and pH values between natural 8.0 and 10.5. The results are summarized in Table 13-49.

The MV Composite 3 and AV Composite 3 demonstrated settling rates ranging from 0.1 to 0.3 mm/sec. For the JW Composite 3, settling rates were higher, ranging between 0.1 and 0.5 mm/sec and the highest settling rate was observed at a flocculant dosage of 80 g/t.

For the detoxified tailings samples, the settling rates ranged between 0.1 and 0.3 mm/sec for all three composites. The turbidity TSS was assessed with some of the samples. The MV Composite 3 detox tailings samples demonstrated very high turbidity even at the higher flocculant dosages. It was observed that the turbidity improved when tests were carried out at the higher pH values. As an alternative to the elevated pH, testing with even higher flocculant dosages or the use of a coagulant should be investigated.

Table 13-49: Static Settling Tests

Sample	Test	pH	Floc Dosage (MF 10) (g/t)	TSS (mg/L)	Settling Rate (mm/sec)	Final Density (% solids)
MV Composite 3	ST1	8.2	58	-	0.13	41.7
	ST2	8.2	77	-	0.13	51.7
	ST3	8.2	53	-	0.11	37.9
	ST4	8.2	70	-	0.12	40.7
	ST5	8.2	60	-	0.13	50.0
	ST6	8.2	80	-	0.27	42.0
	ST7	10.5	60	-	0.05	42.3
	ST8	10.5	80	-	0.08	43.3
AV Composite 3	ST9	7.9	55	-	0.12	52.4
	ST10	7.9	74	-	0.09	51.6
	ST11	7.9	51	-	0.12	39.8
	ST12	7.9	68	-	0.13	41.7
	ST13	7.9	60	-	0.29	51.1
	ST14	7.9	80	-	0.30	43.2
	ST15	10.5	60	-	0.07	45.5
	ST16	10.5	60	-	0.16	44.5
JW Composite 3	ST17	Natural	60	-	0.30	45.9
	ST18	Natural	80	-	0.31	45.9
	ST19	Natural	40	-	0.07	39.8
	ST20	Natural	60	-	0.15	53.9
	ST21	Natural	80	-	0.16	53.1
	ST22	10.5	40	-	0.15	49.1
	ST23	10.5	60	-	0.15	47.9
	ST24	10.5	80	-	0.48	47.1
JW Comp 3 Detox Tailings	ST25	8.2	60	-	0.31	46.5
MV Comp 3 Detox Tailings	ST26	8.5	60	> 2,000	0.13	49.5
	ST27	8.5	80	449	0.16	47.2
	ST28	10.5	60	236	0.14	46.5
	ST29	10.5	80	91	0.14	45.7
AV Comp 3 Detox Tailings	ST30	Natural	60	-	0.16	47.6
	ST31	Natural	80	-	0.32	46.8
	ST32	10.5	60	-	0.28	48.1
	ST33	10.5	80	-	0.32	47.6

Dynamic Settling Tests

Following the static settling tests, dynamic settling tests were conducted using a bench scale thickener on both the feed and the detoxified tailings samples. Flocculant MF10 was selected for these tests, as this flocculant had produced the highest settled densities and overflow clarity from the previous testwork.

Slurry and flocculant were continuously added to the thickener feed well, testing each flocculant at different dosages. The thickener overflow was collected, and the settled bed of solids was built up to a pre-defined height. The settling tests were conducted at a targeted slurry pH of 8, with the exception of the MV detox tailings samples that required a pH of 10.5 or higher due to poor turbidity (TSS) that was observed during the static settling tests at the lower pH values.

The thickener underflow was sampled to determine the slurry density, while the thickener overflow was checked for turbidity (TSS). The dynamic settling test results are presented in Table 13-50.

Table 13-50: Dynamic Settling Test Results

Sample	TEST	pH	Loading Rate (t/m²/h)	Floc Dosage (g/t)	U/F Density (% solids)	TSS (mg/L)
MV feed	D1-A	8.2	0.30	80	51.3	242
	D1-B	8.2	0.70	80	46.5	60
	D1-C	8.2	0.50	60	50.1	115
	D1-D	8.2	0.50	40	35.8	6,736
	D1-E	8.2	0.50	80	52.4	88
	D1-F	8.2	1.00	60	33.3	193
	D1-G	8.2	0.70	60	-	383
	D1-H	10.5	0.50	60	39.8	106
	D1-I	10.5	0.50	80	50.2	67
AV feed	D2-A	7.9	0.50	40	-	97
	D2-B	7.9	0.30	40	47.9	70
	D2-C	7.9	0.70	40	36.3	118
	D2-D	7.9	0.50	30	38.8	242
	D2-E	7.9	0.50	20	28.8	949
	D2-F	7.9	1.00	30	26.4	374
	D2-G	7.9	0.70	30	30.4	498
	D2-H	7.9	0.50	60	45.1	47
	D2-I	7.9	0.70	60	33.8	72
	D2-J	7.9	1.00	60	38.0	35
	D2-K	7.9	0.50	80	51.9	38
	D2-L	10.5	0.50	60	52.1	41
	D2-M	10.5	0.50	40	45.3	96
JW feed	D4-A	8.2	0.50	60	56.5	118
	D4-B	8.2	0.50	80	55.2	96
	D4-C	10.5	0.50	60	53.6	53
MV Detox Tailings	D5-A	10.5	0.50	80	53.8	573
	D5-B	10.5	0.30	80	54.2	3,233
	D5-C	10.5	0.70	80	50.8	568
	D5-D	10.5	0.50	100	50.7	385
	D5-E	11.0	0.50	80	49.4	114
	D5-F	11.5	0.50	80	53.3	96

Sample	TEST	pH	Loading Rate ($t/m^2/h$)	Floc Dosage (g/t)	U/F Density (% solids)	TSS (mg/L)
AV Detox Tailings	D6-A	8.2	0.50	80	54.4	107
	D6-B	8.2	0.30	80	57.3	92
	D6-C	8.2	0.70	80	50.5	56
	D6-D	8.2	0.50	60	53.8	85
	D6-E	8.2	0.50	40	42.4	246
	D6-F	8.2	1.0	60	42.4	106
	D6-G	8.2	0.70	60	42.2	68
JW Detox Tailings	D3-A	8.0	0.50	60	52.4	324
	D3-B	8.0	0.30	60	55.1	428
	D3-C	8.0	0.70	60	44.6	275
	D3-D	8.0	1.00	60	41.9	455
	D3-E	8.0	0.50	40	39.6	23,434
	D3-F	8.0	0.50	80	54.4	91

Thickener underflow densities of up to 52% solids could be achieved on the MV and AV feed samples, whilst 57% solids underflow density at a 0.5 $t/m^2/h$ flux loading rate could be achieved for the JW feed samples.

For the CN detox tailings samples, underflow densities of approximately 54% solids could be achieved for MV, AV, and JW at a loading rate of 0.5 $t/m^2/h$.

The measured TSS was quite variable under the tested conditions. The specification limit should be reviewed to determine a suitable target for the thickener overflow clarity.

It can be concluded that thickener underflow densities of approximately 50% solids w/w are achievable with a flocculant dosage of 60 g/t.

Based on the results presented, a flocculant dosage of 60 g/t and a solids loading rate of 0.23 $t/h/m^2$ is recommended for the Red Mountain pre-leach thickener sizing and design. Based upon the solids settling rate and plant throughput, it is estimated that a 27 m-diameter pre-leach thickener will be required for the thickening duty. In the next phase of the PGP, it is recommended that additional optimization testwork is conducted to improve the thickener overflow clarity.

14 MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimates for the Premier & Red Mountain Gold Project (the Project) includes the resources of the Premier Gold Project (PGP) and the Red Mountain Gold Project (RMP).

The resources at the PGP include the Premier, Big Missouri, Silver Coin, Martha-Ellen, and Dilworth deposits. This work was completed by Susan Bird, P.Eng. (APEGBC) with an effective date of December 12, 2019.

The resource estimation work at the RMP was completed by Dr. Gilles Arseneau, P.Geo. (APEGBC) and the effective date of the RMP Mineral Resource Statement is August 30, 2019.

The following sections present a summary of each of the resource areas, and then describe the modelling methodology for the PGP and the Red Mountain area separately since they were done by different QPs.

14.1 Premier Gold Project Mineral Resource Estimate

The Mineral Resources for the PGP have been updated since the previous estimate in January 2019 (Rennie et al., 2019) due to additional drilling and updated geologic interpretation for the Premier, Big Missouri, and Silver Coin deposit areas.

The Mineral Resource with an effective date of December 12, 2019 is listed in Table 14-1, using a 3.5 g/t gold equivalent (AuEq) cut-off. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards for mineral resources and mineral reserves (CIM, 2014) were followed for the Mineral Resource Estimate.

Table 14-1: PGP Resource Estimate at a 3.5 g/t AuEq Cut-off – Effective date: December 12, 2019

Class	Deposit	In-situ Tonnage (kt)	In-situ Grades			Metal	
			AuEq (g/t)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
Indicated	Premier	1,298	8.90	8.46	64.20	353	2,680
	Big Missouri	1,116	8.48	8.36	16.90	300	607
	Silver Coin	1,597	7.77	7.61	23.00	390	1,181
	Martha-Ellen	130	5.80	5.47	48.00	23	201
	Dilworth	-	-	-	-	-	-
	Total Indicated	**4,141**	**8.25**	**8.01**	**35.1**	**1,066**	**4,669**
Inferred	Premier	1,753	7.00	6.72	39.80	379	2,243
	Big Missouri	1,897	8.44	8.34	14.70	508	896
	Silver Coin	523	7.19	7.03	23.20	118	390
	Martha-Ellen	653	6.36	6.12	34.30	129	720
	Dilworth	235	6.51	6.13	56.10	46	424
	Total Inferred	**5,061**	**7.45**	**7.25**	**28.7**	**1,180**	**4,673**

Notes: **1.** Mineral Resources are estimated at a cut-off grade of 3.5 g/t AuEq based on metal prices of US$1,300/oz Au and US$20/oz Ag. **2.** The AuEq values were calculated using US$1,300/oz Au, US$20/oz Ag, a silver metallurgical recovery of 45.2%, and the following equation: AuEq = Au g/t + (Ag g/t x 0.00695). **3.** A mean bulk density of 2.85 t/m^3 is used for Premier and of 2.80 t/m^3 for all other deposit areas. **4.** A minimum mining width of 2.5 m true thickness is required to be classified as Resource material. **5.** Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate for Premier, Big Missouri, Silver Coin, Martha Ellen, or Dilworth deposits.

14.2 Changes to the Premier Gold Project Mineral Resources

Table 14-2 and Table 14-3 presents the total change in the PGP Resource Estimate by deposit from the previous estimate announced in Ascot's News Release of December 2018 and detailed in the Technical Report (Rennie, et al., 2019). There has been a significant increase in Indicated tonnage for the three deposits in which drilling has taken place and which have been updated since the last estimate. This tonnage increase is partially offset by a drop in grades, resulting in an increase in overall metal content. The changes to the PGP Resource Estimate are due to discovery of additional PGP Mineral Resources through diamond drilling, upgrading of Inferred material, and enhanced geologic interpretation and controls in the modelling conducted during 2019.

Table 14-2: Summary of Changes to the PGP Resource from 2018 to 2019 – Indicated

Deposit	In-situ Tonnes ('000s)	Tonnage Change from 2018 (%)	In-situ Grade		Contained Ounces (koz)	
			Au (g/t)	Change from 2018 (%)	Au	Change from 2018 (%)
Premier	1,298	+4	8.46	+21	353	+26
Big Missouri	1,116	+107	8.36	+2	300	+111
Silver Coin	1,597	+86	7.61	-5	390	+76
Martha Ellen	130	0	5.47	0	23	0
Total	**4,141**	**+49**	**8.01**	**+7**	**1,066**	**+60**

Table 14-3: Summary of Changes to the Total PGP Resource from 2018 to 2019 – Inferred

Deposit	In-situ Tonnes ('000s)	Tonnage Change from 2018 (%)	In-situ Grade		Contained Ounces (koz)	
			Au g/t	Change from 2018 (%)	Au	Change from 2018 (%)
Premier	1,753	+1	6.72	+13	379	+14
Big Missouri	1,897	-16	8.34	+1	509	-15
Silver Coin	523	-55	7.03	-10	118	-59
Martha Ellen	653	0	6.12	0	129	0
Dilworth	235	0	6.13	0	46	0
Total	**5,061**	**-16**	**7.25**	**+1**	**1,180**	**-15**

Notes: The following notes pertain to Tables 14-2 and 14-3:
1. Mineral Resources are estimated at a cut-off grade of 3.5 g/t AuEq based on metal prices of US$1,300/oz Au and US$20/oz Ag. **2.** Percent differences are calculated as: (2020–2018)/2018%. **3.** The AuEq grade was calculated using the same parameters as the last Resource Estimate for comparison purposes. **4.** The AuEq values were calculated using US$1,300/oz Au, US$20/oz Ag, a silver metallurgical recovery of 45.2%, and the following equation: AuEq(g/t) = Au(g/t) + 45.2% x Ag(g/t) x 20 / 1,300. **5.** A mean bulk density of 2.85 t/m^3 is used for Premier and of 2.80 t/m^3 for all other deposit areas. **6.** A minimum mining width of 2.5 m true thickness is required to be classified as Resource material.

14.3 Red Mountain Mineral Resources Estimate Summary

The Red Mountain mineral resource model utilized a total of 699 drill holes, 230 of which were drilled by IDM: 12 in 2014, 62 in 2016, 116 in 2017, and 40 in 2018. The resource estimation work was completed by Dr. Gilles Arseneau, P. Geo. (APEGBC) an appropriate independent Qualified Person (QP) within the meaning of NI 43 101. The effective date of the Mineral Resource statement is August 30, 2019.

Section 14.18 describes the resource estimation methodology and summarizes the key assumptions considered by Arseneau Consulting Services Inc. (ACS). In the opinion of ACS, the resource evaluation reported herein is a reasonable representation of the gold and silver mineral resources found at the RMP at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (2003) and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. Table 14-4 summarizes the resource estimate for the Red Mountain Project. The QP is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that could materially affect the Red Mountain Mineral Resource Estimate.

ACS audited the database used to estimate the Red Mountain mineral resources, and is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the gold mineralization, and that the assay data are sufficiently reliable to support mineral resource estimation.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the Province of BC in terms of environmental, permitting, taxation, socioeconomic, marketing, and political factors.

Table 14-4: RMP Mineral Resource Statement Reported at a 3.0 g/t AuEq Cut-off

		Grade		Contained Ounces	
	Tonnes (t '000s)	Au (g/t)	Ag (g/t)	Au (oz '000s)	Ag (oz '000s)
Total Measured	1,920	8.81	28.3	543.8	1,747
Total Indicated	1,271	5.85	10.01	238.8	409
Measured and Indicated Total	3190	7.63	21.02	782.6	2,156
Total Inferred	**405**	**5.32**	**7.33**	**69.3**	**95.5**

Note: RMP Resources are reported at a 3.0 g/t Au cut-off for underground long hole stoping

14.4 Key Assumptions / Basis of Estimate for the Premier Gold Project Mineral Resource Estimate

The total number of holes completed for the entire PGP property is 4,623 with drilling by deposit area summarized in Table 14-5.

The drilling by area and year within each of the block models is summarized for the pre-Ascot drilling in Table 10-1 through Table 10-5 and for the Ascot drilling in Table 10-6 through Table 10-10.

Table 14-5: Summary of Drilling by Premier Deposit Area

Deposit	Era	Holes	Length (m)	Intervals Assayed	Assayed (m)	% Assayed
Premier	Pre-Ascot	910	78,464	27,581	38,971	50
	Ascot	1,121	288,450	40,933	68,801	24
	Subtotal	**2,031**	**366,914**	**68,514**	**107,772**	**29**
Big Missouri	Pre-Ascot	381	25,085	7,488	11,838	47
	Ascot	763	155,197	64,337	110,835	71
	Subtotal	**1,144**	**180,282**	**71,825**	**122,673**	**68**
Silver Coin	Pre-Ascot	898	112,062	52,550	92,719	83
	Ascot	94	13,546	5,087	8,383	62
	Subtotal	**992**	**125,609**	**57,637**	**101,102**	**80**
Dilworth	Pre-Ascot	13	625	124	221	35
	Ascot	153	30,242	15,407	24,857	82
	Subtotal	**166**	**30,867**	**15,531**	**25,078**	**81**
Martha Ellen	Pre-Ascot	153	10,510	3,095	4,486	43
	Ascot	137	22,353	8,589	16,485	74
	Subtotal	**290**	**32,863**	**11,684**	**20,971**	**64**
Grand Total		**4,623**	**736,535**	**225,191**	**377,597**	**51**

Separate block models were created for each of the five PGP deposits with a block size of 3 m x 3 m x 3 m. Block model extents are presented in Table 14-6.

Table 14-6: Block Model Extents for each PGP Deposit

Deposit	Axis	Minimum	Maximum	Length (m)	Block Size	No. Blocks
Premier	Easting	436,200	437,595	1,395	3	465
	Northing	6,212,100	6,213,510	1,410	3	470
	Elevation	90	690	600	3	200
Big Missouri	Easting	435,750	437,160	1,410	3	470
	Northing	6,218,500	6,220,351	1,851	3	617
	Elevation	700	1,180	480	3	160
Silver Coin	Easting	435,500	436,100	600	3	200
	Northing	6,217,500	6,218,418	918	3	306
	Elevation	710	1,031	321	3	107
Dilworth	Easting	434,500	435,850	1,350	3	450
	Northing	6,222,400	6,224,200	1,800	3	600
	Elevation	800	1,550	750	3	250
Martha Ellen	Easting	435,350	436,100	750	3	250
	Northing	6,220,580	6,221,600	1,020	3	340
	Elevation	850	1,360	510	3	170

14.5 PGP Geological Models

The geological models for each of PGP deposit areas consisted of creating solids for potentially mineralized zones, and for the post-mineral porphyry dykes and faults. Dikes and faults created for the 2018 model were adjusted to adhere to the new drilling. Mineralization within each of the deposits is now interpreted to have been mineralized by sub-vertical structures which acted as conduits to fluid flow. The structures at Premier are interpreted to be preserved in their original geometry whereas at Big Missouri and Silver Coin, previously east dipping structures have been rotated into their current position to now be shallowly dipping, primarily to the west, with a general younging trend in the same direction.

Mineralization and the relation of the geology to the potentially mineralized wireframes used in the block model interpolation are illustrated and discussed in detail in Section 7 of this report with the wireframes and corresponding search ellipses used during interpolation illustrated below in this section.

To model the potentially mineralized zones for underground mining the AuEq grade has been used to aid in tagging the intervals for potential underground mining. The AuEq grade was calculated using the following assumptions:

- Au price = US$1,300/oz
- Ag price = US$20/oz
- Ag recovery = 45.2%.

The resulting equations is:

$$AuEq(g/t) = Au(g/t) + 45.2\% \times Ag(g/t) \times 20 / 1{,}300$$

The grades for both gold and silver vary by as much as five orders of magnitude over fairly short distances (i.e., 5 m to 20 m). Therefore, correlation of higher grades is difficult and has been mitigated by the inclusion of surrounding lower grade mineralization. For this reason, a cut-off grade of approximately 1.0 g/t AuEq was selected for the mineralization envelopes, which is significantly lower than the actual economic cut-off grade for underground mining. This improved apparent continuity between drill hole intercepts, enhanced interpretation and also allowed for the inclusion of model or "internal" smoothing or dilution.

Wireframes have been created by manual tagging of assay intercepts with an AuEq grade of equal to or greater than approximately 1.0 g/t AuEq and a possible true thickness of 1.0 m to 2.0 m. This has been done to include intercepts below the resource cut-off grade of 3.5 g/t AuEq in order to provide continuity of mineralized solids, and to include internal dilution in the interpolations. The tagged intercepts were then used with the Implicit Modelling Tool in MineSight (MSIM®) to create footwalls and hanging walls for the development of mineralized solids. The surfaces have been clipped to a maximum of 50 m from an outer boundary intercept.

The interpretive process involved a great deal of inspection of intercepts to ensure that they were wide enough in true thickness, whether dilution was required to achieve this minimum thickness, and if so, how much and at what grade.

The precise location of void spaces is not known owing to uncertainties in survey control, the poor quality of the mined-out wireframe volumes, and lack of current production records. Consequently, it was necessary to provide a buffer around known void spaces. This buffer was nominally 2 m to 3 m depending on the circumstances. If the void was solely due to development and not stoping, then the buffer was usually reduced and sometimes not applied at all.

Intercepts of voids in the Ascot drilling were used to evaluate the accuracy of the locations of stoped volume models wherever possible. Legacy holes with high-grade intercepts that occurred near stope volumes were assumed to be mined out and ignored. In many instances, Ascot holes pierced voids and then intersected mineralization adjacent or near to the void space. In other, more rare occurrences, a drill hole would appear to intersect a stope or drift model, but in fact intersected a mineralized zone. Each individual intercept of this nature was evaluated and either rejected or accepted depending on the possibility of whether the zone in question was likely to be mineable. As a general rule, intercepts near stopes were ignored as not mineable if they were within 2 m of the logged void space.

A total of 99 zones for Premier, 83 zones for Big Missouri, 84 for Silver Coin, 14 zones for Martha Ellen, and 22 zones for Dilworth have been modelled. The wireframes are illustrated in the 3-D views and in sections for each deposit where they are also compared to the geology models in Section 7 of this report.

14.5.1 Wireframes – Additional Details for Premier-Northern Lights Deposit

There are portions of the Premier deposit where no additional drilling has been done since the previous resource estimate was published in January 2019, and where there has been no change to the previous wireframes built with more traditional 2-D methods using GEMS software. There are 28 of these zones; they occur within the Lunchroom, Obscene, and Premier Main areas.

In the zones from the 2018 modelling, polyline interpretations were first drawn on cross sections spaced at 5 m to 25 m intervals, depending on drill density. GEMS polylines were created such that they were "pinned" to the drill holes in 3-D to ensure that there were no parallax effects owing to holes being off-section. These lines were extruded into solid "slices" and used to re-interpret the zones on level plan views spaced at 10 m to 20 m intervals, again depending on drill density and/or complexity of the models. The level plan polylines were extruded once more and used as guides to rebuild and refine the cross-sectional interpretations. Minimum true widths for these zones is 2.5 m. Adjacent intercepts could be incorporated into a solid, ostensibly without a distance limit, but in practice, only rarely did the distance between intercepts exceed 30 m. Polylines were limited to an external limit of 25 m from the outermost drill hole, but again, due to the drill density, this limit was not reached very often.

14.6 PGP Assay Statistics and Capping

The assay statistics have been examined using boxplots, histograms, and cumulative probability plots (CPP). The grade distribution for gold and silver within the modelled grade shells is generally lognormal except at very low grades approaching the lower detection limits and at the upper end where high-grade outliers are apparent. Capping the assays of both gold and silver has been implemented as summarized in Table 14-7 and Table 14-8 to limit high grade outliers, as indicated by the CPP plots shown as examples for both gold and silver, for each deposit in Figure 14-1 through Figure 14-10.

Assay statistics for each deposit area for both uncapped and capped gold and silver grades are summarized in Table 14-9 through Table 14-12, illustrating the effect cap ping has had on the weighted Mean Grade and Coefficient of Variation (CV). It should be noted that the high CVs within each area are not indicative of the CVs within the wireframes used to limit composites used during interpolation, which were always well below 2.0. The interpolation methodology has been to cap very high outliers, and to use additional Outlier Restriction of high-grade composites during interpolation to limit the distance of influence of higher grades. This is discussed further in the following sections.

Table 14-7: Assay Capping Strategy – Premier

Domain	Area	Cap Value		Domain	Area	Cap Value		Domain	Area	Cap Value	
		Au (g/t)	Ag (g/t)			Au (g/t)	Ag (g/t)			Au (g/t)	Ag (g/t)
1	NW	9,999	45	35	Obscene	30	800	66	NL	9,999	500
2	NW	9,999	80	36	Obscene	130	300	67	NL	9,999	9,999
3	609	9,999	100	37	Obscene	170	300	68	NL	9,999	9,999
4	609	9,999	110	38	Obscene	160	300	69	NL	9,999	9,999
5	609	50	360	39	Obscene	45	1,000	70	NL	9,999	300
6	609	50	220	40	Obscene	9,999	9,999	71	NL	9,999	80
7	609	9,999	9999	41	Obscene	70	1,700	72	NL	9,999	9,999
8	609	9,999	200	42	Obscene	9,999	2,200	73	NL	9,999	105
9	609	9,999	200	43	Obscene	9,999	300	74	NL	9,999	9,999
10	609	120	100	44	Obscene	100	500	75	NL	9,999	9,999
11	609	9,999	9999	45	Obscene	9,999	1,000	76	NL	9,999	9,999
12	609	9,999	9999	46	Obscene	9,999	9,999	77	NL	9,999	9,999
13	609	9,999	120	47	Obscene	9,999	9,999	78	NL	9,999	9,999
14	LunchRm	100	100	48	Obscene	9,999	9,999	79	NL	9,999	9,999
15	LunchRm	100	500	49	Obscene	9,999	9,999	83	Ben	100	400
16	LunchRm	1,000	1500	50	Obscene	9,999	9,999	84	Ben	100	100
17	LunchRm	220	250	51	Prew	75	120	85	Ben	30	9,999
18	LunchRm	9,999	200	52	Prew	80	80	86	Ben	9,999	9,999
19	LunchRm	9,999	4000	53	Prew	310	200	87	Ben	50	9,999
20	LunchRm	9,999	500	54	Prew	9,999	9,999	88	Ben	9,999	300
21	LunchRm	9,999	500	55	Prew	1,000	500	89	Ben	80	80
22	Main	9,999	200	56	Prew	9,999	100	90	Ben	9,999	9,999
23	Main	9,999	9999	57	Prew	9,999	9,999	91	Ben	9,999	9,999
24	Main	9,999	9999	58	Prew	80	80	92	Ben	30	30
25	Main	9,999	1300	59	Prew	60	50	93	Ben	80	80
26	Main	9,999	1000	60	Prew	75	75	100	602	60	300
27	Main	70	1000	61	Prew	9,999	9,999	101	602	9,999	70
28	Main	9,999	1000	62	Prew	9,999	9,999	102	602	9,999	20
29	Main	9,999	2200	63	Prew	9,999	9,999	104	602	9,999	9,999
30	Main	9,999	1000	64	Prew	9,999	9,999	105	602	100	100
31	Main	100	1900	65	Prew	9,999	9,999	107	602	100	200
32	Main	90	4800					108	609	9,999	9,999
33	Main	100	9999					109	Main/Obs	9,999	9,999
34	Main	9,999	9999					110	Main/Obs	9,999	9,999

Table 14-8: Assay Capping Strategy – Big Missouri, Silver Coin, Martha Ellen, and Dilworth

Area	Au		Ag	
	Cap (g/t)	No. Capped	Cap (g/t)	No. Capped
Big Missouri	200	8	1,000	6
Silver Coin	200	7	600	12
Martha Ellen	70	3	1,000	2
Dilworth	100	3	4,000	3

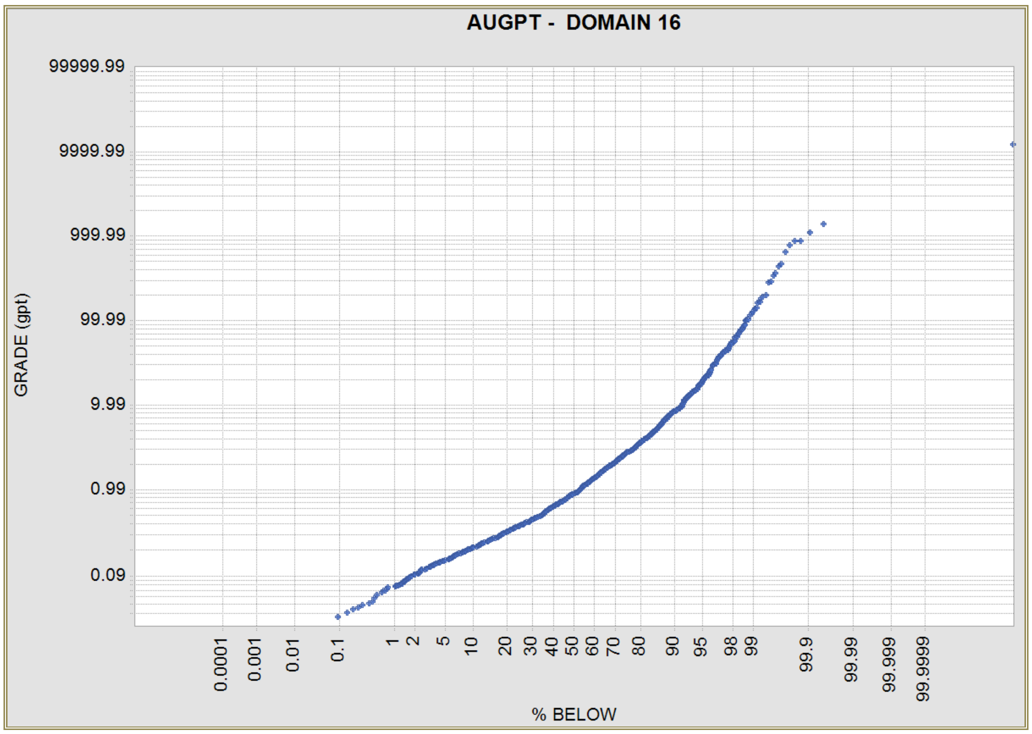


Figure 14-1: Premier – Example of CPP Plot – Domain 16 – Gold

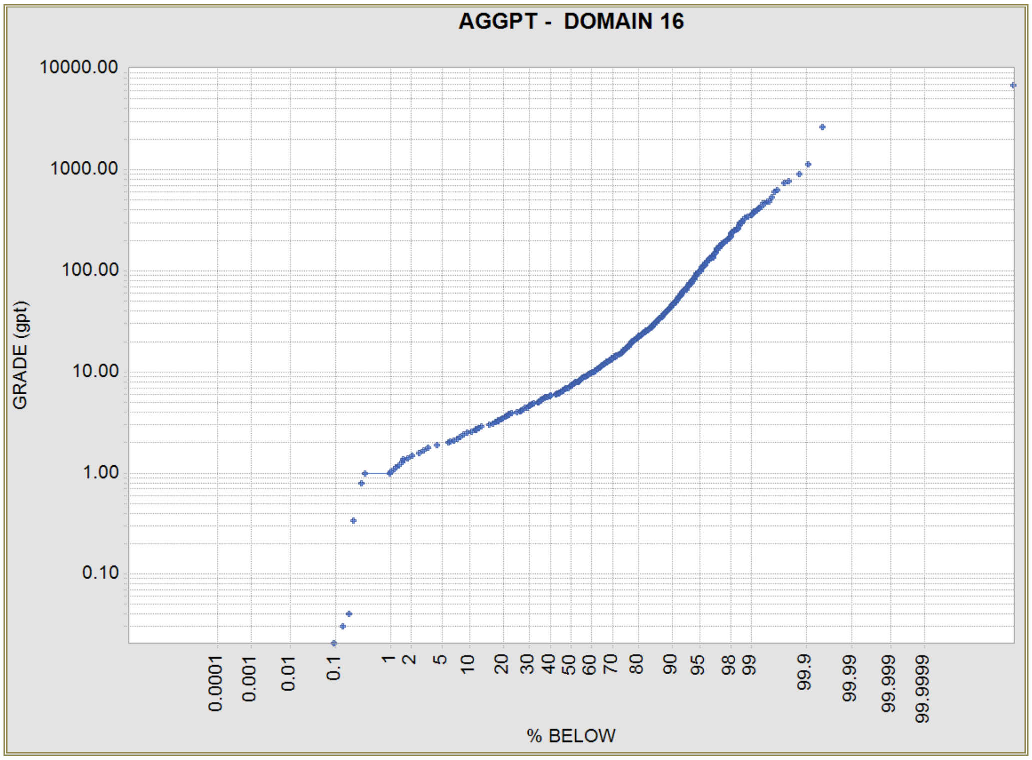


Figure 14-2: Premier – Example of CPP Plot – Domain 16 – Silver

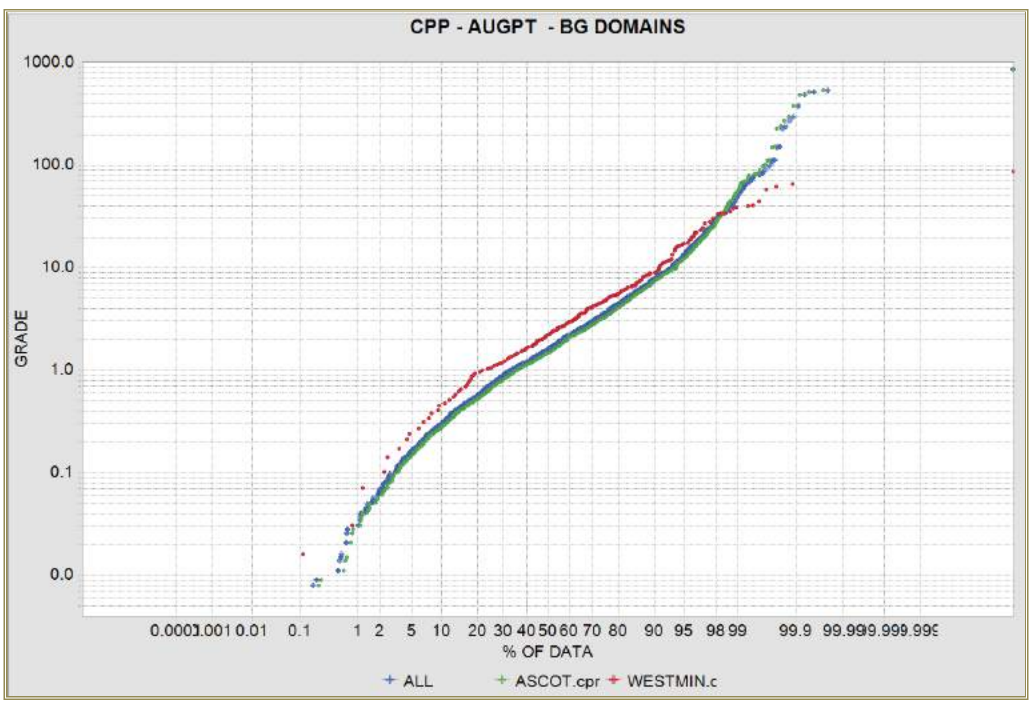


Figure 14-3: Big Missouri Domains – CPP Plot by Owner – Gold

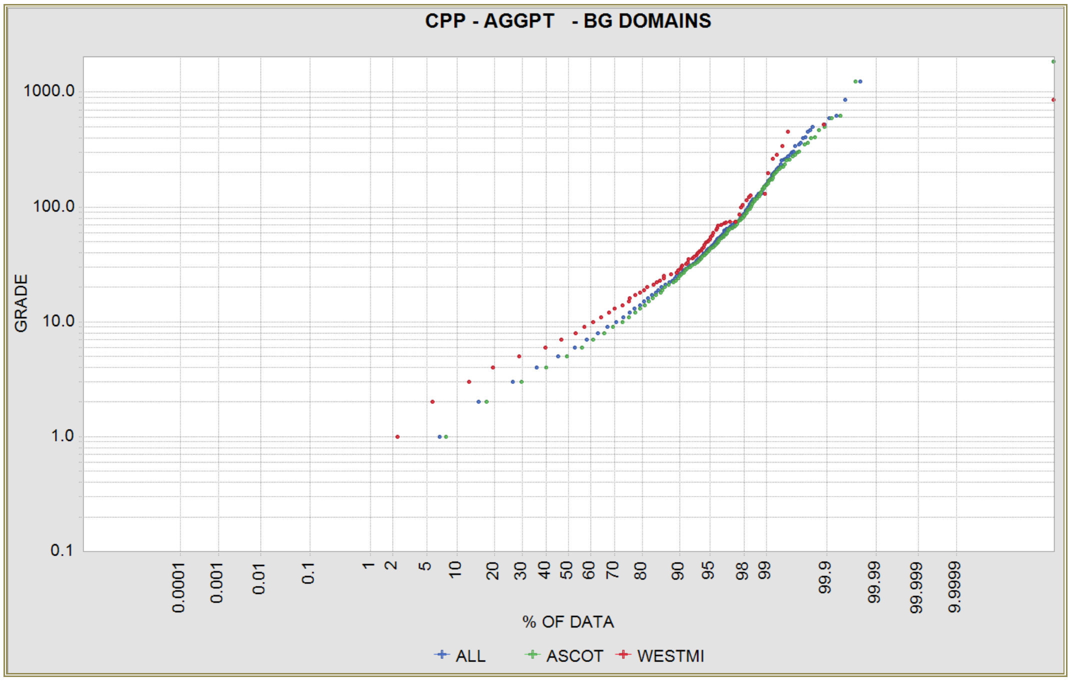


Figure 14-4: Big Missouri Domains – CPP Plot by Owner – Silver

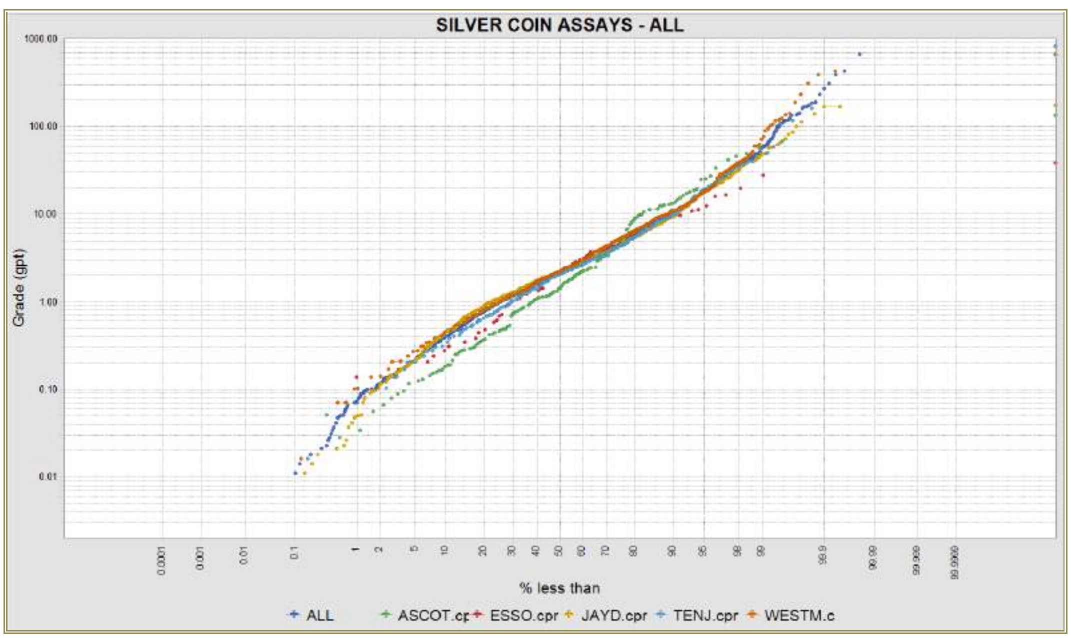


Figure 14-5: Silver Coin – CPP Plot by Owner – Gold

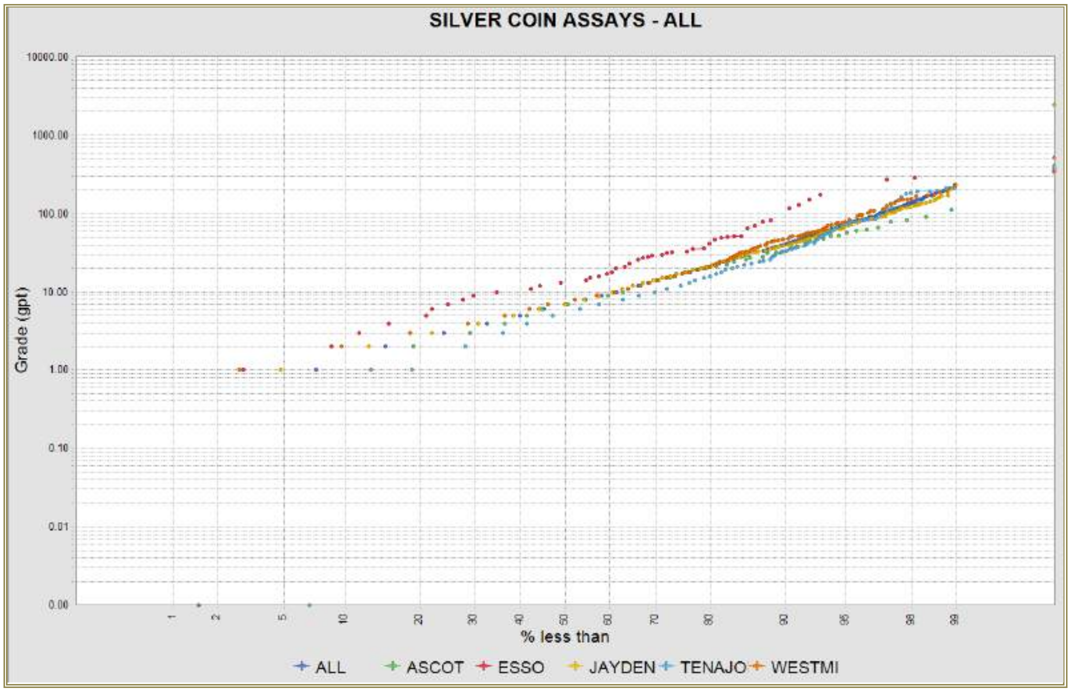


Figure 14-6: Silver Coin – CPP Plot – Silver

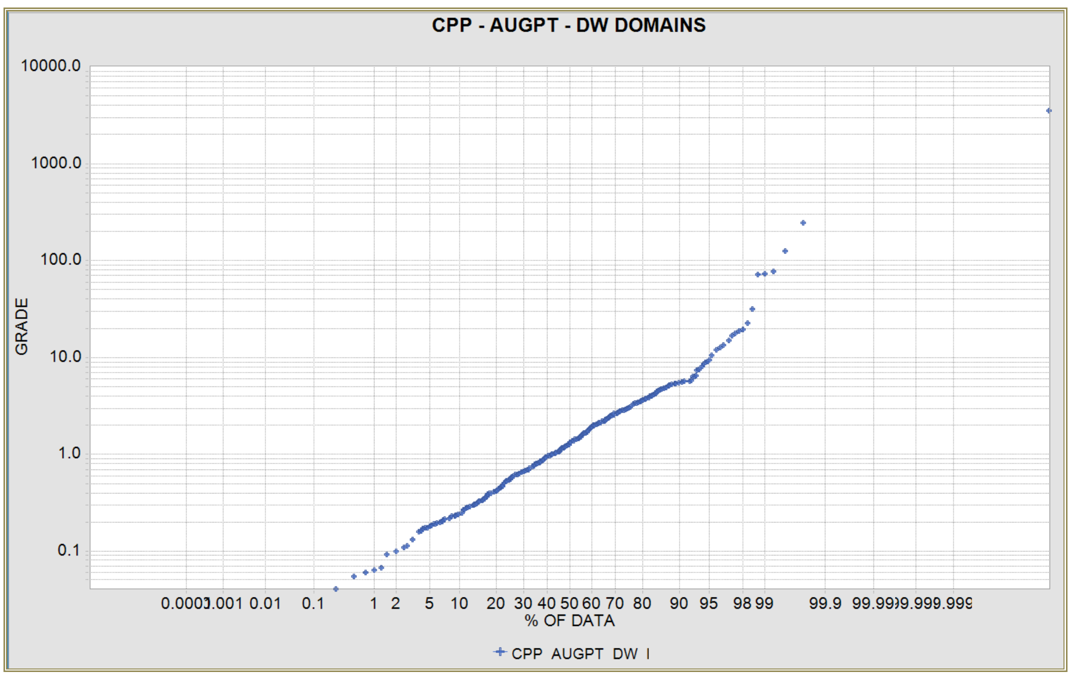


Figure 14-7: Dilworth – CPP Plot – Gold

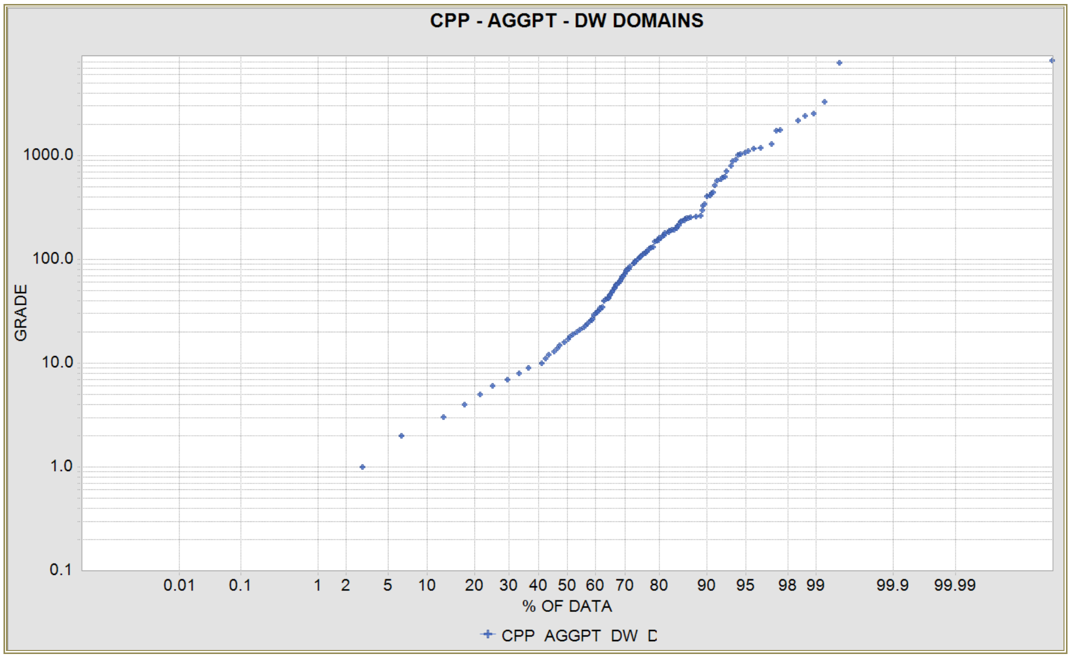


Figure 14-8: Dilworth – CPP Plot – Silver

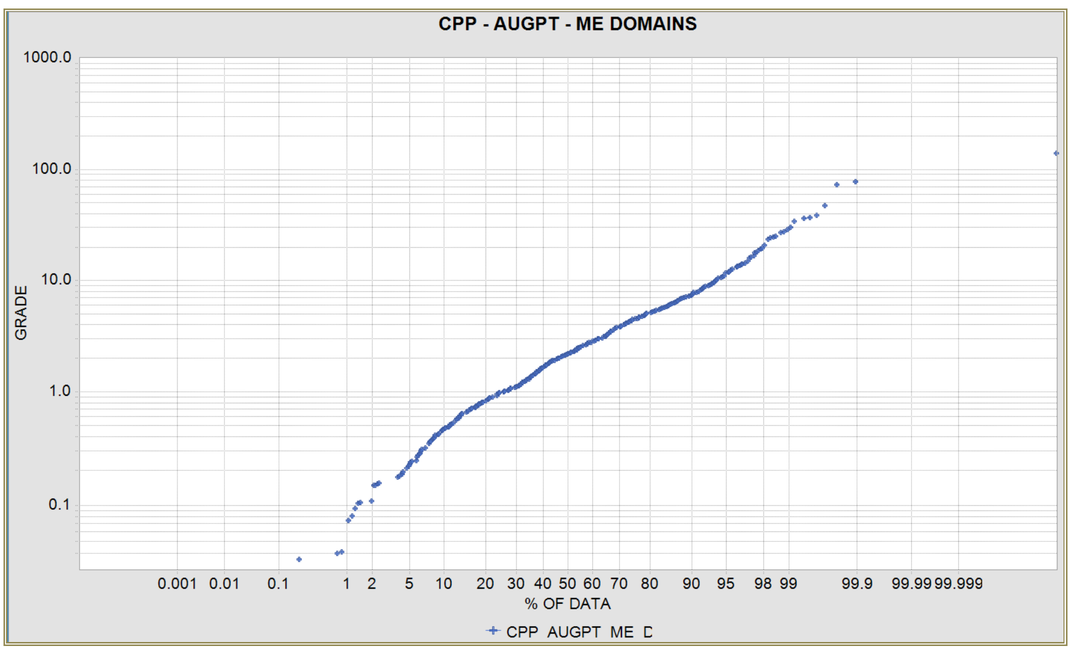


Figure 14-9: Martha Ellen – CPP Plot – Gold

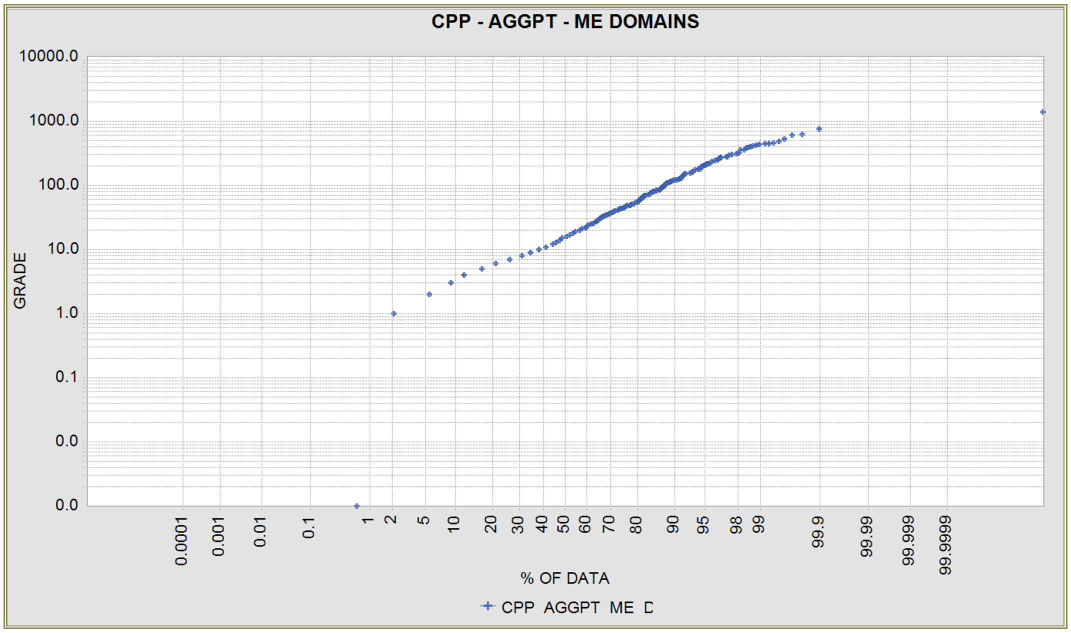


Figure 14-10: Martha Ellen – CPP Plot – Silver

Table 14-9: Assay Statistics – Premier

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	8,736	8,736	8,736	8,736
No. Missing	18	18	18	18
Minimum (g/t)	0.003	0.003	0	0
Maximum (g/t)	12,100	1,000	16,248	4,800
Weighted Mean (g/t)	6.072	5.118	37.920	34.634
Weighted CV	16.48	5.11	6.53	4.28

Table 14-10: Assay Statistics – Big Missouri

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	4,492	4,492	4,492	4,492
No. Missing	0	0	0	0
Minimum (g/t)	0.003	0.003	0	0
Maximum (g/t)	870.88	200	1860	1000
Weighted Mean (g/t)	4.24	4.00	13.40	13.20
Weighted CV	3.98	2.75	3.30	2.90

Table 14-11: Assay Statistics – Silver Coin

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	5,010	5,010	4,110	4,110
No. Missing	0	0	900	900
Minimum (g/t)	0.001	0.001	0	0
Maximum (g/t)	833.1	200	2453	600
Weighted Mean (g/t)	5.008	4.843	18.600	17.700
Weighted CV	3.32	2.48	3.30	2.30

Table 14-12: Assay Statistics – Dilworth

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	480	480	480	480
No. Missing	0	0	0	0
Minimum (g/t)	0.003	0.003	0	0
Maximum (g/t)	3,550	70	8,260	4,000
Weighted Mean (g/t)	6.012	3.015	99.200	88.900
Weighted CV	16.20	3.01	4.80	3.70

Table 14-13: Assay Statistics – Martha Ellen

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	904	904	904	904
No. Missing	0	0	0	0
Minimum (g/t)	0.024	0.024	0	0
Maximum (g/t)	140.503	70	1,395	1,000
Weighted Mean (g/t)	3.911	3.790	43.100	42.900
Weighted CV	2.00	1.61	2.00	1.90

14.7 Premier Gold Project Compositing

Assay sample lengths varied across the drill programs but are generally between 1.0 m and 2.0 m. A histogram of the assay intervals for Premier are shown in Figure 14-11. A base composite length of 1 m has been used for all deposits. Assay data has been coded with a domain value corresponding to the potentially mineralized wireframe prior to compositing. The domain code has been honoured during compositing. Any interval within a domain that was less than 0.5 m was composited with the interval above it, resulting in composite length ranging from 0.5 m to 1.5 m.

A historical 1988 drill hole in the Martha Ellen deposit with incongruously long lengths has been excluded from the grade modelling. The sample data appear to have been composited for use as a metallurgical hole.

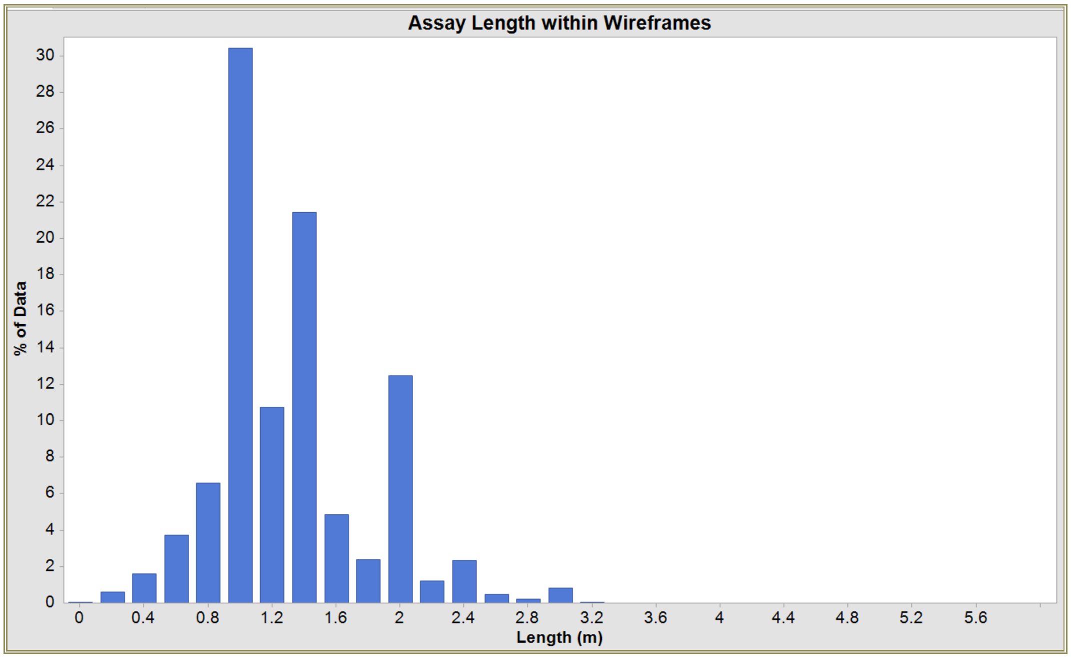


Figure 14-11: Histogram of Assay Lengths – Premier

Composite statistics, for both uncapped and capped values are summarized in Table 14-13 through Table 14-17 for each of the three deposit areas. The tables also provide a comparison of the weighted mean assay grades to the weighted mean composite grade. In each case the grades are virtually the same, indicating that composited grades are representative of the original assay data.

Table 14-14: Composite Statistics – Premier

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	24,160	24,160	24,160	24,160
No. Missing	18	18	18	18
Minimum (g/t)	0.003	0.003	0	0
Maximum (g/t)	12100	1000	16248	4800
Weighted Mean (g/t)	6.075	5.122	37.920	34.635
Weighted C.V.	16.29	4.53	5.93	3.98
Weighted Mean – Assays	6.072	5.118	37.9	34.6
Difference (1-assay/comp)%	0.1	0.1	0.0	0.0

Table 14-15: Composite Statistics – Big Missouri

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	6,028	6,028	6,028	6,028
No. Missing	0	0	0	0
Minimum	0.003	0.003	0	0
Maximum	536	200	1,860	1,000
Weighted Mean (g/t)	4.26	4.02	13.60	13.40
Weighted CV	3.61	2.58	3.20	2.80
Weighted Mean – Assays	4.24	4.00	13.40	13.20
Difference (1-assay/comp)%	0	0	1	1

Table 14-16: Composite Statistics – Silver Coin

Parameter	Within Wireframes			
	Au	Capped Au	Ag	Capped Ag
No. Samples	7,338	7,338	6,046	6,046
No. Missing	0	0	1,292	1,292
Minimum (g/t)	0.001	0.001	0	0
Maximum (g/t)	539.64	200	2453	600
Weighted Mean (g/t)	5.008	4.843	18.600	17.800
Weighted CV	2.87	2.26	3.10	2.20
Weighted Mean – Assays	5.008	4.843	18.6	17.7
Difference (1-assay/comp)%	0.0	0.0	0.0	0.6

Table 14-17: Composite Statistics – Dilworth

Parameter	Within Interpolated Domains			
	Au	Capped Au	Ag	Capped Ag
No. Samples	616	616	616	616
No. Missing	0	0	0	0
Minimum	0.003	0.003	0	0
Maximum	1597.65	70	5516	2828
Weighted Mean (g/t)	5.980	3.005	100.3	90.0
Weighted CV	10.95	2.88	3.80	3.10
Weighted Mean – Assays	6.012	3.015	99.2	88.9
Difference (1-assay/comp)	-0.5	-0.3	1.1	1.2

Table 14-18: Composite Statistics – Martha Ellen

Parameter	Within Interpolated Domains			
	Au	Capped Au	Ag	Capped Ag
No. Samples	1,171	1,171	1,171	1,171
No. Missing	0	0	0	0
Minimum	0.024	0.024	0	0
Maximum	140.503	70	964	766
Weighted Mean (g/t)	3.909	3.788	43.1	42.9
Weighted CV	1.88	1.50	1.90	1.80
Weighted Mean – Assays	3.911	3.790	43.1	42.9
Difference (1-assay/comp)	-0.1	-0.1	0.0	0.0

14.8 Premier Gold Project Outlier Restriction

Table 14-18 summarizes the Outlier Restriction values for Premier and Table 14-19 through Table 14-22 for the other deposits. At a distance greater than 2 m, the restricted value is used in the interpolation for the first two passes of the interpolation, and the value was not used at all for the 3rd and 4th passes.

Table 14-19: Outlier Restriction of Composites during Interpolation

Domain	Area	Outlier Value		Domain	Area	Outlier Value		Domain	Area	Outlier Value	
		Au (g/t)	Ag (g/t)			Au (g/t)	Ag (g/t)			Au (g/t)	Ag (g/t)
1	NW	22	9999	35	Obscene	50	1000	66	NL	30	500
2	NW	9999	9999	36	Obscene	100	100	67	NL	9999	9999
3	609	20	9999	37	Obscene	30	200	68	NL	9	9999
4	609	9999	9999	38	Obscene	9999	30	69	NL	5	9999
5	609	9999	100	39	Obscene	9999	9999	70	NL	7	300
6	609	30	200	40	Obscene	9999	9999	71	NL	9	80
7	609	9999	20	41	Obscene	20	1000	72	NL	9999	9999
8	609	20	9999	42	Obscene	9999	9999	73	NL	30	105
9	609	9999	100	43	Obscene	30	50	74	NL	30	9999
10	609	9999	50	44	Obscene	100	600	75	NL	10	9999
11	609	9999	9999	45	Obscene	9999	9999	76	NL	5	9999
12	609	30	9999	46	Obscene	9999	9999	77	NL	6	9999
13	609	20	9999	47	Obscene	9999	9999	78	NL	3	9999
14	LunchRm	20	70	48	Obscene	9999	100	79	NL	6	9999
15	LunchRm	80	1000	49	Obscene	9999	1000	83	Ben	15	400
16	LunchRm	1000	900	50	Obscene	9999	1000	84	Ben	40	100
17	LunchRm	40	200	51	Prew	35	120	85	Ben	7	9999
18	LunchRm	30	9999	52	Prew	20	80	86	Ben	9999	9999
19	LunchRm	50	3000	53	Prew	100	200	87	Ben	4	9999
20	LunchRm	9999	60	54	Prew	9999	9999	88	Ben	9999	300
21	LunchRm	9999	9999	55	Prew	40	500	89	Ben	3	80
22	Main	50	9999	56	Prew	4	100	90	Ben	5	9999
23	Main	9999	20	57	Prew	10	9999	91	Ben	9999	9999
24	Main	9999	9999	58	Prew	7	80	92	Ben	20	30
25	Main	9999	9999	59	Prew	8	50	93	Ben	12	80
26	Main	9999	900	60	Prew	5	75	100	602	60	9999
27	Main	9999	300	61	Prew	7	9999	101	602	9999	9999
28	Main	9999	9999	62	Prew	5	9999	102	602	9999	9999
29	Main	65	200	63	Prew	30	9999	104	602	9999	9999
30	Main	9999	100	64	Prew	6	9999	105	602	200	9999
31	Main	9999	3000	65	Prew	9999	9999	107	602	60	9999
32	Main	20	1500					108	609	10	9999
33	Main	10	20					109	Main/Obs	9999	9999
34	Main	9999	9999					110	Main/Obs	9999	9999

Table 14-20: Outlier Restriction – Big Missouri

ICODE	Outlier Value		ICODE	Outlier Value		ICODE	Outlier Value	
	Au (g/t)	Ag (g/t)		Au (g/t)	Ag (g/t)		Au (g/t)	Ag (g/t)
1011	100	400	129	30	400	158	50	400
1012	100	400	130	100	400	159	50	400
102	20	200	131	50	400	160	50	400
103	30	400	132	50	400	161	100	400
104	30	400	1331	50	400	162	50	400
105	50	400	1332	50	400	163	50	400
106	50	400	134	50	400	164	50	400
107	50	150	135	50	400	165	50	400
108	50	200	136	50	400	166	50	400
109	50	400	137	50	400	167	50	400
110	50	400	138	30	400	168	50	400
111	50	400	139	30	400	169	100	400
112	50	400	140	50	400	170	50	400
113	50	400	141	50	400	171	50	400
114	50	400	142	50	400	172	50	400
115	50	400	143	50	400	173	50	400
116	50	400	144	100	400	174	50	400
117	50	400	145	50	400	175	50	400
118	50	400	146	100	400	176	100	400
119	50	400	147	50	400	177	50	400
120	100	400	148	50	400	178	50	400
121	50	400	149	100	400	179	50	400
122	50	400	150	50	400	180	50	400
123	50	400	152	50	400	181	50	400
124	50	400	153	50	400	182	50	400
125	50	400	154	50	400	183	50	400
126	70	400	155	100	400	184	50	400
127	50	400	156	100	400			
128	50	400	157	100	400			

Table 14-21: Outlier Restriction – Silver Coin

ICODE	Outlier Value		ICODE	Outlier Value		ICODE	Outlier Value	
	Au (g/t)	Ag (g/t)		Au (g/t)	Ag (g/t)		Au (g/t)	Ag (g/t)
101	50	200	3001	50	200	5901	100	200
102	50	200	3101	50	200	6001	50	200
201	50	200	3201	50	200	6101	50	200
301	50	200	3301	50	200	6201	50	200
401	50	200	3302	50	200	6301	50	200
501	100	200	3303	50	200	6401	50	200
601	50	200	3401	50	200	6402	50	200
701	50	200	3402	50	200	6501	50	200
702	50	200	3501	100	500	6502	50	200
703	50	200	3601	50	200	6601	50	200
704	50	200	3701	50	200	6701	50	200
801	50	200	3801	50	200	6801	50	200
901	50	200	3901	50	200	6901	50	200
1001	50	200	4001	50	200	7001	50	200
1002	50	200	4101	50	300	7101	50	200
1101	50	200	4201	50	200	7301	50	200
1201	50	200	4301	50	200	7401	50	200
1301	50	200	4401	50	200	7501	100	200
1401	50	200	4501	50	200	7601	50	200
1501	50	200	4601	50	200	7701	50	200
1601	50	200	4701	50	200	7801	50	200
1701	50	200	4801	100	200	7901	100	200
1801	50	200	4901	50	200	8001	100	200
1901	50	200	5001	50	200	8101	50	200
2001	50	300	5101	50	300	8201	50	200
2101	50	200	5201	50	400	8301	50	200
2201	50	200	5202	50	400	8401	50	200
2301	50	200	5301	50	200	8501	50	200
2401	50	400	5401	100	200	8601	50	200
2501	50	500	5501	50	200	8701	50	200
2601	50	200	5502	50	200	8801	50	200
2701	50	200	5601	50	200	8901	50	200
2801	50	200	5701	50	200	9001	50	200
2901	50	200	5801	100	300	9101	50	200

Table 14-22: Outlier Restriction – Dilworth and Martha Ellen

Deposit	ICODE	Outlier Value	
		Au (g/t)	Ag (g/t)
Martha Ellen	All	30	300
Dilworth	1103	6	300
	1900	4	300
	All others	30	300

14.9 Premier Gold Project Density Assignment

Model blocks were assigned the mean density value of 2.85 for the Premier deposit and 2.80 for all other deposit within the PGP. A summary of specific gravity (SG) sampling and results is presented in Section 12 of this report.

14.10 Premier Gold Project Block Model Interpolations

Block dimensions are 3 m x 3 m x 3 m. The block model is defined as a Multiple Percent Model, with up to two mineralized zones per block associated with block percent items.

Variogram modelling was not very effective at defining anisotropy due to varying orientations of the mineralized zones across each deposit, and to the multiple stacked lens nature of the mineralization. There are generally too few data pairs in each domain, while downhole variograms are generally across the zone and therefore do not provide data along strike and down-dip of mineralization. Therefore, the orientation of anisotropy has been obtained from the orientation of the domain itself. In some cases, the mineralized domain solids have been further subdivided based on the strike and dip of the solid. In these cases, sharing of samples across the subdivided domains has been allowed during interpolation. Figure 14-12 through Figure 14-16 illustrate the domain solids and corresponding search ellipses used in interpolation for each of the PGP deposits; Premier, Big Missouri, Silver Coin, Martha Ellen, and Dilworth.

Search parameter orientations varied based on the vein orientations as summarized in Table 14-22 through Table 14-26 for each deposit. The rotation values R1, R2, and R3 are the rotation of the principal axes about the Y-axis, X-axis, and Z-axis, respectively, using the right-hand rule with positive rotation upwards.

Interpolation has been done using inverse distance cubed (ID3) in all cases. The restrictions on search distances and composite selection for each of the five passes of the interpolations are given in Table 14-27 through Table 14-29 for Premier, Big Missouri, and Silver Coin and in Table 14-30 for Martha Ellen and Dilworth. It should be noted that no new drilling was completed on Martha Ellen and Dilworth since the previous NI 43-101 Resource Estimate (Rennie, et al., 2019) and therefore the interpolations remain the same for these two deposits.

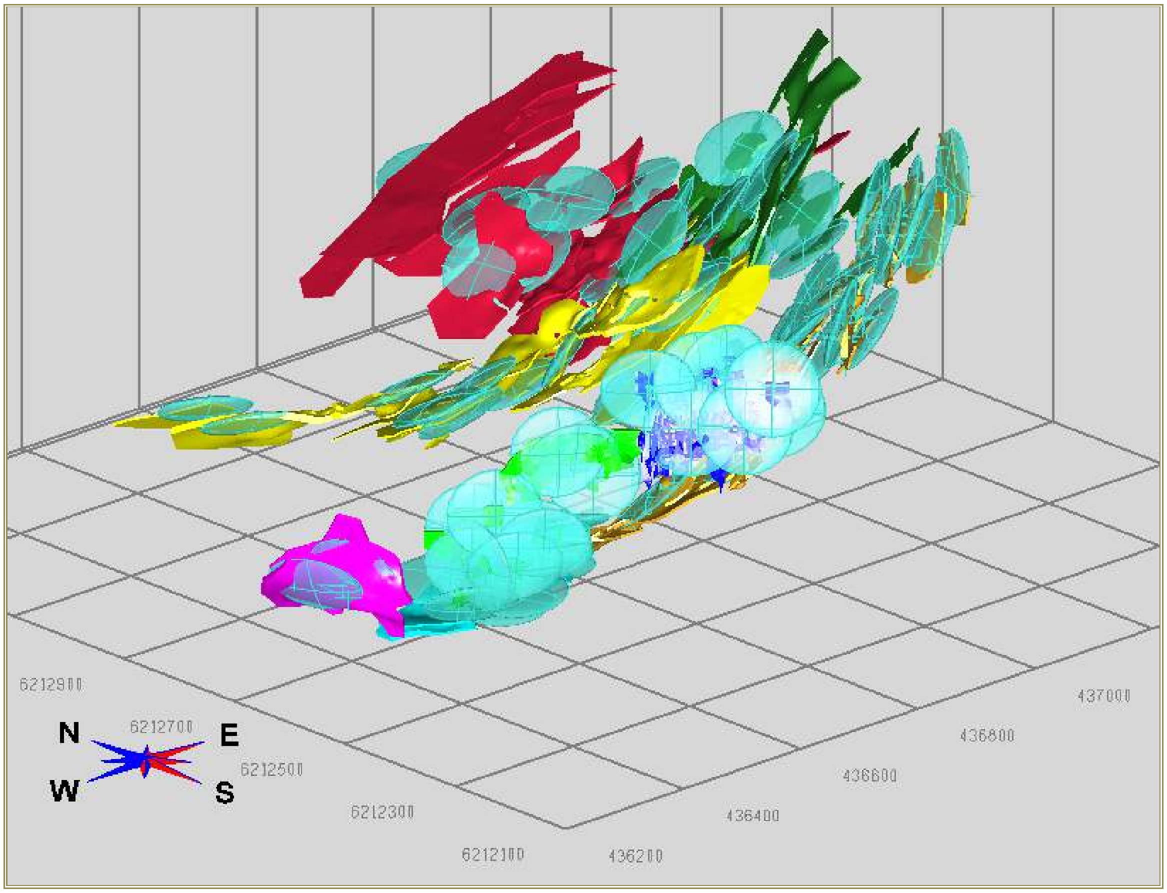


Figure 14-12: 3-D View Looking Northeast of Mineralized Domains and Search Ellipses (cyan) – Premier

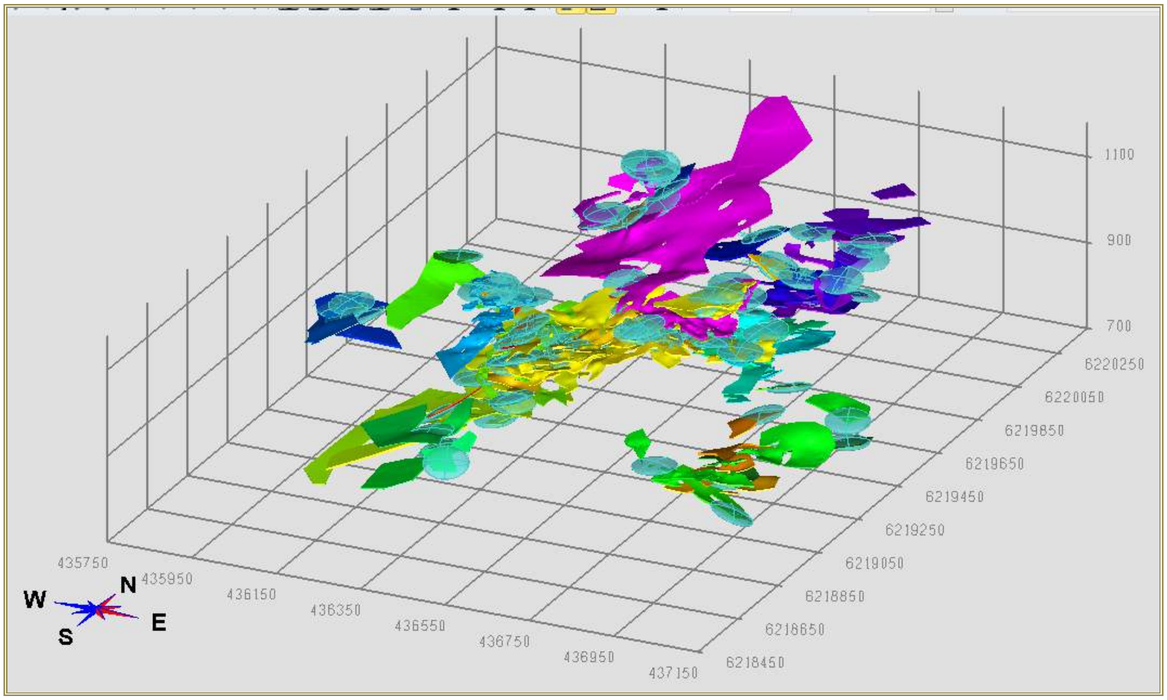


Figure 14-13: 3-D View Looking North of Mineralized Domains and Search Ellipses (cyan) – Big Missouri

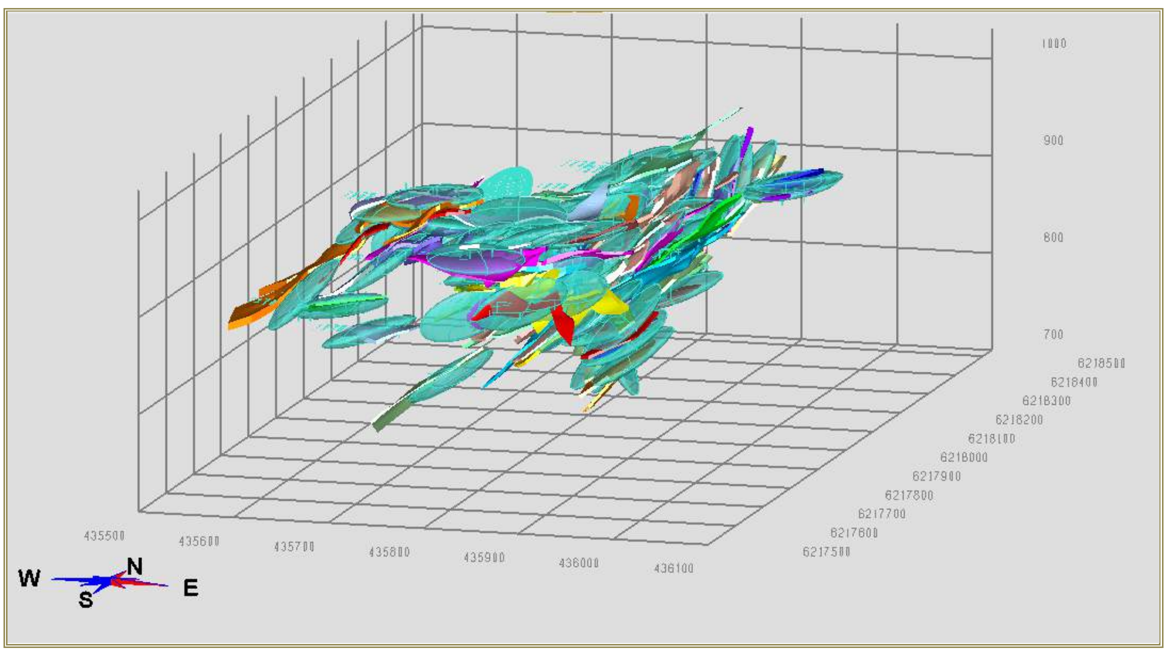


Figure 14-14: 3-D View Looking North of Mineralized Domains and Search Ellipses (cyan) – Silver Coin

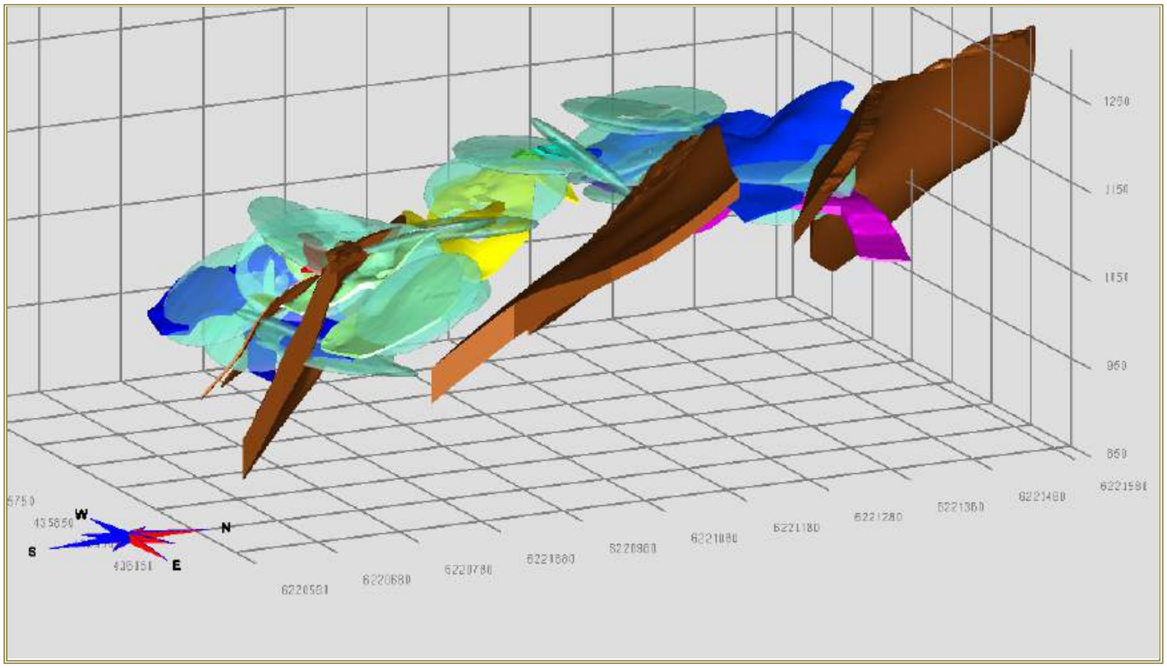


Figure 14-15: 3-D View Looking Northwest of Mineralized Domains, dykes (brown) and Search Ellipses (cyan) – Martha Ellen

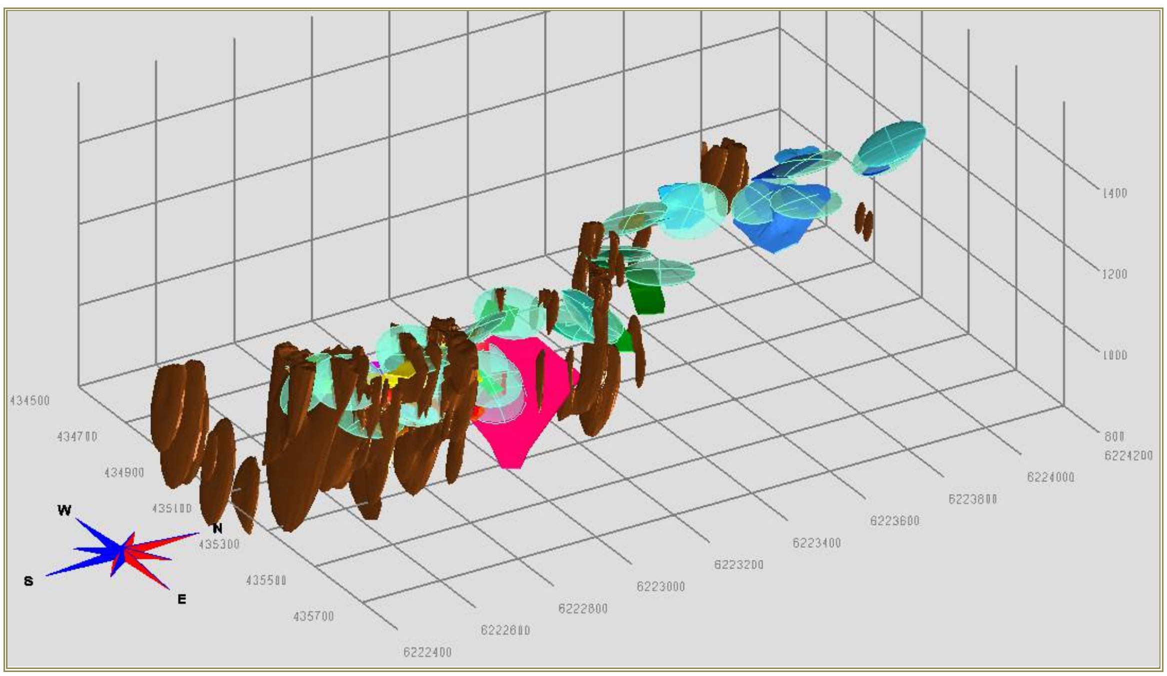


Figure 14-16: 3-D View Looking Northwest of Mineralized Domains, Dykes (brown) and Search Ellipses (cyan) – Dilworth

Table 14-23: Domain Orientations—Premier

ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3
1	330	0	-50	34	210	0	-35	67	270	0	-44
2	340	0	-65	35	250	0	-52	68	250	0	-25
3	335	0	-70	36	200	0	-35	69	210	0	-38
4	303	0	-40	37	225	0	-68	70	210	0	-40
5	315	0	-70	38	220	0	-70	71	178	0	-38
6	300	0	-65	39	220	0	-65	72	304	0	-33
7	130	0	-85	40	205	0	-85	73	295	0	-30
8	310	0	-83	41	240	0	-80	74	260	0	-28
9	300	0	-68	42	240	0	-75	75	245	0	-38
10	305	0	-65	43	240	0	-60	76	218	0	-23
11	305	0	-85	44	225	0	-57	77	240	0	-60
12	305	0	-65	45	215	0	-63	78	255	0	-60
13	305	0	-90	46	223	0	-83	79	235	0	-30
14	130	0	75	47	230	0	-80	83	225	0	-55
15	110	0	-78	48	215	0	-65	84	222	0	-56
16	135	0	-75	49	215	0	-60	85	195	0	-57
17	120	0	75	50	205	0	-65	86	236	0	-67
18	108	0	85	51	200	0	-25	87	177	0	-33
19	120	0	83	52	200	0	-25	88	228	0	-35
20	140	0	-75	53	215	0	-30	89	198	0	-53
21	135	0	-70	54	195	0	-40	90	198	0	-51
22	40	0	50	55	205	0	-40	91	215	0	-61
23	60	0	85	56	160	0	-22	92	230	0	-55
24	55	0	40	57	210	0	-35	93	230	0	-53
25	32	0	40	58	220	0	-40	100	160	0	-10
26	230	0	-55	59	207	0	-28	101	160	0	-10
27	230	0	-40	60	210	0	-45	102	165	0	-10
28	235	0	-40	61	193	0	-36	104	290	0	-28
29	220	0	-55	62	195	0	-50	105	237	0	-33
30	220	0	-55	63	170	0	-45	107	311	0	-40
31	235	0	-75	64	190	0	-20	108	0	0	-85
32	220	0	-75	65	190	0	-25	109	30	0	70
33	210	0	-70	66	235	0	-35	110	30	0	70

Table 14-24: Domain Orientations—Big Missouri

ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3
1011	310	0	20	129	285	0	5	158	205	0	-15
1012	110	0	-5	130	305	0	5	159	215	0	-15
102	315	0	30	131	305	0	5	160	165	0	-20
103	130	0	-10	132	305	0	5	161	185	0	-3
104	15	0	17	1331	290	0	7	162	240	0	-10
105	90	0	-30	1332	275	0	-20	163	315	0	5
106	130	0	-5	134	270	0	3	164	330	0	5
107	60	0	-5	135	250	0	23	165	245	0	10
108	60	0	-10	136	355	0	10	166	30	0	-7
109	80	0	-5	137	340	0	15	167	312	0	15
110	45	0	-10	138	250	0	10	168	312	0	15
111	100	0	10	139	195	0	-2	169	10	0	-25
112	50	0	5	140	150	0	25	170	280	0	20
113	120	0	5	141	70	0	10	171	315	0	-15
114	188	0	9	142	30	0	-23	172	60	0	5
115	130	0	10	143	50	0	20	173	45	0	15
116	87	0	-13	144	50	0	-10	174	345	0	15
117	105	0	12	145	40	0	5	175	230	0	-3
118	110	0	10	146	120	0	-10	176	162	0	-27
119	170	0	10	147	145	0	-10	177	170	0	-5
120	185	0	-20	148	95	0	-7	178	160	0	-25
121	170	0	-30	149	130	0	-8	179	195	0	-36
122	120	0	3	150	75	0	-10	180	30	0	-5
123	0	0	30	152	95	0	-5	181	110	0	-7
124	350	0	22	153	90	0	-15	182	220	0	-6
125	295	0	9	154	115	0	-10	183	150	0	-15
126	315	0	5	155	80	0	-2	184	200	0	-18
127	260	0	8	156	105	0	5				
128	265	0	15	157	85	0	-15				

Table 14-25: Domain Orientations—Silver Coin

ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3
101	45	0	43	3001	325	0	20	5901	280	0	5
102	60	0	-5	3101	350	0	38	6001	320	0	40
201	40	0	33	3201	255	0	10	6101	330	0	20
301	50	0	40	3301	340	0	20	6201	65	0	10
401	45	0	40	3302	30	0	23	6301	0	0	65
501	25	0	20	3303	30	0	23	6401	150	0	80
601	40	0	17	3401	350	0	25	6402	30	0	85
701	0	0	25	3402	26	0	57	6501	35	0	50
702	100	0	15	3501	345	0	33	6502	355	0	85
703	20	0	5	3601	355	0	7	6601	5	0	45
704	352	0	36	3701	340	0	60	6701	350	0	10
801	0	0	12	3801	330	0	75	6801	350	0	7
901	345	0	30	3901	355	0	65	6901	40	0	37
1001	345	0	45	4001	332	0	80	7001	20	0	10
1002	20	0	5	4101	345	0	80	7101	305	0	25
1101	5	0	5	4201	335	0	75	7301	295	0	10
1201	15	0	20	4301	345	0	72	7401	328	0	15
1301	38	0	20	4401	20	0	70	7501	315	0	10
1401	350	0	35	4501	345	0	75	7601	325	0	25
1501	12	0	35	4601	15	0	50	7701	345	0	25
1601	15	0	22	4701	12	0	65	7801	140	0	75
1701	55	0	15	4801	348	0	68	7901	0	0	15
1801	0	0	38	4901	356	0	48	8001	0	0	15
1901	335	0	35	5001	350	0	20	8101	358	0	40
2001	355	0	40	5101	330	0	70	8201	0	0	60
2101	350	0	50	5201	330	0	35	8301	330	0	30
2201	350	0	10	5202	350	0	58	8401	350	0	40
2301	40	0	13	5301	305	0	45	8501	50	0	20
2401	355	0	32	5401	320	0	45	8601	329	0	41
2501	355	0	30	5501	150	0	70	8701	35	0	38
2601	10	0	52	5502	350	0	-67	8801	355	0	30
2701	35	0	47	5601	340	0	45	8901	10	0	55
2801	355	0	38	5701	180	0	10	9001	43	0	45
2901	345	0	35	5801	355	0	15	9101	6	0	58

Table 14-26: Domain Orientations—Martha Ellen

ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3
230	40	0	0	303	80	0	-35
231	35	0	-25	310	132	0	30
232	130	0	-20	320	75	0	-35
240	170	0	-5	321	40	0	-37
250	40	0	-15	340	80	0	-5
260	95	0	40	341	80	0	10
270	100	0	5	350	5	0	-15
280	40	0	-20	351	320	0	-15
290	190	0	-10	352	55	0	-27
300	235	0	30	360	100	0	10
301	345	0	23	361	80	0	25
302	280	0	-5	370	65	0	17

Table 14-27: Domain Orientations—Dilworth

ICODE	ROT1	ROT2	ROT3	ICODE	ROT1	ROT2	ROT3
100	290	0	23	1103	280	0	45
200	290	0	23	100	15	0	-25
300	340	0	20	200	15	0	-25
400	270	0	20	300	335	0	10
401	270	0	0	400	320	0	35
500	237	0	25	500	320	0	20
700	305	0	-40	600	340	0	-10
800	325	0	-12	700	340	0	-33
900	210	0	-15	1801	206	0	33
901	210	0	0	800	240	0	38
1000	5	0	-20	900	185	0	30
1100	5	0	-25	1000	270	0	-20
1101	340	0	-41	1100	325	0	-30
1102	23	0	-23				

Table 14-28: Search Distances and Sample Selection—Premier

Search Parameter	PASS				
	1	2	3	4	5
DIST – Y	20	30	50	60	100
DIST – X	20	30	50	60	100
DIST – Z	5	5	10	15	15
Minimum No. Comps	6	6	6	2	1
Maximum No. Comps	12	12	12	12	16
Maximum / DH	3	3	3	4	4
Quadrant Restriction	Split	Split	none	Split	none
Maximum/Quad	6	6	12	6	16

Table 14-29: Search Distances and Sample Selection—Big Missouri

Search Parameter	PASS				
	1	2	3	4	5
DIST – Y	20	30	50	50	100
DIST – X	20	30	50	50	100
DIST – Z	5	5	10	10	15
Minimum No. Comps	6	6	6	2	1
Maximum No. Comps	12	12	12	6	16
Maximum / DH	3	3	3	2	5
Quadrant Restriction	Split	Split	none	Split	none
Maximum/Quad	6	6	12	6	16

Table 14-30: Search Distances and Sample Selection—Silver Coin

Search Parameter	PASS				
	1	2	3	4	5
DIST – Y	20	30	50	60	100
DIST – X	20	30	50	60	100
DIST – Z	5	5	10	15	15
Minimum No. Comps	6	6	6	2	1
Maximum No. Comps	12	12	12	12	16
Maximum / DH	3	3	3	4	4
Quadrant Restriction	Split	Split	none	Split	none
Maximum/Quad	6	6	12	6	16

Table 14-31: Search Distances and Sample Selection—Dilworth and Martha Ellen

	PASS			
	1	2	3	4
DIST - Y – AU	20	30	50	80
DIST - X – AU	20	30	50	80
DIST - Z – AU	5	5	10	10
DIST - Y – AG	10	20	30	80
DIST - X – AG	10	20	30	80
DIST - Z – AG	5	5	10	10
Minimum No. Comps	6	6	6	2
Maximum No. Comps	12	12	12	6
Maximum / DH	3	3	3	2
Quadrant Restriction	Split	Split	Split	Split
Maximum / Split Quadrant	6	6	6	6

14.11 Premier Gold Project Block Model Validation

A Nearest Neighbour model (NN model) has been created in each deposit area in order to compare the ID3 modelled grades with the de-clustered composite grades. The NN model has been created using composites of 3 m intervals, which is approximately the minimum mining width, and using the uncapped values are used in the comparison.

14.11.1 Global Bias Check

A comparison of global mean values with the de-clustered composite data for each deposit area is provided in Table 14-32 through Table 14-36. The tables indicate good agreement for each deposit with the de-clustered composite data in all cases. The generally lower mean grades (at zero cut-off) for all deposits is due to the inherent model smoothing introduced during interpolation as well as the capping and outlier restriction applied to the modelled grades, which was not applied to the de-clustered composite (NN model) data. Grade-Tonnage curves plotted in the next section on validation provide a better comparison of the mean grade distribution throughout a range of cut-off grades.

The slightly higher silver grade for the modelled Martha Ellen deposit in the Inferred category is immaterial because the value of the silver in this deposit is less than 4% of the value of the Au Equivalence used for the resource. The lower grade of gold for Dilworth for Inferred values (all blocks in Dilworth are classed as Inferred) is due to lack of drill density in some domains.

Table 14-32: Global Mean Grade Comparison—Premier

		AU	AUNN	AG	AGNN	Difference (%)	
						1-AUNN/AU	1-AGNN/AG%
MI	Number Samples	83,283	83,283	83,283	83,283	-	-
	Number Missing	0	0	0	0	-	-
	Minimum	0.00	0.00	0	0	-	-
	Maximum	504.35	559.16	4584	8187	-	-
	Mean	4.27	5.50	35.5	39.6	-28.9	-11.5
MII	Number Samples	23,8736	23,8736	23,8736	23,8736	-	-
	Number Missing	0	0	0	0	-	-
	Minimum	0.00	0.00	0	0	-	-
	Maximum	504.35	559.16	4,584	8,187	-	-
	Mean	3.74	4.85	27.6	27.2	-30	1

Table 14-33: Global Mean Grade Comparison—Big Missouri

	Parameter	AU	AUNN	AG	AGNN	Difference (%)	
						1-AUNN/AU	1-AGNN/AG%
MI	Number Samples	85,459	85,459	85,459	85,459	-	-
	Number Missing Samples	0	0	0	0	-	-
	Minimum	0.02	0.00	0	0	-	-
	Maximum	192.60	536.00	486	857	-	-
	Weighted Mean	4.11	4.72	12.6	13.2	-14.8	-4.8
MII	Number Samples	25,9947	25,9947	25,9947	25,9947	-	-
	Number Missing Samples	0	0	0	0	-	-
	Minimum	0.01	0.00	0	0	-	-
	Maximum	192.60	536.00	486	857	-	-
	Weighted Mean	3.89	5.00	12.4	12.3	-29	1

Table 14-34: Global Mean Grade Comparison—Silver Coin

	Parameter	AU	AUNN	AG	AGNN	Difference (%)	
						1-AUNN/AU	1-AGNN/AG%
MI	Number Samples	100,136	100,136	100,136	100,136	-	-
	Number Missing Samples	0	0	0	0	-	-
	Minimum	0.02	0.01	0	0	-	-
	Maximum	126.59	232.52	496	955	-	-
	Mean	4.25	4.58	15.8	17.6	-7.6	-11.4
MII	Number Samples	142,643	142,643	142,643	142,643	-	-
	Number Missing Samples	0	0	0	0	-	-
	Minimum	0.02	0.00	0	0	-	-
	Maximum	126.59	232.52	496	955	-	-
	Mean	4.16	4.41	16	18.3	-5.9	-14.4

Table 14-35: Global Mean Grade Comparison—Martha Ellen

	Parameter	AU	AUNN	AG	AGNN	1-AUNN/AU	1-AGNN/AG%
MI	Number Samples	8,275	8,275	8,275	8,275	-	-
	Number Missing Samples	1	1	1	1	-	-
	Minimum	0.106	0.012	0	0	-	-
	Maximum	55.118	84.617	407	546	-	-
	Mean	3.1052	3.3645	34.2	35.1	-8	-3
MII	Number Samples	45,735	45,735	45,735	45,735	-	-
	Number Missing Samples	1	1	1	1	-	-
	Minimum	0.057	0.004	0	0	-	-
	Maximum	55.118	84.617	453	546	-	-
	Mean	3.7329	3.5288	28.3	25.5	5	10

Table 14-36: Global Mean Grade Comparison—Dilworth

						Difference (%)	
	Parameter	AU	AUNN	AG	AGNN	1-AUNN/AU	1-AGNN/AG%
MII	Number Samples	24,667	24,588	24,420	23,789	-	-
	Number Missing Samples	37	116	284	915	-	-
	Minimum	0.06	0.115	0	1	-	-
	Maximum	69.701	54.563	1,316	1,892	-	-
	Mean	2.7515	3.2992	42.8	41.1	-20	4

14.11.2 Grade—Tonnage Curves

Grade-tonnage curves have been created to be sure that the amount of internal model “smoothing”, or dilution of grades is appropriate; these are presented below in Figure 14-17 through Figure 14-19. The final modelled grades are compared to the de-clustered composites NN models, both with and without Outlier Restrictions applied during interpolations. For each deposit, the final modelled grades indicate slightly higher tonnages and lower grades than the NN with Outlier Restrictions, indicating that interpolations are not optimistic, but rather have some conservatism built in; nevertheless, final diluted grades must be determined during mine planning.

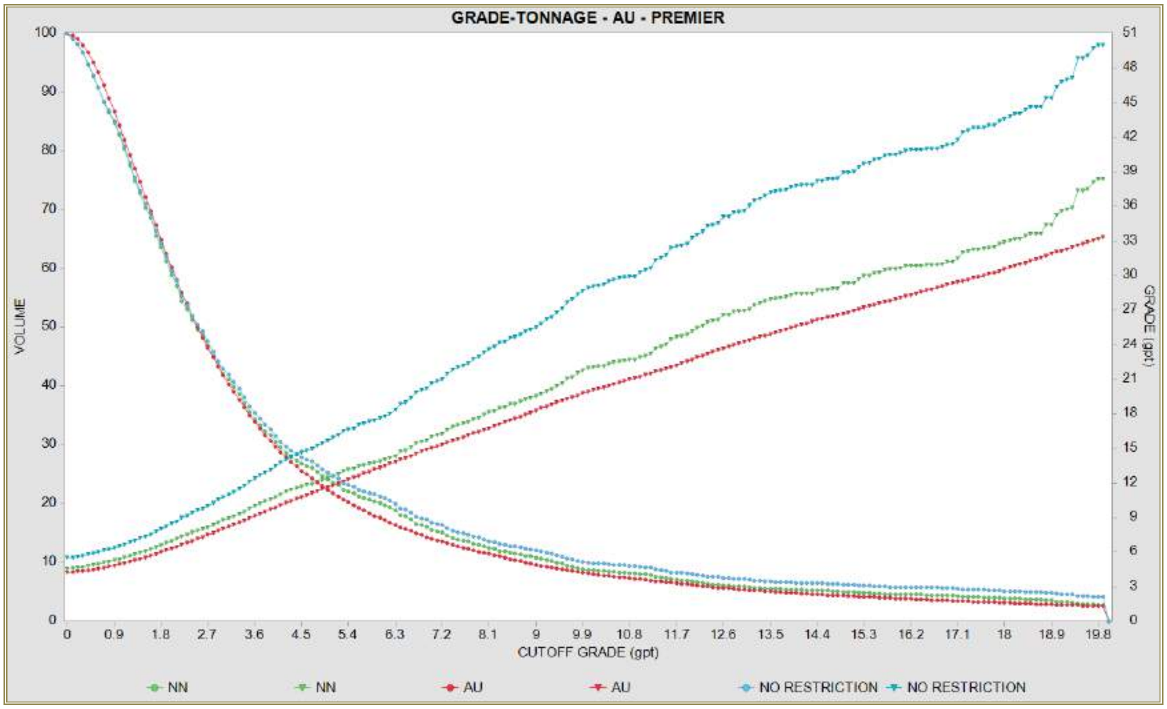


Figure 14-17: Grade-Tonnage Curve Comparison – Premier

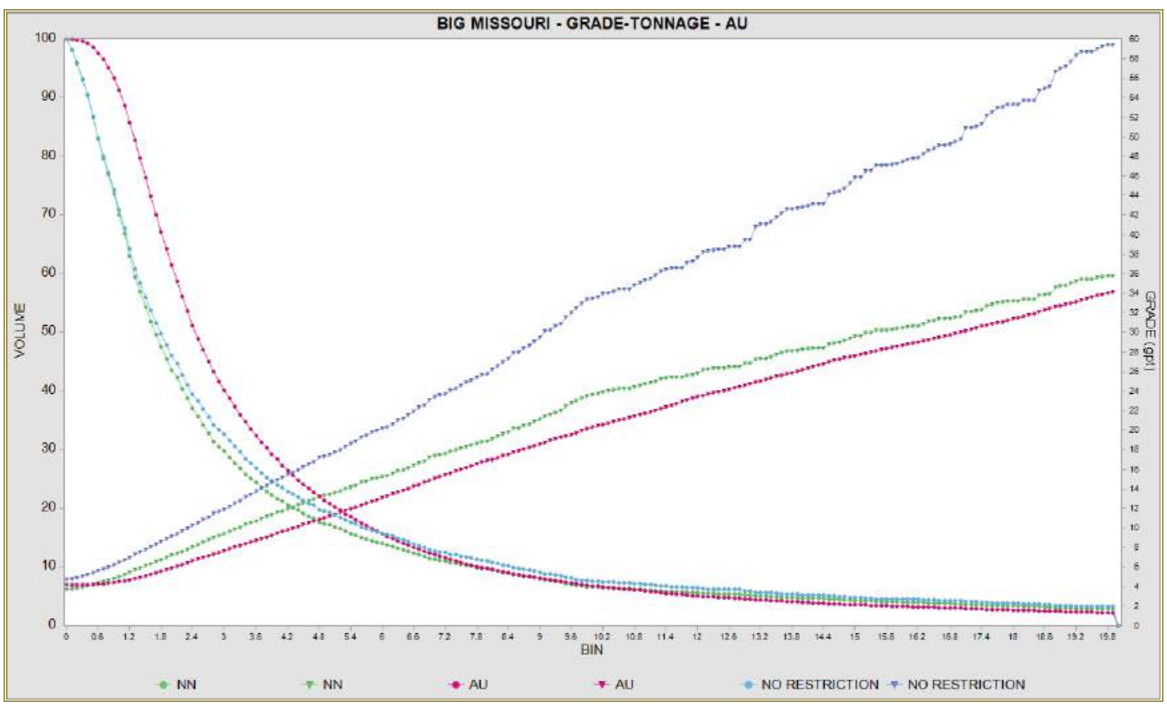


Figure 14-18: Grade-Tonnage Curve Comparison – Big Missouri

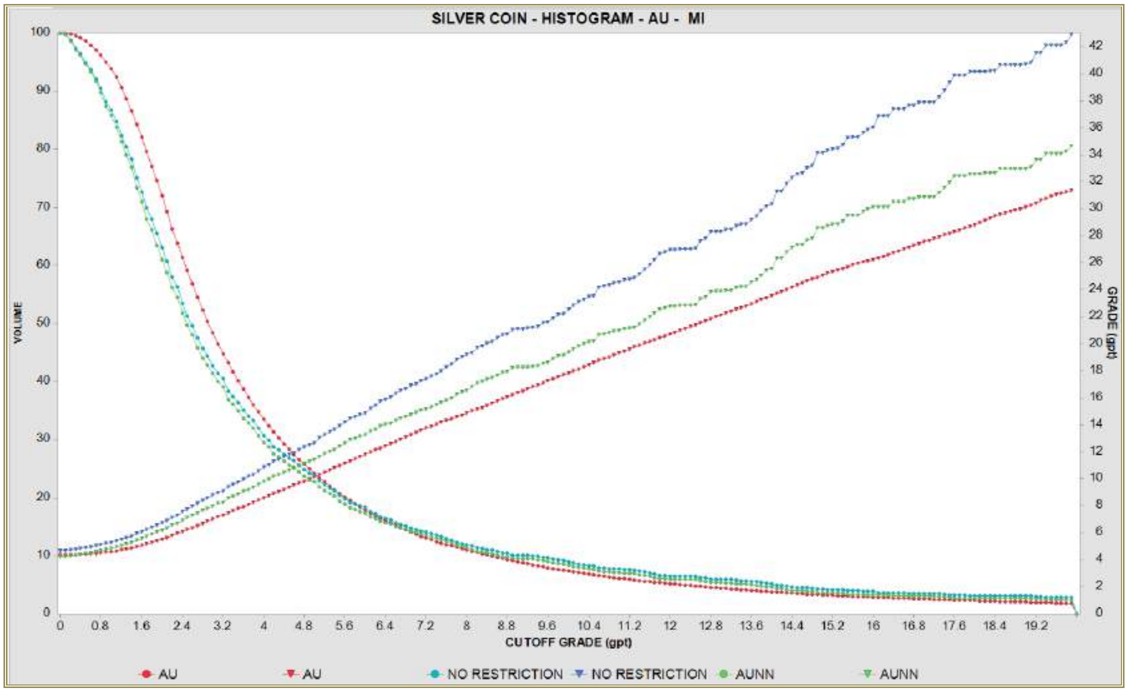


Figure 14-19: Grade-Tonnage Curve Comparison – Silver Coin

14.11.3 Swath Plots

Swath plots were generated to assess the model for local bias by comparing the ID3 and NN estimates on panels through the deposits. The results show good comparison between the methods, with the final model grades generally just below the de-clustered (NN) composite grades, particularly for the areas of the model with significant tonnage. Examples are presented in Appendix B.

14.11.4 Visual Inspection

Model verification was initially carried out by visual comparison of blocks and sample grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent assay and composite grades. Block model Au grade distribution is illustrated in Figure 14-20 to Figure 14-30.

Drill hole traces display the original assay grades which plot the Au or Ag grade using the same grade cut-offs as the blocks. The plots illustrate:

- The wireframes, labelled by area for Premier
- The block grades scaled by total percent of the block within the wireframe
- The assay traces and values as histograms with the maximum grade scaled to 30 g/t at a 10 m histogram width
- Assays are shown for ±10 m from the section
- Underground drifts, raises, and stopes (shown in black) to illustrate that no wireframes are within 10 m of a known underground opening.

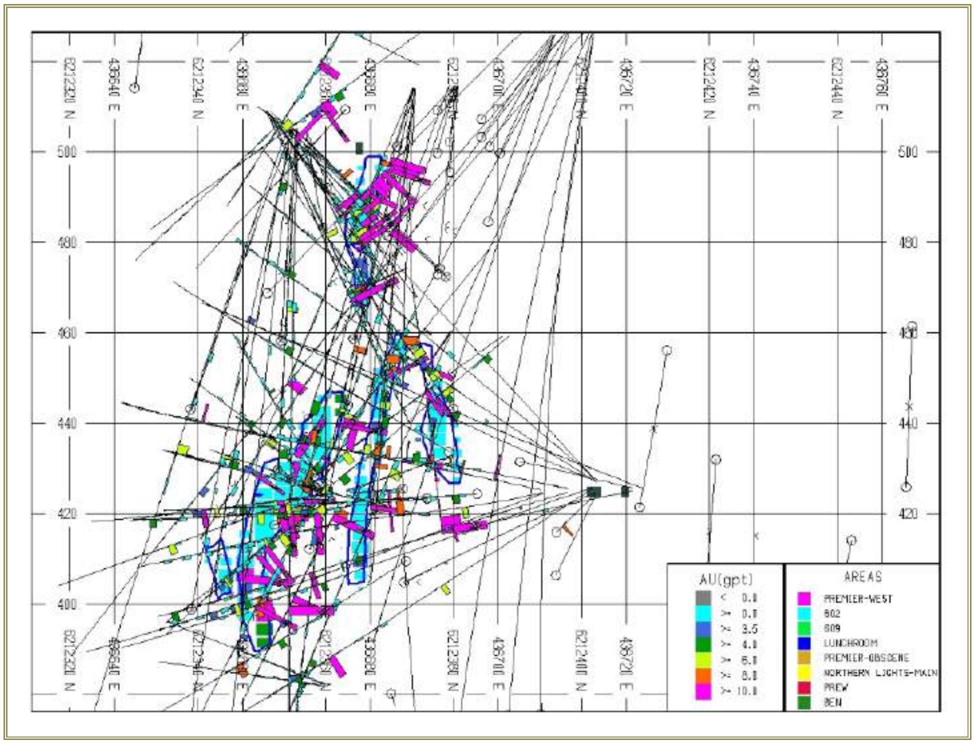


Figure 14-20: Block Model vs. Assay Grade – Premier – Section B-B' – Lunchroom Area – Gold

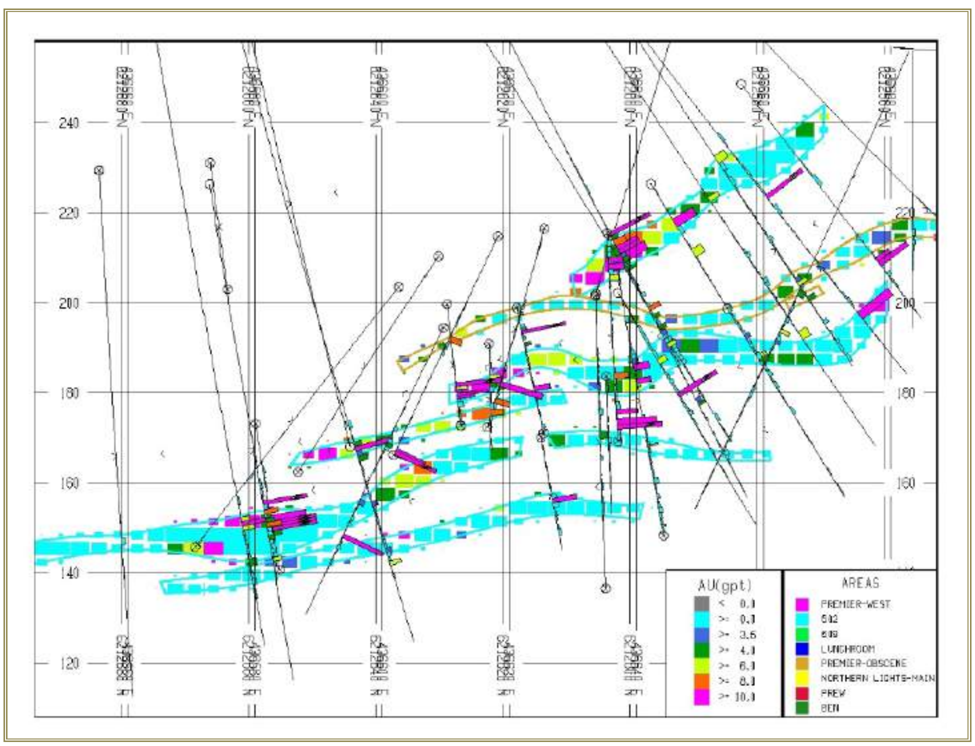


Figure 14-21: Block Model vs. Assay Grade – Premier – Section A-A' – 602 Area – Gold

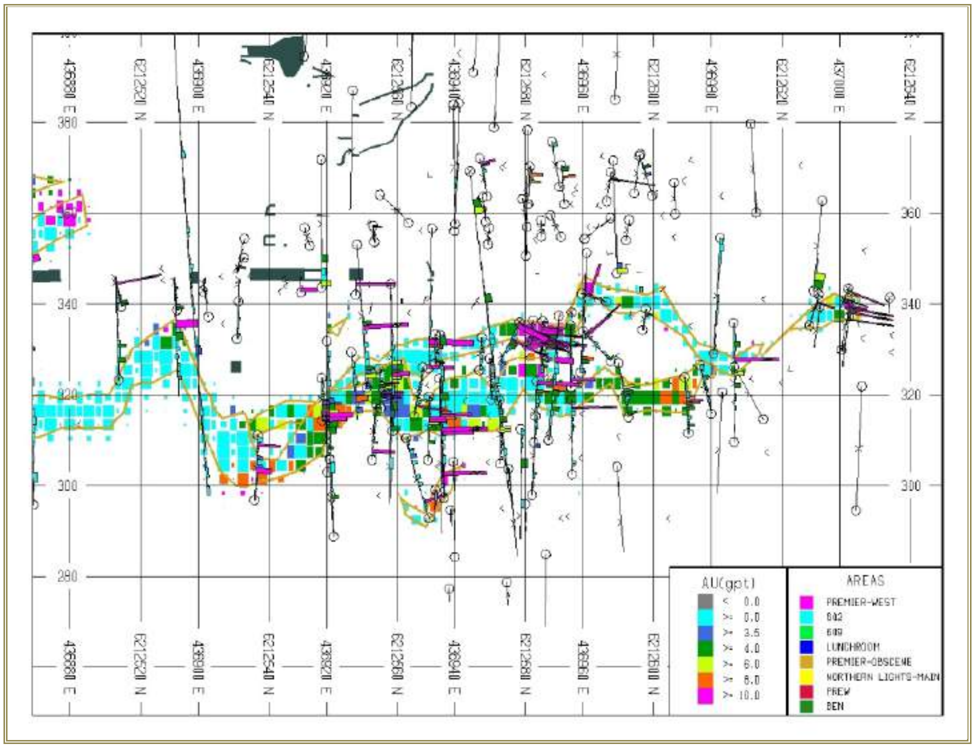


Figure 14-22: Block Model vs. Assay Grade – Premier – Section B-B' – Obscene Area – Gold

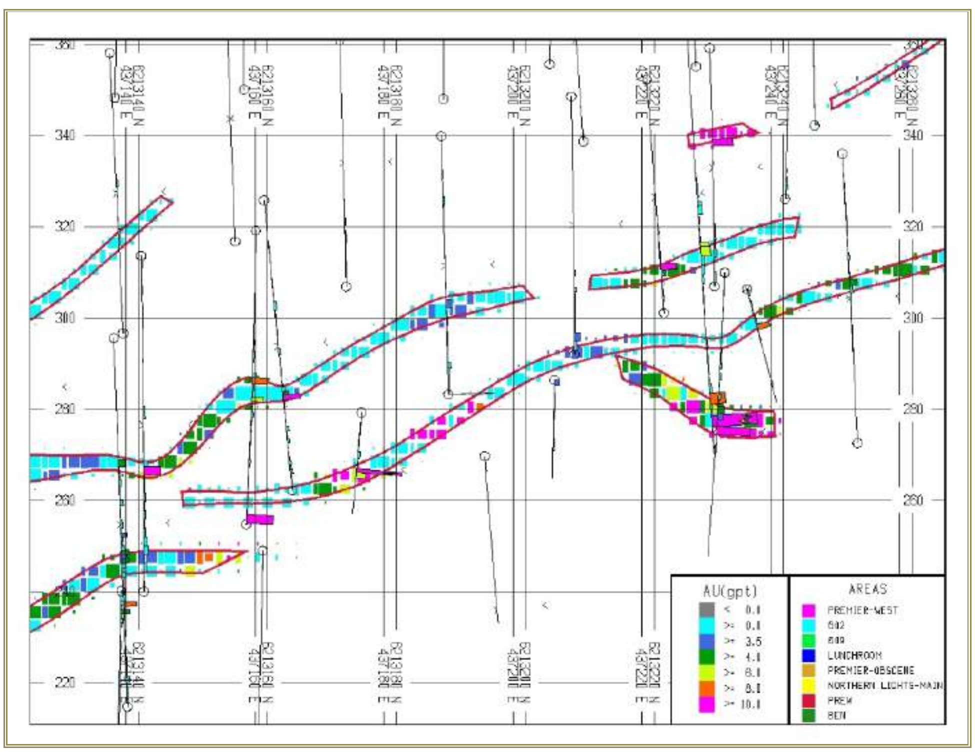


Figure 14-23: Block Model vs. Assay Grade – Premier – Section F-F' – Prew Area – Gold

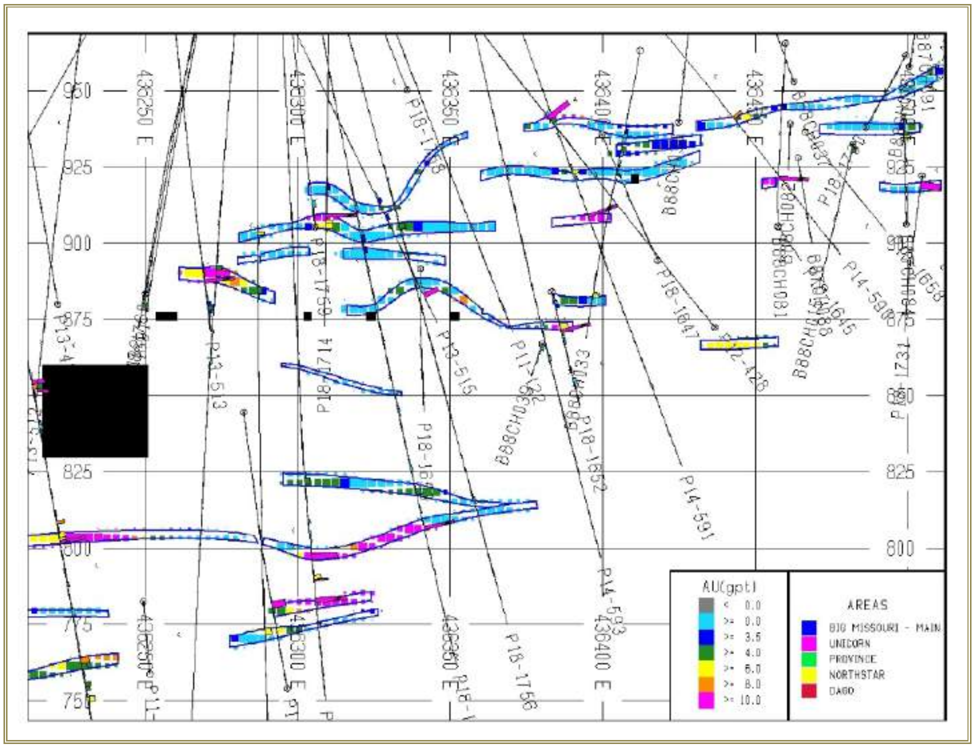


Figure 14-24: Block Model vs. Assay Grade – Big Missouri – Section 6219251 – Main Area – Gold

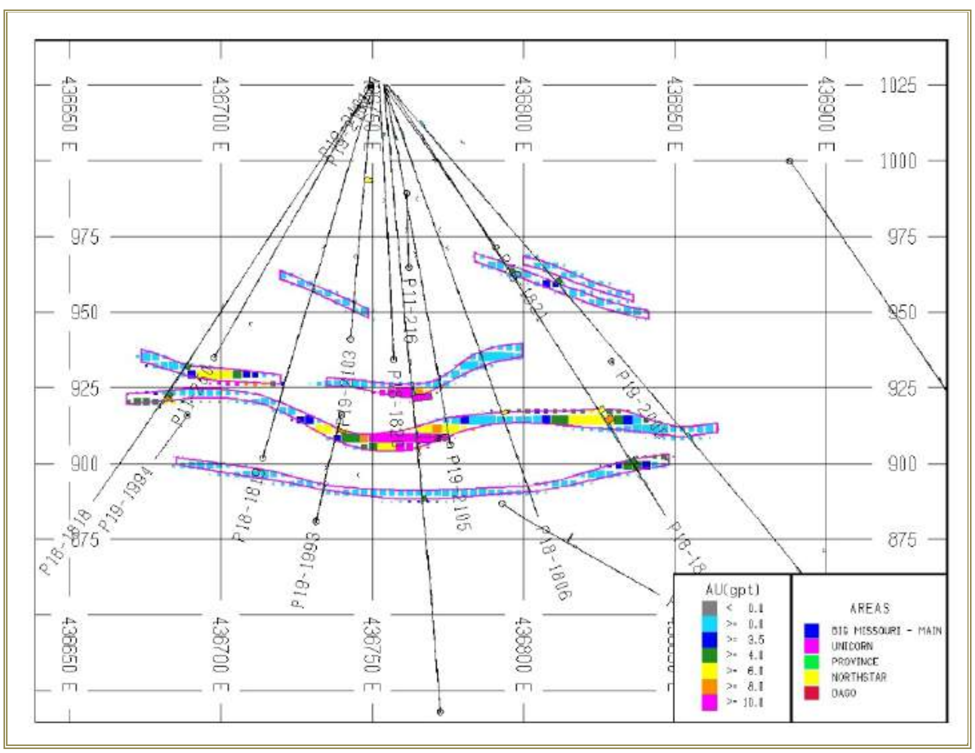


Figure 14-25: Block Model vs. Assay Grade – Big Missouri – Section 6219836 – Unicorn – Gold

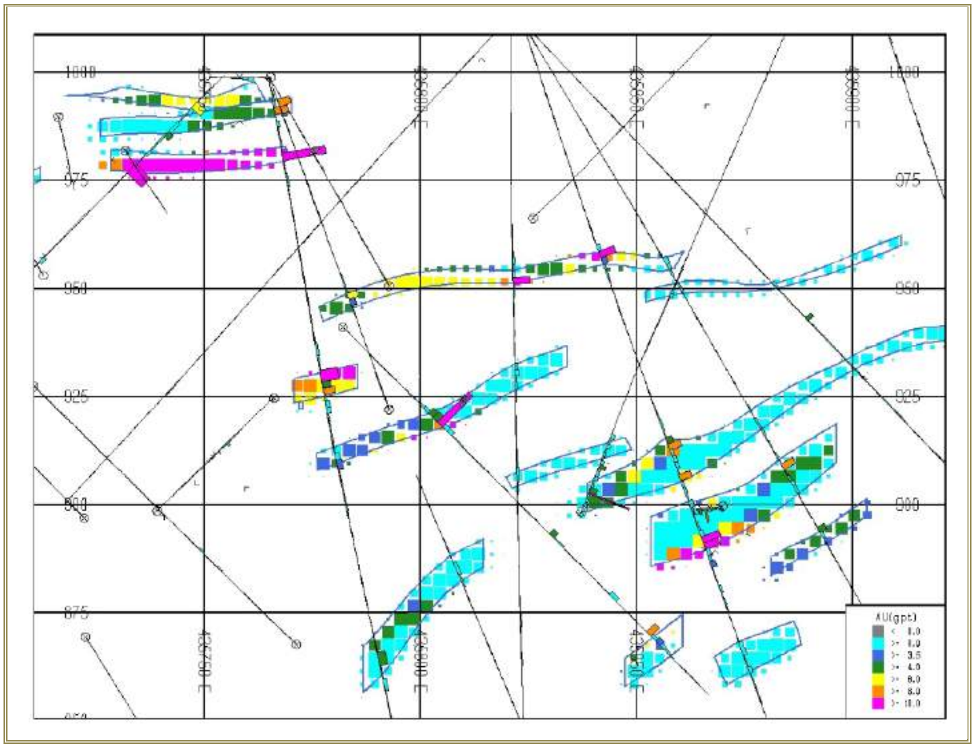


Figure 14-26: Block Model vs. Assay Grade – Silver Coin – Section 6217746N – Gold

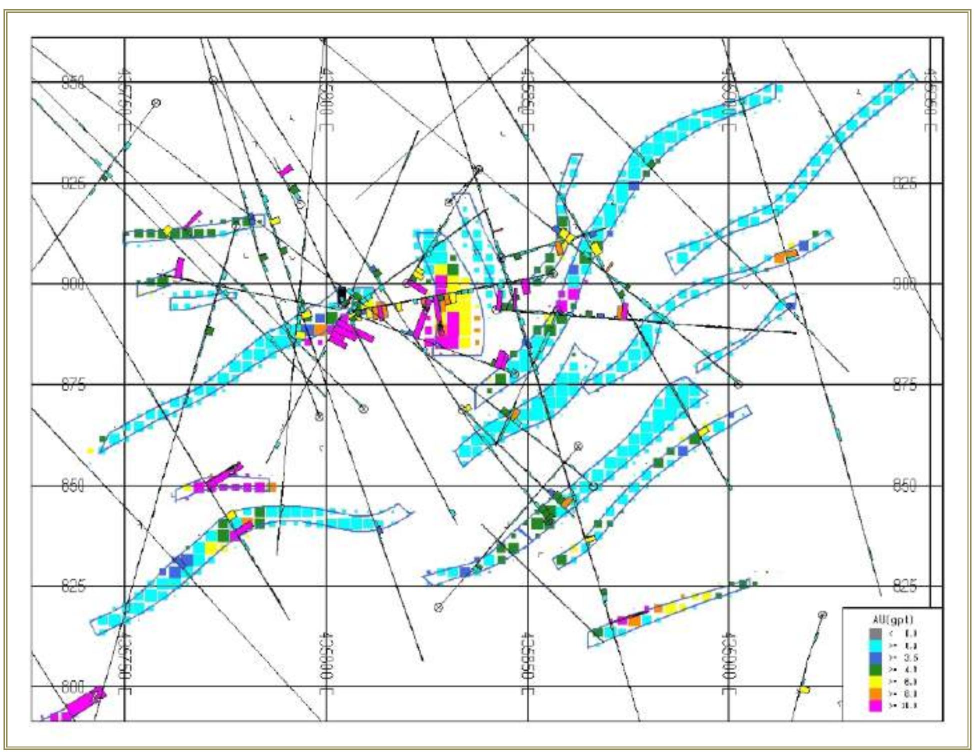


Figure 14-27: Block Model vs. Assay Grade – Silver Coin – Section 6217866N – Gold

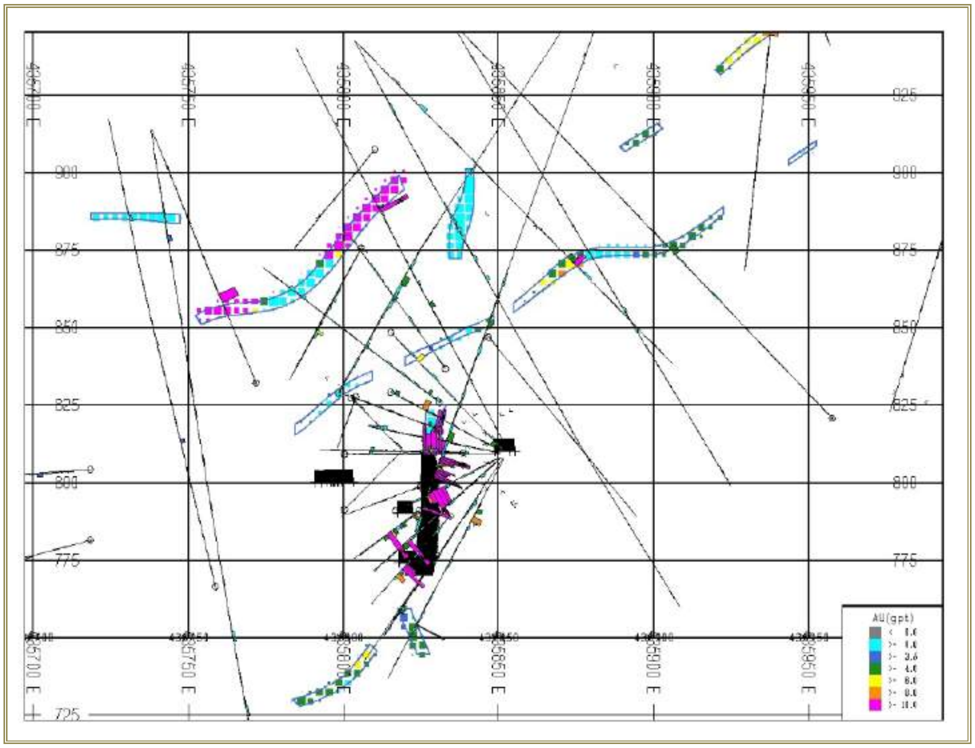


Figure 14-28: Block Model vs. Assay Grade – Silver Coin – Section 6218142N – Gold

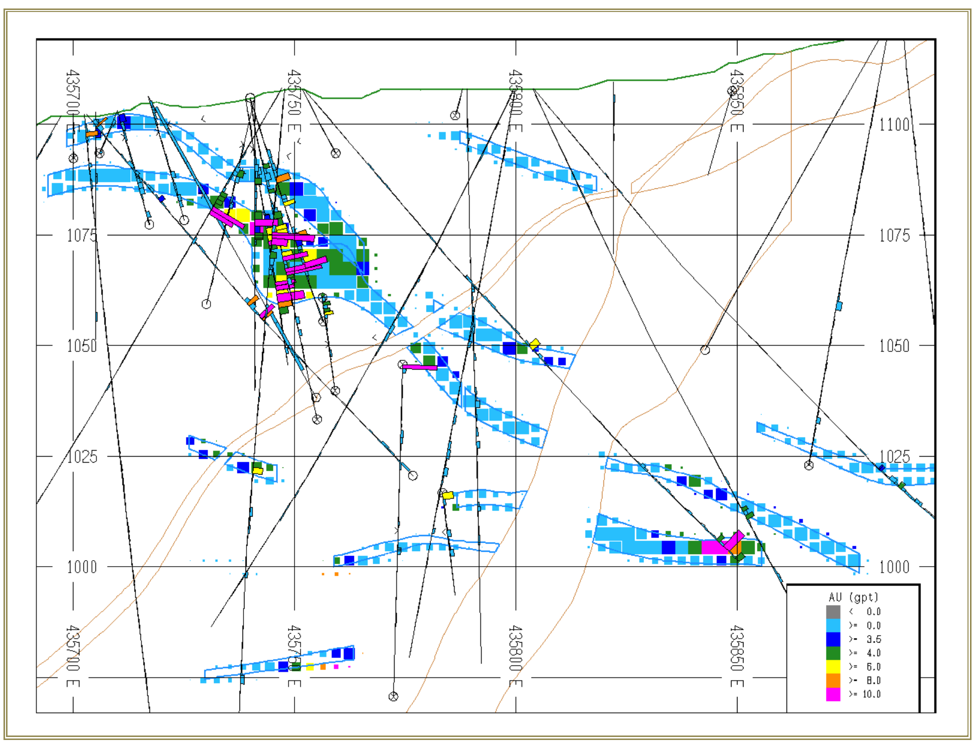


Figure 14-29: Block Model vs. Assay Grade – Martha Ellen – Section 6220793N – Gold

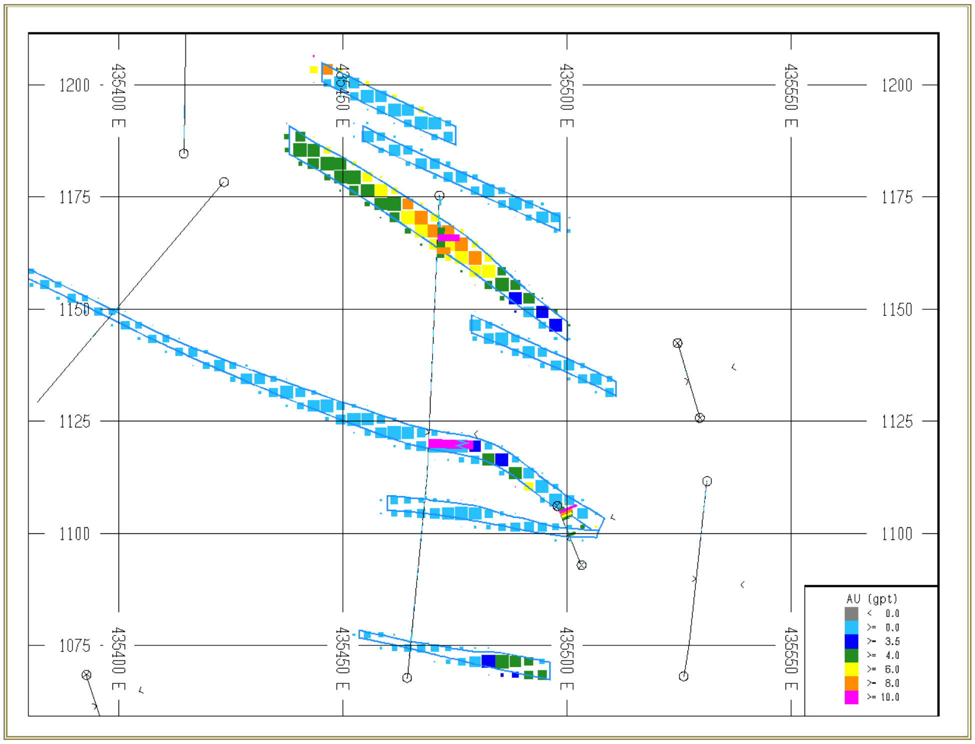


Figure 14-30: Block Model vs. Assay Grade – Dilworth – Section 6222649N – Gold

14.12 Classification of Premier Gold Project Mineral Resources

The blocks were classified according to CIM (2014 and 2019) definitions as follows:

- All Classified material must be within a potentially mineralized wireframe and have a minimum minable true thickness of 2.5 m.
- Blocks within a wireframe and within an anisotropic search ellipse with dimensions of 100 m x 100 m x 15 m are assigned a preliminary classification of Inferred.
- Indicated blocks are required to have at least one of the following criteria:
  - The average distance to the nearest three drill holes is less than 35 m, with none further than 35 m, and there are samples from at least two split quadrants
  - the average distance to the nearest two drill holes is less than 17.5 m, and there are samples from at least two split quadrants, or
  - the distance to the nearest drill hole is less than 10 m and at least two drill holes have been used in the estimate.
  - A cut-off grade of 3.5 g/t AuEq was applied to the block model for reporting of Mineral Resources. This cut-off grade was derived from a preliminary analysis of current mining and processing costs for underground mining operations.

For Dilworth and Martha Ellen, the following classification has been implemented:

- The Inferred classification is based on the anisotropic distance to the nearest drill hole with data of less than or equal to 50 m.
- Blocks are classified as Indicated if they had an average distance to the nearest three drill holes of less than 17.5 m, or an average distance to the nearest two drill holes of less than 10 m.
- Due to limited QA/QC for assays from the Assayers Canada era of drilling at Martha Ellen and Dilworth (2007–July 2010), this data has not been used in the classification of material. Therefore, sections of Indicated blocks have been down-graded to Inferred in some areas of Dilworth and Martha Ellen. This drilling did not have a significant effect on the classification of the Big Missouri resource. See the section of this report entitled Data Verification for a discussion of the QA/QC results.

14.13 Reasonable Prospects of Eventual Economic Extraction

For determination of a resource cut-off grade at the PGP deposits Ascot conducted a very preliminary analysis in April 2018, including a review of cost information from similar projects. The following assumptions were used:

- Gold price of US$1,300/oz (no contribution from silver)
- Underground mining
- Processing at a rate of 1,000 t/d
- US$ exchange rate of US$0.78:C$1.00
- Operating costs of:
  - Mining – US$62.43/t
  - Mill & Services – US$45.00/t
  - G&A – US$25.00/t.

Metallurgical recovery of 89% for gold (based on historical mill performance; silver was not included in the analysis).

The mineralized zones at Premier, and throughout the Project area, embrace a wide range of orientations and thicknesses which would require different mining methods depending on geometry. The following assumptions were made concerning the relative proportions of the mineralization that would be mined by each method and unit costs of those methods:

- Cut-and-fill – 20%, US$88.23/t
- Longhole – 30%, US$50.00/t
- Inclined room and pillar – 20%, US$40.00/t
- Alimak – 20%, US$60.00/t
- Shrinkage – 10%, US$97.83/t.

The implied cut-off grade, based on the above assumptions, was 3.55 g/t Au. Ascot's analysis has been reviewed by the QP and is considered to be reasonable for the purposes of determining a resource cut-off grade. A block cut-off grade of 3.5 g/t AuEq was applied to the block models at Premier for reporting of Mineral Resources.

In addition to the cut-off grade, a 2.5 m minimum true thickness constraint was used to exclude material considered too thin to warrant underground mining. True thickness values have been determined from the assay intervals by using the dip of the mineralized zone and the dip of the drill hole. The true thickness has then been interpolated for the block using the majority zone of mineralization.

Although a 3.5 g/t AuEq cut-off grade has been used, it should be noted that this is essentially a 0.0 g/t cut-off grade within gradeshells of minable shapes created at a 3.5 g/t AuEq, and therefore conforms to the updated CIM best practices guidelines (CIM, 2019). Figure 14-31 illustrates the wireframes used for interpolation (blue) and the 3.5 g/t AuEq grade shells (yellow) used to define the Resource Estimate for the Premier-Northern Lights area. The continuity of the Resource above 3.5 g/t AuEq is evident.

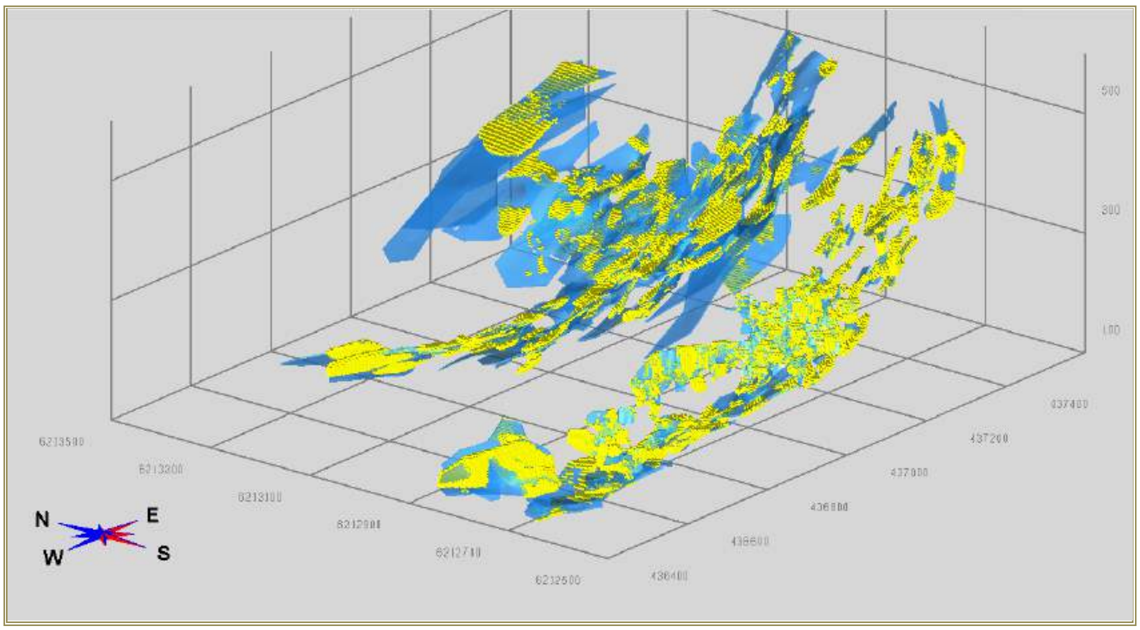


Figure 14-31: ***Continuity of the 3.5 g/t AuEq Grade Shell—Premier***

14.14 Premier Gold Project Mineral Resource Statement and Sensitivity to Cut-off Grade

Table 14-1 presents the Mineral Resource Estimate for each of the PGP deposits at a base case cut-off grade of 3.5 g/t AuEq; Table 14-37 summarizes the sensitivity of the Total PGP Resource to cut-off grade with the base case cut-off grade of 3.5 g/t AuEq highlighted.

Table 14-37: PGP Resource Sensitivity to Cut-off Grade, Effective Date of December 12, 2019

Class	Cut-off AuEq (g/t)	In-situ Tonnage (kt)	In-situ Grades			Metal	
			AuEq (g/t)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
Indicated	2.5	6,015	6.60	6.39	30.12	1,237	5,825
	3.0	4,958	7.43	7.20	32.7	1,148	5,207
	3.5	4,141	8.25	8.01	35.1	1,066	4,669
	4.0	3,483	9.11	8.85	37.6	990	4,215
	4.5	2,954	9.98	9.70	40.0	921	3,797
	5.0	2,545	10.82	10.53	42.0	861	3,439
Inferred	2.5	7,565	5.97	5.79	26.1	1,408	6,342
	3.0	6,176	6.70	6.51	27.2	1,292	5,402
	3.5	5,061	7.45	7.25	28.7	1,180	4,673
	4.0	4,071	8.36	8.15	30.0	1,067	3,925
	4.5	3,364	9.22	9.01	31.0	974	3,352
	5.0	2,890	9.96	9.74	31.7	905	2,942

14.15 Factors That May Affect the Premier Gold Project Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Commodity price assumptions
- Metal recovery assumptions
- Mining and processing cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the Province of BC in terms of environmental, permitting, taxation, socioeconomic, marketing, and political factors.

14.16 Risk Assessment of the Premier Gold Project Mineral Resource Estimate

The identified risk factors have been split into technical and non-technical groups, with technical meaning those identified and discussed in this report and non-technical having to do with assumptions on prices, costs, and outside influences. A description of each factor is given in Table 14-37 along with either the justification for the approach taken or mitigating factors in place to reduce any risk. A matrix of the risk factors mentioned above, and additional potential risks known for the PGP specifically are summarized in the matrix of Table 14-38. As illustrated, there are no adverse risks that are in the possible, likely, or known categories that have a detrimental impact on the overall project. It is considered, however, that the low metallurgical recoveries used will have a positive impact on the PGP deposits.

Table 14-38: List of Risks/Rewards and Mitigations Justifications

	No.	Description	Justification / Mitigation
Technical Factors	1	QA/QC Standards for Ag assayed high in 2019	Re-assay standards for 2019 or do checks
	2	Silver Coin QA/QC not to the same level as other deposits for legacy drilling	Check assays have been done where possible
	3	Surveys of legacy holes inaccurate	Definition drilling applied prior to mining
	4	Classification criteria	2019 drilling indicates veins are continuous to 35 m distances used for Indicated classification
	5	Unknown geologic structures	Continuous mapping of structures and ongoing exploration drilling
	6	Capping and outlier restriction	CPP, Swath Plots, and G-T curves show model validates well with composite data
Non-Technical Factors	7	First Nations treaty issues	Follow Nisga'a Treaty
	8	Au price falls below US$1,300	Conservative price has been used
	9	Recoveries used for the current resource are conservative	Additional metallurgical testing
	10	Processing and mining costs are low	Lower costs are used to include all mineralization with "reasonable prospect of eventual economic extraction"
	11	Claims boundary issues	Legal Consul has been hired
	12	Environmental permitting issues	Ongoing with input from Nisga'a
	13	Areas of Resource are not conducive to underground mining	Geotechnical and mine planning studies are underway

Table 14-39: Matrix of Potential Risk Factors

		Impact to the Overall Project				
		Positive Impact	Neutral / Immaterial	Slight Negative Impact	Somewhat Detrimental	Very Detrimental
Probability	Known Factor	9	1, 2			
	Likely		3, 10			
	Possible			5		
	Not Likely			4, 11	6	13
	Almost No Chance			8	12	7

Additional details on the justification for the updated classification to Indicated for the three deposits drilled in 2019 (Premier, Big Missouri, and Silver Coin) are provided in Figure 14-32 to Figure 14-34. These figures illustrate the 2019 drilling compared to the 2018 model. The 2019 AuEq assay grades are compared to the 2018 grade shells of Inferred material above a 3.5 g/t AuEq cut-off. These figures show clearly that the wireframing process and interpolation methods predicted well the grade and location of mineralization. Continuity of mineralization above 3.5 g/t AuEq up to at least 75 m is evident, providing justification for classification to Indicated when three drill holes are within 35 m for these three deposits. Note that the classification did not change in Martha Ellen and Dilworth where there was no 2019 drilling and remains with the restriction of three drill holes within 17.5 m.

In each case the location of the wireframes has been adjusted slightly based on the 2018 drilling, but the modelling methodology remained very much the same.

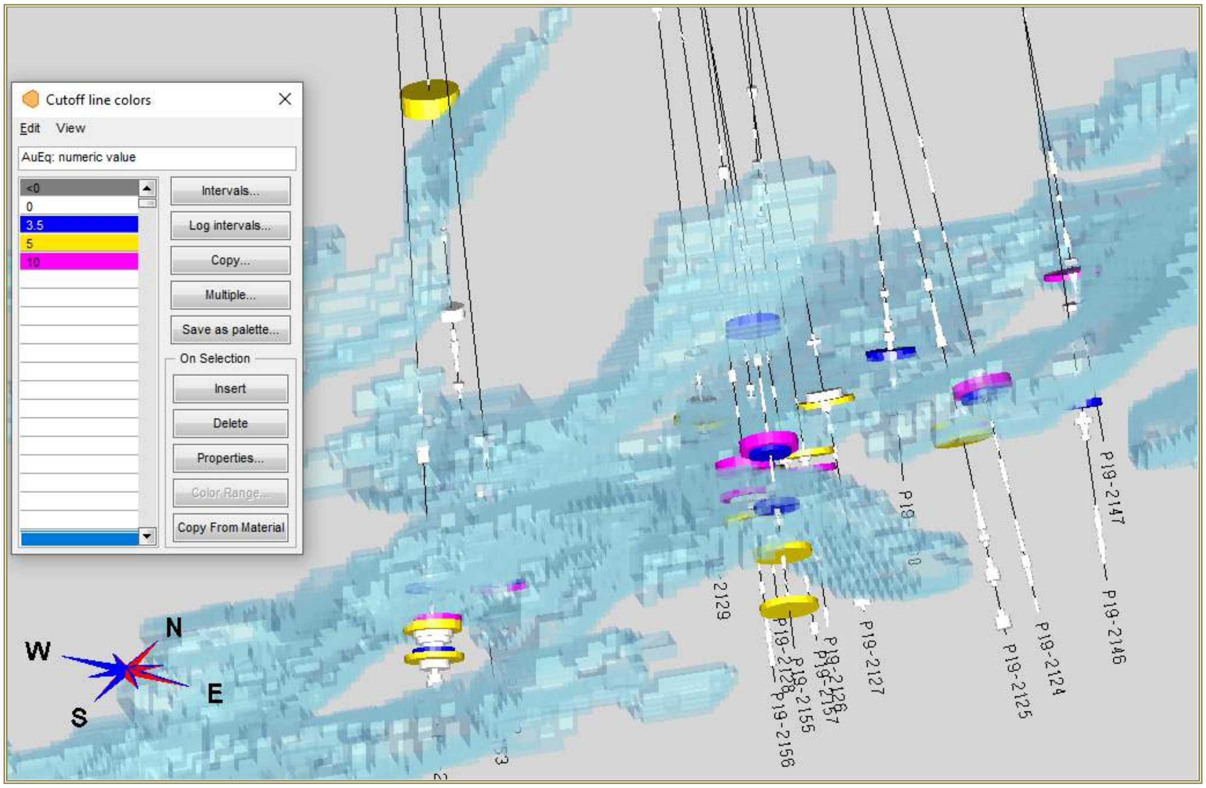


Figure 14-32: 2019 Drilling Compared to Inferred Material > 3.5 g/t AuEq – Premier (Prew)

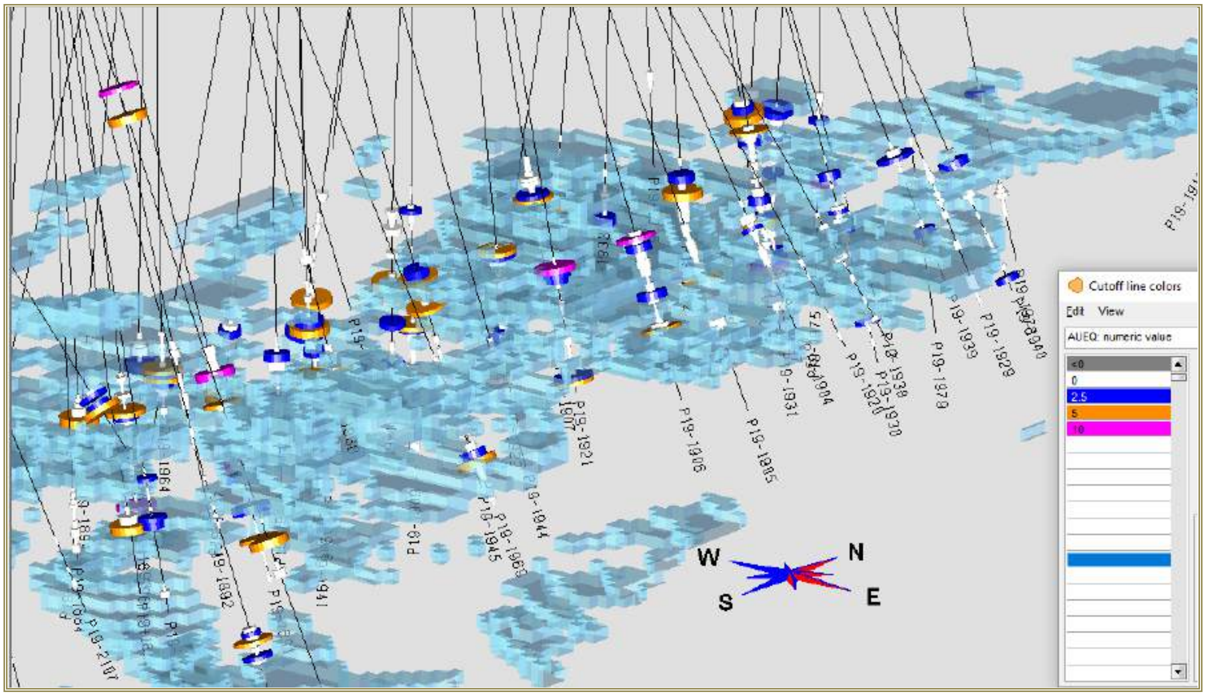


Figure 14-33: 2019 Drilling Compared to Inferred Material > 3.5 g/t AuEq – Big Missouri

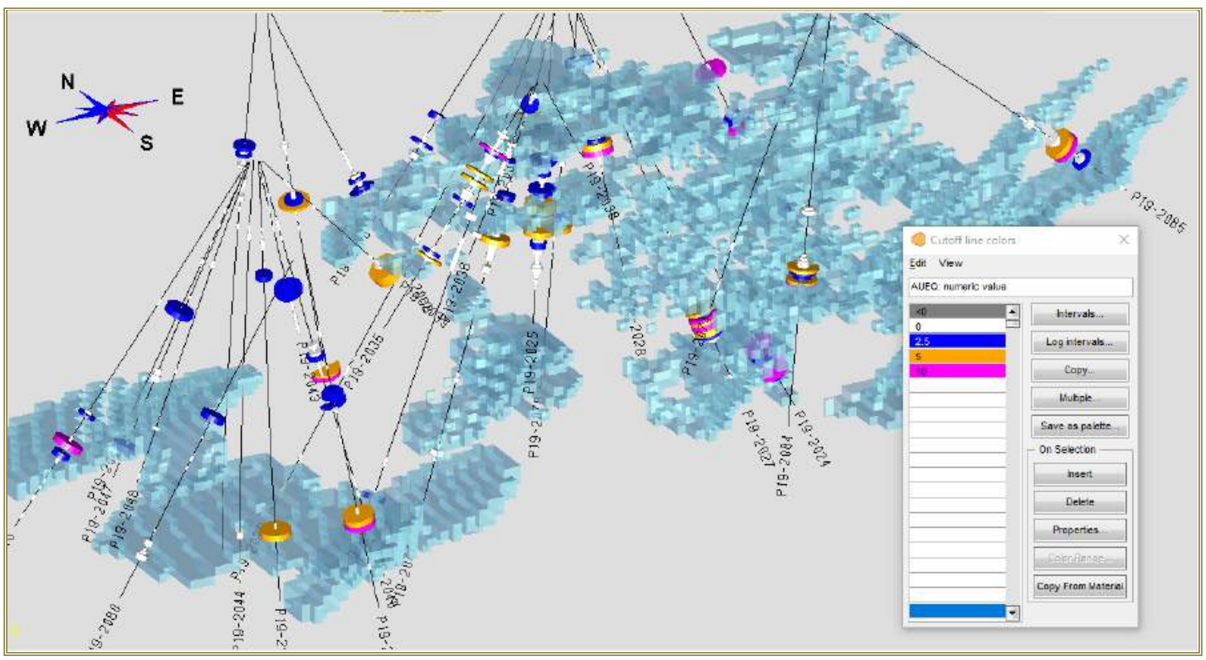


Figure 14-34: 2019 Drilling Compared to Inferred Material > 3.5 g/t AuEq – Silver Coin

14.17 Peer Review of the Premier Gold Project Mineral Resource Estimate

The assumptions, data, methodology, and results of this mineral resource estimate have been reviewed by the following members Ascot's geology and engineering team:

- Mr. John Kiernan, P.Eng., Chief Operating Officer
- Mr. Lars Beggerow, M.Sc., Vice President Geoscience and Exploration
- Mr. Lawrence Tsang, P.Geo., Senior Project Geologist
- Mr. George Dermer, P.Eng., Consulting Mining Engineer.

14.18 Red Mountain Resource Estimation Procedures

The resource evaluation methodology involved the following procedures:

- Database compilation and verification
- Validation of wireframe models for the boundaries of the gold mineralization
- Definition of resource domains
- Data conditioning (compositing and capping) for geostatistical analysis and variography
- Block modelling and grade interpolation
- Resource classification and validation
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades
- Preparation of mineral resource statement.

14.19 Red Mountain Drill Hole Database

The drilling database consists of historical drilling most of which has been carried out by LAC in the early 1990s. Between 2000 and 2001, North American Metals Corporation (NAMC) relogged all of the mineralized intervals and carried out an extensive database validation of the drill database. Banks Island Gold drilled two holes in the Marc zone in 2013 and IDM drilled five holes in the deposit in 2014, three holes targeting the 141 zone, and two holes targeting the AV zone. IDM drilled 62 holes in 2016 to better defined the mineralization, collect some samples for metallurgical tests and upgrade some of the Inferred mineralization to Indicated. IDM also drilled seven exploration holes targeting other areas on the RMP in 2014. The 2017 and 2018 drill programs concentrated on upgrading the mineral resource classification for the Marc, AV, and JW zones, and better defining the SF zone.

There is a total of 77,280 records in the assay database, excluding the chip samples collected from the underground development; of these, 4,654 represent samples taken from the mineralized horizons. Table 14-40 summarizes the basic statistical data for all the assays in the database and Table 14-41 summarizes the assays contained within the mineralized zones only.

Table 14-40: Basic Statistical Information for all Assays in the Red Mountain Database

Assays	Au (g/t)	Ag (g/t)
Valid cases	77,280	77,280
Mean	0.86	2.88
Variance	81.67	653.13
Std. Deviation	9.04	25.56
Variation Coefficient	10.53	8.89
Minimum	0.00	0.00
Maximum	1,462.00	2,152.00
1st percentile	0.01	0.00
5th percentile	0.01	0.00
10th percentile	0.02	0.00
25th percentile	0.03	0.10
Median	0.11	0.37
75th percentile	0.37	1.06
90th percentile	1.15	2.67
95th percentile	2.62	7.53
99th percentile	13.01	46.80

Table 14-41: Basic Statistical Information of Gold Assays within the Red Mountain Mineralized Zones

Assays	Au (g/t)	Ag (g/t)
Valid cases	4,654	4,654
Mean	9.19	29.85
Variance	1,262.64	8,386.25
Std. Deviation	35.53	91.58
Variation Coefficient	3.87	3.07
Minimum	0.00	0.00
Maximum	1,462.00	2,152.00
1st percentile	0.06	0.00
5th percentile	0.33	0.05
10th percentile	0.68	0.33
25th percentile	1.61	1.60
Median	3.63	8.81
75th percentile	8.10	27.50
90th percentile	17.81	59.25
95th percentile	31.29	103.15
99th percentile	92.39	386.02

14.20 Design of Red Mountain Modelling Criteria

A significant amount of time and effort was invested during the 2000 field season to develop modelling criteria for the mineralization at Red Mountain. Areas of investigation included general lithology, nature of sulfide occurrences, relationship of pyrite to gold grade, and structural control on mineralization.

The results of the studies suggested that the following were important modelling criteria:

- Basic lithology, including major structural features, with appropriate textural modifiers.
- The limits of pyrite, and more rarely pyrrhotite, stockwork. These limits are often, but not always coincident with a 1 g/t Au assay outline. Inside this outline, sulfide occurs as disseminations, micro-veinlets, planar and irregular veins, and irregular masses. Average pyrite content in lower gold grade sections of the stockwork is at least 4%. Outside the stockwork limits, sulfide occurs as disseminations and sparse micro-veinlets with an average pyrite content of 1.5%.
- The shift from a pyrite-dominated stockwork to a pyrrhotite-dominated alteration halo is sharp and often corresponds to a 1 g/t Au outline, except in rare cases where pyrrhotite abundance, style, and gold content mimics the pyrite stockwork.
- The cumulative thickness of pyrite in a given interval has the best correlation to gold grade regardless of the width or number of veins and represents the most important data that can be collected to constrain gold distribution. The data collected suggest that cumulative pyrite thickness could be used to delineate high and low-grade domains.
- Brecciation of pyrite veins is also related to gold distribution and can be measured by qualitative measurements, although in practical terms such measurements are time-consuming and very subjective.

After the compilation of the 2016 drilling, IDM decided to review and modify the geological wireframes defining the mineralized zones at Red Mountain. While a similar geological approach to the 2000 modelling was followed, a stronger emphasis was placed on including grade that may have been excluded because of strict geological modelling rules. Furthermore, the base cut-off was raised from a nominal 1 g/t to 2.5 g/t. Some of the lesser defined zones remained modelled at a 1 g/t cut-off.

Following the 2017 drilling, IDM re-evaluated the geological model for the mineralization at Red Mountain and proposed a different geological model that incorporated both stratigraphic as well as structural controls. The re-interpretation resulted in only minor changes to the Marc, AV, and JW zones, but significant changes to the 141 zone and the 132 zone (now named Smit). The new interpretation seems to open new ground for exploration, but still needs additional drilling to be fully confirmed.

14.21 Red Mountain Solid Modelling

New 3-D solids were generated for the mineralized zones using the following process:

- Cross-sections were plotted at 25 m intervals showing all surface and underground diamond drill holes. The sections were plotted with one side of the drill hole trace showing the primary lithology and its modifiers, and the other side showing the assay interval and gold grade.

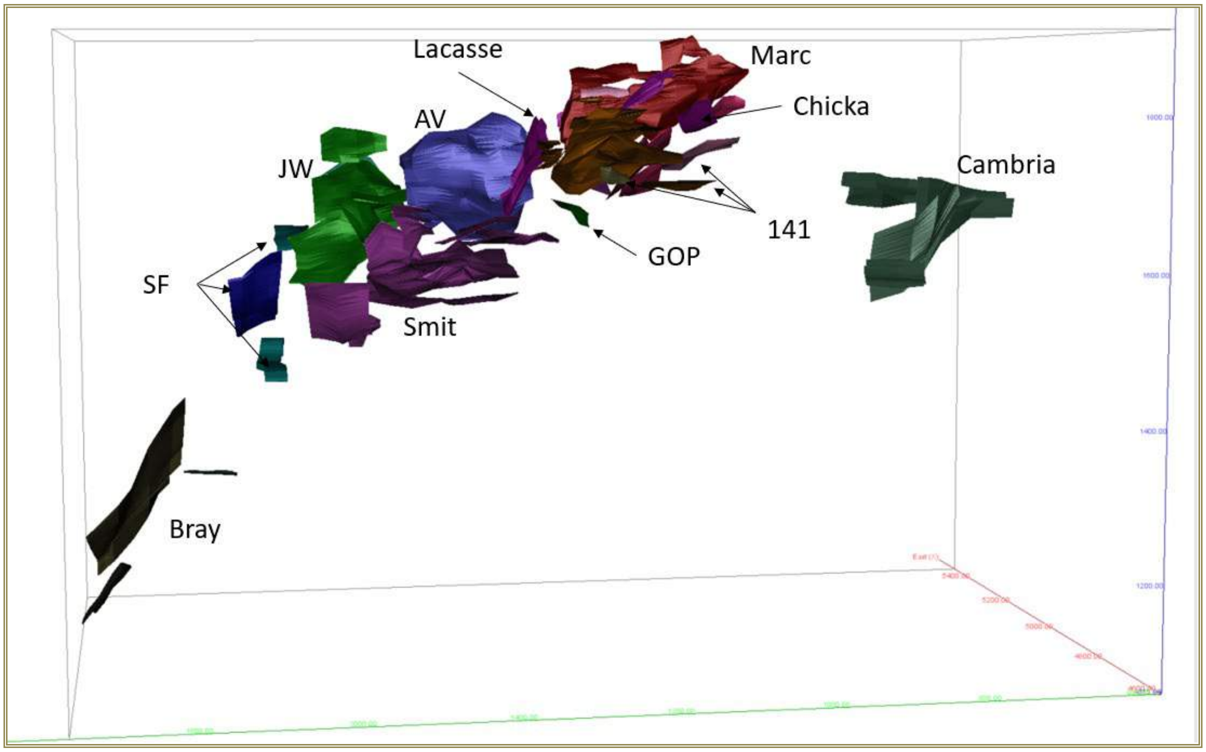
ACS reviewed all of the 3-D solids prior to resource estimation and agrees with the general modelling criteria selected. The outlines are generally based on gold cut-off that for the most part coincide with the limits of pyrite and pyrrhotite stockwork. The boundaries of the stockwork are very abrupt in some places and gradational into the wall rock in others. The stockwork outlines often, but not always, corresponded to areas of intense quartz sericite alteration that give the rock a bleached appearance.

The outlines were derived from vertical sections in Geovia software. The vertical section outlines were digitized as closed polylines that were snapped to the actual 3-D locations of the drill holes. The closed polylines were then "wobbled" (splined) in order to smooth the transition to off-section drill holes while maintaining the integrity of the interpretation.

Each wireframe was assigned a unique rock code as outlined in Table 14-42 and shown on Figure 14-35.

Table 14-42: Rock Codes Assigned to Red Mountain Wireframes

Zone	Rock Code	Zone	Rock Code
Marc	101	Cambria	601
AV	201	Chicka	701
JW	301	Bray	801 & 802
Marc Footwall	102, 105, & 106	SF	901 & 902
Marc Hanging Wall	103	GOP	1001
Marc NK	104	Marc Low Grade	1101
JW FW	302	AV Low Grade	1201
JW HW	303	JW Low Grade	1301
141	401 & 402	Lacasse	2001
Smit	501		



Note: Markers on axes are 200 m apart

Source: ACS, 2019

Figure 14-35: 3-D Perspective View Looking East Showing Location of Red Mountain Mineralized Zones

14.22 Red Mountain Bulk Density

The bulk density of the Red Mountain gold deposits has been evaluated by LAC in 1993–1994 when 4,225 SG determinations estimated from drill were submitted to the Eco-Tech lab in Stewart. In 2000, NAMC collected 58 samples that were subjected to bulk density analysis. IDM has been routinely collecting bulk density data as part of their drilling programs. Collectively, there are 5,316 bulk density readings in the drill hole database; 1,517 are from sample intervals within the solids used for resource calculation. Average SG values for different subsets of the entire data set are given in Table 14-43.

Table 14-43: Bulk Density Sample Results

Zone	No. Samples	Minimum	Maximum	Average
All Samples	5,316	1.44	4.87	2.88
Within Mineralized Zones	1,517	1.85	4.87	2.99
Marc Zone	1,021	2.03	4.39	2.96
AV Zone	215	2.73	4.42	3.01
JW, JW HW & JW FW	57	2.67	4.39	3.00
Marc FW	10	2.89	3.01	2.93
Marc Hanging Wall	5	2.74	3.05	2.74
141	27	2.70	3.88	2.92

Zone	No. Samples	Minimum	Maximum	Average
Smit	67	2.72	4.87	3.52
Cambria	11	2.96	4.10	3.28
Chicka	0	No Data	No Data	No Data
Bray	0	No Data	No Data	No Data
SF	1	3.66	3.66	3.66
GOP	1	3.65	3.65	3.65
Marc Low Grade	68	2.66	3.34	2.87
AV Low Grade	20	1.85	3.16	2.85
JW Low Grade	10	2.76	3.12	2.91
Lacasse	4	2.88	2.90	2.89
Waste	3,799	1.44	4.76	2.83

14.23 Top Cut Applied to Red Mountain Assay Data

Block grade estimates may be unduly affected by high grade outliers. Therefore, assay data were evaluated for high-grade outliers. Based on the analysis of the assay distribution, ACS decided that capping of high-grade assays was warranted. ACS evaluated each of the mineralized lenses separately to define appropriate capping level for each zone prior to compositing. Table 14-44 summarizes the various gold capping levels and Table 14-45 summarizes the silver capping levels.

Table 14-44: Gold Assay Capping Levels

Rock Code	Zone Name	Capping Level (g/t)	Number of Assays Capped
101	Marc	75	34
102, 103, 104, 105 & 106	Marc FW, HW & Outlier	No Capping	0
201	AV	55	22
301, 302 & 303	JW & JW FW & HW	40	11
401 & 402	141	40	2
501	Smit	20	5
601	Cambria	No Capping	0
701	Chicka	No Capping	0
801 & 802	Bray	20	1
901 & 902	SF	20	2
1001	GOP	20	1
1101	Marc Low Grade	8	3
1201	AV Low Grade	No Capping	0
1301	JW Low Grade	No Capping	0
2001	Lacasse	65	2

Table 14-45: Silver Assay Capping Levels

Rock Code	Zone Name	Capping Level (g/t)	Number of Assays Capped
101	Marc	500	22
102, 103, 104, 105 & 106	Marc FW, HW & Outlier	70	2
201	AV	200	7
301, 302 & 303	JW & JW FW & HW	200	10
401 & 402	141	80	2
501	Smit	No Capping	0
601	Cambria	No Capping	0
701	Chicka	No Capping	0
801 & 802	Bray	60	5
901 & 902	SF	60	4
1001	GOP	No Capping	0
1101	Marc Low Grade	60	3
1201	AV Low Grade	90	1
1301	JW Low Grade	45	2
2001	Lacasse	310	2

14.24 Red Mountain Composite Statistics

14.24.1 Composite Statistics

All assay data were composited to a fixed length prior to estimation. ACS evaluated the assay lengths for the various deposits and found that most samples had an average length of less than 1.5 m. ACS therefore decided to composite all assay data to 1.5 m prior to estimation. Table 14-46 summarizes the basic statistical data for capped gold composites used in the resource estimates and Table 14-47 shows the statistics of the capped silver composited data.

Table 14-46: Descriptive Statistics of Red Mountain 1.5 m Gold Composites

Zone	Marc Au (g/t)	AV Au (g/t)	JW Au (g/t)	141 Au (g/t)	Smit Au (g/t)	Marc Low Grade Au (g/t)	AV low Grade Au (g/t)	JW Low Grade Au (g/t)	All Other Zones Au (g/t)
Valid cases	3,452	483	231	282	458	83	117	37	305
Mean	7.32	7.78	7.49	3.66	3.06	1.78	1.66	1.74	4.60
Variance	1.36	87.25	51.51	18.26	8.52	1.93	1.20	0.66	44.29
Std. Deviation	9.94	9.34	7.18	4.27	2.92	1.39	1.10	0.81	6.65
Variation Coefficient	1.36	1.20	0.96	1.17	0.95	0.78	0.66	0.47	1.45
Minimum	0.00	0.04	0.12	0	0.00	0.02	0.01	0.19	0.00
Maximum	75.00	55.00	40.00	33.06	20.00	8.00	7.32	3.73	63.69
1st percentile	0.03	0.18	0.40	0.00	0.17	0.00	0.01	0.21	0.00
5th percentile	0.43	0.88	1.25	0.20	0.56	0.31	0.07	0.24	0.03
10th percentile	0.83	1.72	2.15	0.51	0.71	0.49	0.42	0.45	0.07
25th percentile	1.91	3.05	3.14	1.15	1.18	0.93	1.02	1.23	0.68

Zone	Marc Au (g/t)	AV Au (g/t)	JW Au (g/t)	141 Au (g/t)	Smit Au (g/t)	Marc Low Grade Au (g/t)	AV low Grade Au (g/t)	JW Low Grade Au (g/t)	All Other Zones Au (g/t)
Median	3.86	4.90	5.07	2.44	2.13	1.41	1.49	1.69	2.66
75th percentile	8.24	8.00	9.24	4.45	3.77	2.36	2.17	2.16	5.48
90th percentile	17.85	18.23	14.09	9.10	6.76	3.54	2.84	3.09	12.25
95th percentile	26.52	23.85	25.65	13.24	8.42	4.80	3.21	3.32	16.37
99th percentile	53.16	54.88	39.76	21.46	16.83	6.25	7.08	3.56	33.32

Table 14-47: Descriptive Statistics of Red Mountain 1.5 m Silver Composites

Zone	Marc Ag (g/t)	AV Ag (g/t)	JW Ag (g/t)	141 Ag (g/t)	Smit Ag (g/t)	Marc Low grade Ag (g/t)	AV low Grade Ag (g/t)	JW Low Grade Ag (g/t)	All Other Zones Ag (g/t)
Valid cases	3,452	483	231	282	458	83	117	37	305
Mean	25.31	21.25	25.61	6.52	2.14	16.76	20.77	9.13	9.05
Variance	2,524.60	757.64	1,571.06	127.11	30.50	226.00	312.70	216.09	589.12
Std. Deviation	50.25	27.53	39.64	11.27	5.52	15.03	17.68	14.70	24.27
Variation Coefficient	1.99	1.30	1.55	1.73	2.59	0.90	0.85	1.61	2.68
Minimum	0.00	0.00	0.00	0.00	0.00	0.10	0.60	0.00	0.00
Maximum	500.00	200.00	200.00	79.84	72.22	55.18	82.21	45.00	309.61
1st percentile	0.00	0.05	0.00	0.00	0.00	0.00	0.60	0.00	0.00
5th percentile	0.05	0.38	0.05	0.00	0.05	0.45	0.99	0.00	0.00
10th percentile	0.36	1.92	0.05	0.10	0.14	1.82	1.44	0.00	0.00
25th percentile	1.80	5.40	3.94	0.70	0.33	4.07	8.65	0.05	0.49
Median	9.92	11.96	12.55	2.14	0.70	11.92	15.22	0.05	1.97
75th percentile	27.49	26.36	26.37	6.58	1.76	25.09	28.44	13.31	7.40
90th percentile	57.91	51.73	61.77	20.33	4.70	42.39	46.33	37.89	22.23
95th percentile	94.03	80.37	138.85	33.28	8.51	49.66	59.51	42.77	46.48
99th percentile	305.75	152.68	195.69	57.82	26.02	54.04	80.80	44.11	114.31

14.25 Red Mountain Spatial Analysis

Spatial continuity of gold and silver was evaluated with correlograms developed using SAGE 2001 (Version 1.08). The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram is close to zero is called the “range of correlation,” or simply “the range.” The range of the correlogram corresponds roughly to the more qualitative notion of the “range of influence” of a sample or composite.

Directional correlograms were generated for composited data at 30-degree increments along horizontal azimuths. For each azimuth, correlograms were calculated at dips of 0, 30, and 60 degrees. A vertical correlogram was also calculated, using the information from these 37 correlograms. Sage then determines the best fit model using the least square fit method. The correlogram model is described by the nugget (Co), the variance contribution of the two nested structures (C1, C2), and the range of each of the structures.

Experimental correlograms were obtained for drill hole directions for which sufficient data existed for the mineralized zones at Red Mountain. The Marc zone is the most densely drilled and provides the greatest opportunity for determining the short-range character of the correlogram. Correlograms developed from the Main Marc zone was utilized for all the Marc lenses. Correlograms were also derived for AV, JW, 141, and Smit zones.

Because of the sparse drilling in the Cambria, Chicka, Bray, SF, GOP, and Lacasse zones, ACS decided to estimate these zones with inverse distance to the second power (ID^2) as they contained insufficient data to develop any significant correlograms.

Table 14-48 summarizes the correlogram parameters used to estimate gold and silver in the block model.

Table 14-48: Correlogram Parameters Used for Grade Estimation

Zone	Metal	Model Type	Nugget (C_0)	C_1 & C_2	Rotation			Range		
					(Z)	(Y)	(Z)	Rot X	Rot Y	Rot Z
Marc	Au	Exponential	0.2	0.452	-55	29	47	4	4	25
				0.347	-55	29	47	10	156	34
	Ag	Exponential	0.2	0.8	-66	26	61	6.5	38	21
				NA	NA	NA	NA	NA	NA	NA
AV	Au	Exponential	0.2	0.537	72	51	23	14	8	7
				0.262	72	51	23	16	12.5	41
	Ag	Exponential	0.25	0.75	14	-42	-14	52	26	8
				NA	NA	NA	NA	NA	NA	NA
JW	Au	Exponential	0.3	0.542	-80	14	49	4	22	4
				0.157	-80	14	49	12	89	154
	Ag	Exponential	0.1	0.748	-12	29	-5	4	12	74
				0.151	-12	29	-5	39	123	156
141	Au	Exponential	0.35	0.65	31	-86	-6	10	32	5
				NA	NA	NA	NA	NA	NA	NA
	Ag	Exponential	0.27	0.611	27	46	-17	27	60	5
				0.118	27	46	-17	28	120	78
Smit	Au	Exponential	0.5	0.5	22	34	4	85	18	7
				NA	NA	NA	NA	NA	NA	NA
	Ag	Exponential	0.2	0.57	-28	-13	55	15	80	18
				0.23	-28	-13	55	59	135	62

14.26 Red Mountain Block Model

A 3-D block model was created using Geovia GEMs (Version 6.8.2) to represent the lithological and structural characteristics specific to the Red Mountain deposit. This model was used as a framework for the grade model, which relied on geostatistical analysis of the sample data and a detailed understanding of the geology to produce an estimate of the contained resource.

The parameters for the block model are listed in Table 14-49. Block model coordinates are in local grid coordinates to be consistent with historical data. Block size was set to 4 m x 4 m x 4 m to better define the mineralized zones and to stay consistent with previous resource estimates. The rock type element in the block model was coded for all zones using a 0.001% selection process. The rock and percent models were then updated with specific codes for each of the mineralized zones as outlined in Table 14-42. All waste blocks were assigned a default rock code of 99.

Table 14-49: Model Parameters for the Red Mountain Block Model

Coordinates		Origin Coordinates	Block Size (m)	Number of Blocks
Axis	Model			
Easting	Column	4,500	4	250
Northing	Row	500	4	375
Elevation	Level	1,000	4	250

Gold grades were interpolated within the individual zones using ordinary kriging or ID^2 and multiple passes as outlined in Table 14-50. Grades were interpolated into blocks only if the blocks had not been interpolated by a previous pass. Both passes one and two required at least two drill holes within the search volume to interpolate a block grade. Pass three was run with a smaller search radius and required only a single drill hole to assured that all blocks pierced by drill holes were interpolated.

Table 14-50: Interpolation Parameters Used for Red Mountain Grade Interpolation

Zone/Rock Code	Pass	Rotation			Search Ellipse Size			No of composites		Maximum No. per Hole
		Z	Y	Z	X	Y	Z	Minimum	Maximum	
Marc/101	1	0	-75	0	30	30	10	5	15	3
Marc/101	2	0	-75	0	60	60	15	2	15	1
Marc/101	3	0	-75	0	20	20	20	2	15	none
AV/201	1	0	-50	0	30	30	10	5	15	3
AV/201	2	0	-50	0	60	60	15	2	15	1
AV/201	3	0	-50	0	20	20	20	2	15	none
JW/301	1	0	-36	0	30	30	10	5	15	3
JW/301	2	0	-36	0	60	60	15	2	15	1
JW/301	3	0	-36	0	20	20	20	2	15	none
141/401	1	0	33	0	30	30	10	5	15	3
141/401	2	0	33	0	60	60	15	2	15	1
141/401	3	0	33	0	20	20	20	2	15	none
Marc FW/102	1	40	50	0	30	30	10	5	15	3
Marc FW/102	2	40	50	0	60	60	15	2	15	1
Marc FW/102	3	40	50	0	20	20	20	2	15	none
Marc HW/103	1	0	-60	0	30	30	10	5	15	3
Marc HW/103	2	0	-60	0	60	60	15	2	15	1
Marc HW/103	3	0	-60	0	20	20	20	2	15	none
Marc NK/104	1	46	23	0	30	30	10	5	15	3
Marc NK/104	2	46	23	0	60	60	15	2	15	1

Zone/Rock Code	Pass	Rotation			Search Ellipse Size			No of composites		Maximum No. per Hole
		Z	Y	Z	X	Y	Z	Minimum	Maximum	
Marc NK/104	3	46	23	0	20	20	20	2	15	none
Marc FW_2/105	1	-10	-75	0	30	30	10	5	15	3
Marc FW_2/105	2	-10	-75	0	60	60	15	2	15	1
Marc FW_2/105	3	-10	-75	0	20	20	20	2	15	none
Marc FW_3/106	1	0	-22	0	30	30	10	5	15	3
Marc FW_3/106	2	0	-22	0	60	60	15	2	15	1
Marc FW_3/106	3	0	-22	0	20	20	20	2	15	none
JW FW/302	1	-48	-39	0	30	30	10	5	15	3
JW FW/302	2	-48	-39	0	60	60	15	2	15	1
JW FW/302	3	-48	-39	0	20	20	20	2	15	none
JW HW/303	1	-45	-58	0	30	30	10	5	15	3
JW HW/303	2	-45	-58	0	60	60	15	2	15	1
JW HW/303	3	-45	-58	0	20	20	20	2	15	none
141L/402	1	40	-60	0	30	30	10	5	15	3
141L/402	2	40	-60	0	60	60	15	2	15	1
141L/402	3	40	-60	0	20	20	20	2	15	none
Smit/501	1	90	20	0	30	30	10	5	15	3
Smit/501	2	90	20	0	60	60	15	2	15	1
Smit/501	3	90	20	0	20	20	20	2	15	none
Cambria/601	1	-45	-75	0	30	30	10	5	15	3
Cambria/601	2	-45	-75	0	60	60	15	2	15	1
Cambria/601	3	-45	-75	0	20	20	20	2	15	none
Chicka/701	1	-25	-25	30	30	30	10	5	15	3
Chicka/701	2	-25	-25	30	60	60	15	2	15	1
Chicka/701	3	-25	-25	30	20	20	20	2	15	none
Bray/801	1	76	56	0	30	30	10	5	15	3
Bray/801	2	76	56	0	60	60	15	2	15	1
Bray/801	3	76	56	0	20	20	20	2	15	none
Bray_Flat/802	1	0	8	0	30	30	10	5	15	3
Bray_Flat/802	2	0	8	0	60	60	15	2	15	1
Bray_Flat/801	3	0	8	0	20	20	20	2	15	none
SF/901	1	-45	-20	0	30	30	10	5	15	3
SF/901	2	-45	-20	0	60	60	15	2	15	1
SF/901	3	-45	-20	0	20	20	20	2	15	none
SF_2/902	1	0	50	0	30	30	10	5	15	3
SF_2/902	2	0	50	0	60	60	15	2	15	1
SF_2/902	3	0	50	0	20	20	20	2	15	none
GOP/1001	1	0	-8	0	30	30	10	5	15	3
GOP/1001	2	0	-8	0	60	60	15	2	15	1
GOP/1001	3	0	-8	0	20	20	20	2	15	none
Lacasse/2001	1	-74	-50	0	30	30	10	5	15	3

Zone/Rock Code	Pass	Rotation			Search Ellipse Size			No of composites		Maximum No. per Hole
		Z	Y	Z	X	Y	Z	Minimum	Maximum	
Lacasse/2001	2	-74	-50	0	60	60	15	2	15	1
Lacasse/2001	3	-74	-50	0	20	20	20	2	15	none
Marc Low Grade/1101	1	0	-75	0	30	30	10	5	15	3
Marc Low Grade/1101	2	0	-75	0	60	60	15	2	15	1
Marc Low Grade/1101	3	0	-75	0	20	20	20	2	15	none
AV Low Grade/ 1201	1	0	-50	0	30	30	10	5	15	3
AV Low Grade/ 1201	2	0	-50	0	60	60	15	2	15	1
AV Low Grade/ 1201	3	0	-50	0	20	20	20	2	15	none
JW Low Grade/1301	1	0	-36	0	30	30	10	5	15	3
JW Low Grade/1301	2	0	-36	0	60	60	15	2	15	1
JW Low Grade/1301	3	0	-36	0	20	20	20	2	15	none

Bulk density was interpolated using Inverse distance weighted to the second power. For those blocks that had insufficient density data to generate a block estimate, the block densities were assigned the average density for the rock type as defined in Table 14-51.

Table 14-51: Red Mountain Block Model Default Densities by Rock Codes

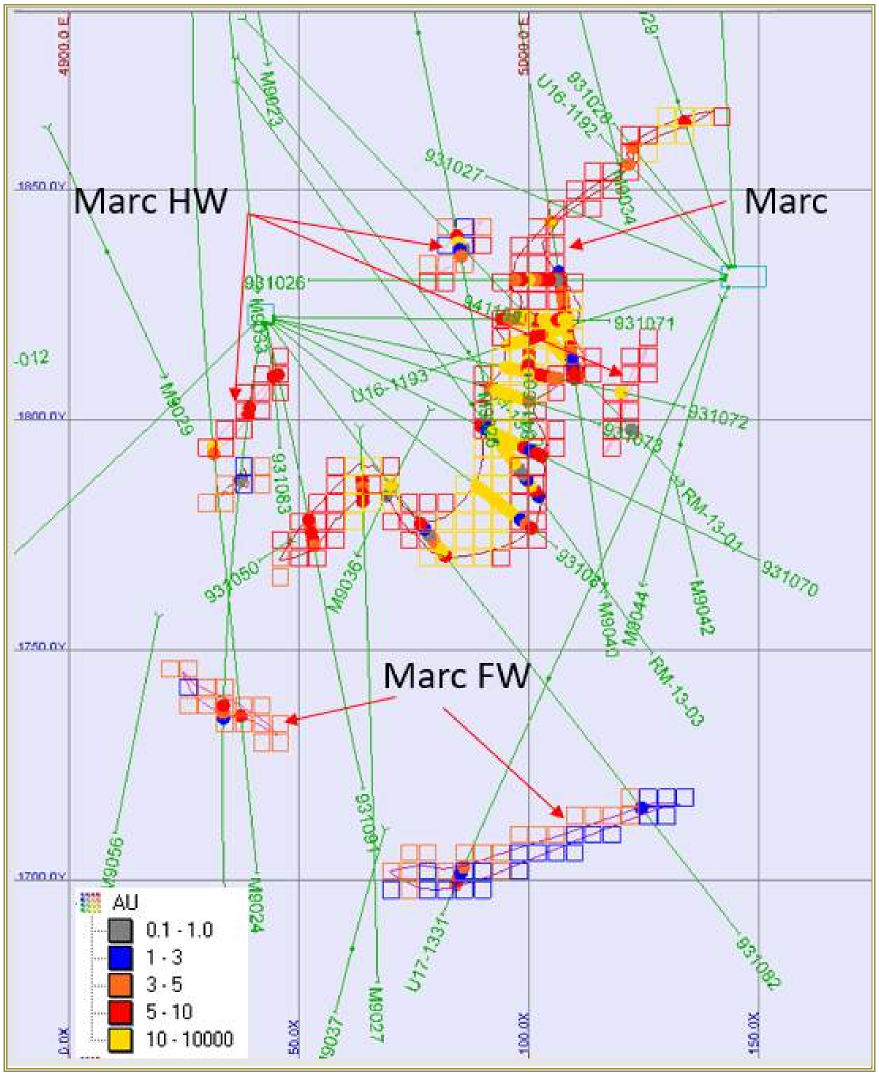
Rock Code	Average Density	Rock Code	Average Density
99	2.83	501	3.52
101	2.96	601	3.28
102	2.96	701	2.99
103	2.96	801 & 802	2.99
104	2.96	901 & 902	2.99
105	2.96	1001	2.99
106	2.96	1101	2.87
201	3.01	1201	2.85
301, 302 & 303	3.00	1301	2.91
401 & 402	2.92	2001	2.89

14.26.1 Red Mountain Model Validation

The zones were validated by completing a series of visual inspections, and by comparison of average assay grades with average block estimates along different directions—swath plots.

Visual Comparison

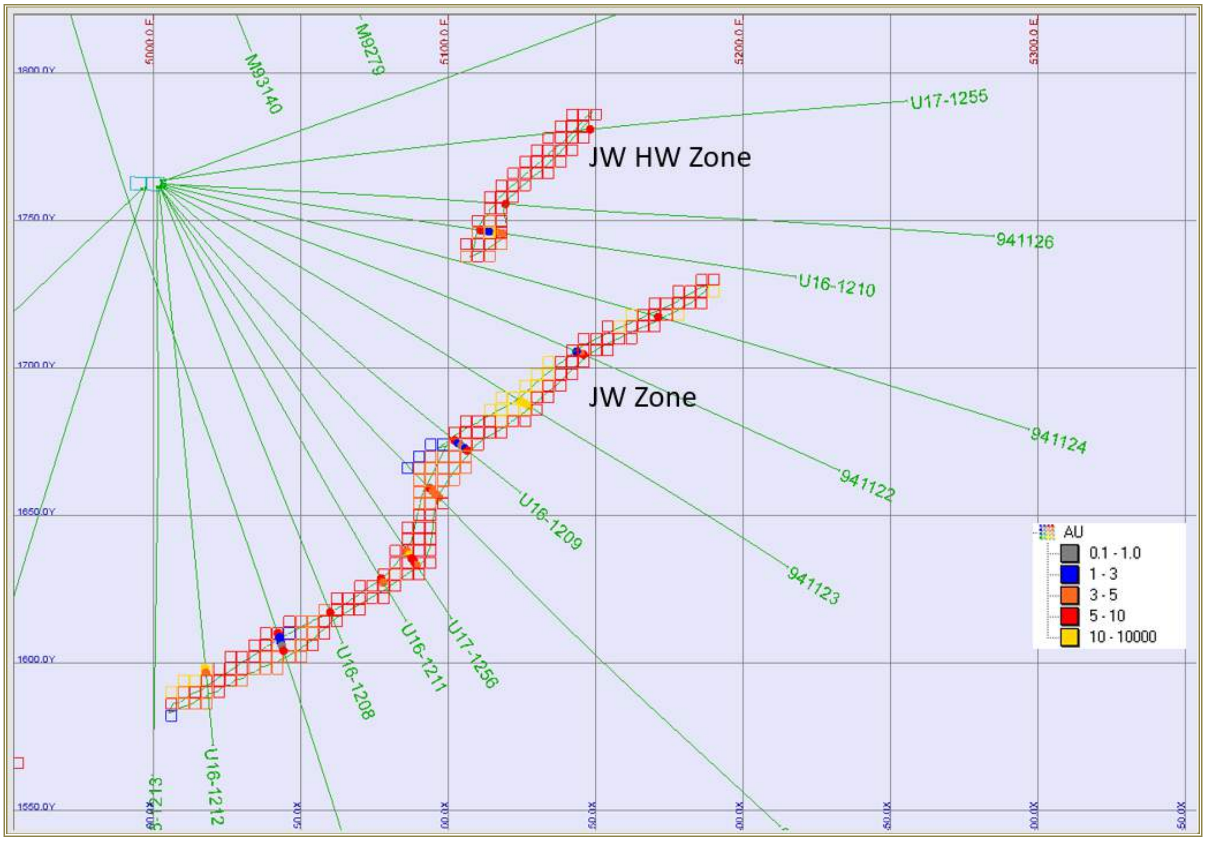
The model was checked for proper coding of drill hole intervals and block model cells. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values (Figure 14-36 and Figure 14-37).



Note: Grid lines are 50 m apart and blocks are 4 m x 4 m

Source: ACS, 2018

Figure 14-36: Section 1250N (Red Mountain Local Grid) Showing Block Drill Hole Composites and Estimated Gold Grades



Note: Grid lines are 50 m apart and blocks are 4 m x 4 m

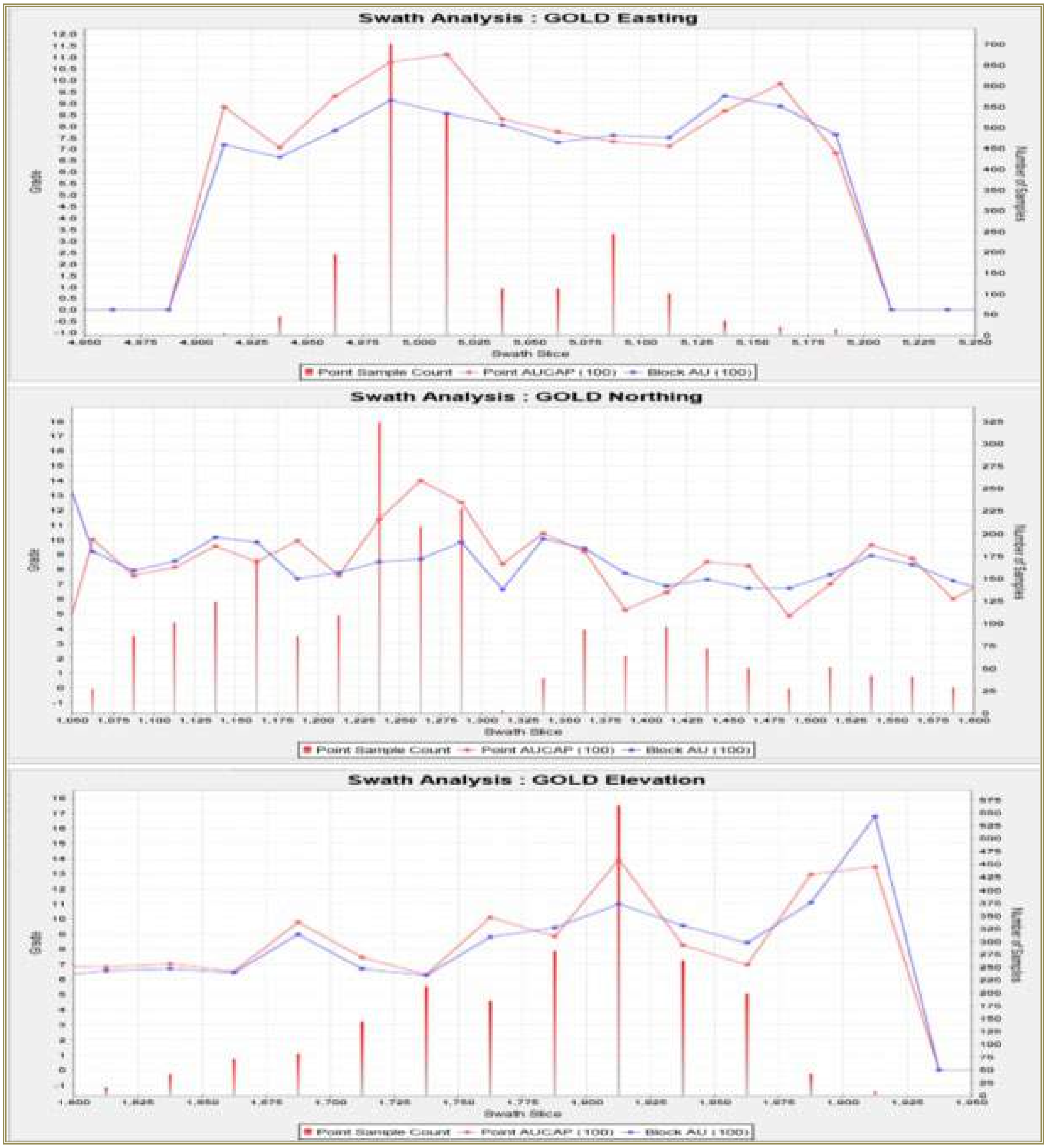
Source: ACS, 2018

Figure 14-37: Section 1600N (Red Mountain Local Grid) Showing Drill Hole Composite and Estimated Gold Grades

Red Mountain Swath Plots

Average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south, and horizontal (by elevation) swaths.

Figure 14-4 shows the swath plot for gold for the Marc, AV, and JW zones. On average, the estimated data agree well with the composited data, with the estimated values being slightly more smoothed than the composite data.



Source: ACS, 2018

Figure 14-38: Swath Plot for Gold for the Marc, AV, and JW Zones

14.27 Red Mountain Resource Classification

Block model quantities and grade estimates for the RMP were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (the CIM Definition Standards, May 2014) by Dr. Gilles Arseneau, P.Geo. (APEGBC), an independent QP for the purpose of NI 43-101.

Mineral resource classification is typically a subjective concept; however, industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

ACS is satisfied that the geological modelling reflects the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drill holes. Drilling samples were from sections spaced between 20 m to 80 m.

ACS considers that blocks in the Marc, AV, and JW zones estimated during pass one, and from at least three drill holes, could be assigned to the Measured category. All other blocks interpolated during pass 1 in the Marc, AV, and JW zones were assigned to the Indicated category. Blocks estimated with at least three holes during Pass 2 in all zones were classified as Indicated. All other estimated blocks were classified as Inferred.

14.28 Red Mountain Mineral Resource Statement

CIM Definition Standards defines a Mineral Resource as:

> *…a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds, and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, ACS considers that major portions of the Red Mountain deposits are amenable for underground extraction by long-hole stoping method.

In order to determine the quantities of material satisfying "reasonable prospects for economic extraction," ACS assumed a minimum mining cut-off of 3 g/t Au representing an approximate mining cost of $160 and a minimum mining width of 2 m. The reader is cautioned that there are no mineral reserves at the RMP.

ACS is unaware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, or political issues that may adversely affect the Mineral Resources presented in this study.

ACS considers that the blocks with grades above the cut-off grade satisfy the criteria for "reasonable prospects for economic extraction" and can be reported as a Mineral Resource. Mineral resources for each deposit at the RMP are summarized in Table 14-52.

Table 14-52: Red Mountain Mineral Resource Statement at a 3 g/t Au Cut-off Effective August 30, 2019

Zone	Tonnage (t)	In-situ Gold Grade (g/t)	In-situ Silver Grade (g/t)	Contained Gold (troy oz)	Contained Silver (troy oz)
Marc Zone					
Measured	726,400	10.61	41.08	247,700	959,500
Indicated	11,900	8.63	28.33	3,300	10,800
Inferred	600	10.08	9.53	200	200
AV Zone					
Measured	844,000	7.82	19.95	212,200	541,300
Indicated	108,100	9.61	22.04	33,400	76,600
Inferred	400	10.65	13.36	100	200
JW Zone					
Measured	308,300	7.79	19.32	77,200	191,500
Indicated	132,600	6.89	16.30	29,400	69,500
Inferred	400	6.92	8.00	100	100
141 Zone					
Indicated	233,700	4.68	6.14	35,200	46,100
Inferred	61,000	4.75	3.74	9,300	7,300
Smit				***0***	***0***
Indicated	461,300	4.42	3.58	65,500	53,200
Inferred	100,800	4.89	2.28	15,800	7,400
Marc Footwall					
Indicated	78,000	5.97	5.75	15,000	14,400
Inferred	700	3.33	0.66	100	0
Marc Outlier Zone					
Indicated	4,200	6.10	8.50	800	1,100
Marc NK Zone					
Indicated	37,900	7.37	8.25	9,000	10,000
Inferred	100	8.57	13.59	0	0
JW HW					
Measured	41,000	5.09	41.53	6,700	54,700
Indicated	20,000	5.83	10.58	3,700	6,800
Bray					
Indicated	93,100	5.02	12.87	15,000	38,500
Inferred	59,000	5.11	14.14	9,700	26,800
Chicka					
Indicated	14,300	9.94	3.98	4,600	1,800
Inferred	2,300	5.30	2.18	400	200
JW FW					
Indicated	4,700	15.46	26.95	2,300	4,100
SF					
Indicated	36,100	7.12	30.26	8,300	35,100
Inferred	40,400	4.88	6.05	6,300	7,900

Zone	Tonnage (t)	In-situ Gold Grade (g/t)	In-situ Silver Grade (g/t)	Contained Gold (troy oz)	Contained Silver (troy oz)
Cambria					
Inferred	80,600	6.85	4.53	17,700	11,700
GOP					
Inferred	12,100	7.04	3.03	2,700	1,200
Lacasse					
Indicated	34,600	11.97	36.68	13,300	40,800
Inferred	700	5.72	13.83	100	300
Marc Low Grade					
Inferred	8,500	4.90	19.59	1,300	5,400
AV Low Grade					
Inferred	16,700	4.95	26.52	2,700	14,200
JW Low Grade					
Inferred	21,100	4.09	18.55	2,800	12,600
Measured Total	**1,919,600**	**8.81**	**28.30**	**543,800**	**1,747,000**
Indicated Total	**1,270,500**	**5.85**	**10.01**	**238,800**	**408,800**
Measured + Indicated Total	**3,190,100**	**7.63**	**21.02**	**782,600**	**2,155,800**
Inferred Total	**405,400**	**5.32**	**7.33**	**69,300**	**95,500**

Note: *3 g/t Au is calculated as the cut-off grade for underground long-hole stoping. Note numbers may not add up due to rounding.

Mineral resources were estimated in conformity with generally accepted CIM *Estimation of Mineral Resources & Mineral Reserves Best Practices Guidelines*. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socioeconomic, and other factors.

Mineral reserves can only be estimated based on the results of an economic evaluation as part of a preliminary feasibility study or feasibility study. As such, no Mineral Reserves have been estimated by ACS. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined; however, ACS is of the opinion that it is reasonable to expect that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Mineral resources that are not mineral reserves have no demonstrated economic viability.

14.29 Red Mountain Grade Sensitivity Analysis

The mineral resources at Red Mountain are sensitive to selection of the reporting cut-off grade. To illustrate this sensitivity, the global model quantities and grade estimates of the Measured and Indicated resource are presented in Figure 14-39, and the Inferred resources are presented in Figure 14-40. The reader is cautioned that the grade and tonnages presented in these figures should not be misconstrued as a mineral resource statement. The figures are presented only to show the sensitivity of the block model estimates to the selection of cut-off grade.

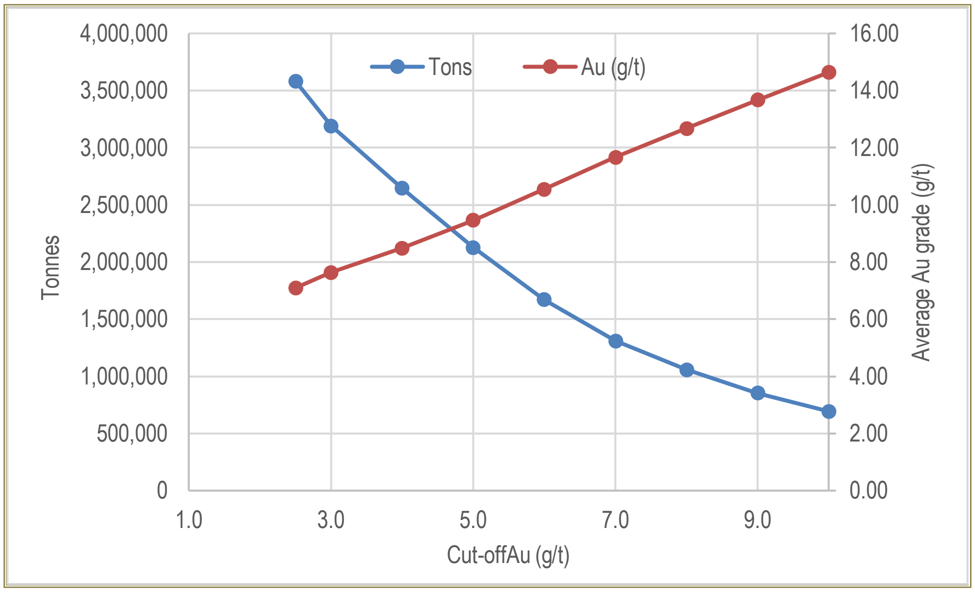


Figure 14-39: Grade Tonnage Curve for Measured and Indicated Mineral Resource at Red Mountain

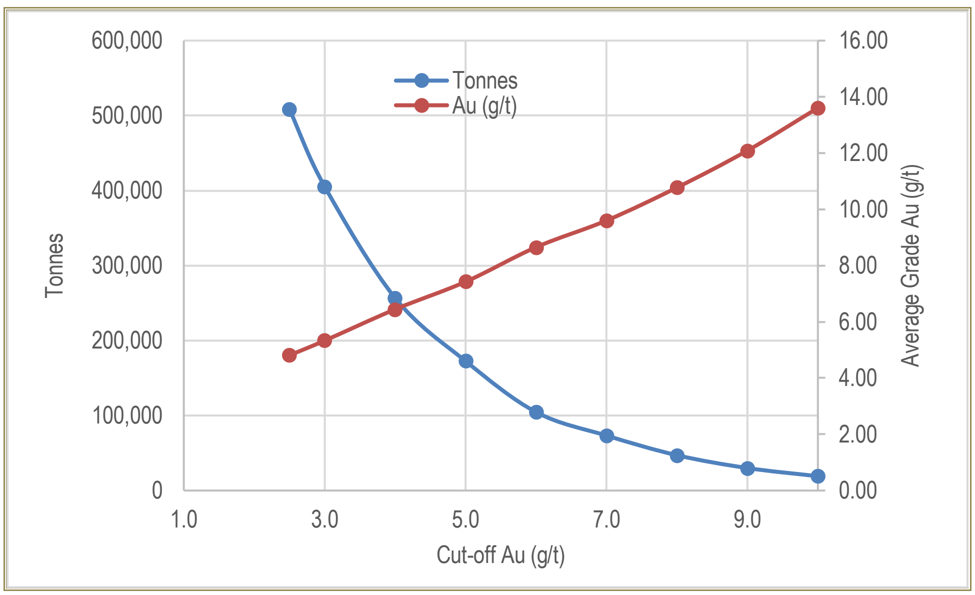


Figure 14-40: Grade Tonnage Curve for Inferred Mineral Resource at Red Mountain

The 2019 mineral resource estimate is reported at a 3.0 g/t Au cut-off grade. Cut-off grades may be re-evaluated considering prevailing market conditions (including gold prices, exchange rates, and mining costs). Table 14-53 summarizes the variability of the tonnage and grade at various cut-off selection. The mineral resource bases case is highlighted in bold and the reader is cautioned that the grade and tonnages presented in these figures should not be misconstrued as a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14-53: Variability of Red Mountain Mineral Resource at Various Cut-off Grades

Class	Cut Off (g/t)	Tonnes	Au g/t	Ag g/t	Oz Au	Oz Ag
Measured	>9.0	700,900	13.80	39.95	310,900	902,800
	>8.0	852,200	12.86	37.29	352,200	1,024,600
	>7	1,034,600	11.91	34.76	396,100	1,159,700
	>6.0	1,270,400	10.90	32.51	445,100	1,331,600
	>5.0	1,537,800	9.96	30.46	492,400	1,510,400
	>4.0	1,786,400	9.20	28.95	528,600	1,667,400
	>3.0	1,919,600	8.81	28.31	543,800	1,752,100
	>2.5	1,951,400	8.71	28.14	546,700	1,770,800
Indicated	>9.0	152,800	13.10	29.79	64,400	146,800
	>8.0	204,200	11.93	26.30	78,300	173,100
	>7	275,100	10.78	22.89	95,300	203,000
	>6.0	401,100	9.42	19.40	121,500	250,900
	>5.0	588,100	8.16	16.03	154,400	304,000
	>4.0	858,800	7.01	12.88	193,400	356,700
	>3.0	1,270,500	5.85	10.01	238,800	410,100
	>2.5	1,631,500	5.16	8.36	270,500	439,800
Inferred	>9.0	30,000	12.09	11.38	11,700	11,000
	>8.0	47,100	10.77	10.91	16,300	16,600
	>7	73,300	9.60	10.86	22,600	25,700
	>6.0	104,600	8.65	10.67	29,100	36,000
	>5.0	172,600	7.42	8.99	41,200	50,100
	>4.0	256,700	6.45	8.28	53,200	68,500
	>3.0	405,300	5.33	7.32	69,500	95,700
	>2.5	508,200	4.81	7.29	78,600	119,500

14.30 Previous Red Mountain Mineral Resource Estimates

Mineral resources have been estimated for the RMP in the past. IDM reported mineral resources in a technical report dated July 31, 2018. The 2018 mineral resources are summarized in Table 14-54.

Table 14-54: Previous Mineral Resource Statement for Red Mountain

Class	Tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Measured and Indicated	2,771,300	7.91	22.75	704,600	2,026,800
Inferred	316,000	6.04	7.6	61,400	72,000

The previous mineral resources are presented here only as a means of comparing the previous estimate with the current estimate presented in Table 14-52. As can be seen, the tonnage of the mineral resource has increased in the measured and indicated categories as a result of the 2018 drilling, and the gold and silver grades have dropped slightly in all categories due to the inclusion of lower-grade mineralized zones.

15 MINERAL RESERVE ESTIMATES

15.1 Mineral Reserve Block Model Preparation and Sub-blocking

The basic tenet of CIM guidelines and NI 43-101 reporting is that both a Resource and a Reserve must have the potential for economic extraction, which differentiates ore from a mineral occurrence. Based on this premise, the resources in Section 14 were determined using conservative cut-off grade (COG) estimates and incorporating mining thicknesses into wireframing.

The resource block models, including data on rock quality designation (RQD) were verified and tested.

Contemporary stope optimization software requires the use of sub-block models for improved spatial precision; therefore, the original resource models were 3 m x 3 m x 3 m these were sub-blocked to minimum block size parameters of 0.3 m x 0.3 m 0.3 m. The sub-block models were compared to the original ore percent resource models and found to be 99.64% in agreement.

PGP and RMP reserves were completed at a level of detail appropriate for a feasibility study, based on development and stope designs scheduled appropriately using the mineral resource models outlined in Section 14. All mining in the plan at both PGP and RMP employs underground methods from planned and existing adits that provide side-hill access for underground development.

15.2 Introduction

A Mineral Reserve is defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) within the CIM Definition Standards on Mineral Resources and Mineral Reserves, as adopted by CIM Council on May 10, 2014, as follows:

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

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource demonstrated by at least a pre-feasibility study (PFS). This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the Proven Mineral Reserve category implies that the Qualified Person (QP) has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A Probable Mineral Reserve is the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource, demonstrated by at least a PFS. Information on mining, processing, metallurgical, economic, and other relevant factors must be incorporated that demonstrate, at the time of reporting, that economic extraction of material can be demonstrated.

Reserves are subdivided based on the level of confidence in the classification. A Proven Mineral Reserve has a higher confidence level than a Probable Mineral Reserve.

CIM guidelines require that only material categorized as Measured or Indicated Resources be considered for potential Mineral Reserves. The reserve classifications used in this report conform to the CIM classification of NI 43-101 resource and reserve definitions and Companion Policy 43-101CP.

15.3 Cut-off Grade

At an operating mine, each mining method has its own reserve COG criteria predominantly based on differences in estimated cost and ability to recover ounces efficiently. PGP resources are stated at a 3.5 g/t COG (based on metal prices and preliminary cost estimates at the time of modelling) and 3.0 g/t COG for RMP, the block model wireframes in each area include material down to 2 g/t to allow for mining flexibility. The resources at RMP appeared more cohesive in nature, and looked to be amenable to bulk methods, so they were originally reported at a lower COG. PGP resources were varied and so they were conservatively reported at a higher Resource COG.

15.3.1 Premier Gold Project

Gold equivalent (AuEq) values were calculated using US$1,400/oz Au and US$17/oz Ag according to the following equation: AuEq(g/t) = Au(g/t) + Ag(g/t) x 17 / 1,400. On this basis the following COGs based on AuEq grade were used to estimate the economic potential of the stopes: long-hole (LH) = 2.85 g/t; inclined undercut long-hole (IULH) = 3.44 g/t; cut-and-fill (C&F) = 3.44 g/t; room-and-pillar (R&P) = 3.82 g/t; and development = 2.85 g/t.

15.3.2 Red Mountain Project

Most of the mine plan at RMP was completed in the fall of 2019, so there are minor differences in the COG approach.

AuEq values were calculated using US$1,300/oz Au and US$15/oz Ag according to the following equation: AuEq(g/t) = Au(g/t) + Ag(g/t) x 15 / 1,300. On this basis the following COGs based on AuEq grade were used to estimate the economic potential of the stopes: LH = 3.11 g/t; IULH = 4.0 g/t; C&F = 4.1 g/t; and development = 3.11 g/t.

A future reserve upgrade will consolidate the methodology at both sites. Analysis has shown that for similar direct mining methods, the differences between the sites is somewhat immaterial once losses, dilution and stope smoothing has been applied. The biggest difference between the two sites is that RMP COG must reflect longer haulage distances and lower mill recovery (Table 15-1).

Table 15-1: Preliminary Cut-off Grade Methodology—PGP

	L-H	I U L-H	C & F	R & P
Recovery Factors				
Mining Dilution	15.00%	25.00%	20.00%	15.00%
Mining Recovery	95.00%	95.00%	95.00%	82.00%
Processing Recovery	95.00%	95.00%	95.00%	95.00%
Gold Sale				
FOREX	1.316	1.316	1.316	1.316
Gold price (US$)	1,400.00	1,400.00	1,400.00	1,400.00
Gold price (C$)	1,842.11	1,842.11	1,842.11	1,842.11
Royalty (%)	2.00%	2.00%	2.00%	2.00%
Royalty ($/oz)	36.84	36.84	36.84	36.84
Refinery charges (/oz)	7	7	7	7
Selling Cost ($/oz)	43.84	43.84	43.84	43.84
OPEX $/t)				
Mining Costs	25	39	45	45
Mine Dev	35	35	35	35
G&A	8	8	8	8
U/G Haulage	7.5	7.5	7.5	7.5
U/G Services	8	8	8	8
Northern Logistics	5	5	5	5
Defin. Drilling	4	4	4	4
Rd Haulage	8	8	8	8
Processing Costs	29	29	29	29
All in Mining Cost ($/t)	129.5	143.5	149.5	149.5
Cut-off Grade (g/t)	2.85	3.44	3.44	3.82

15.4 Mining Shapes

To convert Resources to Reserves, mining economics and operational considerations must be applied to the resources, including allowing for recovery and dilution underground. As part of the mine planning exercise, the Measured and Indicated Mineral Resource block models were run through the Deswik Stope Optimizer (DSO) and evaluated under a number of different mining methods. The various results were then evaluated using standard operating practices and estimated mine costs, to determine the optimal approach in each instance. The overarching philosophy was to maximize the extraction of the ounces contained within the mining shapes at the lowest possible cost per ounce.

The Mineral Reserves included in the mine plan occur at two sites. The first is the Premier Gold Project (PGP) which has three separate mining areas: Silver Coin (SC) and Big Missouri (BM) about 7 km to the north of the Premier Mill, and Premier/Northern Light (PNL) adjacent and south of the mill. The second site is the Red Mountain Project (RMP) about 23 km southeast of the PGP Mill, which has one active mining area with three major zones: Marc, AV, and JW.

15.5 Dilution

Dilution comes in from both planned and unplanned sources. Planned dilution is material taken within the bounds of a stope layout, while unplanned material is that which comes from the hangingwall (HW) or footwall (FW) outside the stope boundary, often due to poor drilling and blasting practices (operational factors) or due to localized ground conditions (geotechnical factors). Unplanned dilution can also include a minor amount of backfill depending on mining method, backfill type and stope sequencing.

Unplanned dilution from the HW and FW was calculated as a percentage based on two parameters; the mining method; and the geotechnical characteristics in the area where the stope is located (e.g., at PGP the RQD value in the block model). In general, low RQD values incurred higher dilution and LH methods had higher dilution than R&P methods, mainly due to operational factors such as proximity to the production areas and selectivity of mining method. The R&P method is essentially mined with development-type crews and mobile equipment and ground support practices, utilizing short small diameter drill holes and bolting/screening ground support in smaller exposures to stabilize the rock mass, while LH stopes are drilled over greater distance with blind holes which can deviate, and the larger diameter longer holes can cause greater damage to the HW or FW in a stoping area. LH drilling accuracy can be well-managed, and LH stope sizes limited to ensure economic trade-off between stope height and dilution (higher stopes = less development = more dilution).

At PGP, stope by stope decisions were made weighing the economics and practicality of extracting parallel and sub-parallel ore zones; in some instances, decisions were made to take two inclined wireframes as a long hole stope rather than as separate C&F stopes because it maximized the ounces recovered at the lowest cost per ounce. In this circumstance the waste parting between the two wireframes is planned dilution but has some grade. If the dilution is in an Inferred block within a stope shape, the grade of the Inferred block can be utilized by the mine geologist or QP to assign a dilution grade to this material. If the material is not in an Inferred block, a background grade will be assigned to the material according to its location. The background grade was determined through a rigorous examination of the grade adjacent to resource wireframes using pierce points of diamond drill holes into the respective wireframe. The resource wireframes reflect a 2 g/t cut-off grade envelope, so material less than that cut-off was not included within the wireframe.

The following tables show parameters for determining dilution grades. Table 15-2 shows grades for unplanned dilution and Table 15-3 the method used for determining the grade of Inferred material captured as planned dilution within the boundaries of a stope. On initial evaluation, the unplanned dilution was given a grade of zero in order to not influence the mining method selection.

Table 15-2: Dilution by Mining Method and RQD

	Long-Hole (%)	IULH (%)		Dilution (g/t)		
		Ore (HW)	Waste (FW)	Premier	Silver Coin	Big Missouri
RQD: 90 -100%	14	10	8	0.67	0.81	0.52
RQD: 75 -90%	19	16	8	0.67	0.81	0.52
RQD: 50 -75%	25	23	8	0.67	0.81	0.52
RQD: 25 -50%	32	33	8	0.67	0.81	0.52
RQD: 0 -25%*	40	42	8	0.67	0.81	0.52

Table 15-3: Factor Applied to Inferred Dilution Tonnes within Stope Shape

Distance to Two DDHs		
Minimum (m)	Maximum (m)	Grade Factor
17.5	20	1
20	22	0.75
22	24	0.5
24	26	0.25
>26	-	0

Table 15-3 provides the methodology at PGP for varying the grade factor for internal stope dilution roughly based on variography where possible and is based on the distance of the block from the nearest two resource diamond drill holes (DDH). On the advice of the Resource QP, a block with an average distance of between 17.5 m to 20 m to two drill holes is assigned the Inferred block grade. There are three distance to drill hole categories that assign 75%, 50%, and 25% of the Inferred block grade until at an average distance of more than 26 m to two drill holes, the assigned grade for dilution material becomes zero.

At the PGP sites, the ore contacts are gradational, meaning that the HW and FW generally contain gold values, as opposed to a sharp geological contact between gold-bearing zones and barren rock. Therefore, dilutive material at the site generally has some grade associated with it. As long as this lower grade (marginal) ore does not displace higher grade material available to process in the mill, then it has a positive impact on net present value (NPV) and the material should be processed. This is the usual approach in underground mines to calculate marginal cut-off grade.

Similarly, at RMP the country rock immediately adjacent to the mining blocks has some grade which was assigned appropriately to dilution material based on location.

When stopes are successively mined in a vertical manner, backfill required as a floor to extract the mineralized material within LH and C&F stopes may cause some dilution to occur from mucking surfaces. Some material may be purposely taken to allow recovery of ore fines within the top layer of fill. Dilution from vertical exposures of backfill will be controlled by various levels of cement in the rockfill; typically, backfill dilution of 0.5 m has been allocated from vertical exposures and 0.25 m from the floor.

Dilution from backfill is a bigger consideration at RMP where the majority of the ore is mined in transverse LH stopes. This is mitigated by carefully planned use of CRF in primary stopes.

15.6 Recovery

Mining recovery is a function of the selected mining methods and incorporates potential losses from operating practices. In a LH situation, recovery is largely dependent on the accuracy of drilling and blasting techniques. When done improperly, it can result in material left behind in the planned stope that never reports to the mucking horizon. In production stoping with R&P or C&F, this is a lesser concern, as the headings can be visually inspected and rectified.

In all methods, minor amounts of material can be lost due to mucking practices caused by uneven floors, difficulty in remote mucking due to limited visibility or on occasion operator error or inexperience. Full-width development extraction under geological control of the bottom drill/mucking sill mitigates most of the floor condition issues, as well as mucking on top of rockfill (i.e., not relying on large-diameter stope slash holes to widen the bottom sill).

Enforcing standard practices and procedures and training can further mitigate many of the instances noted above. Based on general industry experience with the methods being applied at PGP and RMP, the expectation is that mining recovery will be in the range of 95%.

15.7 Mineral Reserves

The calculated Proven and Probable reserves based on the mine plans at PGP and RMP total 6.2 Mt at a grade of 5.9 g/t Au and 19.7 g/t Ag. The majority of precious metals recovered in this plan are from LH stoping methods. Reserve details are shown in Table 15-4 to Table 15-6. Previously stated resources are inclusive of reserves.

Table 15-4: Reserves by Category

Reserves by Category	Ore (t)	% of Tonnage	Grade			Ounces		
			Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
PGP								
Proven	-	-	-	-	-	-	-	-
Probable	3,631,898	100	5.45	19.11	5.69	636,805	2,230,964	663,920
PGP Total	**3,631,898**	**100**	**5.45**	**19.11**	**5.69**	**636,805**	**2,230,964**	**663,920**
RMP								
Proven	2,193,599	86.2	6.68	21.69	6.93	471,368	1,530,052	489,023
Probable	351,234	13.8	5.51	13.76	5.67	62,241	155,340	64,033
RMP Total	**2,544,833**	**100**	**6.52**	**20.60**	**6.76**	**533,609**	**1,685,392**	**553,056**
PGP & RMP								
Proven	2,193,599	35.5	6.68	21.69	6.93	471,368	1,530,052	489,023
Probable	3,983,133	64.5	5.46	18.63	5.68	699,046	2,386,304	727,954
PGP & RMP Total	**6,176,732**	**100**	**5.89**	**19.72**	**6.13**	**1,170,414**	**3,916,356**	**1,216,976**

Notes: CIM Definition Standards were followed for classification of Mineral Reserves
The Qualified Person for the Mineral Reserve Estimate is Frank Palkovits, P.Eng., of Mine Paste
AuEq values for PGP were calculated in the spring 2020 using $1,400/oz Au and $17/oz Ag with no allowance for silver recovery
AuEq values for RMP were completed in the fall 2019 at $1,300/oz Au and $15/oz Ag with no allowance for silver recovery
Based on current mining areas, Silver is an immaterial contributor to overall economic, but is recovered in the mill
Rounding may result in minor differences.

Table 15-5: Reserves by Source

	Ore	% of Tonnage	Grade			Ounces		
			Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
PGP								
Silver Coin								
Proven	-	-	-	-	-	-	-	-
Probable	1,794,374	49.4	4.83	14.47	5.01	278,646	834,495	288,863
Big Missouri								
Proven	-	-	-	-	-	-	-	-
Probable	809,282	22.3	7.15	12.20	7.30	186,073	317,390	189,897
Premier								
Proven	-	-	-	-	-	-	-	-
Probable	1,028,243	28.3	5.21	32.64	5.60	172,087	1,079,079	185,161
PGP Total								
Proven	-	-	-	-	-	-	-	-
Probable	3,631,898	100.0	5.45	19.11	5.69	636,805	2,230,964	663,920
RMP								
Marc Zone								
Proven	842,630	38.4	7.71	30.40	8.06	208,944	823,566	218,446
Probable	49,687	14.1	4.08	8.51	4.18	6,525	13,592	6,682
AV Zone								
Proven	958,899	43.7	6.34	16.50	6.53	195,570	508,694	201,439
Probable	126,854	36.1	7.22	19.50	7.44	29,428	79,510	30,345
JW Zone								
Proven	392,070	17.9	5.30	15.69	5.48	66,855	197,793	69,137
Probable	174,694	49.7	4.68	11.08	4.81	26,288	62,238	27,006
RMP Total								
Proven	2,193,599	100.0	6.68	21.69	6.93	471,368	1,530,052	489,023
Probable	351,234	100.0	5.51	13.76	5.67	62,241	155,340	64,033
PGP & RMP								
Proven	2,193,599	35.5	6.68	21.69	6.93	471,368	1,530,052	489,023
Probable	3,983,133	64.5	5.46	18.63	5.68	699,046	2,386,304	727,954
PGP & RMP Total	**6,176,732**	**100.0**	**5.89**	**19.72**	**6.13**	**1,170,414**	**3,916,356**	**1,216,976**

Table 15-6: Reserves by Mining Method

	Ore (t)	% of Tonnage	Grade			Ounces		
			Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
PGP								
Development Ore	258,142	7.1	5.73	22.94	6.01	47,577	190,417	49,889
Production Ore								
Longhole	2,108,982	58.1	4.53	21.56	4.79	307,076	1,462,095	324,881
IULH	390,841	10.8	6.30	16.28	6.50	79,190	204,606	81,675
C&F	144,300	4.0	6.00	11.10	6.14	27,854	51,513	28,479
R&P	729,633	20.1	7.46	13.74	7.63	175,108	322,334	178,996
PGP Total	**3,631,898**	**100**	**5.45**	**19.11**	**5.69**	**636,805**	**2,230,964**	**663,920**
RMP								
Development Ore	191,391	7.5	6.81	21.37	7.06	41,887	131,519	43,414
Production Ore						-	-	-
Longhole	1,837,576	72.2	6.47	21.64	6.72	382,291	1,278,557	397,044
IULH	500,991	19.7	6.57	15.67	6.76	105,900	252,401	108,813
C&F	14,876	0.6	7.38	47.91	7.91	3,530	22,914	3,785
R&P	-	0.0	-	-	-	-	-	-
RMP Total	**2,544,833**	**100**	**6.52**	**20.60**	**6.76**	**533,609**	**1,685,392**	**553,056**
PGP& RMP								
Development Ore	449,533	7.3	6.43	23.86	6.46	92,995	344,850	93,304
Production Ore								
Longhole	3,946,558	63.9	5.43	21.60	5.69	689,367	2,740,652	721,925
IULH	891,832	14.4	6.46	15.94	6.64	185,090	457,007	190,488
C&F	159,176	2.6	5.44	10.07	6.30	27,854	51,513	32,264
R&P	729,633	11.8	7.46	13.74	7.63	175,108	322,334	178,996
PGP & RMP Total	**6,176,732**	**100**	**5.89**	**19.72**	**6.13**	**1,170,414**	**3,916,356**	**1,216,976**

16 MINING METHODS

16.1 Introduction

The mine plan is based on the fully diluted Proven and Probable Reserves of 6.2 million tonnes (Mt) for the combined Premier Gold Project (PGP) and the Red Mountain Project (RMP) sites as presented in Section 15. The four mine areas at the two sites are optimally operated through a haulage ramp system with side-hill access from surface using new and refurbished existing underground infrastructure. Ramps will be utilized to haul ore and waste and provide access for personnel, equipment, materials, and services, while also forming part of the mine dewatering and air ventilation circuit.

As shown schematically in the Figure 16-1, production will commence at Silver Coin (SC) and Big Missouri (BM), followed by the Marc zone at RMP and the Premier/Northern Light (PNL) zone. Silver Coin and Big Missouri will be interconnected and will share systems, while Premier and the main zones at Red Mountain will operate independently.

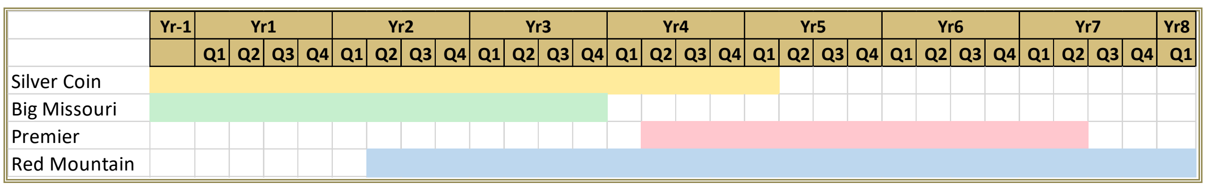


Figure 16-1: Generalized Life-of-Mine Development Schedule

As an operating philosophy, the approach taken was to use methods, systems, and approaches in a similar manner where appropriate for both sites to maximize operating efficiency while minimizing both capital and operating costs.

The entire underground mobile fleet including development, drilling, mucking and truck haul fleets will be shared amongst the four project areas. As each zone enters into development, then production followed by depletion, the appropriate equipment will be moved according to the overall combined schedule. It was conducive to this approach to use one common set of designs for shape and size of ramps, drifts, cross-cuts and production drill sills to ensure operational effectiveness across all project areas.

At PGP the mining strategy is predominately based on long-hole mining methods (~61%) with different variants of the long-hole technique required to adapt to particular geometries in SC and PR, while BM is primarily room-and-pillar (R&P) mining. At RMP, nearly 56% of all the reserves particularly within the AV-JW zone are amenable to using the Transverse Long-hole method. Reserves within the Marc zone will be mined by either the Longitudinal Retreat Long-hole or the Inclined Undercut Long-hole methods. The geometry of the Marc Zone is massive with a moderately steep dip whilst the AV-JW zones show a blend of moderately to shallow dipping (20° to 45°) and narrower ore lenses that require more flexible and intricate techniques.

Mining at SC will initially be accessed from existing portals, while BM will be accessed by a new portal and ramp developed from a bench in the existing S1 pit. The SC and BM areas will be joined by a connector ramp which will allow sharing of services and allow all ore haulage from both SC and BM to come out the same portal and travel down the Big Missouri Haul Road (BMHR). Water will also be directed out of this portal to be piped down the BMHR to the water treatment plant near the Premier Mill.

At Premier/Northern Light (PNL), initial access will be via slash and development of an existing access stub at the 4.5 level horizon, while water will continue to be directed to the existing 6 Level portal adjacent to the current water treatment plant.

Zones within Red Mountain will be serviced by a main ramp that connects to the existing upper portal and to the newly designed lower portal. A total of 17 levels are required to extract the reserves over the vertical extent of the orebody. The total lateral development requirements are estimated at 19,900 metres (m) over the life-of-mine (LOM). The AV-JW and Marc zones share a focal point located on Level 1700 where both sections of the main ramp servicing the upper and lower portal converge.

The dimensions of the main ramps at each area were determined according to required clearances for the selected mobile equipment, while also considering ventilation requirements during development and production. It was determined that a 4.5 m wide by 4.5 m high profile would be suitable for a 30-tonne haul truck, although it would be possible to use 42-tonne trucks upon removal of the ventilation ducts. In general, all the ramps are designed to be driven at a maximum of ±15% gradient while minimizing intersections with major geological structures and optimized to reduce development requirements to access each production area. The main ramps are fully serviced for ventilation, power (compressed air and electricity), communications, industrial water, and dewatering.

Re-muck bays are laid along the main ramp typically at 125 m intervals (to a maximum of 150 m) and will eventually be converted to store equipment and materials, sumps, refuge stations or explosives magazines. Level access crosscuts and haulage drifts are planned to be developed off the ramp following a similar profile of 4.5 m x 4.5 m. All infrastructure development, which would not be used for access, i.e., re-mucks, ventilation drifts and sumps, were designed with a 4.0 m x 4.0 m profile, (Figure 16-2).

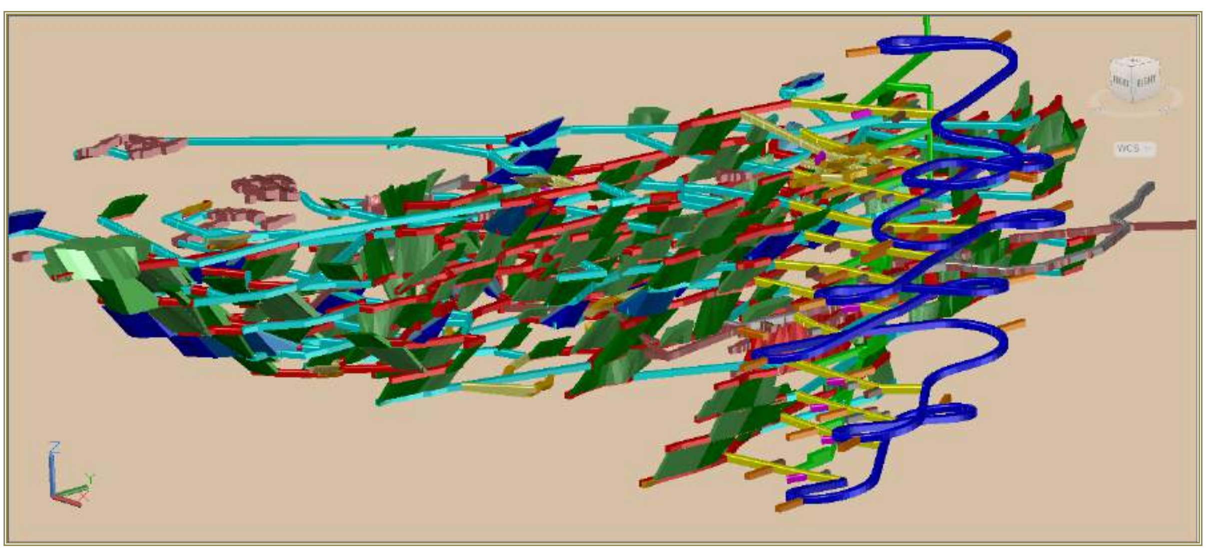


Figure 16-2: Isometric View of the Silver Coin Deposit and Related Development

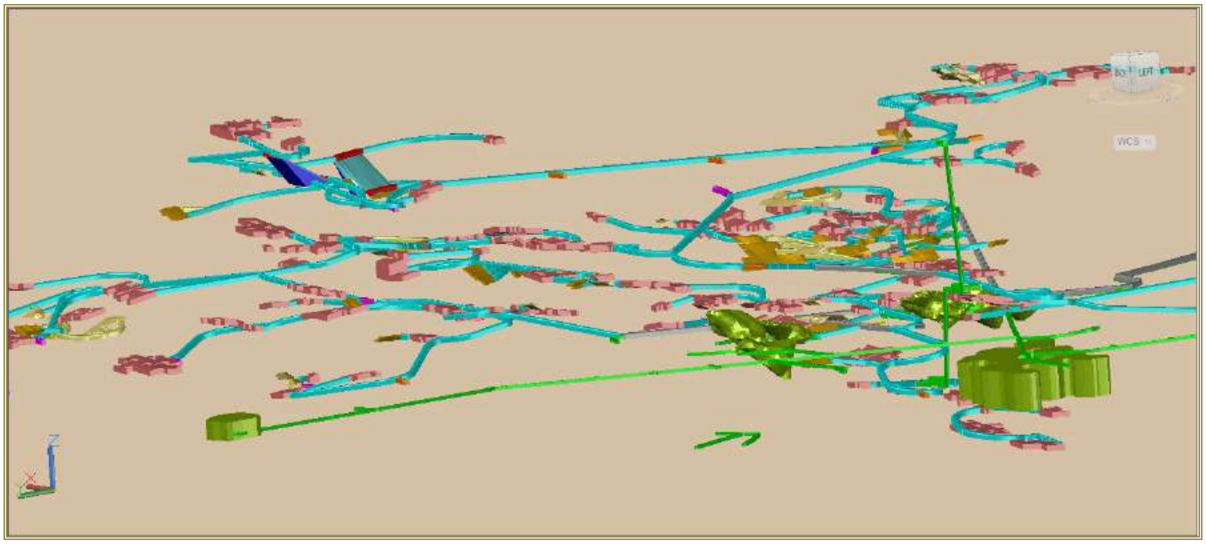


Figure 16-3: Isometric View of the Big Missouri Deposit and Related Development

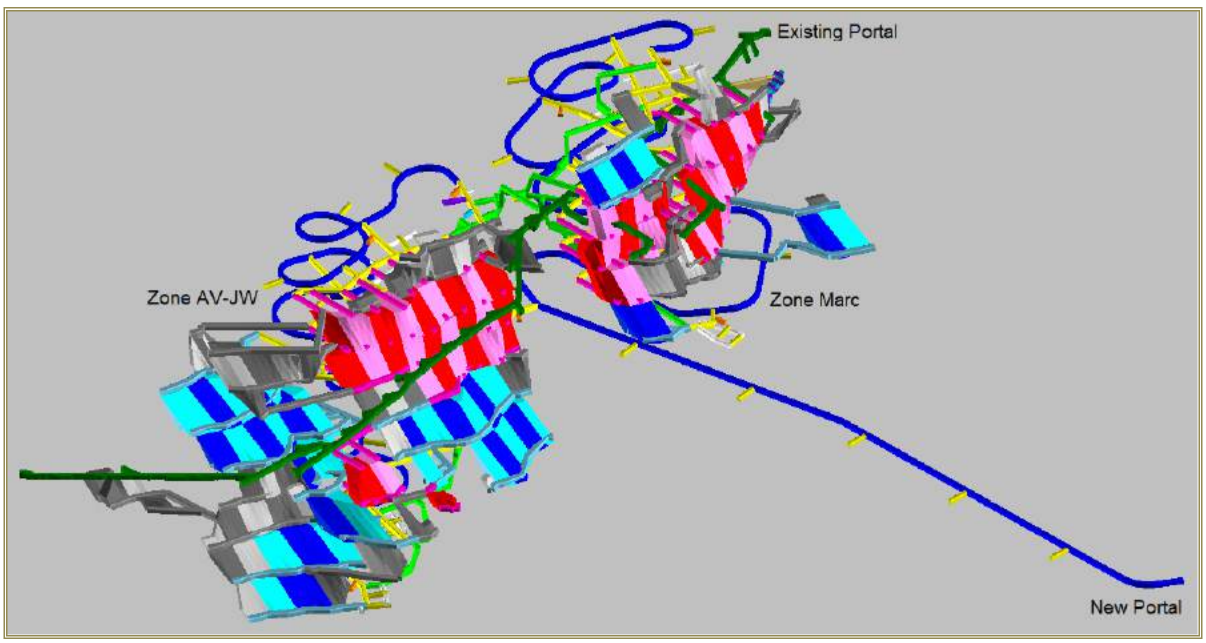


Figure 16-4: Isometric View of the Red Mountain Deposit and Related Development

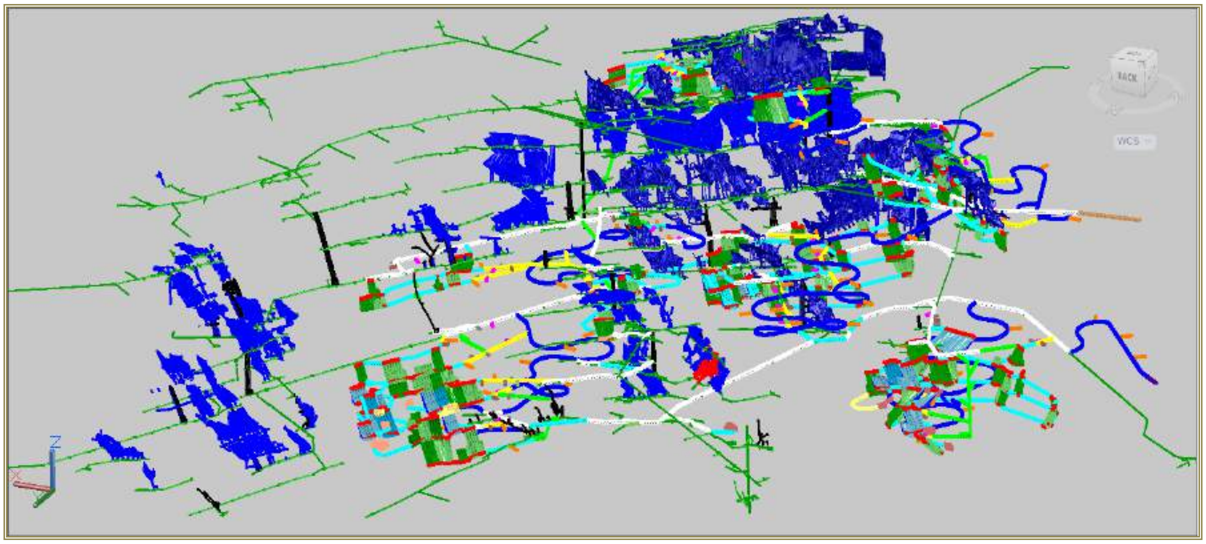


Figure 16-5: Isometric View of the Premier Deposit and Related Development

A common philosophy across all projects relies on a planned network of ventilation raises to bring fresh air from surface to key points in the mine where it will be distributed to every level, through new development or refurbished existing infrastructure. Within Red Mountain, this network will be composed of one fresh air intake located at the upper portal which in turn will be connected to raises independently serving AV-JW and Marc zones. Each section of raise will be equipped with ladders and safety bulkheads to provide a means of emergency egress. Exhaust Return Air will travel through the main ramp only. Raises are designed to be 3.5 m x 3.5 m square using Alimak mining equipment (Table 16-1). Similar approaches will be taken on common infrastructure for SC/BM, and at PNL existing levels will be slashed and connected to a common raise system, aided by existing natural ventilation and historical development.

Table 16-1: Design Parameters

Parameter	Design Criteria
Minimum mining width	2.0 m
Vertical distance between sublevels	25.0 m
Vent raise dimensions	3.5 m x 3.0 m square
Main ramp and haulage ramp	4.5 m wide x 4.5 m high (max 15% grade)
Main haulage drive and x-cuts	4.5 m wide x 4.5 m high (max 3% grade)
Ore drive	6.0 m wide x 4.5 m high (max 3% grade)

The overall LOM based on Proven and Probable Reserves is estimated at approximately eight years, including one year of pre-production followed by five years of full commercial production, which ramps down in year seven to close in Year 8. Production mining in each of the areas will progress according to the schedule shown in Table 16-2 and Figure 16-6.

Table 16-2: Premier Gold Project Generalized Production Schedule

Ore Production	Silver Coin	Big Missouri	Premier	Red Mountain
Transverse LH	N/A	N/A	N/A	Year 3 to 8
Longitudinal LH	Year 1 to 5	N/A	Year 5 to 7	Year 3 to 8
Incl. Undercut LH	Year 1 to 5	Year 3 to 4	Year 5 to 7	Year 3 to 8
Room-and-Pillar	Year 1 to 3	Year 1 to 4	Year 5 to 6	N/A
Cut-and-Fill	Year 1 to 5	Year 1 to 3	Year 6 to 7	Year 6

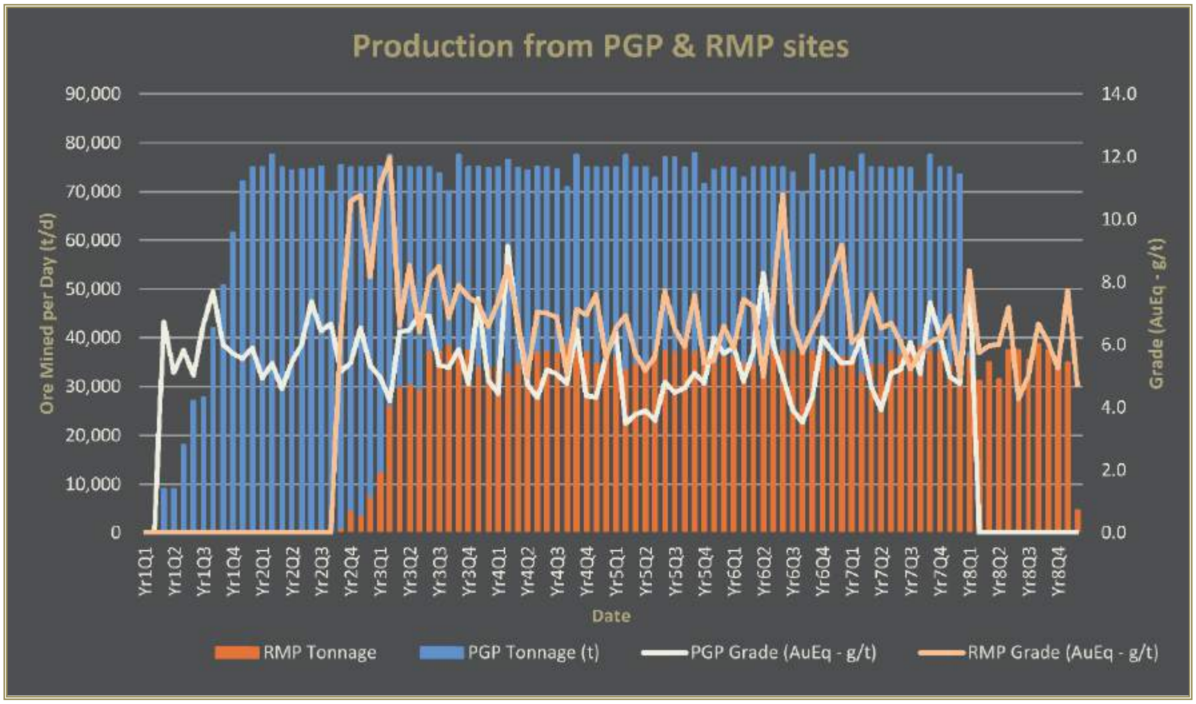


Figure 16-6: Production Schedule

16.2 Mining Method

16.2.1 Silver Coin and Big Missouri

Silver Coin (SC) and Big Missouri (BM) consist of numerous smaller isolated zones that require a strategy to minimize development while ensuring a cost-effective mining approach. Zones vary in dip, width, and length such that a flexible approach is needed. As development continues, additional definition drilling will further define and delineate the zones for optimization. Approximately 78% of the ore at SC can be extracted by longitudinal long-hole retreat, while flatter zones will be approached with inclined undercut long-hole (19%), at BM 81% will be mined by R&P and 14% by C&F methods. Old mining voids which were not historically filled will be used to allow quick disposal of waste rock, while also using these as a storage from which to backfill in effective larger campaigns.

16.2.2 Premier / Northern Light

Premier/Northern Light (PNL) had been widely developed and mined, such that the approach will be to use existing development as much as possible, initially for a portion of the ventilation and second egress. As development begins, definition drilling will further identify and delineate reserves and resources, as well as confirm historical stopes, allowing for greater detail and optimization. Most production is planned as long-hole (70%) and inclined undercut long-hole (19%) with minor amounts of R&P and C&F, depending on configuration of each stope; development waste will be readily available and used for rockfill or cemented rock fill to increase effectiveness of the approach. Historical unfilled stopes will be used to mitigate extensive rock haulage and disposal on surface, while also used for ground stability.

16.2.3 Red Mountain

The RMP mine plan will primarily use mechanized long-hole mining techniques (72% LH, 14% IULH) on most of the three mining areas, the zones share a general NE-SW strike, with dips varying between 30° and 80°. The zones range from 2 m to 40 m in width, a length of 70 m to 200 m in strike, and 60 m to 100 m in height.

The orebody geometry allows for a combination of transverse and longitudinal long-hole stoping as well as areas which can be mined on a pillarless basis where most of the stopes will be backfilled using cemented rock fill (CRF) and plain rock fill. A minor portion of the reserves (~1%) in very shallow dipping areas such as the upper part of Marc will be mined using the overhand C&F mining method.

Ore will be hauled by 30 tonne underground mining trucks via the main access ramp to the surface ore pad located at the lower portal. The broken ore will then be transported off-road by a local contractor from Red Mountain ore pad down to the PGP mill site.

16.3 Mining Method Selection

The proposed mining methods to be used at PGP and RMP have been used successfully in numerous operations globally and regionally that share similar ore body geometry. Local labour force and mining professionals alike are familiar with these methods and possess an extensive underground mining background in highly mechanized and narrow vein mining operations. In consideration of practicalities, the minimum mining width for long-hole stoping has been set to 2.0 m.

The various zones will be further subdivided into local production centers in order to increase the inventory of working areas to sustain the target production level. From the main ramp, a crosscut will be driven to reach the main level infrastructure where the ore drive will be developed easterly and westerly from the approximate midpoint of each production center.

16.3.1 Long-Hole

LH mining is a common practice in the industry, generally utilizing a top sill for drilling and a bottom sill for production mucking. The top and bottom sills are accessed from mine levels connected by central ramp systems.

LH stopes can generally be oriented in two directions, longitudinal or transverse. Transverse is used where the ore is continuous and generally of >10 m thickness, such that the stopes are oriented perpendicular to strike, and stopes are mined as panels in a primary-secondary configuration or a primary-primary configuration, generally working from centre out to the perimeter, nominally dictated by rock mechanics. A majority of the ore at RMP is mined in this way, but at PR, SC, and BM where the ore is narrower, most of the LH material comes from longitudinal retreat sequence, where the mining follows the strike (main orientation of the ore zones). In this method, stopes must retreat sequentially as the ore is extracted from the outer perimeter back towards a central access ramp, nominally dictated by logistics of accessing ore (LH longitudinal retreat). There are some minor amounts of ore in areas where wireframes congregate or are in proximity to each other where LH stoping is configured in whatever manner required to maximize ore extraction at minimal cost, while minimizing dilution.

In general, and regardless of stope orientation, a stope can be drilled from either top or bottom drill sills, for either downhole or up-hole drilling respectively, and in some cases, a combination of both; the decision is based on a configuration to maximize efficiency (drill and blast optimization), ore recovery and minimize dilution and impact on ground conditions. Drill hole size is determined by optimizing the same parameters, including drill hole accuracy and explosives placement optimization, in addition to drill-type selection (top hammer or down-the-hole hammer (DTH) equipment). Irregular and narrow zones will require shorter length and smaller diameter holes, whereas large bulk stopes can be extended in height to reduce development costs without the expense of loss of drilling accuracy. PGP and RMP will be optimized by orebody configuration and the above parameters, with drill hole dimensions from the 2-1/8" to 4" diameter holes in either ring or parallel hole layouts (rings) depending on the width of the material to be blasted and orientation of the ore body (dip and plunge), and irregularity. In all cases, production drill rings are loaded with ANFO or emulsion and slashed into a void created by a slot raise.

Due to the nuggety nature of the material at PGP, top and bottom sills will be developed under geological control, with each face channel sampled and high-grade material kept centered in the face as much as possible. As the drift is developed, stope definition drilling will follow behind the working face utilizing a bazooka drill or similar on a mobile rubber-tired self-propelled carrier. Final stope layouts and geometry will then be made upon which drill and blast layouts are produced.

When the ore from the production stope has been completely mined out by LHD, rock fill (RF) and CRF will be deposited into the stope. In some cases where there is no adjacent mining, RF from development waste will be placed in the stope rather than on surface. In some instances, limited exposure of the fill will require a portion of the fill to be cemented such that it will remain stable when the adjacent stope is blasted. The CRF is used to sparingly to create containment for the lower cost RF, or for future undercutting of previous stope bottom sills to recover ore within sill pillars.

Approximately 58% of the ore at PGP and 72% of the ore at RMP is taken with this method, for a combined total of 64% from both sites.

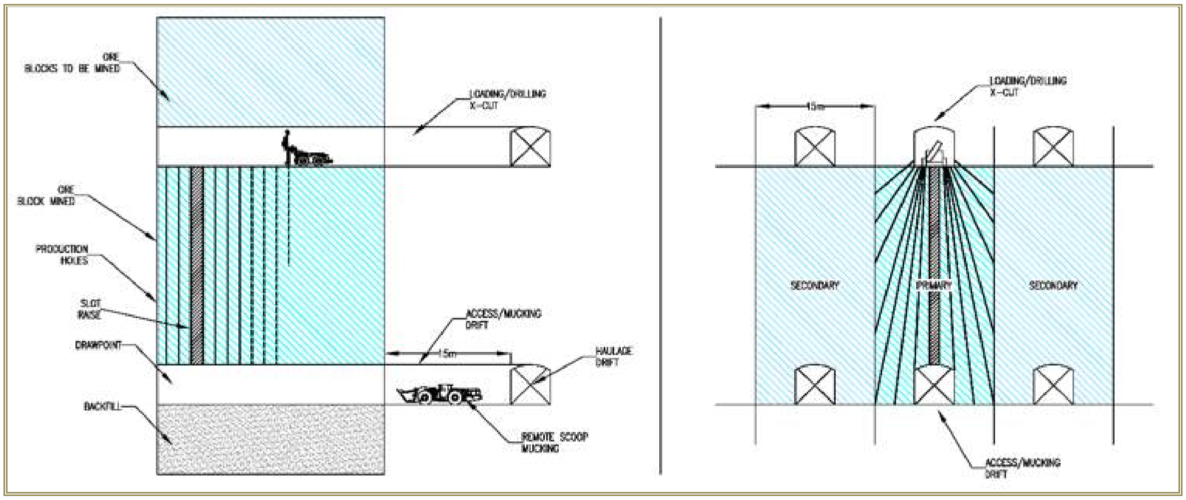


Figure 16-7: Typical Long-Hole Configuration

16.3.2 Inclined Undercut Long-Hole (IULH)

IULH is a long hole method that is sometimes referred to as the "Holloway" method due to its usage at that mine. It is basically a mechanized resuing method, used when ore is dipping at angles less than 55 degrees where material would not be able to fall or slide down along the footwall after blasting to the extraction point.

As in conventional LH mining, there are top and bottom sills developed in ore, but there is an additional secondary bottom footwall drift in waste, running parallel to the bottom sill ore drift. Initially waste material is drilled and blasted from the secondary FW drift and the rock removed and used for RF or CRF in a previously completed stope. A skin of waste or low-grade mineralization is left below the FW of the designed stope, accepting some dilution to maximize the recovery of the economic-grade material within the stope boundary. The amount of dilution is dependent on operational controls (drill and blast accuracy) and local geotechnical conditions.

Once the waste material has been mucked remotely leaving a void, long holes drilled from both top and bottom sills are blasted to break and fall (slashed) into the void and the ore material is remotely mucked out and trucked to the mill.

This method is generally more cost effective, requires considerably less waste development and has higher productivity than mechanized cut-and-fill (MCF), which is often used in narrow deposits with shallow dips in the range of 10 to 50 degrees.

Approximately 11% of the ore at PGP and 20% of the ore at RMP is taken with this method, for a combined total of 14% from both sites.

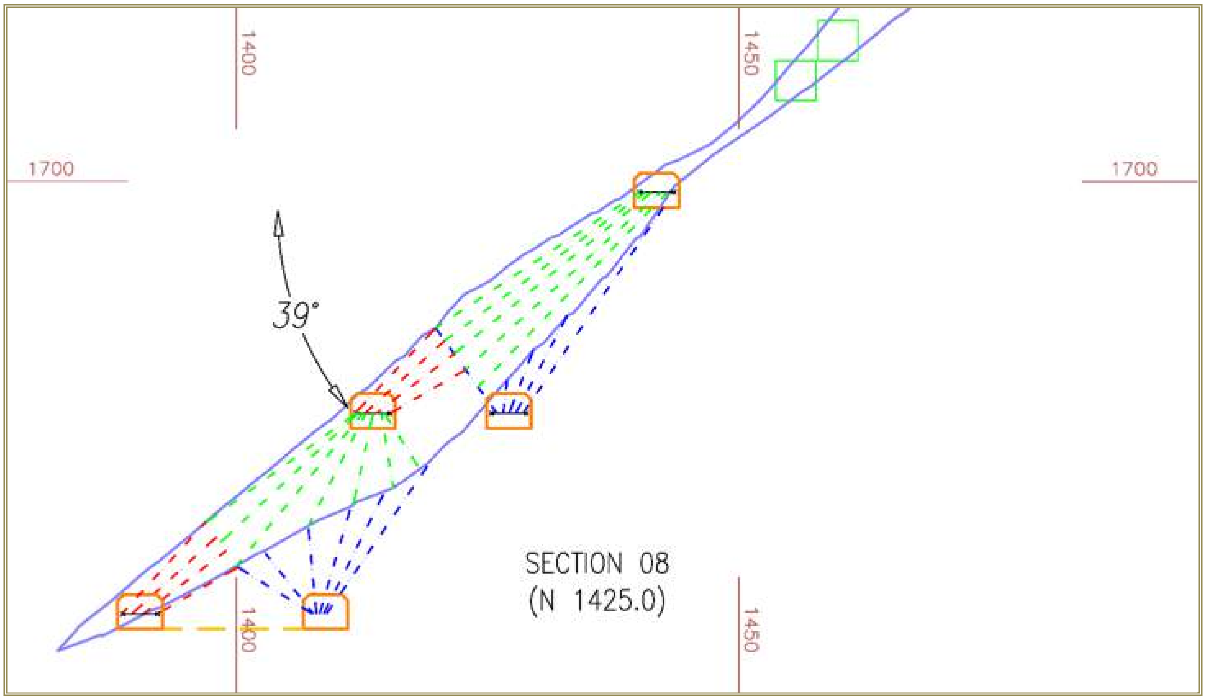


Figure 16-8: Typical IULH Stope Section

16.3.3 Mechanized Room-and-Pillar

In areas where the ore has a generally regular HW, limited thickness and is flat lying or gently inclined such as western deeper areas at Premier and most of Big Missouri, mechanized room-and-pillar (MRP) or inclined MRP will be used. The conventional approach is to drive a development ore heading into the stope with crosscuts between pillars forming a checkerboard pattern of mined out larger rooms with smaller pillars left for support. In operational practice, although the stope is planned as a regular checkerboard, and where a single cut is to be made, pillars are usually left in low grade or waste areas, this is done under the guidance of a geotechnical engineer who assesses the local conditions and acceptable spans.

On a first pass basis, 60% to 70% of the ore in an area will be mined, with a total of about 85% potentially mined near the end of the mine life utilizing a pillar recovery program retreating out of the mining area. (Note, where ore thickness is greater than that which can be extracted by a single cut, multiple cuts can be taken, thus this method is then referred to as post-pillar C&F (PPC&F), and pillars do not shift from cut to cut. It is not envisioned that this method will be needed at this time).

Productivity can be fairly high because the jumbo and mobile fleet of LHDs, scissor-lifts/bolting trucks, and haul trucks will be used in a multi-heading situation maximizing their utilization. Jumbos being used in capital development nearby can also be temporarily used to supplement production equipment if there is not a waste heading requiring immediate use of that equipment.

Approximately 20% of the ore at PGP (mostly in Big Missouri, 81%) and 0% of the ore at RMP is taken with this method, for a combined total of 12% from both sites.

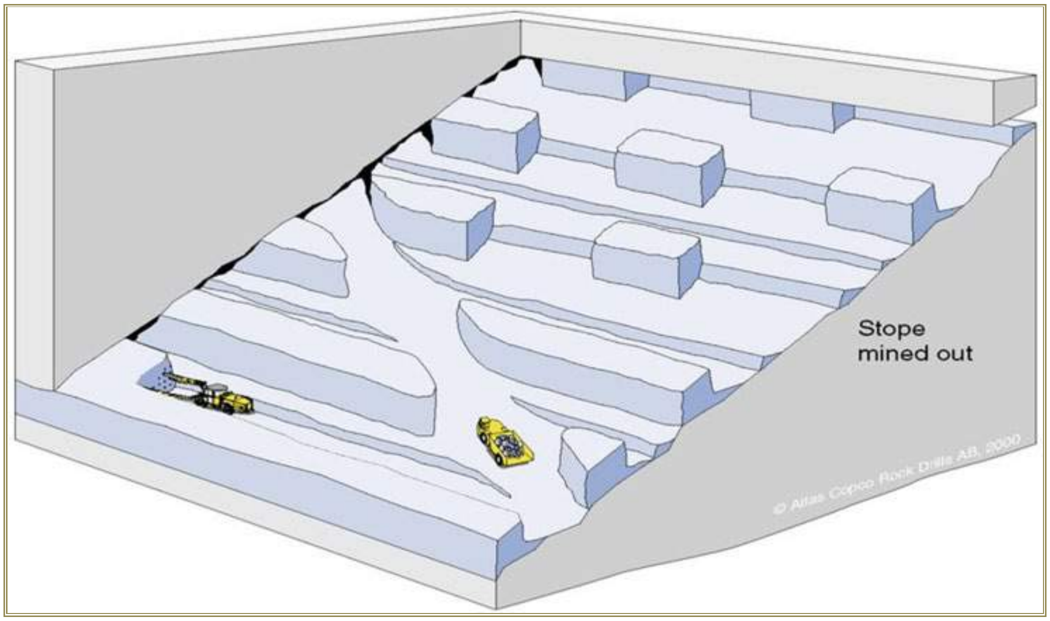


Figure 16-9: Typical MRP

16.3.4 Mechanized Cut-and-Fill

Cut-and-fill (C&F) mining has a long history of use in the industry and can be done as a captive stope with small equipment (jacklegs and slushers), mining and filling smaller cuts of 2 m to 2.4 m or as a MCF operation utilizing a ramp access and typically jumbos and LHDs taking bigger cuts.

MCF utilizes a footwall ramp to access successive stope lifts or cuts, typically 4 m or 5 m high depending on ore orientation and geotechnical considerations. The grade of the FW ramp will often depend on the strike length of a stope or group of stopes, with the general idea being to drive a crosscut from the ramp to the ore somewhere near the centre of the stope to allow for two ore headings in opposite directions along strike to reach higher productivity and cost-effectiveness.

The FW ramp is driven at a standoff distance from the ore zone that is determined by local ground conditions and the number of lifts in the stope. The first crosscut access from the ramp is driven downward to the first cut typically no greater than 15% depending on what a fully loaded LHD is capable of moving out of the stope. The ore is driven on the initial access along strike perpendicular to the FW crosscut in both directions to the extent of the stope design; when this is completed the stope is filled with CRF if it is a bottom cut (MC&F from below will extract ore up to the base of the CRF, thus requiring competent consolidated fill on the bottom cut). If there is no ore to be mined below a sill, RF without cement can be used to mine each successive cut).

After backfilling is complete the back of the access drift is slashed (Take-Down-Back or TDB) and the rock provides a floor to access the next stope cut, where the stoping process is repeated. This continues until the ramp and crosscut access reaches the maximum grade that an LHD can work on, approximately 20%. Then the next group of stope cuts is accessed from either above or below this series of cuts. This is dependent on the ore geometry and approach to mining.

Approximately 4% of the ore at PGP and <1% of the ore at RMP is taken with this method, for a combined total of 2% from both sites. The remaining ore comes from development (~8%).

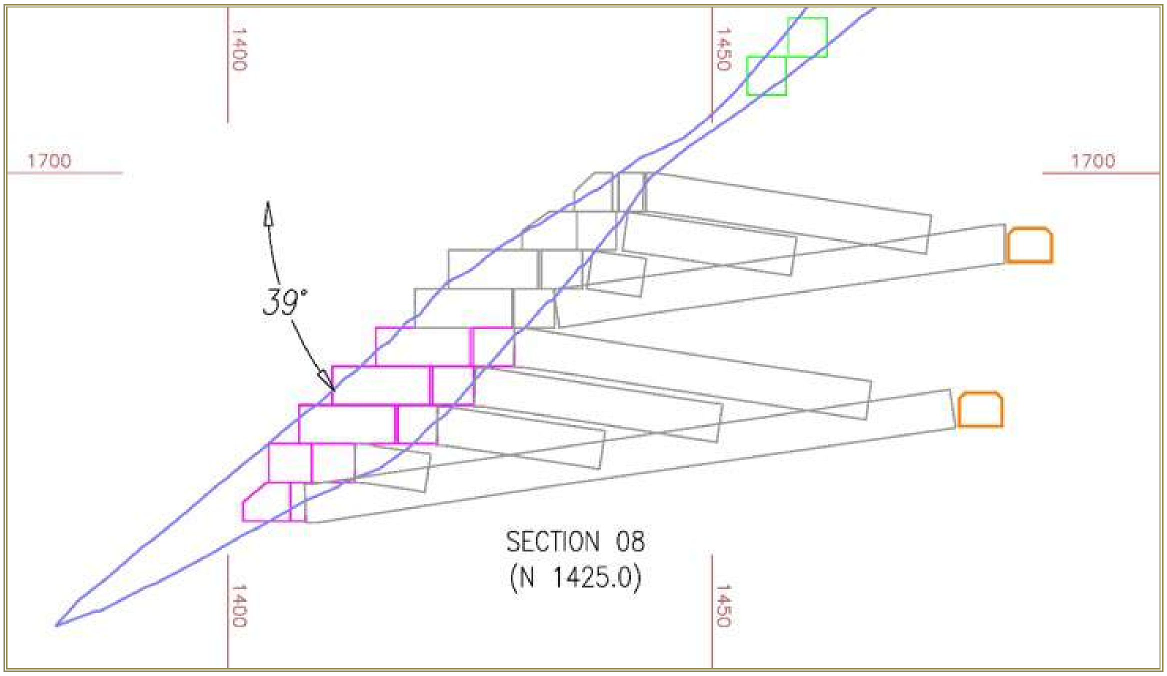


Figure 16-10: Typical MCF Stope

As mentioned above, productivity also has an impact on a stoping choice. Drilling longer holes with a specialized drill rig over a longer distance to take out a large volume is generally more productive than using a jumbo to mine along strike in a conventional narrow MCF stope. In this specific case, there are only two headings available in a stope and they must sequence/interact with each other in terms of unit operations such as drilling, blasting and ground control, so many more stopes are needed to fully use the equipment, limiting the ability to enhance productivity/lower unit costs. In addition, a FW ramp is required to access each successive cut in the stope with significant additional barren rock development required to regain stope access

16.4 Geotechnical Assessment

SRK conducted field investigations at the Red Mountain Project site to characterize the structural and rock mass conditions and undertook the appropriate analysis of the underground mine and infrastructure design in support of a feasibility level study. The geotechnical assessment focused on the three primary ore bodies, namely Marc, AV, and JW, which were targeted during the 2016 drill program. A three-dimensional structural model and representative geotechnical domains were developed for the advancement of the geotechnical studies and the development of geotechnical design recommendations. Geotechnical drilling and underground mapping from the 2016 Red Mountain field program formed the basis of the geotechnical interpretation. Historical geotechnical data from preliminary resource investigations in the early 1990's was also considered during the geotechnical investigations and evaluations.

At PGP Ascot relied on Dr. Paul Hughes P.Eng. at the Norman B. Keevil Institute at UBC for geomechanical strength testing of core material at the PGP site, as well as historical site information to determine expected mine conditions for cost and productivity estimates.

16.4.1 Rock Mass Characterization

Red Mountain Project

SRK conducted field investigations at RMP to characterize the structural and rock mass conditions and undertook the appropriate analysis of the underground mine and infrastructure design in support of a feasibility level study. The 2016 feasibility geotechnical data acquisition/investigation program included the following:

- Geotechnical Core Logging: Eleven drill holes had a combined resource and geotechnical purpose. Three boreholes were drilled into Marc, three into AV, and five were drilled into JW.
- Drilling was primarily conducted in eastern or western (mine grid) trending orientations. Five boreholes were orientated, and one additional borehole was surveyed by tele-viewer. Major faults were characterized by changing drill hole orientation to intersect these structures.
- Rock Strength Testing: Point load testing was undertaken approximately every 3m during core logging and core samples from the ore zone and surrounding host rock (hanging wall and footwall) were collected for laboratory strength testing. A total of 29 unconfined compressive strength (UCS) tests and 8 triaxial compression tests were conducted.
- Core Photo Logging: Core photos of the entire 2016 resource drilling program were reviewed and qualitatively geotechnically assessed.
- Underground Mapping: Accessible areas of the existing underground excavations (approximately 875 m) were geotechnically and structurally mapped.

Premier Gold Project

In early October 2019, Ascot commissioned EXYN technologies of Philadelphia to execute an autonomous drone survey of portions of the existing underground mining areas at Silver Coin and Big Missouri. The observations of these workings showed large excavations (50 m long x 30 m wide and 50 m high x 35 m with a thickness that varies from 3.5 to 5.5 m on strike) to be long term stable without any backfill.

Litho-structural modelling, using Leapfrog Geo and Edge software, and block model generation was achieved by InnovExplo team to assist the geotechnical engineering study for the three mine areas (Big Missouri, Silver Coin and Premier-Northern Lights). The main goal was to identify geological features related to low-RQD domains (RQD <50) and model the extent of these features. The resulting shapes can be used to avoid planning infrastructure or development in or near domains of poor ground quality. A second objective was to create and implement RQD block models to evaluate the average stope RQD value and ground support requirements while providing guidance on dilution.

In addition to the observations at Silver Coin and Big Missouri, Ascot has relied on observations of the historical Premier Mine drifts and accessible stopes, in addition to the rock strength analysis; the approach within this study for future mining has been to use conservative stope sizes and shapes to maintain safe operations with reasonable productivity and control of dilution and recovery. Within those limited areas where notable ground structures are known to be present, Ascot has taken a pragmatic approach to employ both diamond drilling and down-hole inspection technologies to assess stope-by-stope geotechnical analysis and reconfigure stope designs to ensure ore can be recovered safely and economically.

Since historical stope sizes and shapes are larger than those used within the study, optimization of stope sizes and configurations will continue as part of the continuous process/quality improvement typically used in the industry.

16.4.2 In-Situ Stress

In situ stresses have not been directly measured for either of the main project sites. Stress orientations have been assumed based on a review of the World Stress Map (Heidbach et al., 2014) while magnitudes have been estimated from typical common ratio tables for stress near ground surface (Table 16-3). Lithostatic stress has been assumed as the base case to determine the stress magnitudes. A sensitivity analysis was performed on the stress conditions using numerical modelling for both higher stress and isotropic stress conditions for the models made for RMP.

Table 16-3: Principle Stress Ratios and Orientations Relative to the Mine Grid

Stress	Trend	Plunge	K Ratio
σ_1	Horizontal	0 °	1.5
σ_2	Horizontal	0 °	1.5
σ_3	0 °	90 °	1

Source: SRK, 2017

16.4.3 Intact Rock Strength Testing

Red Mountain

The intact rock strength testing was carried out at Queen's University using laboratory rock tests and geotechnical rock mass classification data to determine the rock mass material properties and the backfill applied in the numerical modelling. Core samples were selected form mineralized zones and surrounding rock mass. The intact rock strength parameters based on laboratory testing and interpreted Hoek-Brown strength parameters are presented in Table 16-4.

Table 16-4: Intact Rock Strength Parameters

	Parameters	Rock Results
Laboratory Intact Rock Strength Results	Unconfined Compressive Strength (UCS) (MPa)	105
	Young's modulus (E) (GPa)	24.9
	Poisson's ratio (n)	0.14
Hoek-Brown Intact Rock Strength Parameters	Geological strength index (GSI)	60
	Hoek-Brown material coefficient (mi)	16

Source: SRK, 2017

Premier Gold Project

The intact rock strength test was carried out at UBC for samples from various PGP zones to serve as a basis for the design of the portals. A limited number of tests were completed generally distant from the mining sites. The results of these tests although informative, and generally confirmative of observed ground conditions were of limited use in the design and the sequence of the mine, so other methods were employed. Average parameters for PGP are summarized in Table 16-5.

Table 16-5: Average Intact Rock Strength Parameters

	Parameters	Rock Results
Laboratory Intact Rock Strength Results	Unconfined Compressive Strength (UCS) (MPa)	112
	Young's modulus (E) (GPa)	43.5
	Poisson's ratio (n)	0.30

Source: Ascot, 2019

16.4.4 Rock Mass Classification

Red Mountain

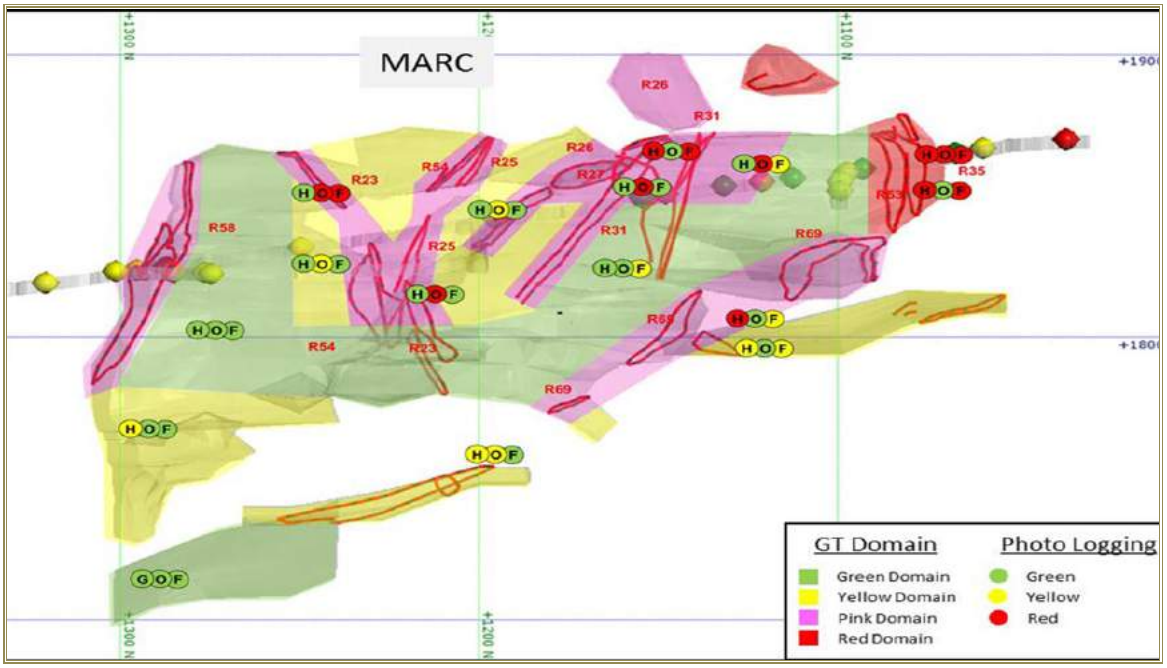
The rock mass classification is based on detailed geotechnical core logging, underground geotechnical mapping, and laboratory testing. The rock mass classification was carried out by systems such as RMR and RQD, IRS, Q' and rock mass jointing, to represent the geotechnical domain. Thus, four main geotechnical domains were identified and named green, yellow, pink and red. Table 16-6 presents a summary of the rock mass classification results.

Table 16-6: Rock Mass Parameters for Each Geotechnical Domain

Geotech Domain	Photo Rating	Intact Rock Strength (IRS)	RQD (%)	Jn	Jr	Ja	Q'	RMR_{90}
Green	1	105	95	6	2	2 – 4	7.9 – 15.8	60 – 65
Yellow	2	105	80	9	2	2 – 4	4.4 – 8.9	55 – 60
Pink	3	105	80	9	2	2 – 4	4.4 – 8.9	55 – 60
Red	4 – 5	50	20 – 60	9 – 15	1 – 2	3 – 4	0.1 – 4.4	25 – 55

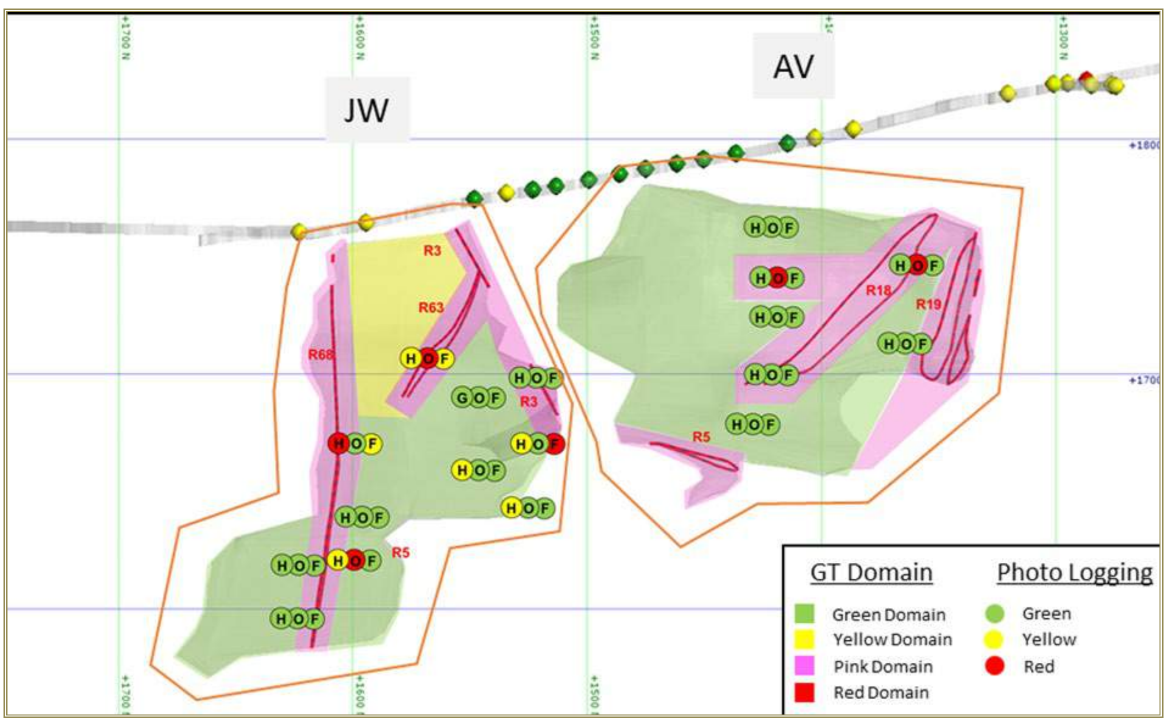
Source: SRK, 2017

The geotechnical domains were assessed for each of the orebodies: Marc (Figure 16-11), AV, and JW (Figure 16-12). The geotechnical data, core photographs, structural model, and recommended mining methods were considered.



Source: SRK, 2017

Figure 16-11: Geotechnical Domains within the Marc Zone



Source: SRK, 2017

Figure 16-12: Geotechnical Domains for the JW and AV Zones

Premier Gold Project

Spatial analysis of RQD and block model estimation were used to design mining infrastructures and evaluate the requirement of ground support. The combination of lithological modelling, structural modelling and block model calculation using Leapfrog Geo and Edge software packages was used to develop mine design and infrastructure layouts. The Ascot database includes tables of drill hole intervals (Au and Ag assays, lithologies, rock strength laboratory measurements, geotechnical parameters, RQD values, year of drilling, and names of geologists), oriented structural measurements (n: 2012), and a photographic drill core library for the 2019 DDH (n: 307). A ductile-brittle fault network has been modelled for the Big Missouri, Silver Coin and Premier-Northern Lights mine areas. Some fault zones developed at lithological contacts, whereas others are associated with mineralized ore zones.

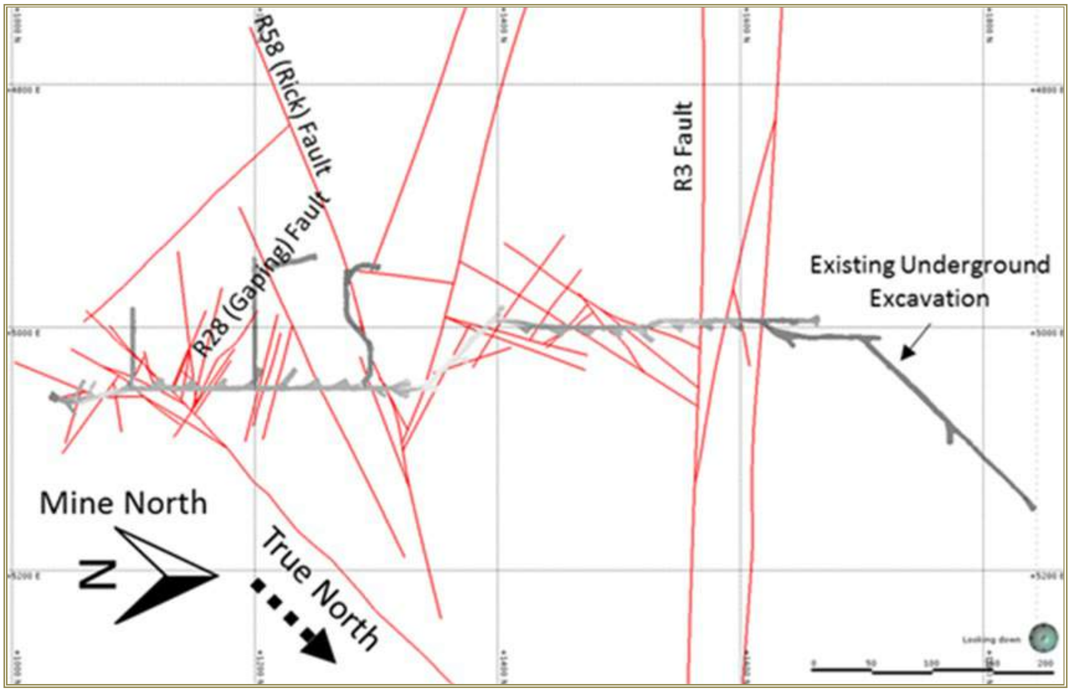
The RQD block models suggest a near-surface fracture zone (up to 100 m thick), identified mainly in the Premier-Northern Lights (PM_Fault 6) and Silver Coin areas (SC_Fault 7). At Premier-Northern Lights, this fracture zone is associated with the dacitic volcaniclastic units of the Betty Creek Formation and its contact with the underlying andesite flows of the Unuk River Unit. In addition, low-RQD domains with metre-scale thickness and low lateral continuity are likely part of an anastomosing network of ductile-brittle fault zones. Block modelling results for each area indicate that the fault domain RQD ranges from 26 to 74, with an average value of 55.

Empirically, recent visual observations and drone surveys (see Section 16.4.8) were undertaken within historical development and production stopes; these illustrate good ground stability after >20 years of non-backfilled and unsupported openings with stope dimensions of 50 m x 30 m x 35 m

16.4.5 Major Structures and Rock Mass Jointing

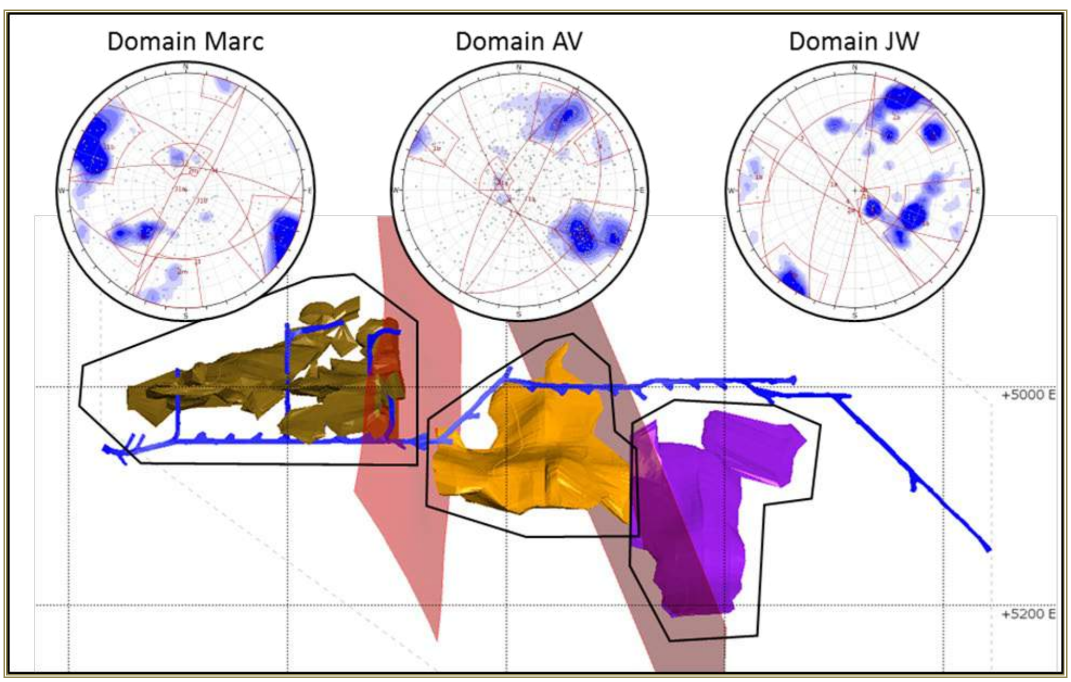
Red Mountain

Based on a three-dimensional structural model, the structural geology revealed three dominant trends for the deposit (Figure 16-13). Thus, three individual structural domains were identified based on joint set orientation and the impact of the joint sets on kinematic stability assessed. Data collected through underground mapping, oriented core logging, and the tele-viewer survey indicated similar joint orientations across the Project, with local rotation of some features. The domains are bound by the offsetting faults and correspond with the mineralized zones Marc, AV, and JW (Figure 16-13).



Source: SRK, 2017

Figure 16-13: Plan Section through Existing Infrastructure Showing the Traces of the 2017 Structural Model and the Existing Underground Excavations)



Source: SRK, 2017

Figure 16-14: Structural Domains for Red Mountain Orebody Areas

Premier Gold Project

Based on spatial and statistical analysis, low-RQD domains were highlighted according to each of zones, with a total of 24 brittle-ductile fault zones modelized (Table 16-7), including their damage zones. Thus, 9 faults were identified for Big Missouri, 7 for Silver Coin, and 8 for Premier. Due to lack of orientated drill holes and recent underground mapping no joint orientation were obtained or validated for the PGP area. RQD value and historical knowledge of the site were used to estimate rock mechanics requirements.

Faults generally have shallow to moderate dips to the west and strike to the north or northeast (Table 16-7). Faults typically contain a core zone (0.5 m to 1 m) with moderate RQD values (50 to 70), and a surrounding fracture zone (2 m to 3 m) with low RQD values (25 to 50). In the Big Missouri area, fault surfaces are also constrained by oriented-core structures. Most of those faults occur at the contact between porphyry dykes and andesitic units or within quartz-pyrite-sericite ore zone breccias.

In the Premier-Northern Lights area, three fault domains – PM_Fault 8, PM_Fault 2, and PM_Fault 1 – appear to have a major control on orebody geometry. The core zones (1 m or less) are characterized by cohesive schistose material with alteration minerals such as sericite, chlorite, and quartz. The damage zones contain breccia, gouge and mud that are often associated with quartz breccia, chlorite-filled fractures, quartz-pyrite veinlet stockworks and quartz-sericite-pyrite-carbonate-(kaolinite-chlorite) alteration zones.

Table 16-7: Low-RQD Domains for Each Zone

Deposit	Fault	Mean RQD	Standard Deviation	True Thickness of Fault Zone (m)	Main Direction	Structure Type
All deposits	N/A	68,8	25,9	N/A	N/A	N/A
Big Missouri (All)	N/A	65,9	25,8	N/A	N/A	N/A
Big Missouri (Faults)		29,6	21,2			
	BM_Fault1	30,6	22,7	0,8	N-S	Lithological brecciated contact, ductile fault zone
	BM_Fault3	26,2	19,6	2,0	N-S	fragile-ductile fault zone
	BM_Fault4	33,8	22,9	1,5	NW-SE	fragile-ductile fault zone
	BM_Fault5	21,6	17,4	0,3	NW-SE	fragile-ductile fault zone
	BM_Fault7	25,2	16,2	0,5	N-S	fragile-ductile fault zone
	BM_Fault8	42,1	24,6	1,4	N-S	Mineralized breccia zone
	BM_Fault9	22,2	14,4	2,0	NW-SE	Mineralized breccia zone
	BM_Fault2	34,3	21,0	0,3	N-S	Near-surface fracture zone
	BM_Fault6	16,4	8,3	0,2	N-S	fragile-ductile fault zone
Premier-Northern Lights (All)		69,7	25,8	N/A	N/A	N/A
Premier-Northern Lights (Faults)		33,5	23,1			
	PM_Fault1	20,5	16,7	2,5	N-S	Lithological contact
	PM_Fault2	34,8	23,0	0,1	NE-SW	Mineralized breccia zone
	PM_Fault3	27,5	19,4	0,3	NE-SW	Lithological contact
	PM_Fault4	29,6	20,6	0,4	NE-SW	Mineralized breccia zone
	PM_Fault5	33,0	19,6	1,4	N-S	Lithological contact

Deposit	Fault	Mean RQD	Standard Deviation	True Thickness of Fault Zone (m)	Main Direction	Structure Type
	PM_Fault6	35,2	24,4	0,3	NE-SW	Near-surface fracture zone
	PM_Fault7	38,0	23,8	0,3	N-S	Lithological contact
	PM_Fault8	41,1	19,5	0,2	NE-SW	Mineralized breccia zone
Silver Coin (All)	N/A	75,0	25,4	N/A	N/A	N/A
Silver Coin (Faults)		42,8	24,3			
	SC_Fault2	47,3	20,1	0,8	N-S	Lithological contact
	SC_Fault3	40,9	20,5	1,0	N-S	Mineralized breccia zone
	SC_Fault4	52,2	19,0	1,0	NE-SW	Mineralized breccia zone
	SC_Fault6	37,8	21,5	0,4	N-S	fragile-ductile fault zone
	SC_Fault5	52,5	20,8	0,1	N-S	fragile-ductile fault zone
	SC_Fault7	46,6	25,1	0,7	N-S	fragile-ductile fault zone
	SC_Fault1	21,0	16,4	2,0	NE-SW	Near-surface fracture zone

RQD block models and the fault zone models were generated for the three deposits (Big Missouri, Premier, and Silver Coin) in the PGP area and served as the basis for the design of mining infrastructure. Figure 16-15 to Figure 16-17 show the mining infrastructure modelling carried out using Deswik software.

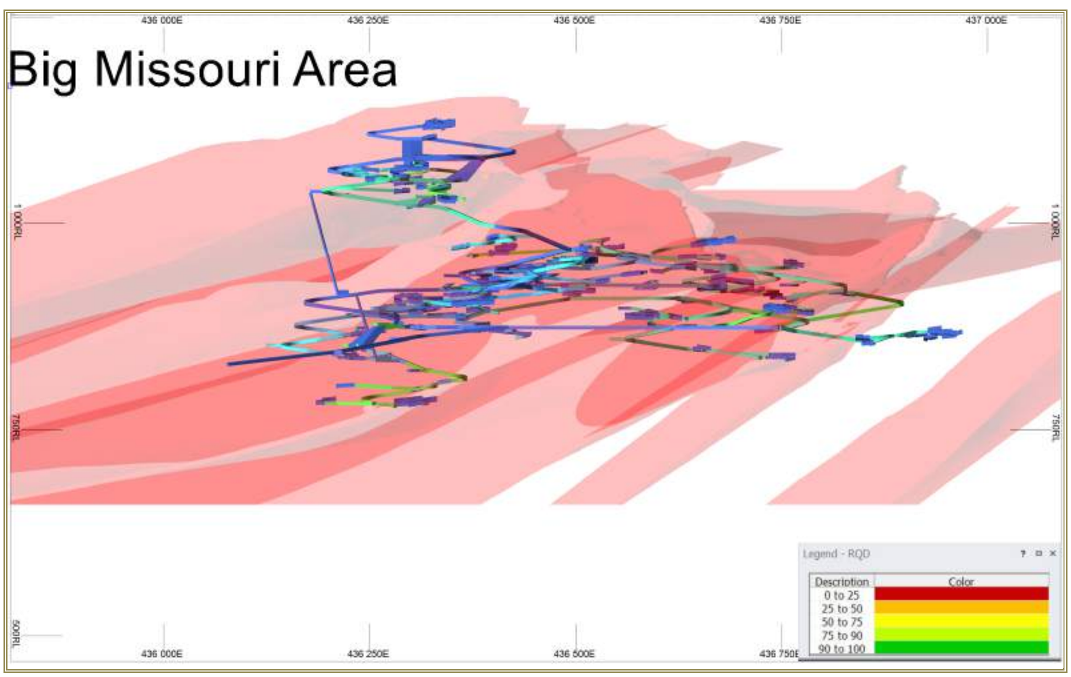


Figure 16-15: Big Missouri Area Infrastructure and Fault Model

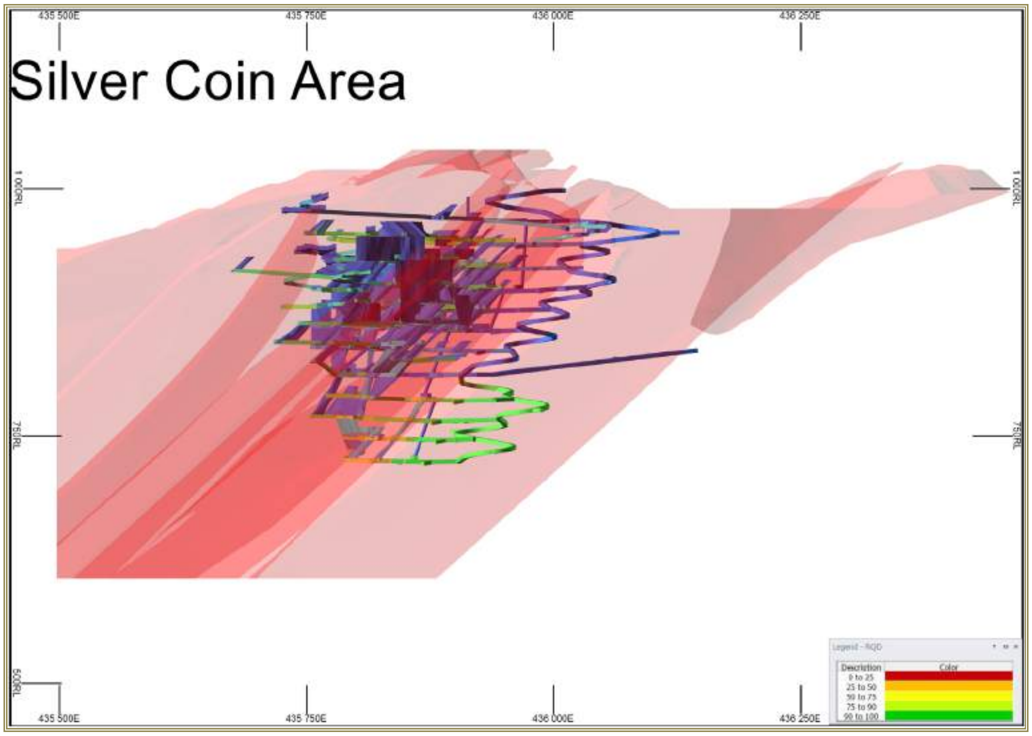


Figure 16-16: Silver Coin Area Infrastructure and Fault Model

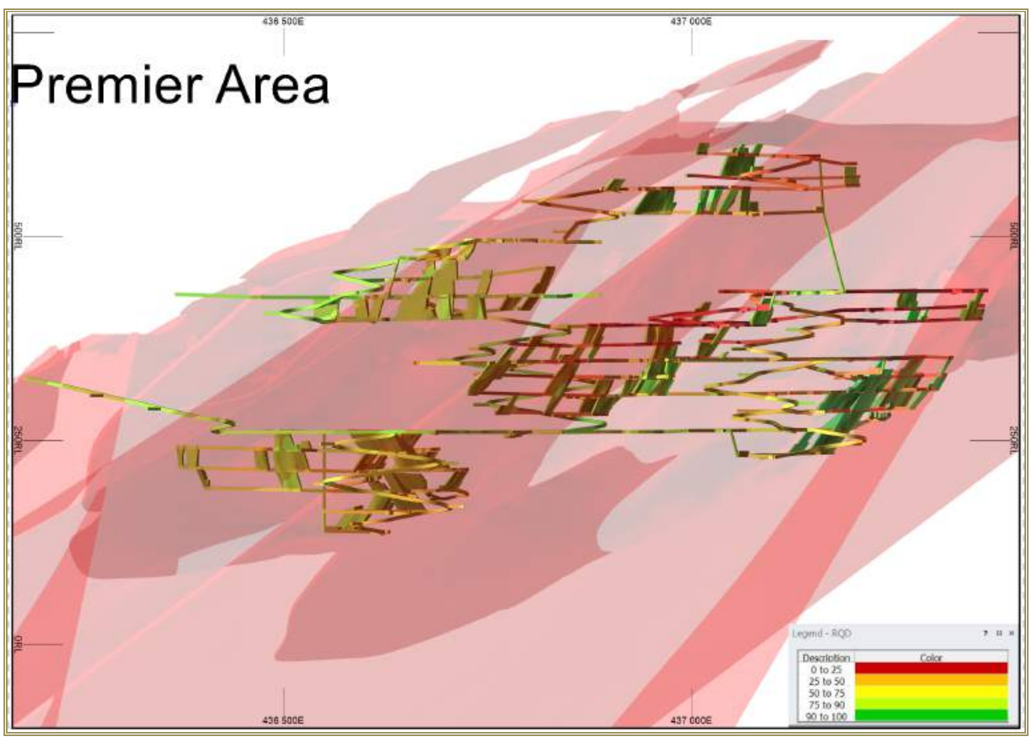


Figure 16-17: Premier Area Infrastructure and Fault Model

16.4.6 Anticipated Rock Mass Behaviour

Red Mountain

At first, numerical modelling was conducted using the 3D elastic boundary element code Map3D to assess the interaction between planned excavations using the proposed mining extraction sequence for a first pass of the mining design and sequence. The evaluation of the induced stresses around production and development excavations provides an understanding of the level of induced stress and the zones of reduced confinement due to excavation interaction and expected level of damage that may occur during mining. No major issues were identified.

Once the mine design and sequence were complete, validation was made with the 2D plastic finite element code RS2 from RocScience. A representative section was analysed for each of the three main zones of the mine to validate the design and the sequence. Again, no major issue was identified under the studied conditions, but some minor elements were detected.

The mine sites are in a relatively low stressed environment in the Canada Alaska Cordillera and no zones of excessively high stress are expected around the open stopes. The risk related to mining induced seismicity is considered low.

Premier Gold Project

Due to the existence of a vast network of historical drifts and stopes that were available for inspection, a dedicated geotechnical campaign was not prioritized. Ascot conducted a small geotechnical program in key locations to obtain rock-type specific geotechnical data. Numerical modelling will be conducted when sufficient data is collected in the future. The design and mine planning were executed in accordance with standard operating procedures used in underground mining and validated by the experience of the authors. The anticipated rock behavior will be confirmed as part of the continuous process/quality improvement typically used in the industry.

16.4.7 Premier Gold Project Rock Strength Testing

Ascot provided two sets of samples to Dr. Paul Hughes P.Eng. at the Norman B Keevil Institute at UBC for geomechanical strength testing. The first set was tested in November 2019 and the second in December 2019.

The initial set of ten samples was chosen from lengths of representative NQ core that focused on distinct rock units within the PGP, representative of the Premier, Silver Coin, and Big Missouri deposits. These samples were subjected to Unconfined Compressive strength (UCS) and Indirect tensile (Brazilian)tests which are summarized Table 16-8.

The second set of twelve samples was obtained from material that was to be sent for crusher suitability testing. The samples were chosen by Dr Hughes in conjunction with Lars Beggerow VP Geoscience for Ascot as representative of the rock and ore types found at the PGP site.

The samples collected were solid HQ core from four drill holes and were shipped in bulk sample bags for UCS testing only. Results are shown in Table 16-8.

Table 16-8: PGP Rock Strength Testing Summary of Results from Round 1 and Round 2

Batch	Sample ID	Drill Hole ID	Depth		Rock Type	Height (mm)	Diameter (mm)	Ratio (H/d)	Bulk Density (kg/m³)	UCS		Young Modulus E (GPa)	Poisson Ration n ( )	Average Brazilian Tensile Strength	
			From (m)	To (m)						(MPa)	(psi)			(MPa)	(psi)
Round 1	PRM-01	P19-2180	58.18	58.58	Porphyry	128.98	47.38	2.72	2,772	150.8	21,864	43.4	0.31	12.4	1,801
Round 1	PRM-02	P19-2180	131.45	131.98	Andesite	128.96	47.41	2.72	2,803	157.1	22,781	69.4	0.30	10.4	1,503
Round 1	PRM-03	P19-2180	93.66	94.15	Andesite	127.59	47.40	2.69	2,842	110.4	16,015	41.1	0.35	10.5	1,523
Round 1	PRM-04	P19-2128	221.40	222.03	Andesite	130.17	47.59	2.74	2,850	17.9	2,596	18.3	0.14	8.6	1,250
Round 1	SCRM-01	P19-2136	35.00	35.55	Andesite	131.38	47.55	2.76	2,811	63.2	9,157	24.9	0.37	5.5	798
Round 1	SCRM-02	P19-2111	16.25	16.69	Andesite	126.55	47.50	2.66	2,778	104.1	15,090	40.4	0.38	7.4	1,076
Round 1	SCRM-03	P19-2076	25.76	26.37	Andesite	128.04	47.21	2.71	2,815	49.1	7,113	35.7	0.22	5.9	860
Round 1	BMRM-01	P19-1930	158.14	158.60	Andesite	127.17	47.41	2.68	2,787	130.4	18,911	49.7	0.32	10.1	1,460
Round 1	BMRM-02	P19-2107	127.54	128.10	Andesite	129.12	47.20	2.74	2,837	130.5	18,921	38.0	0.22	10.9	1,588
Round 1	BMRM-03	P19-1968	29.63	30.10	Andesite	129.87	47.46	2.74	2,797	117.2	16,987	48.8	0.20	11.2	1,618
Round 2	QB-1	P19-2181	3.00	13.00	Quartz Breccia	119.36	62.80	1.901	2,848	98.1	14,227	N/A	N/A	N/A	N/A
Round 2	QB-2	P19-2181	3.00	13.00	Quartz Breccia	130.08	62.87	2.069	2,840	101.0	14,647	N/A	N/A	N/A	N/A
Round 2	QB-3	P19-2181	3.00	13.00	Quartz Breccia	117.67	62.87	1.872	3,041	134.8	19,552	N/A	N/A	N/A	N/A
Round 2	QB-4	P19-2181	3.00	13.00	Quartz Breccia	122.25	62.86	1.945	2,857	107.4	15,571	N/A	N/A	N/A	N/A
Round 2	PP-1	P19-2172	54.91	67.90	Premier Porphyry	117.67	62.89	1.871	2,797	118.3	17,160	N/A	N/A	N/A	N/A
Round 2	PP-2	P19-2172	54.91	67.90	Premier Porphyry	109.73	62.82	1.747	2,799	53.5	7,753	N/A	N/A	N/A	N/A
Round 2	PP-3	P19-2172	54.91	67.90	Premier Porphyry	121.53	62.88	1.933	2,788	117.0	16,968	N/A	N/A	N/A	N/A
Round 2	BMA-1	P19-2068	151.06	168.03	Big Missouri Andesite	111.89	62.43	1.792	2,861	77.9	11,292	N/A	N/A	N/A	N/A
Round 2	PA-1	P19-2149	47.62	76.73	Premier Andesite	109.51	60.51	1.81	2,801	91.2	13,231	N/A	N/A	N/A	N/A
Round 2	PA-2	P19-2149	47.62	76.73	Premier Andesite	115.92	60.50	1.916	2,844	102.3	14,840	N/A	N/A	N/A	N/A
Round 2	PA-3	P19-2149	47.62	76.73	Premier Andesite	109.42	60.62	1.805	2,846	110.2	15,978	N/A	N/A	N/A	N/A
Round 2	PA-4	P19-2149	47.62	76.73	Premier Andesite	109.07	60.52	1.802	2,871	74.5	10,795	N/A	N/A	N/A	N/A

Source: Ascot, 2020

Aside from one sample that tested low (P19-2128), the majority of the rock, 14 of 22 samples (64%) would be rated high (100 to 250 MPa) followed by 32% rated medium (50 to100 MPa) as per ISRM standard terminology. The one low result likely had an unseen defect prior to testing.

These results are consistent with field observations by drill site geologists as to general quality of core recovered supported by underground management observation of good and stable ground conditions seen in old development headings and open stopes.

Figure 16-18 shows a sample of the testwork done. Full information can be found in Appendix D.

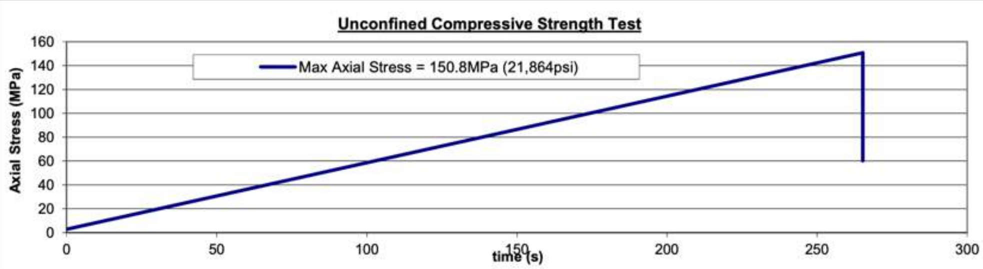
ELASTIC MODULUS AND UNCONFINED COMPRESSIVE STRENGTH TEST
(ISRM-1979)

Sample ID	PRM-01
Drillhole ID	P19-2180
From (m)	58.18
To (m)	58.58
Rock Type	Porphyry
Diameter (mm):	47.38
Height (mm):	128.98
Bulk Density(kg/m³)	2,772
Failure Type:	Single Shear
Tested By:	P.Hughes
Date Tested:	07-Nov-19

Before Test

After Test

Area, (A) (mm²)	Ratio (h/ϕ)	Peak Load (kN)	σ_{UCS} (MPa)	σ_{UCS} (psi)	Young Modulus (GPa)	Poisson Ratio
1,763	2.72	265.9	150.8	21,864	43.4	0.31



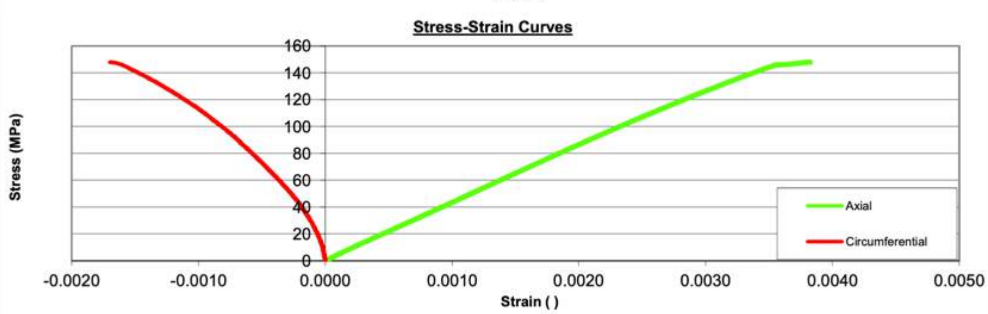


Figure 16-18: Unconfined Compressive Strength Test

16.4.8 EXYN Drone Survey

In early October 2019, Ascot commissioned EXYN technologies of Philadelphia to execute an autonomous drone survey of portions of the existing underground mining areas at Silver Coin and Big Missouri. The drone uses proprietary technology to navigate underground openings utilizing LiDAR and High-Res 3D mapping in real time to avoid typical mine hazards. The resulting output is similar to CMS surveys done at stope brows, but because the instrument can safely go into the stope itself, there are no shadows and accurate stope volumes can be rendered.

Two examples are shown below, the first (Figure 16-19) for an area on the 2850 level at Big Missouri showing stopes that were mined in the 1940s and the second from a stope on the main 810 level at Silver Coin that was mined and processed at the Premier Mill in 1991.

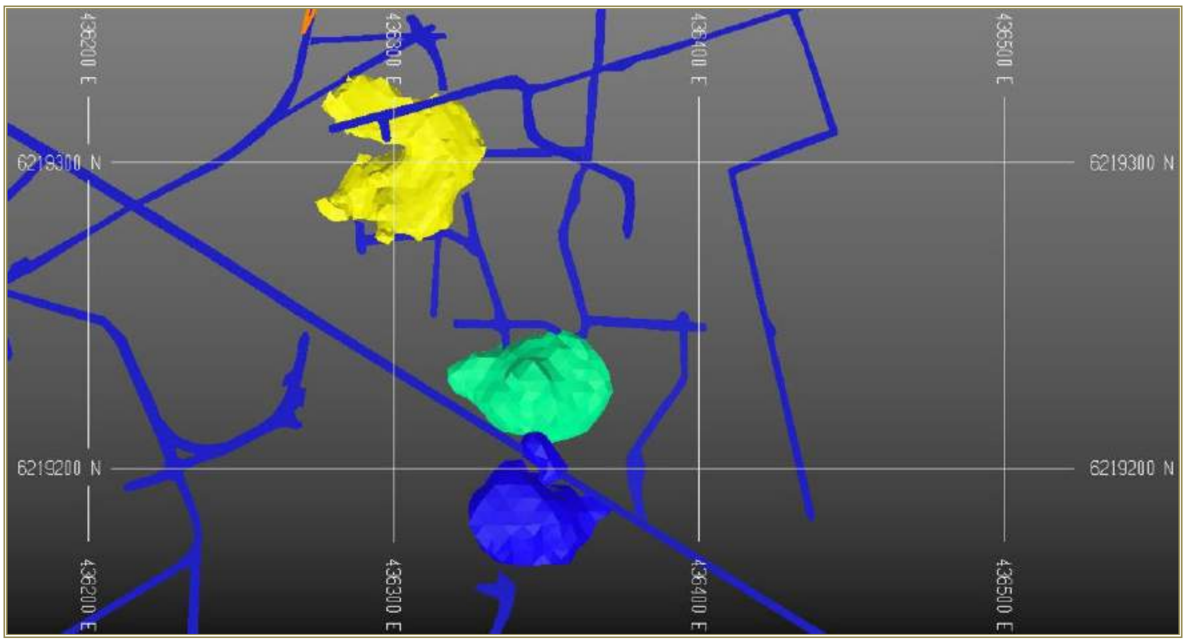


Figure 16-19: Big Missouri Drone Survey Shapes

Inspection demonstrates that the green stope 2850-2 is approximately 50 m long and 30 m wide and measured 35 m high. The other stopes are of similar or even greater dimensions. These stopes were mined in around 1941 and have been left open without backfill and without ground support. A benching method was used from top to bottom, with ore channeling down finger raises to a haulage level. Observations by mine operating personnel confirm that the stopes and adjacent development are stable and in good shape despite the large spans and many years since mining.

The Silver Coin stope Face Cut 35 Block A that is shown in Figure 16-20, is approximately 50 m high, and 35 m on strike with a thickness that varies between 3.5 m to 5.5 m. Underground inspection confirmed that there was very little spalling from the stope, and the ground had stood up well without backfill, since the stope was mined in 1991.

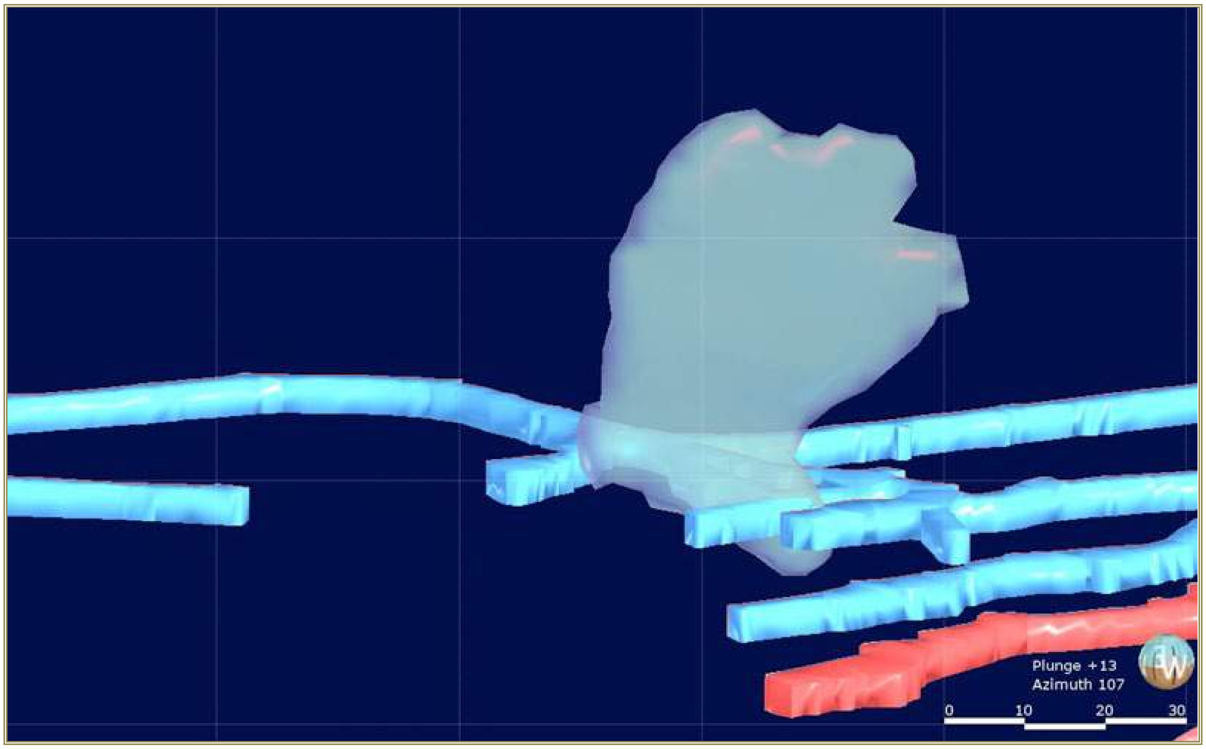


Figure 16-20: Silver Coin Drone Survey Shapes

In addition to the observations at Silver Coin and Big Missouri, Ascot has relied on observations of the historical Premier Mine drifts and accessible stopes, in addition to the rock strength analysis; the approach within this study for future mining has been to use conservative stope sizes and shapes to maintain safe operations with reasonable productivity and control of dilution and recovery. Within those limited areas where notable ground structures are known to be present, Ascot has taken a pragmatic approach to employ both diamond drilling and down-hole inspection technologies to assess stope-by-stope geotechnical analysis and reconfigure stope designs to ensure ore can be recovered safely and economically.

Since historical stope sizes and shapes are larger than those used within the study, optimization of stope sizes and configurations will continue as part of the continuous process/quality improvement typically used in the industry.

16.4.9 Hydrogeology

Red Mountain

Between 1990 and 2016, several technical hydrogeological field programs were carried out in support of exploration and permitting. The field methods included borehole drilling and logging, installation and development of monitoring wells, hydraulic conductivity testing (packer tests and slug tests), measurements of groundwater levels, and measurements of inflow rates and pressure heads during dewatering events of the decline.

Based on the hydrogeological assessment of the mine site conducted by SRK in 2017, the majority of mine inflows will come from intersection of, or connection to, faults or areas of broken ground through open joints. Inflows are predicted to be relatively low, reaching an annual average rate of about 3,810 m³/d in Year 2 and then decreasing from this point onward to about 2,640 m³/d (i.e., Base Case), while under more conservative assumptions (i.e., Upper Case), predictions were respectively 6,400 m³/d and 4,400 m³/d. Inflows of this magnitude are not expected to impact geotechnical assessments or mining conditions, except in localized areas. Seasonal water inflows into structurally complex stoping areas could complicate mining and backfill operations.

Dewatering of the underground mine will be achieved using gravity, where possible, via the lower access ramp. Pumping will be used, as needed, to assure positive dewatering in decline headings, and to route water to settling sumps and holding ponds prior to discharge or further treatment, as required. At mine closure, the ventilation shafts, adits, and portals will be sealed to limit the potential for direct mine water discharge to surface waters and limit the ingress of oxygen; the groundwater system is then expected to return to baseline conditions.

Premier Gold Project

In 2019, Palmer undertook a Baseline Hydrogeology Study for the Premier Gold Project to establish the baseline conditions of the groundwater flow system and to evaluate the potential effects of the proposed Mine Plan for operation at the Premier, Silver Coin and Big Missouri Mines. The following is an extracted summary of their observations.

A MODFLOW model was used to simulate groundwater flow in the fractured and unfractured bedrock, as well as in the waste rock areas. Although MODFLOW was originally developed to simulate flow in porous media, it is widely used for modelling groundwater in fractured rocks if they behave as equivalent porous media at the scale of study. This assumption was used for this study. Groundwater Vistas was used as pre- and post-processor, simulations were conducted by using the MODFLOW-NWT version of MODFLOW.

The domain of the groundwater model includes the Cascade Creek watershed plus a portion of the Salmon River watershed to capture the area influenced by the underground workings at the Silver Coin and Big Missouri Mines. Relevant hydrological features were incorporated into the model (e.g., lakes, rivers, seeps) as well as the existing underground workings.

The underground mine workings corresponding to the ultimate mine development were simulated using the Mine Plan data prepared in 2019 by SRK Consulting. Groundwater seepage into the underground mine workings was simulated by using MODFLOW “drain” nodes (McDonald and Harbaugh, 1988). Drain elevations were specified based on the elevation of the proposed underground workings at that location.

Results presented in Table 16-9 show that predicted seepage rates (existing and operation conditions) are dependent on climatic conditions with lower seepage occurring during dry years and higher seepage during wet years.

The model results predict an increase in groundwater seepage rates to the proposed underground workings ranging from approximately a 17% increase under high flow conditions to a 46% increase under low flow conditions at the Premier site. At the Silver Coin and Big Missouri sites, the model predicts an approximately 24% increase in flow under the high flow condition and a 30% increase under the low flow condition.

Table 16-9: Seepage Rates

	Flow Conditions (L/s)		
	High	Average	Low
Premier			
Existing	108	64	32
Operations	126	78	47
Silver Coin and Big Missouri			
Existing	34	30	20
Operations	42	38	26

16.4.10 Typical Ground Support Patterns

Red Mountain

The ground support requirements were evaluated for the planned development and production excavations in each of the geotechnical domains using empirical criteria, empirical design charts and experience from similar operations. The empirical support requirements will be adjusted based on expected conditions during mining. Spans exceeding the recommended planned dimensions will require a case by case assessment and adjustment to the support specifications. Otherwise, #6 welded wire mesh are required at very roof and shoulder drifts. 0.9 m split sets should be used as required to pin the mesh to the wall at the back. The general support recommendations for development are summarized in Table 16-10.

Table 16-10: Development Ground Support Requirement for Red Mountain

Domain	Excavation and Dimensions	Planned Ground Support
Green, Yellow, and Pink	Permanent (4.5 mH x 4.5 mW)	Roof: 1.8 m fully resin grouted rebar (1.2 m x 1.2 m) Walls: 1.5 m galvanized split sets (1.2 m x 1.2 m), screen support to within 2.1 m of floor
	Temporary (4.5 mH x 4.5 mW)	Roof: 1.8 m fully resin grouted rebar (1.2 m x 1.2 m) Walls: 1.5 m split sets (1.2 m x 1.2 m), screen support to within 2.1 m of floor
	Temporary (4.0 mH x 6.0 mW)	Roof: 1.8 m fully resin grouted rebar (1.2 m x 1.2 m) Walls: 1.5m split sets (1.2 m x 1.2 m), screen support to within 2.1 m of floor
	Intersection	Roof: 2.1 m fully resin grouted rebar (1.2 m x 1.2 m) added 4.0 m cable bolts (2.0 m x 2.0 m) Walls: 1.5 m split sets (1.2 m x 1.2 m), screen support to within 2.1 m of floor
Red	Permanent (4.5 mH x 4.5 mW)	Roof and walls: 2.1 m plastic coated swellex (1.2 m x 1.2 m), welded wire mesh and 50 mm shotcrete down to 0.5 m from the floor. Add cable bolts around the faults.
	Temporary (4.5 mH x 4.5 mW)	Roof and walls: 2.1 m plastic coated swellex (1.2 m x 1.2 m), welded wire mesh and 50 mm shotcrete down to 0.5 m from the floor. Add cable bolts around the faults.
	Temporary (4.0 mH x > 4.5 mW)	Exposed span should be limited to 4.0 m. Fill first drift with CRF before mining the second.
	Intersection	Not applicable

Secondary ground support requirement has been studied to reduce dilution in the stopes. According to the authors' experience and empirical rules, it is estimated that most of the stopes of the mine will be supported. Depending on the dip of the stope and its span, a case by case ground support design is recommended. For example, stopes with a span less than 5.0 m and a dip around 80° will probably not required any cables but a wider stope with a lower dip like inclined undercut LH will required additional support. Overall, cables quantities required for average stopes have been estimated for planning purposes.

Cable length from 4.0 m to 12.0 m should be required with spacing at middle length of the cable between 2.0 m to 2.5 m. For the planning, when a stope should require cables, an average 15 t/cable meters, 30 t/cable meters and 13 t/cable meters are estimated for respectively longitudinal, transversal, and inclined undercut long holes stopes. Increased ground support will be required to manage stability around the fault damage zones.

Premier Gold Project

Ground support patterns for both temporary and permanent excavations required at PGP area are based on the recommendation made for Red Mountain. For mine planning and cost estimation requirements, the average RQD value based on RQD block models were the main tools used for the design and the evaluation of ground support requirements.

Table 16-11: Secondary Support Requirement based on Average RQD and Mining Method

RQD Interval	Long-Hole Mining Method (tonne / cable meters)	IULH Mining Method (tonne / cable meters)
90 – 100%	15	13
75 – 90%	15	13
50 – 75%	12	11
25 – 50%	10	9
0 – 25 %	10 (Smaller stopes recommended)	9 (Smaller stopes recommended)

For development drifts two main ground support patterns were estimated. The first one for routine situations while the second one is a reinforced pattern used for areas of faults and low rock mass quality. Ground support patterns must be validated and corrected with updated geotechnical data once available and before the development of PGP areas.

Finally, cable quantities required for average stopes have been estimated for planning purposes. Every long hole and IULH stope is expected to be cable bolted for a better dilution control. The estimation of the cable requirements for stopes was based on mining method and average RQD of the stopes for conditions similar to Red Mountain.

16.4.11 Stope Dimensions and Dilution

Red Mountain

The results of the stope stability assessment concluded that overhand open stoping was recommended for the Green, Yellow and Pink domains (outlined in Figure 16-11 and Figure 16-12) based on the 25 m sublevel spacing (floor to floor). The Red domains are more suited to a C&F mining method.

The mineralized zone is not defined by discrete structures; dilution from both the hanging wall and footwall will be influenced by the ground conditions, joint orientation, and stress state. The level of unplanned dilution is expected to vary between the various mineralized zones due to the difference in joint orientation, depth below surface, and structural complexity. Empirical estimates and bench marking have been used to determine the combined unplanned dilution (equivalent linear overbreak) for the different geotechnical domains:

- Green Domain: Transverse (0.55 m to 0.90 m), Longitudinal (0.85 m) and Inclined undercut (0.75 m – 1.05 m) stopes
- Yellow and Pink Domain: Transverse (0.80 m to 1.40 m), Longitudinal (1.15 m) and Inclined undercut (1.05 m – 1.45 m) stopes.

Kinematic wedge analysis using UNWEDGETM identified potential unstable wedges forming in the hanging wall of the long-hole stoping. These wedges are expected to increase dilution depending on the zones and the mining method used.

The stope dimensions and dilution expected for Red Mountain are summarized in Table 16-12.

Table 16-12: Stope Dimension Summary for Marc Zone

Sub-level Spacing		Transverse – Primary Stope 25	Transverse – Secondary Stope 25	Longitudinal 25	IULH 25
Yellow Domain	Stope Width	15	10	10	12
	Stope Height	29	29	29	29
	Stope Length	20	20	20	20
	HW Dilution	0.45	0.6	0.7	0.7
	FW Dilution	0.35	0.35	0.45	0.35
	Total Dilution	0.80	0.95	1.15	1.05
Green Domain	Stope Width	15	10	10	12
	Stope Height	29	29	29	29
	Stope Length	20	20	25	20
	HW Dilution	0.35	0.45	0.50	0.55
	FW Dilution	0.20	0.30	0.35	0.2
	Total Dilution	0.55	0.75	0.85	0.75

Premier Gold Project

Stope sizing for the PGP areas was designed based on empirical methods, and experience of the authors based on the rock mechanics information for the site. This methodology is used to determine stope dimensions which are appropriate based on the level of geotechnical knowledge but somewhat conservative when compared to historical workings. Thus, the findings of the stope stability assessment concluded that the suitable mining methods, depending on the ground conditions were long-hole, inclined undercut long-hole (IULH), C&F and R&P methods. The main stope dimensions are summarized in Table 16-13.

Table 16-13: PGP Stopes Dimensions

Stope Parameters	L-H Retreat Max	IULH Max	C&F Max	R&P Max
Stope Width	15	15	10	12
Stope Height	30	25	4	6
Stope Length	35	35	4	17.5

Based on expected conditions similar to Red Mountain and historical stope sizes, the dilution values were evaluated according to RQD values and are summarized in Table 16-14.

Table 16-14: Dilution Data Distributed According to RQD Data

		IULH (%)		Dilution (g/t)		
	Long Hole (%)	Ore (HW)	Waste (FW)	Premier	Silver Coin	Big Missouri
RQD: 90–100%	14	10	8	0.67	0.81	0.52
RQD: 75–90%	19	16	8	0.67	0.81	0.52
RQD: 50–75%	25	23	8	0.67	0.81	0.52
RQD: 25–50%	32	33	8	0.67	0.81	0.52
RQ: 0–25%*	40	42	8	0.67	0.81	0.52

16.4.12 Crown Pillar Dimensions

Red Mountain

Crown pillar stability is only considered within the Marc zone. Mineralization in the Marc zone extends near surface in the vicinity of the existing portal. C&F and long-hole stoping will be used to mine in proximity to surface. The crown pillar will not be stable long term and will require sufficient support to excavate. The planned C&F excavations have a minimum crown pillar of 12.5 m thickness, and long-hole stoping will be conducted with a minimum pillar thickness of 20 m that results in a temporary stable crown pillar for a maximum 8 m to 10 m span. The current mine plan honors these recommendations. Long-term crown pillar stability will require tight filling of the C&F and long-hole excavations with CRF.

Premier Gold Project

Crown pillar assessment in the PGP area was designed based on historical data, observations and expected conditions similar to Red Mountain. Crown pillars are forecasted to be long-term stable assuming stope dimensions at a minimum of 20 m in thickness and secondary support. Upon project start-up, a primary focus will be to further develop relevant geotechnical data and models for detailed planning, scheduling, and costing purposes. This includes exploration and definition diamond drilling, geological and geotechnical mapping and geotechnical testing and analysis.

16.4.13 Backfill Strength Requirements

Backfill Methods

The principle method of backfilling at PGP and RMP will be with CRF comprised of waste development rock or quarried talus from surface. RF will be used to fill ore voids wherever structural fill is not required.

As soon as production begins the waste used to backfill the stopes can be managed underground in the re-muck, which will reduce the transport and storage of waste on the surface. All of the development waste produced at both sites will be used as mine backfill, with any excess deposited in historical open stopes. As such, aside from a temporary waste pile at the start of production, no underground waste will remain on surface.

CRF consists of waste development rock mixed with cement slurry, a cement binder content of 5% was assumed for general cost purposes. CRF would be produced on the site with a Mobile Cemented Rock Fill Plant carried out with a cassette-based carrier as outlined in Section 16.2.7.3. The mobile plant will be filled by a cement silo located at the mine portal and transported underground to be mixed with waste according to the backfill recipe.

Cemented Rock Fill

The selected mining methods, long-hole open stoping (LHOS), inclined undercut stopes (IULH) and C&F require backfill to achieve the planned extraction and manage stability. CRF will be used in all longitudinal, transverse and Inclined Undercut long-hole stopes at different cement ratios with a fill factor of between 70% and 85% depending on the dip. A higher cement ratio will be used in primary transverse stopes, as well as in some of the secondary transverse stopes when a mining sequence consists of more than three stopes.

Geotechnical considerations do not require CRF in secondary stopes and C&F mining provided the fill will not be exposed by future mining, and in some instances, stopes can remain empty provided they are not required for development rock disposal.

CRF strength (UCS >2 MPa) is based on benchmarking from similar operations and the requirement that it remains stable for the maximum exposed span (20 m) and a sublevel height of 25 m. In some instances, at RMP sill mining is planned, so the need to work under previously placed backfill exists. In this case, CRF is required for the sill.

The main requirement of the cemented fill is for it to remain stable (and self-supporting). When a fill wall is exposed by mining an immediately adjacent stope, a CRF with sufficient binder content can be determined for the exposed fill dimensions (height, width, and length) and duration of exposure. Variability of CRF strength occurs due to many factors including site conditions (i.e., rock type and underground rock temperature), uncontrolled nature of development waste rock particle gradation, binder and water mix control, use of admixtures, and degree or lack of actual mixing; hence factor of safety typically results in a minimum of 4 wt% to 6 wt% for vertical exposures.

For higher risk exposures such as under C&F, typically crushing and classification of waste rock and pre-mixing of CRF (also termed cemented aggregate fill, or CAF) will be used to ensure greater certainty of reaching a predicted strength. This estimated strength considers the partial backfilling of the upper portion of the stope (Figure 16-21). The binder content takes into account the site-specific factors and process/placement methodology to lessen the variability of the cemented rockfill mix during mining operations. With the implementation of a QA/QC program, and through experience, the mine will be in a position to evaluate the opportunity to reduce the binder content through operational and material controls.

Gradation of rock can be readily controlled by use of a grizzly to partially mitigate gap gradation, if required. For relatively narrow mining at the PGP sites, lower strengths and lower binder usage is possible, especially with longitudinal retreat mining, whereas RMP will have greater exposures with transverse mining; the benefits of additional rock processing or mixing (sorting, crushing, binder use, pre-mixing) will be assessed with regards to transverse mining (primary or secondary stopes) or longitudinal retreat with minimal exposure, and based on sequences and schedules

For planning purposes, it has been assumed that there will be a delay of seven days after backfilling before mining of an adjacent stope commences. The delay for binder curing can be extended or shortened to facilitate sequence and schedule demands; where accelerated mining is needed; the incremental cost of increased binder will be considered to augment production throughput. Lower binder usage to uncemented rock fill will be employed in certain situations such as secondary stopes or a final stope within a sequence having no future exposure. In all cases, barren pillars (sills and vertical) can be used in pillar layout to mitigate use of binder.

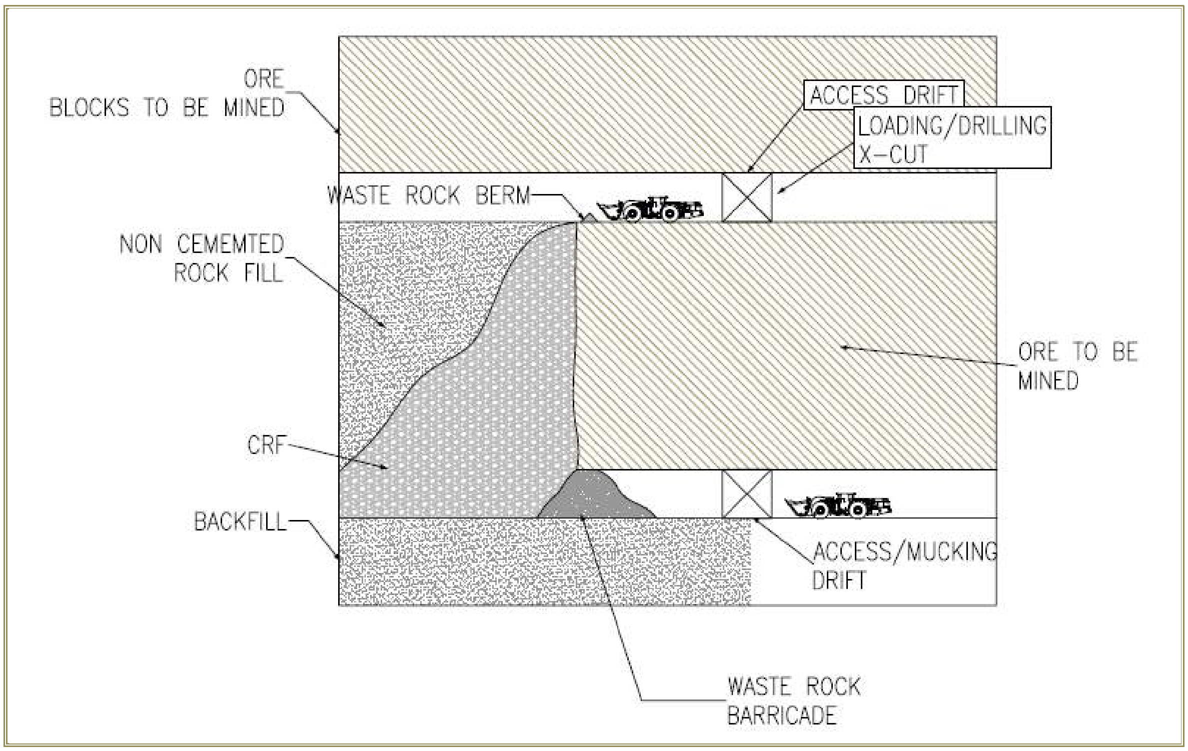


Figure 16-21: Schematic Representation of the Proposed Partial Use of Cemented Rockfill

Stopes located above the temporary sill pillar will be backfilled with a higher strength CRF to allow future sill pillar recovery. The estimated CRF strengths varies with the stope span (or width) from about 1 MPa to 3 MPa for spans ≤3 m to up to 10 m, respectively.

Based on the current laboratory test results, it was recommended that a 6% and 7% binder content be used for excavated stope spans (or widths) of ≤3 m and 3 m to 10 m, respectively, this binder level is more appropriate for the larger transverse LH stopes at RMP. A minimum delay of 28 days curing time was used for planning purposes. Reduction in binder is possible with greater quality control of rock through classification (i.e., screening), crushing and increased mixing. The cost-benefit determined stope-by-stope will require a trade-off evaluation. For PGP, 4% to 6% binder for CRF is more appropriate with the smaller non-contiguous stopes.

Any future change to the minimum binder content for the stopes located above sill pillars will be based upon; a) laboratory test results with specimens taken directly from the CRF mix at the backfill bay, b) actual performance of CRF and with minimum to no unravelling after the sill pillar is recovered and c) mine experience gained during the recovery of the upper sill pillars

Test of CRF Strength

There are opportunities to fine-tune the CRF recipe with the mobile cement plant to achieve CRF strength (UCS >2 MPa) that is based on benchmarking from similar operations and the requirement that it remains stable for the maximum exposed span (20 m) and a sublevel height of 25 m.

In-situ testing is suggested for optimization of the CRF operations at the mine. Collecting data on the 28-day UCS test with development waste rock and locally sources talus and different ratio at 4%, 5%, and 6% cement allowing a better control of the CRF quality. Various companies have executed compression tests in similar underground conditions in British Columbia, and future work in this respect will be required at the PGP site.

Cement Requirement

For the mine, it is expected that daily binder delivery will be available from the port of Stewart. At least two cement suppliers have been identified with the capability to supply the site. We propose the installation of a cement silo with a nominal capacity of 200 tonnes, to ensure a continuous supply at the site.

16.5 Underground Mine Design

PGP and RMP mobile fleets will be obtained from a single common pool, with equipment distributed according to ramp up and ramp down of operations. Furthermore, all development headings will have common dimensions, as shown in Table 16-15. Waste development will be standardized according to main haulage fleet, while ore development will range in size depending on the zone being mined and the mining method. Narrow longitudinal retreat stopes will follow geological control and ore dip to facilitate drilling and blasting, while transverse stopes will be developed to facilitate effective drilling and blasting approach of bulk vertical primary and secondary stopes. The access ramps are driven typically at a maximum gradient of ±15%. Narrow zones will be driven at dimensions suitable for good drilling and blasting operations and to minimize dilution.

At RMP the various zones will be accessed through the main ramp from the two portals located on the Southern side of Red Mountain. From the ramp, sublevels will be developed to access mineralized material. Excavation in mineralized material is generally 4.5 m high by 6.0 m wide as a minimum, but the width varies in relation to vein thickness to provide the optimal geometry for production drilling. Ore and waste will be hauled by LHDs from the production area to either a re-muck bay or a loading point close to the ramp and then loaded into trucks to be hauled directly to surface.

BM and SC are located adjacent to one another with common access through the main adit from the S1 pit, while Premier which is located about 500 m below and adjacent to the Mill is accessed by its own network of new and historical adits and portals. Each deposit will be accessed independently from various surface haulage routes. The common connection between SC and BM will be established early in year one of the project.

Initially, ore and waste will be loaded into underground haul trucks and transported to the portal, with waste rock stockpiled and later used for backfill. At PGP where old void stopes exist, waste may be readily disposed when possible and if not needed for current backfill schedules. Rehabilitation work will be completed in areas where mine access is required in historical underground workings. Most historical PGP development had drift sizes with approximate dimensions of 1.8 m wide by 2.1 m high, Level 6 had previously been slashed by Westmin to approximately 2.5 m x 2.5 m. Rehabilitation work will expand the drift sizes to 5.0 m x 5.0 m in the main ramp, and 4.0 m x 4.0 m in any secondary haulage development.

The compilation of mine development quantities considered in the mine plan is presented in Table 16-15.

Table 16-15: PGP and RMP Mine Development Quantities (metres)

	Year -1 2021	Year 1 2022	Year 2 2023	Year 3 2024	Year 4 2025	Year 5 2026	Year 6 2027	Total
Premier Gold Project – PGP								
Horizontal Development (CAPEX m)								
Ramp (4.5 x 4.5)	1,589	4,937	687	1,995	2,593	1,353	-	13,153
Footwall & Level Access (4.5 x 4.5)	171	1,821	188	593	1,327	1,052	-	5,151
Level Drift (4.0 x 4.0)	148	500	127	139	584	794	22	2,314
Large Drift (6.0 x 4.0)	-	-	-	-	-	-	-	-
Rehabilitation (4.5 x 4.5)	838	1,372	5	1,013	1,490	1,581	269	6,567
Alimak nest (4.5 x 8.0)	11	56	33	16	32	51	14	212
MISC (Infrastructures) (4.0 x 4.0)	206	1,032	232	393	1,163	811	84	3,922
Horizontal Development (OPEX m)								
Footwall & Level Access (4.5 x 4.5)	-	972	545	117	313	271	-	2,218
Level Drift (4.0 x 4.0)	766	14,345	6,348	449	4,907	3,683	278	30,776
Large Drift (6.0 x 4.0)	45	1,423	830	33	1,025	356	-	3,711
Rehabilitation (4.5 x 4.5)	42	12	167	213	342	51	-	827
Vertical Development (CAPEX m)								
Alimak (3.5 x 3.5)	-	350	152	-	205	193	22	922
Red Mountain Project – RMP								
Horizontal Development (CAPEX m)								
Ramp (4.5 x 4.5)	-	2,203	942	370	321	-	-	3,836
Footwall & Level Access (4.5 x 4.5)	-	1,377	909	556	138	-	-	2,980
Vent Access (4.0 x 4.0)	-	322	327	260	83	-	-	993
Alimak Nest (4.5 x 8.0)	-	80	77	33	46	-	-	235
Ore Drift (4.0 x 4.0)	-	-	20	250	-	81	-	352
Draw Point (4.0 x 4.0)	-	268	207	72	-	-	-	546
MISC - Waste (Infrastructures)	-	701	561	286	228	-	-	1,775
Horizontal Development (OPEX m)								
Footwall & Level Access (4.5 x 4.5)	-	20	45	30	-	-	-	95
Access Cut & Fill (4.0 x 4.0)	-	-	-	-	36	128	-	164
Cut & Fill (4.0 x 4.0)	-	-	-	-	-	-	-	-
Ore Drift (4.5 x 6.0)	-	1,193	628	2,007	853	649	-	5,330
Draw Point (4.0 x 4.0)	-	1,470	1,210	458	55	-	-	3,194
Backfill - CRF (4.0 x 4.0)	-	-	-	-	158	126	44	328
Vertical Development (CAPEX m)	-	101	185	44	79	-	-	409
Alimak (3.5 x 3.5)	-	101	185	44	79	-	-	409

16.5.1 Mining Sequence

Development and production will follow the schedule outlined below in Figure 16-22, commencing at the BM and SC deposits with RMP and PNL sites following through a logical program to fill the Mill at 2,500 t/d while shifting both labour force and mobile fleets as each site transitions through development, production and into completion.

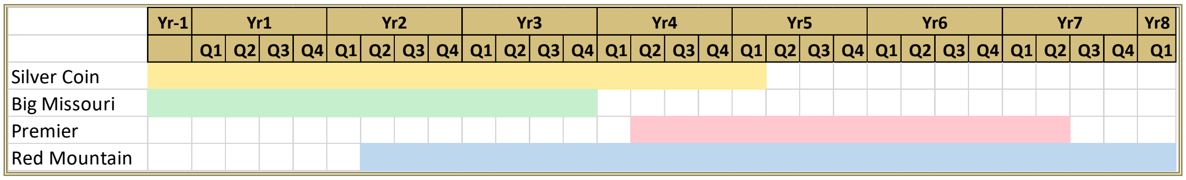


Figure 16-22: Production Schedule

At PGP, two crews will start initial development at Silver Coin and Big Missouri, ramping up to three crews working two 10-hour shifts per day over the first two years. Teams will be assigned as needed between the various zones at either of two operating sites.

As it ramps up to full production RMP will require three development crews who will be working 10-hour shifts day and night. Two of these teams will be dedicated to ramp development from the upper and lower portals. The third team will be focussed on developing level access and ore drifts.

An Alimak team will develop the ventilation raise network.

Figure 16-23 and Figure 16-24 are longitudinal views of the Premier and BM deposits respectively, showing the outline of each production center within each deposit.

Historically, the main haulage levels in Premier were 6 and 4 levels, they will now act as return air ways where air will be exhausted out the portal, and in an emergency, Level 6 can provide secondary egress. In general, production will begin from the west and progress eastwards.

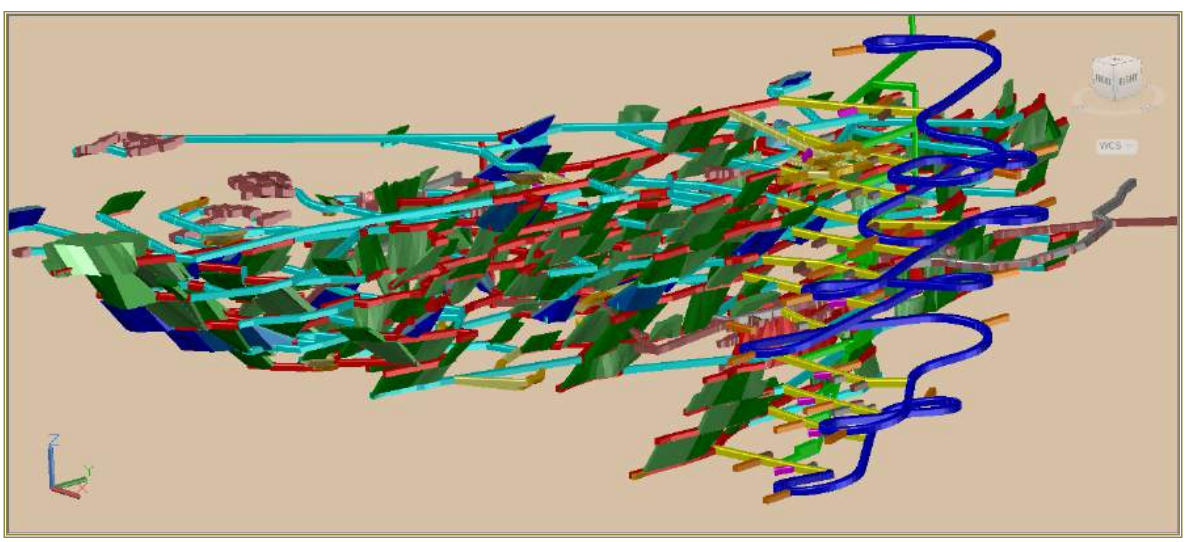


Figure 16-23: Longitudinal View of Silver Coin Production Zones

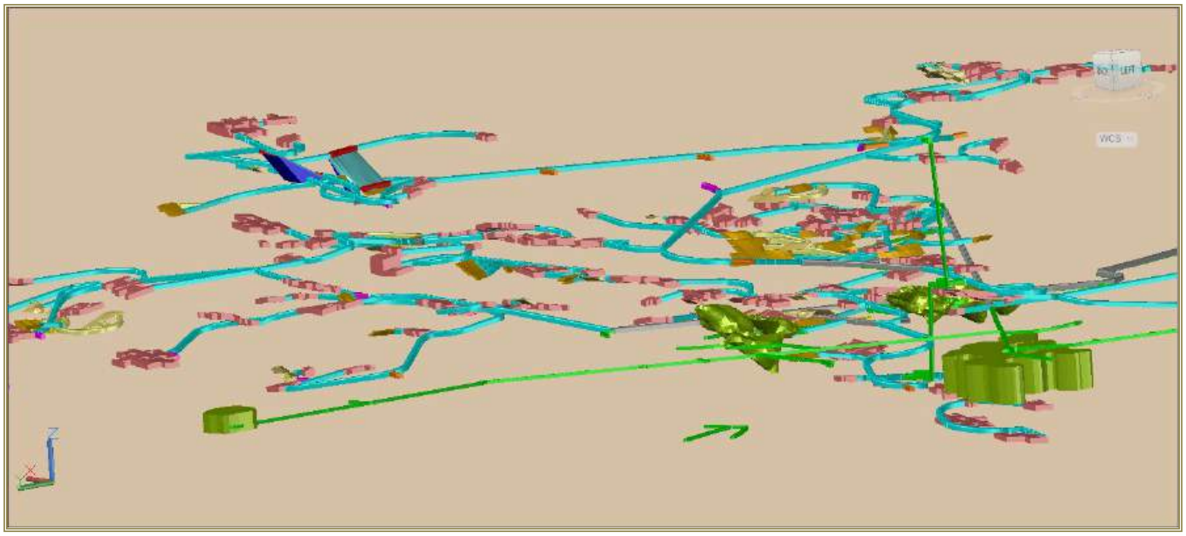


Figure 16-24: Longitudinal View of Big Missouri Production Zones

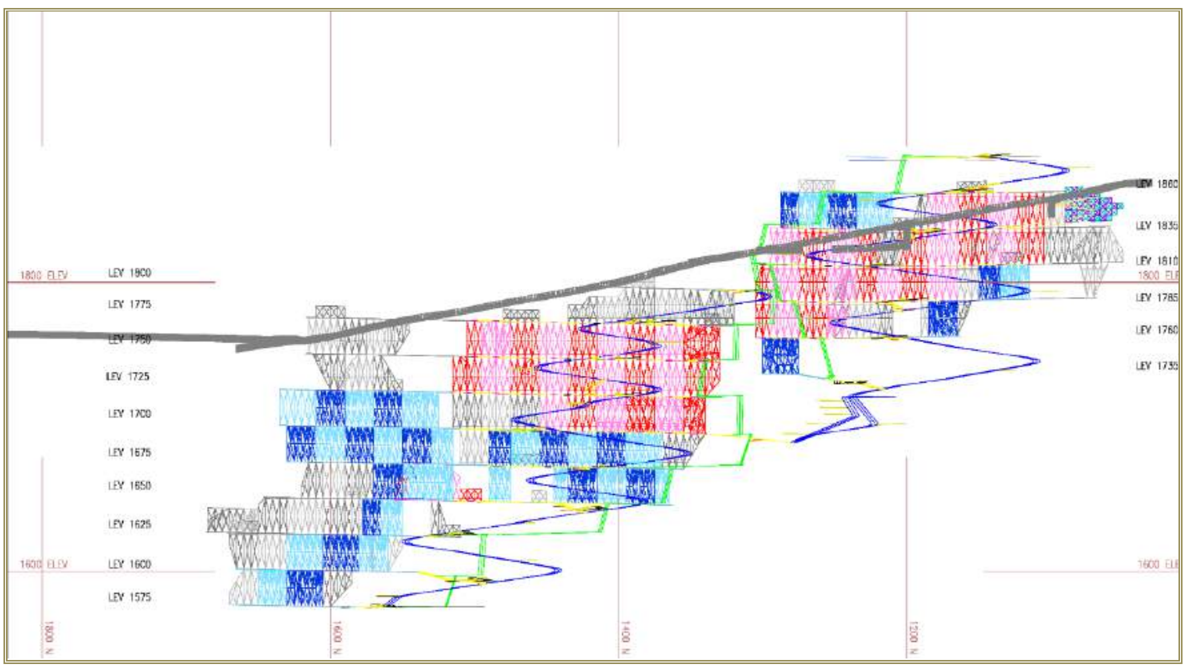


Figure 16-25: Longitudinal View of the AV-JW and Marc Production Zones

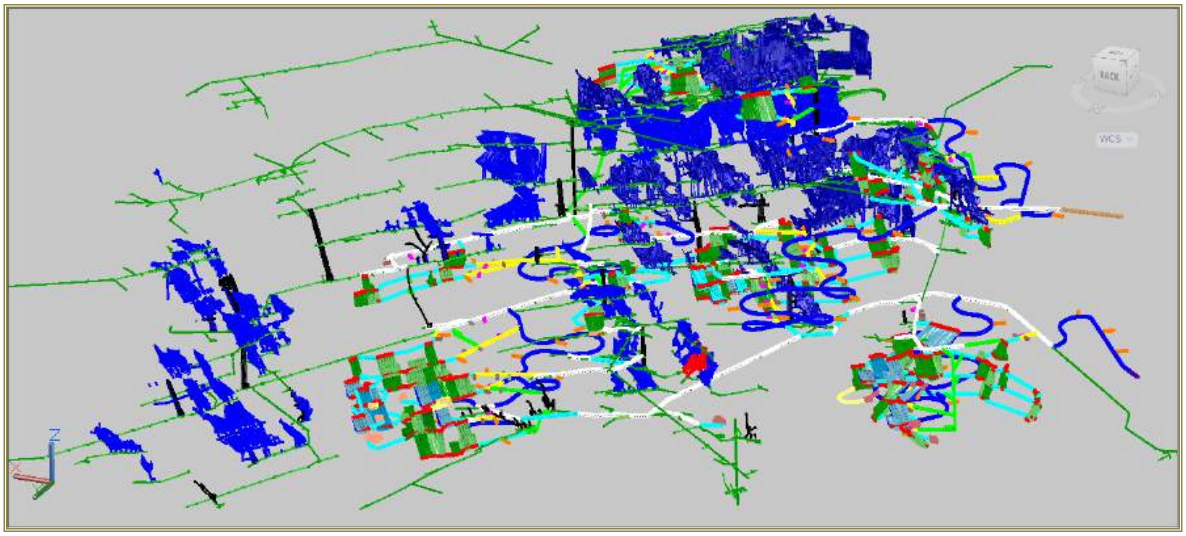


Figure 16-26: Longitudinal View of Premier Production Zones

16.6 Mining Rate

Mill throughput of 2,500 t/d nominal is planned through the scheduling of PGP and RMP operations, while transitioning labour and equipment through pre-production, production, and closure.

At PGP the production rate will ramp-up from pre-production to reach 1,700 t/d average in Year 1, climbing to 2,500 t/d average in Year 2, until RMP comes online Year 3, when steady state production from each site will be about 1,250 t/d. The mine plan has a duration of approximately eight years not including the three months pre-production period and starts ramping down in Year 7, finishing in mid-Year 8.

Limited development ore production will occur during the pre-production stage. This material will be stockpiled and used for mill commissioning.

The underground mine plan provides for recovering 6.2 Mt of ore over the LOM, producing 1.2 Moz AuEq from the combined PGP and RMP, as shown in Table 16-16 , with AuEq ounces and tonnage distributed by mining method and site shown in Table 16-17.

Table 16-16: Mining Plan—Yearly Tonnage and Gold Production Breakdown

	Year -1 2021	Year 1 2022	Year 2 2023	Year 3 2024	Year 4 2025	Year 5 2026	Year 6 2027	Year 7 2028	Year 8 2029	Total
Total Development Waste (m)	3,804	16,553	14,278	11,224	7,022	13,898	11,452	2,989	23	81,244
Total Development Waste (t)	205,406	866,983	793,895	672,394	436,104	825,337	758,942	312,556	46,913	4,918,530
Total Development Ore (m)	12	1,754	1,320	1,723	822	1,457	1,042	228	-	8,358
Total Development Ore (t)	530	81,015	73,092	94,173	48,673	83,111	73,717	10,099	-	464,409
Grade (g/t)	3.60	5.56	7.14	6.46	7.36	5.97	5.45	5.21	-	6.23
In-situ Grade (g/t)	10.46	7.82	9.21	7.29	8.26	7.56	6.16	6.82	-	7.65
Total Stoping Ore (t)	8,470	529,866	823,405	803,655	851,210	815,525	823,398	763,604	293,190	5,712,323
Au Grade (g/t)	6.99	5.61	6.08	6.71	5.42	5.43	6.17	5.57	5.83	5.87
In-situ Au Grade (g/t)	11.29	8.08	8.87	9.19	7.72	7.34	8.40	7.06	6.94	8.05
Ag Grade (g/t)	10.18	11.00	14.24	21.00	17.41	21.65	21.79	28.20	15.75	19.45
AuEq Grade (g/t)	7.11	5.74	6.25	6.96	5.62	5.69	6.43	5.91	6.01	6.10
Total Production ore (t)	9,000	610,881	896,497	897,828	899,883	898,636	897,115	773,703	293,190	6,176,732
Total Gold Mined (oz) (AuEq)	2,000	112,770	182,924	200,160	165,763	165,755	184,057	146,923	56,624	1,216,976

Table 16-17: Ore Production by Mining Method

	Ore (t)	% of Tonnage	Ounces		Ore (kt)	% of Tonnage	Ounces (koz)	
			AuEq	% of Ounces			AuEq	% of Ounces
PGP								
Development Ore	258,142	7.1	49,889	7.5	258	7.1	50	7.5
Production Ore								
Longhole	2,108,982	58.1	324,881	48.9	2,109	58.1	325	48.9
IULH	390,841	10.8	81,675	12.3	391	10.8	82	12.3
C&F	144,300	4.	28,479	4.3	144	4.0	28	4.3
R&P	729,633	20.1	178,996	27.0	730	20.1	179	27.0
PGP Total	**3,631,898**	**100.0**	**663,920**	**100**	**3,632**	**100**	**664**	**100**
RMP								
Development Ore	191,391	7.5	43,414	7.8	191	7.5	43	7.8
Production Ore								
Longhole	1,837,576	72.2	397,044	71.8	1,838	72.2	397	71.8
IULH	500,991	19.7	108,813	19.7	501	19.7	109	19.7
C&F	14,876	0.6	3,785	0.7	15	0.6	4	0.7
R&P	-	0.0	-	0.0	-	0.0	-	0.0
RMP Total	**2,544,833**	**100**	**553,056**	**100**	**2,545**	**100**	**553**	**100**
PGP& RMP								
Development Ore	449,533	7.3	93,304	7.7	450	7.3	93	7.7
Production Ore								
Longhole	3,946,558	63.9	721,925	59.3	3,947	63.9	722	59.3
IULH	891,832	14.4	190,488	15.7	892	14.4	190	15.7
C&F	159,176	2.6	32,264	2.7	159	2.6	32	2.7
R&P	729,633	11.8	178,996	14.7	730	11.8	179	14.7
PGP & RMP Total	**6,176,732**	**100**	**1,216,976**	**100**	**6,177**	**100**	**1,217**	**100**

16.7 Mine Development and Production Assumptions

The development and production assumptions have been prepared based on historical databases and benchmarked performance. Stope cycle times were estimated from a first principles build of activities and associated productivities:

- Lateral development: (as shown Table 16-18)
- Alimak raises: 120 m/month
- Long-Hole transverse stoping: from 340 t/d to 550 t/d depending on the stope length
- Long-Hole longitudinal stoping: from 400 t/d to 500 t/d depending on the stope width
- Inclined undercut LH stoping: 400 t/d
- MRP and MCF stoping: 340 t/d.

Table 16-18: Lateral Development Assumptions

Type of Heading	Unit	Projected Development Rate			
		First Month	Second Month	Third Month	Fourth Month and more
Efficiency factor (learning curve)	%	40	60	80	100
Single face	m/month	70	105	140	175
Double face	m/month	82	123	164	205
Triple Face	m/month	94	141	188	235
Multiple face (4 +)	m/month	108	162	216	270

16.8 Mobile Equipment

The selection of underground mining equipment is based on the mining methods, drift and stope dimensions, production rate, operating costs, and capital costs. Given the overall life of mine is eight years, it was assumed that only new equipment would be purchased under lease financing agreements. The list of equipment considered for the Project is presented in Table 16-19. It should be noted that there is an existing fleet of equipment at Red Mountain, consisting of a 2 Boom jumbo, two 10-tonne LHDs and three haul trucks. Tier 4 engines have previously been purchased to upgrade equipment to current standards.

Table 16-19: Mining Equipment for the Project

Mining Equipment	Max	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9
2 Booms Jumbo	3	3	3	3	3	3	3	3	3	3
30-t Mine Truck	5	5	5	5	5	5	5	5	5	5
10-t LHD	5	5	5	5	5	5	5	5	5	5
Bolter	2	2	2	2	2	2	2	2	2	2
ITH Drill – Long Hole	2	2	2	2	2	2	2	2	2	2
Explosives Truck	2	2	2	2	2	2	2	2	2	2
Scissor Lift	2	2	2	2	2	2	2	2	2	2
Shotcrete Sprayer	1	1	1	1	1	1	1	1	1	1
Personnel Carrier	1	1	1	1	1	1	1	1	1	1
Lube Service Truck	1	1	1	1	1	1	1	1	1	1
Boom Truck	1	1	1	1	1	1	1	1	1	1
Motor Grader	1	1	1	1	1	1	1	1	1	1
Utility Vehicle	1	1	1	1	1	1	1	1	1	1
Backhoe with Rockbreaker	1	1	1	1	1	1	1	1	1	1
Telehandler	1	1	1	1	1	1	1	1	1	1
Mechanics Truck	2	2	2	2	2	2	2	2	2	2
Toyota PC	5	5	5	5	5	5	5	5	5	5
Jumbo Drifter – Spare	1	1	1	1	1	1	1	1	1	1
Jackleg	8	8	8	8	8	8	8	8	8	8
Stoper	8	8	8	8	8	8	8	8	8	8

16.9 Shift Schedule

The mine will operate seven days a week, night, and day shifts. This annual schedule is equivalent to 360 days per year of operation.

- Technical services and administration will be on a schedule of 5 days work – 2 days rest. Some workers from technical services will be on a 4 days work – 4 days rest schedule to support development crews.
- Development and production crews will be on a schedule of 4 days work –4 days rest – 4 days work (night shift) – 4 days rest, for 10 hours per shift.

16.10 Underground and Service Labour force

The town of Stewart, BC is well situated for the PGP and RMP Projects, having the community services and facilities available for a healthy lifestyle supported by well-paid mining workforce. Management and technical services will be managed out of Stewart, with daily visits to one or both the RMP and PGP sites, depending on day-to-day needs. Mining, processing, and services teams will report to their respective operations daily and on shifts as scheduled, bused from town.

Technical staff, such as geological teams, will be required daily at each development heading to guide development under geological control, while also managing grade, dilution, and reconciliation tasks. Similarly, ventilation, surveying and mine planning teams will be required to travel between the operations dictated by needs, following the production schedule as shown previously.

Ascot will employ its own mine crews to fill all the positions related to mine development, production, construction, and mine services by the end of pre-production. Most positions in operations will require a day and night shift, while technical positions typically only require a day shift. Table 16-20 lists the steady state underground personnel requirements of mine operations. At steady state 110 people (total on payroll) will be employed within the mining technical, and production departments. Some redundancy has been built into the personnel requirements to account for training, sickness, and absenteeism.

The labour force resources on each working shift used to prepare the mine schedule includes:

- 2 long-hole crews
- 5 truck drivers
- 5 LHD operators
- 3 development crews, for a total of 3 jumbos
- Each development crew consists of 1 jumbo operator and 2 workers for ground support and services; the truck operator is not included as part of the crew count.

Details of the labour force requirements for the duration of the LOM is listed in Table 16-20.

16.11 Technical Staff – Engineering, Geology, and Surveying

The engineering and geology departments will provide the technical support to the mine operations. A senior mine engineer will be in charge of both technical departments. A team composed of senior and junior engineers, geologists, mine and geological technicians will be hired, along with engineering students for work terms. Surveying will be done by a dedicated surveying group who will split duties on a rotation. Geologists will maintain grade control and outline the valuable mineralization to keep dilution to a minimum and guide development under geological control. Mine exploration and definition drilling campaigns will also be managed and coordinated by geological teams.

Table 16-20: Labour Force Requirements—Operation and Engineering Services

Manpower	max	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Superintendant	1	1	1	1	1	1	1	1	1	1
Captain	2	2	2	2	2	2	2	2	2	2
Shifters	8	4	8	8	8	8	8	8	8	4
Jumbo/Drill/Bolter Operator	8	4	8	8	8	8	8	8	8	4
Blaster	8	4	8	8	8	8	8	8	8	4
LHD Operator	8	4	8	8	8	8	8	8	8	4
Truck Operator	8	4	8	8	8	8	8	8	8	4
Services operator	8	4	8	8	8	8	8	8	8	4
Miner	8	4	8	8	8	8	8	8	8	4
Services operator	8	4	8	8	8	8	8	8	8	4
Maintenance Manager	1	1	1	1	1	1	1	1	1	1
Maintenance supervisor	4	4	4	4	4	4	4	4	4	4
Electrician	8	8	8	8	8	8	8	8	8	8
Mechanic	8	8	8	8	8	8	8	8	8	8
Helper	8	8	8	8	8	8	8	8	8	8
Technical Services Manager	2	2	2	2	2	2	2	2	2	2
Mine Engineer	2	2	2	2	2	2	2	2	2	2
Mine Technician	4	4	4	4	4	4	4	4	4	4
Production Geologist	2	2	2	2	2	2	2	2	2	2
Geology technician	2	2	2	2	2	2	2	2	2	2
Heatlh and Safety Coordinator	1	1	1	1	1	1	1	1	1	1
Training Coordinator	1	1	1	1	1	1	1	1	1	1
Total	110	78	110	110	110	110	110	110	110	78

16.12 Mining Services

16.12.1 Mine Ventilation

Red Mountain

To achieve the desired production rate for this deposit, it was determined that 387 kcfm will be needed. The ventilation demand has been determined to satisfy the Canadian ventilation standards. Table 16-21 lists the equipment and their airflow needs. The airflow needs are based on the CANMET engine certificate.

In estimating the rate of fresh air flow for the entire mine, a utilization rate has been applied to account for time when machines may be mechanically unavailable, or simply not in use. The utilisation rates are, respectively, 100% for production equipment, 80% for most service equipment, and 50% for machinery that functions primarily with electricity.

Table 16-21: Mobile Equipment List with Related Volume Requirements

Equipment	Airflow CFM	Quantity	Utilization Rate %	Total Airflow CFM
Jumbo – 2 Booms	9,200	2	50%	9,200
LHD – 10 tonne	17,000	5	100%	85,000
Mine Truck – 30 tonne	21,100	4	100%	84,400
ITH Drill – Long Hole	6,900	2	50%	6,900
Bolter	4,800	2	50%	4,800
Explosive Truck	6,900	2	80%	11,040
Scissor Lift	6,900	1	50%	3,450
Shotcrete Sprayer	6,900	1	80%	5,520
Personnel Carrier	6,900	1	80%	5,520
Lube Service Truck	6,900	1	80%	5,520
Boom Truck	6,900	1	80%	5,520
Motor Grader	11,500	1	80%	9,200
Utility Vehicle	4,000	1	80%	3,200
Backhoe	4,000	1	80%	3,200
Telehandler	7,300	1	80%	5,840
Mechanics Truck	7,300	2	80%	11,680
Toyota PC	7,300	6	80%	35,040

A contingency of 10% has been applied on the total estimated fresh air requirements to allow for additional equipment that could be added during the life of mine. It also allows a quantity of air for potential leaks in the system.

The mine fresh air demand was evaluated according to the equipment on hand annually and is presented in Table 16-22. The maximum demand of 387 kcfm is expected to be reached in Year 2 of production.

Table 16-22: Airflow Requirements for the Project

	Initial	Y1	Y2	Y3	Y4	Y5	Y6
Required Airflow	147,970	228,250	295,030	295,030	295,030	295,030	295,030
Contingency 10%	14,797	22,825	29,053	29,053	29,053	29,053	29,053
Total Airflow	162,767	251,075	324,533	324,533	324,533	324,533	324,533

A typical level will have one LHD and one truck, therefore, requiring 40,000 cfm on every active level. Each level has a drift access to a ventilation raise, which will have a ventilation door to control the air entry. A 100 hp booster fan will be mounted on the ventilation door and vent ducting will bring the fresh air to the face of the stopes that are being mined. This flow will be sufficient to clear the blast fumes and the contaminants generated during production.

Ventilation Infrastructure

Fresh Air Supply

The fresh air will be pushed from the upper portal into the exploration drift, where two sets of raises will bring the air to the various levels. Each zone has its set of raises to facilitate the distribution and control of fresh air.

Main Fans

The main fans were selected with the assistance of a Ventsim™ ventilation simulation model. To reach a sufficient airflow, two fans of 300 hp working at 70% of capacity will be able to push 387 kcfm at a total pressure of at 4.6 w.g. (1.14 kPa). The most appropriate fan arrangement is considered to be two forcing fans in parallel.

Mine Exhaust

The air on the level will travel to the ramp (4.5 m x 4.5 m) exhaust and will exhaust through the lower portal for both Marc and AV and JW zones (Figure 16-27).

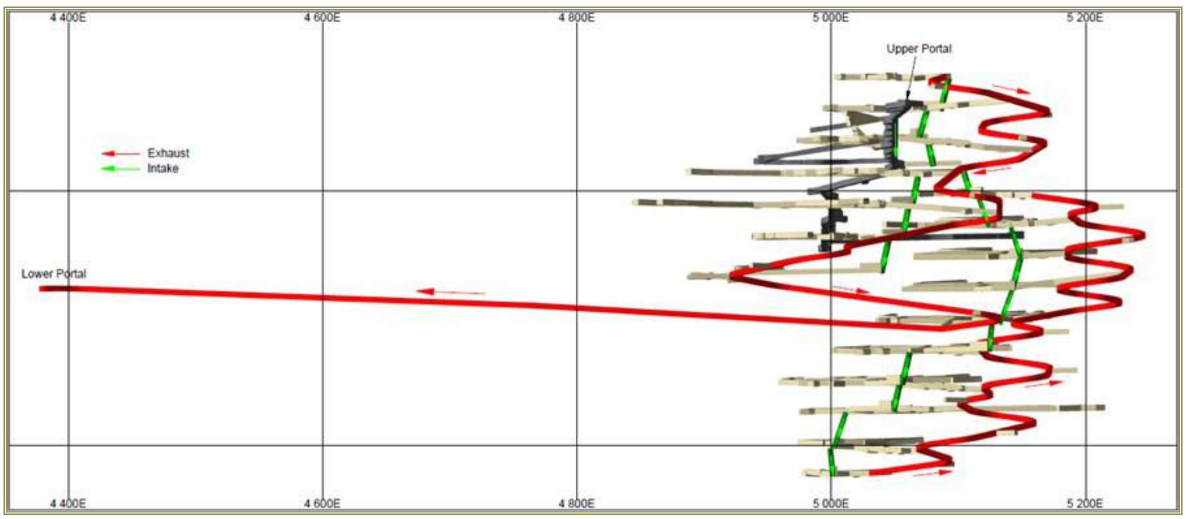


Figure 16-27: Ventilation Network Overview (Looking South)

Mine Air Heating

The heating system consists of direct flame burners and has been sized to heat a total flow of 387 kcfm (182.6 m³/s) and a temperature differential of 10°C which corresponds to the average maximum of the city at a set point of 4°C.

Mine air will be heated by propane gas with two burners, each rated at 7,500 MBTU/H, located upstream of the main fans.

Ventilation Network

The Red Mountain Project consists of the following zones: Marc, AV, and JW.

Marc Zone

The Marc Zone is the zone closest to the existing portal and the exploration drift. Level 1835 is accessed via the existing drift and will allow to start production faster. Once the development of the ramp is completed, ventilation doors will have to be installed on each side of the existing drift where the level connects to not contaminate the mine's intake air with the exhaust of Level 1835 (Figure 16-28). The ventilation doors will be synchronized to limit the contamination from the exhaust air when vehicles travel on this level.

When the permanent network is completed, a ventilation door will be installed in the access to the raises with a secondary fan mounted on it. A second fan will be installed in the existing drift upstream from the level connection. Vent ducting will bring the air to the working face. An "exhaust fan" will pull the air from the level and push it in a vent duct to the other side of the existing drift, where it will exit the level via the ramp.

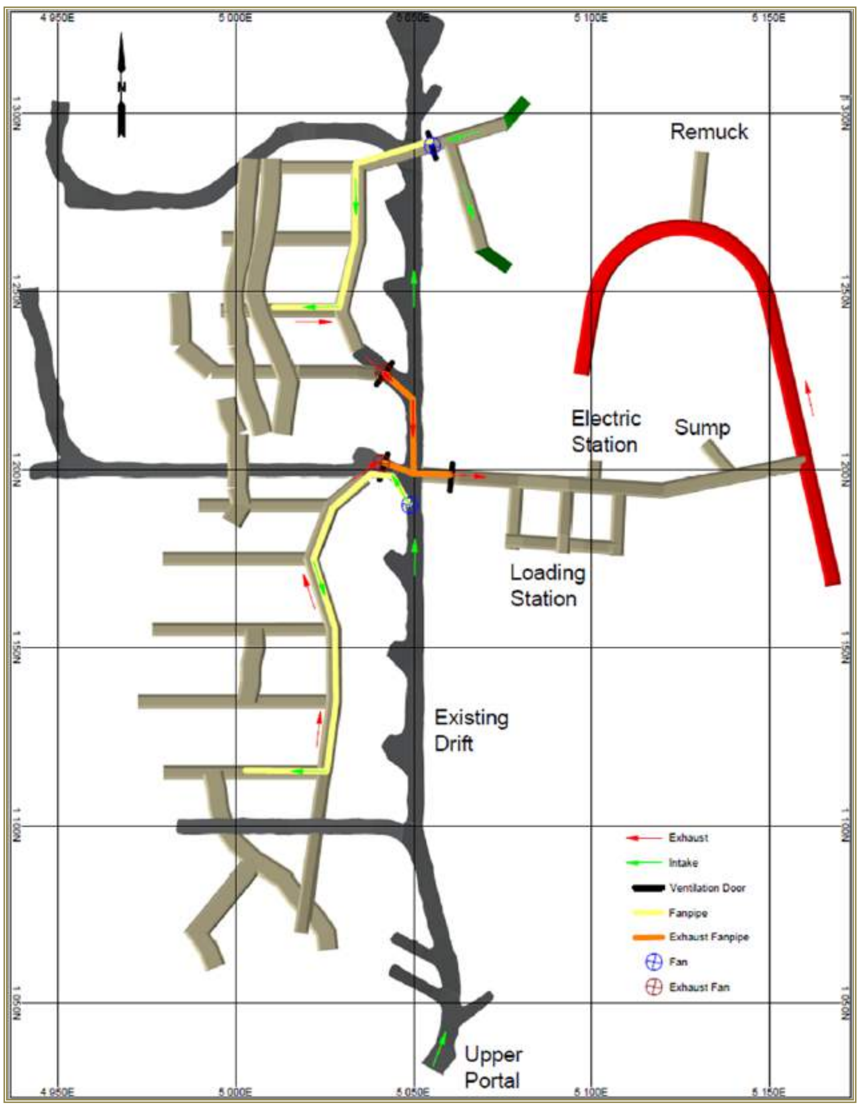


Figure 16-28: Ventilation Setup on Level 1835 in the Marc Zone

For the other levels of the zone, the fresh air will arrive from the raise, located north of the level. A ventilation door will be installed in the access drift with a fan mounted on it. Vent ducting will bring the air to the working face and the contaminated air will then circulate on the level up to the ramp (see Figure 16-29).

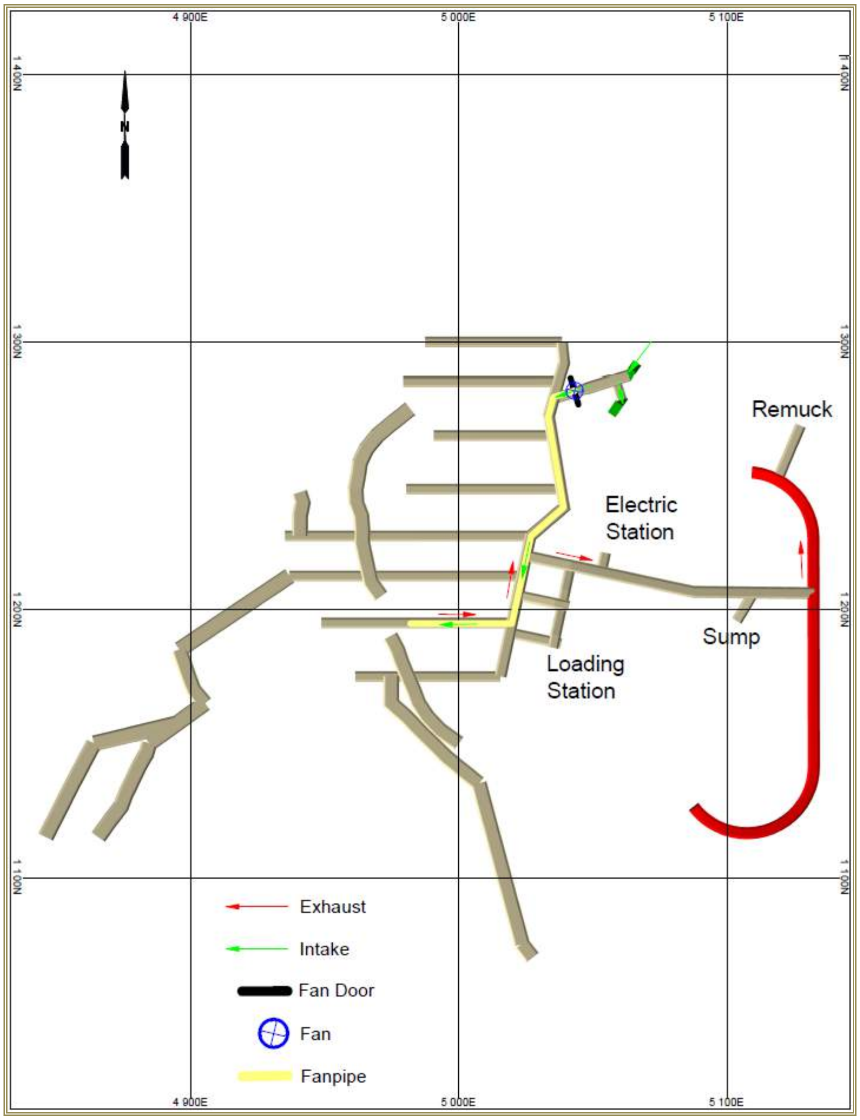


Figure 16-29: Ventilation on a Typical Level in the Marc Zone

Premier

To achieve the desired production rate for this zone, it was determined that 265 kcfm will be needed. The ventilation demand has been determined to satisfy the Canadian ventilation standards. Table 16-23 lists the equipment and their airflow needs. The airflow needs are based on the CANMET engine certificate.

In estimating the rate of fresh air flow for the entire mine, a utilization rate has been applied to account for time when machines may be mechanically unavailable, or simply not in use. The utilization rates are, respectively 100%

for production equipment, 80% for most service equipment, and 50% for machinery that functions primarily with electricity.

A contingency of 10% has been applied on the total estimated fresh air requirements to allow for additional equipment that could be added during the life of mine. It also allows a quantity of air for potential leaks in the system.

Table 16-23: Mobile Equipment List with Related Volume Requirements

Equipment	Airflow CFM	Quantity	Utilization Rate %	Total Airflow CFM
Jumbo – 2 Booms	9,200	1	50	4,600
LHD – 10-t	17,000	3	100	51,000
Mine Truck – 30-t	21,100	3	100	63,300
ITH Drill – Long-Hole	6,900	1	50	3,450
Bolter	4,800	1	50	2,400
Explosive Truck	6,900	1	80	5,520
Scissor Lift	6,900	1	50	3,450
Shotcrete Sprayer	6,900	1	80	5,520
Personnel Carrier	6,900	1	80	5,520
Lube Service Truck	6,900	1	80	5,520
Boom Truck	6,900	1	80	5,520
Motor Grader	11,500	1	80	9,200
Utility Vehicle	4,000	1	80	3,200
Backhoe with Rockbreaker	4,000	1	80	3,200
Telehandler	7,300	1	80	5,840
Mechanics Truck	7,300	1	80	5,840
Toyota PC	7,300	3	80	17,520

The mine fresh air demand was evaluated according to the equipment on hand annually and is presented in Table 16-24. The maximum demand of 265 kcfm is expected to be reached in Year 2 of production.

Table 16-24: Airflow Requirements for the Zone

	Initial	Y1	Y2	Y3	Y4	Y5	Y5	Y6	Y7	Y8
Required Airflow	82,630	156,660	200,600	194,760	194,760	194,760	194,760	194,760	194,760	194,760
Contingency 10%	8,263	15,666	20,060	19,476	19,476	19,476	19,476	19,476	19,476	19,476
Total Airflow	**90,893**	**172,326**	**220,660**	**214,236**	**214,236**	**214,236**	**214,236**	**214,236**	**214,236**	**214,236**

A typical level will have 1 scoop and 1 truck, meaning it will be necessary to deliver 40,000 cfm on every active level. Each level has a drift access to a ventilation raise, which will have a ventilation door to control the air entry. A 20 hp secondary fan will be mounted on the ventilation door and a vent duct will bring the fresh air to the face of the stopes that are being mined. This flow will be sufficient to clear the blast fumes and the contaminants generated during production. The air will then exhaust through the level access to the ramp.

Ventilation Infrastructure

Fresh Air Supply

The fresh air will be pushed from the upper portal in an existing drift, where 183 kcfm will continue through this drift to vent the upper and further areas of the zone. The rest will travel down through one raise to the middle and the lower parts of the zone.

Main Fans

The main fan was selected with the assistance of Ventsim™, a ventilation simulation model. To reach a sufficient airflow, 1 fan of 265 hp working at 90% of its capacity will be able to push 265 kcfm at a total pressure of at 2.0 w.g. (0.5 kPa).

Mine Exhaust

The air on the levels will travel to the ramp (4.5 m x 4.5 m) and will exhaust through the lower portal of the zone (Figure 16-30).

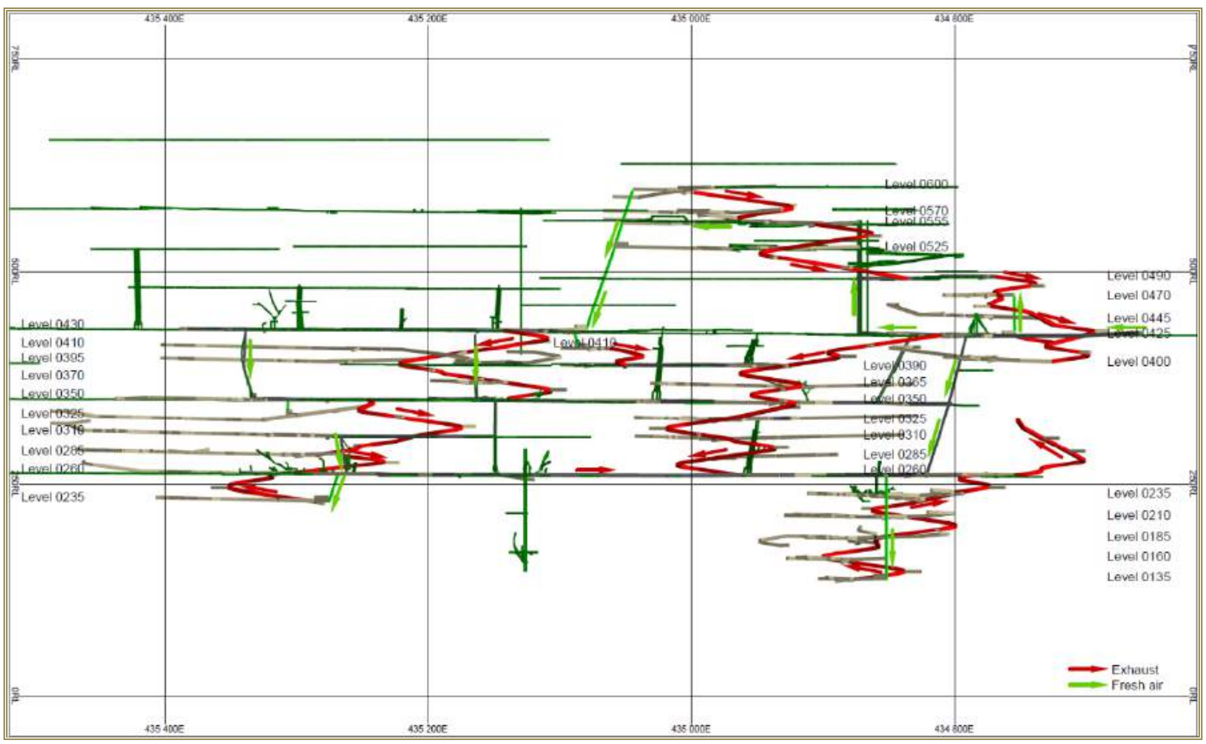


Figure 16-30: Ventilation Network Overview (Looking South-East)

Mine Air Heating

The heating system consists of direct flame burners and has been sized to heat a total flow of 265 kcfm (107.0 m³/s) and a temperature differential of 10°C, which corresponds to the average maximum of the city at a set point of 4°C.

Mine air will be heated by propane gas with one burner, rated at 7.5 MBTU/h, located upstream of the main fans.

Silver Coin and Big Missouri

SC and BM share an access through the historical S1 Pit and to achieve the desired production rate for these zones, it was determined that 350 kcfm will be needed. The ventilation demand has been determined to satisfy the Canadian ventilation standards. Table 16-23 lists the equipment and their airflow needs. The airflow needs are based on the CANMET engine certificate.

In estimating the rate of fresh air flow for the entire mine, a utilization rate has been applied to account for time when machines may be mechanically unavailable, or simply not in use. The utilisation rates are, respectively 100% for production equipment, 80% for most service equipment, and 50% for machinery that functions primarily with electricity.

A contingency of 10% has been applied on the total estimated fresh air requirements to allow for additional equipment that could be added during the life of mine. It also allows a quantity of air for potential leaks in the system.

Table 16-25: Mobile Equipment List with Related Volume Requirements

Equipment	Airflow CFM	Quantity	Utilization Rate %	Total airflow CFM
Jumbo – 2 Booms	9,200	2	50%	9,200
LHD – 10-t	17,000	5	100%	85,000
Mine Truck – 30-t	21,100	6	100%	126,600
ITH Drill – Long Hole	6,900	2	50%	6,900
Bolter	4,800	2	50%	4,800
Explosive Truck	6,900	2	80%	11,040
Scissor Lift	6,900	1	50%	3,450
Shotcrete Sprayer	6,900	1	80%	5,520
Personnel Carrier	6,900	1	80%	5,520
Lube Service Truck	6,900	1	80%	5,520
Boom Truck	6,900	1	80%	5,520
Motor Grader	11,500	1	80%	9,200
Utility Vehicle	4,000	1	80%	3,200
Backhoe with Rockbreaker	4,000	1	80%	3,200
Telehandler	7,300	1	80%	5,840
Mechanics Truck	7,300	2	80%	11,680
Toyota PC	7,300	6	80%	35,040

The mine fresh air demand was evaluated according to the equipment on hand annually and is presented in Table 16-26. The maximum demand of 371 kcfm is expected to be reached in Year 2 of production.

Table 16-26: Airflow Requirements for the Zones

	Initial	Y1	Y2	Y3	Y4	Y5
Required Airflow	147,970	249,350	337,230	337,230	337,230	337,230
Contingency 10%	14,797	24,935	33,723	33,723	33,723	33,723
Total Airflow	**162,767**	**274,285**	**370,953**	**370,953**	**370,953**	**370,953**

A typical level will have 1 scoop and 1 truck, meaning it will be necessary to deliver 40,000 cfm on every active level. Each level has a drift access to a ventilation raise, which will have a ventilation door to control the air entry. A 20 hp secondary fan will be mounted on the ventilation door and vent duct will bring the fresh air to the face of the stopes that are being mined. This flow will be sufficient to clear the blast fumes and the contaminants generated during production. The air will then exhaust through the level access to the ramp.

Ventilation Infrastructure

Fresh Air Supply

Early in the project, Silver Coin and Big Missouri will initially require separate systems given their distant location from each other. Once they get connected midway into their development stage, fresh air will be pushed down a new raise in the Silver Coin Zone and will be routed to Big Missouri via the connecting drift developed from SC on Level 0810.

During the pre-production period at Big Missouri, a fan will be installed close to the portal with adequate vent ducting and will push the air to the desired areas. The contaminated air will flow back and exhaust through that portal.

As in the other zones, 1 scoop and 1 truck will be needed by stopes and will demand 40,000 cfm. Fans will be positioned at strategic places and will push the air through vent ducting to the active stopes. The contaminated air will then travel back and exhaust through the upper portal.

Main Fans

The main fans were selected with the assistance of Ventsim™, a ventilation simulation model. The final setup to reach a sufficient airflow will consist of two fans of 300 hp working at 95% of their capacity. They will be able to push 350 kcfm at a total pressure of at 6.4 w.g. (1.6 kPa). The fans will be installed in series.

At the beginning, two fans will be installed of top of the raise in the Silver Coin zone and once the connection to Big Missouri is completed, the fan that was installed in the Big Missouri portal will be moved to the Silver Coin raise to increase the quantity of airflow available.

Mine Exhaust

For Silver Coin, the air on the levels will travel to the ramp (4.5 m x 4.5 m) and will exhaust through the portal (Figure 16-31).

Once Big Missouri is connected, the air will travel via the ramp (4.5 m x 4.5 m) and will exhaust through the upper portal (Figure 16-32).

Mine Air Heating

The heating system consists of direct flame burners and has been sized to heat a total flow of 350 kcfm (165.2 m^3/s) and a temperature differential of 10°C, which corresponds to the average maximum of the city at a set point of 4°C.

Mine air will be heated by propane gas with two burners, each rated at 7,500 MBTU/H, located upstream of the main fans.

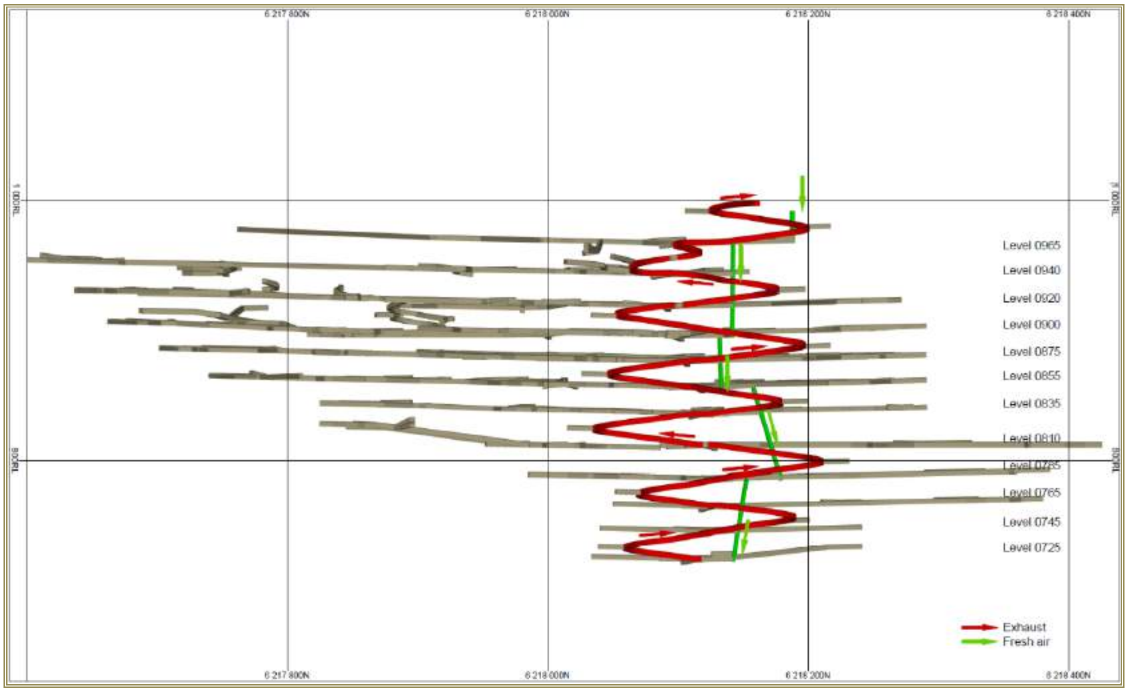


Figure 16-31: Ventilation Network Overview Silver Coin (Looking East)

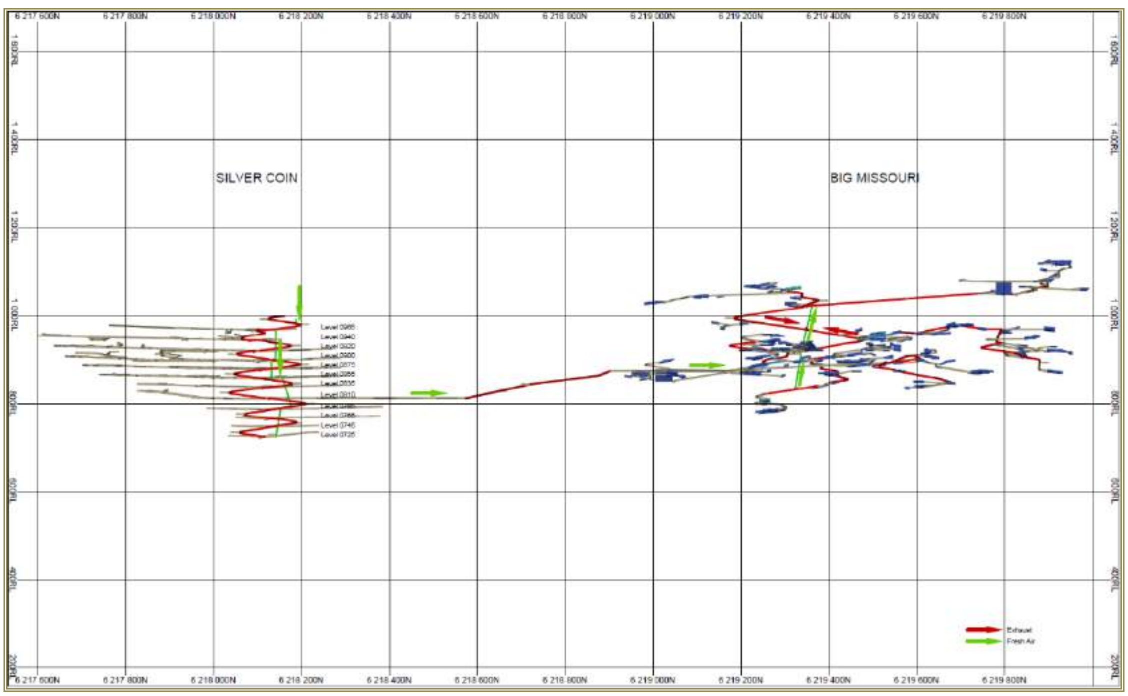


Figure 16-32: Ventilation Network Overview Silver Coin and Big Missouri (Looking East)

16.12.2 Dewatering

Red Mountain

Groundwater inflows into the mine will vary throughout the year. Increased flow rates can be expected during the snow melt in spring. Peak flow rates were modelled and estimated at 6,400 m^3/d and a minimum flowrate of 42 m^3/h is estimated for the winter months. Once developed, all mine water will be handled through the lower portal. Prior to this, water is to be pumped out the existing portal.

Used mine water and ground water is collected at sumps located at the entrance of each level. Water is fed from the footwall drive sumps and production/development faces using small portable 13 kW submersible pumps; discharging into steel pipes with 76 mm diameter. Once the used mine water and ground water reaches the main dewatering sumps, it will first enter the dirty water collection sump, water will then filter over to the bulk headed pump station collection sump via drain holes.

Table 16-27: Dewatering System – Main Design Criteria

Properties	Unit	AV.	Max.
Production	m^3/h	20	73
Seepage Water	m^3/h	63	
Total Dewatering Capacity	**m^3/h**	**83**	**136**
Solid Content	%	0.3	15
Total Solid in the Slurry	**Dry t/month**	**170**	

A total of five pumping stations will be installed during the Red Mountain mine life. The location of each pumping station is relevant to the deepest development for Year 0 to Year 3; development does not mine deeper in Year 4 to Year 6.

Pump setup #1 is located at 1,860 masl elevation, situated near the existing portal (Figure 16-33). The pump station consists of two 56 kW centrifugal pumps, each with their own independent 200 mm discharge lines. A 58 hp submersible pump will also be installed with a 3" discharge line purposed for mine supply water.

Pump setup #2 is located at 1,785 masl, situated at the deepest part of development at the end of Year 0 (Figure 16-34). The pump station is equipped with two 188 kW centrifugal pumps, each with their own 200 mm discharge lines. Pump setup #2 is designed to pump water to pump setup #1.

Pump setup #3 is installed at the end of Year 0 when the lower portal development reaches Level 1700 in the AV/JW Zone (Figure 16-33). The sump is situated near the junction with the ramp. The pump station is equipped with two 58 hp submersible pumps each with their own 200 mm discharge line.

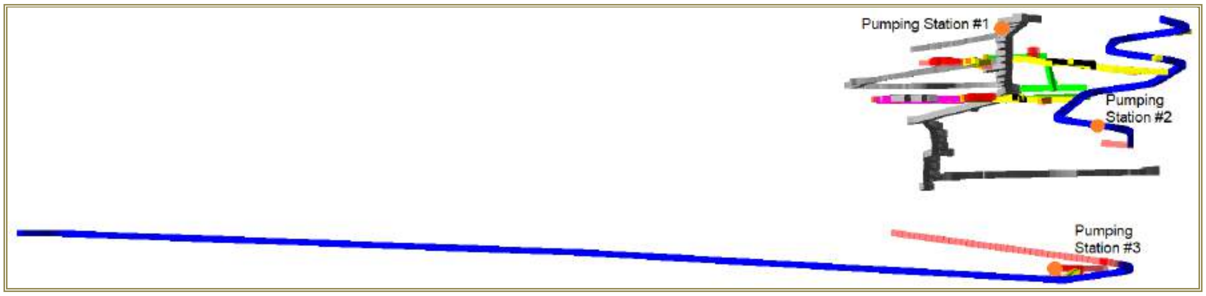


Figure 16-33: Progression of the Development at the End of Year 0

Pump setup #4 is located at 1,746 masl, which is the location of the sump on Level 1750 in the AV/JW zone (see Figure 16-34). The development is going upward and reaches that level at the end of the Year 1. The pump station is setup with two 56 kW centrifugal pumps each equipped with their own 200 mm discharge lines. Water is pumped from pump setup #4 to pump setup #3 before being pumped out via the lower portal.

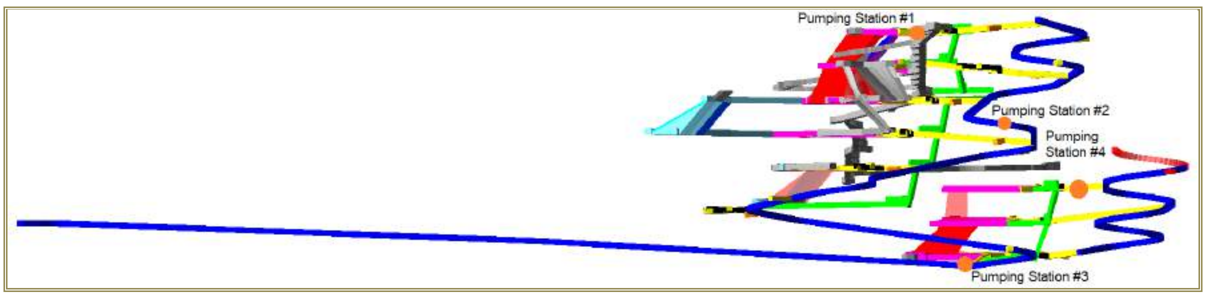


Figure 16-34: Progression of the Development at the End of Year 1

Pump setup #5 is located at 1,606 masl near the deepest part of development during the Red Mountain mine life (Figure 16-35) at the end of Year 2. This pump station consists of two 250 hp centrifugal pumps with two 200 mm discharge lines. Water is pumped from pump setup #5 to pump setup #3.

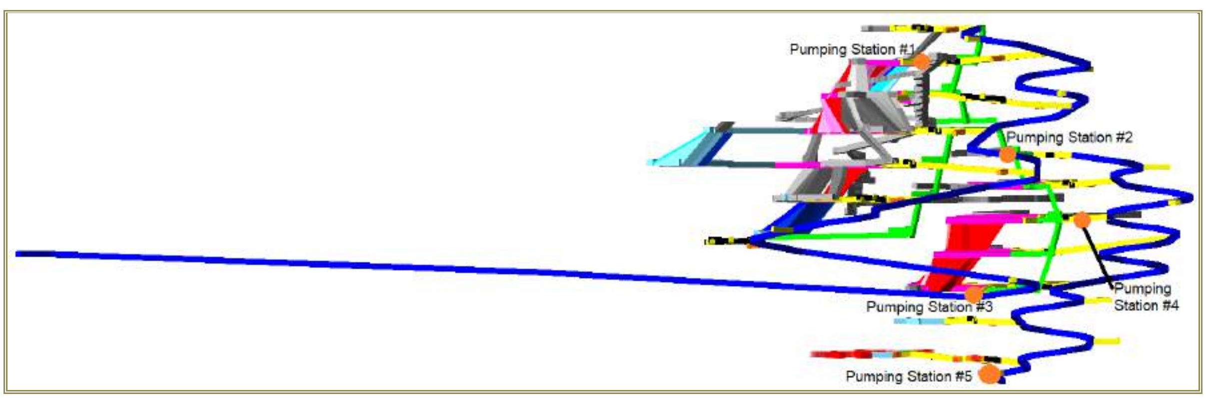


Figure 16-35: Progression of the Development at the End of Year 2

Pump setup #5 will be moved to its permanent place at the end of Year 3 at elevation 1565, after which the development will not go deeper (Figure 16-36).

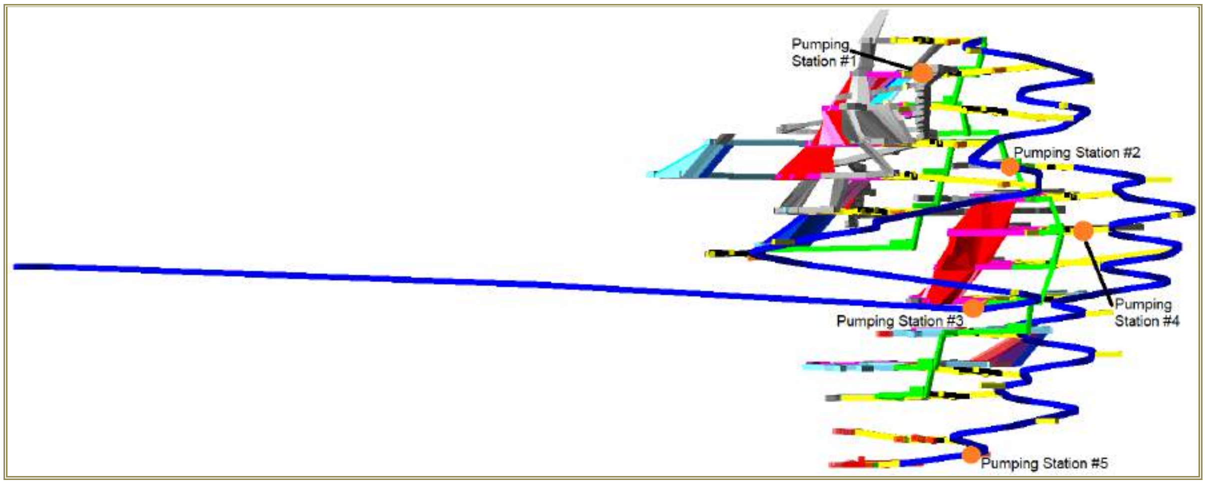


Figure 16-36: Progression of the Development at the End of Year 3

Figure 16-37 shows the network of the pump station locations for the final years of mining. All water discharged from the main dewatering sumps is collected at the water treatment plant.

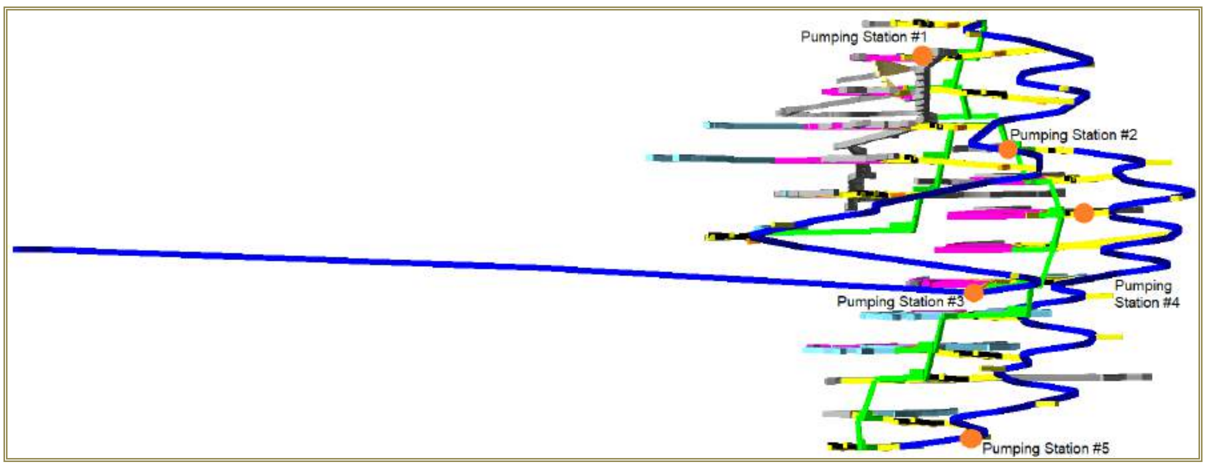


Figure 16-37: Permanent Dewatering Network

Premier Gold Project

<u>Premier Northern Light</u>

A total of 11 pumping stations are needed to ensure production at the Premier Zone. The location of each pumping station is dependant on the deepest development in each period from Year 2024 to Year 2026. Beyond this period, additional development does not progress any deeper and therefore no further modification is required.

Each station will be equipped with a pump or a set of pumps that will ensure sufficient dewatering capacity on the various levels without compromising production. A hydrogeological study has determined that this zone will have a high level of water inflow. Piping will be sized to control this seepage and channel the water to pumping station #1, so it can be directed through the 6Level (6L) Portal to the water treatment plant and settling ponds.

Pump setup #1 is located on production Level 0260, close to the new ramp leading to the Production Portal. The ramp connects to the existing 6L and Portal, where water will be exiting the mine.

Pump setup #2 is located on production Level 0425, an existing level close to the 4L Portal. This pumping station will be the main pumping station for this area of the mine, and will collect the water from the upper levels and direct it to the pumping station #1.

Pump setup #3 is located on production level 0210 at the end of Year 2024, which is the lowest level that mine development will reach. This pumping station will direct water to pumping station #1.

Pump setup #4 is located on production Level 0400. This level is only one level below pumping station #2. There are no levels below it, so water will accumulate in this area. Figure 16-39 shows the mine in a longitudinal view, looking North-West, and demonstrates the relative position of this level relative to the rest of the zone. Setup #4 will direct water to pumping station #2.

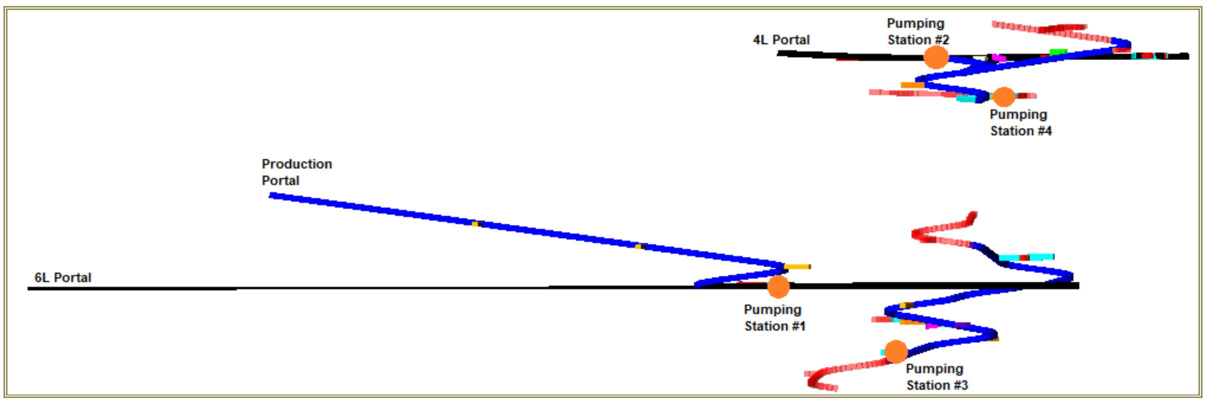


Figure 16-38: Progression of Development at the End of Year 2024, Looking NE

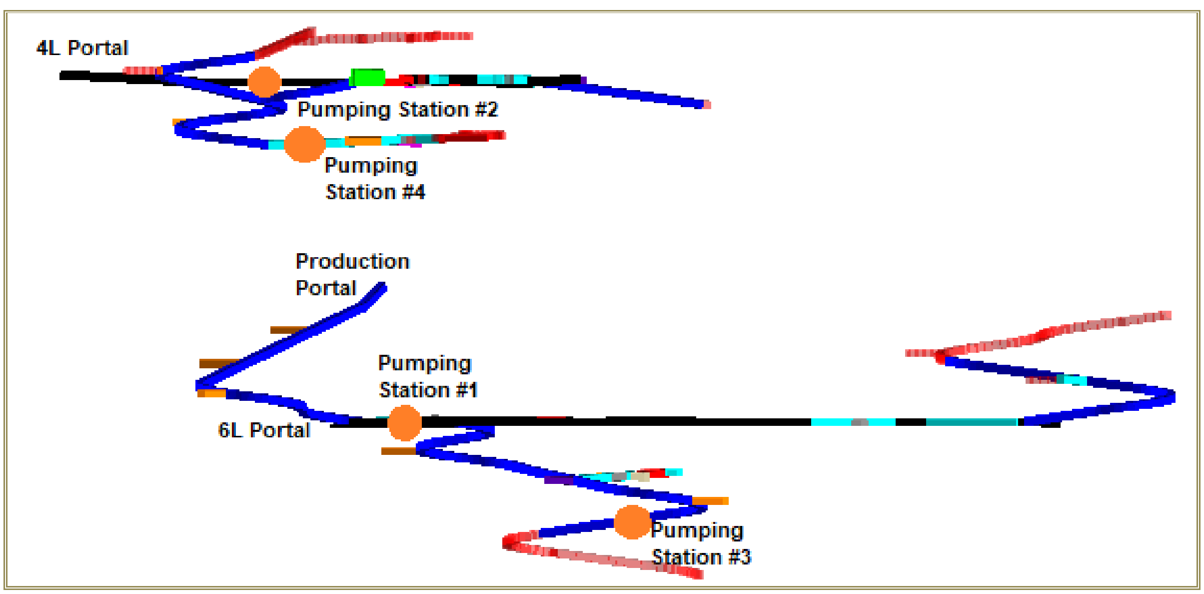


Figure 16-39: Progression of Development at the End of Year 2024, Looking NW

Pump setup #5 is located on production Level 0135, the lowest level of the mine. It will direct water to pumping station #3. This pumping station will be ready by the end of Q3 of Year 2025.

Pump setup #6 is located on production Level 0350, which is an existing level that will be rehabilitated for production. By the end of Year 2025, Level 0260 and Level 0425 will be connected by a ramp at the middle part of the mine. This will ease the water management from production Level 0425 above. Pumping station #6 will direct water to pumping station #1.

Pump setup #7 is located on Level 0490, which is an existing level that will be rehabilitated for production. That station will collect water from these two levels and will direct water to pumping station #2.

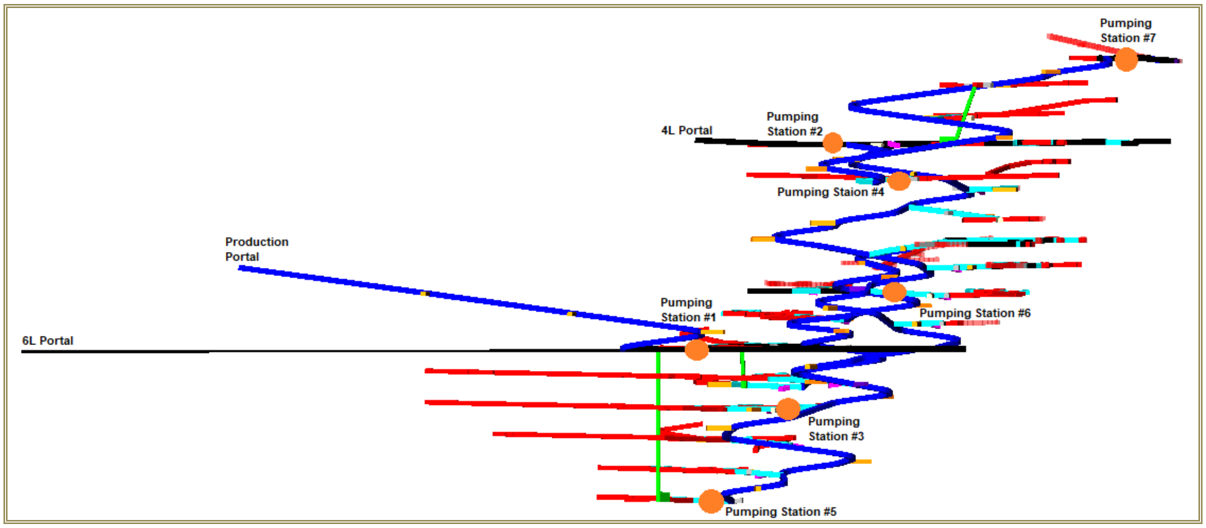


Figure 16-40: Progression of Development at the End of Year 2025, Looking NE

Pump setup #8 is located on production Level 0260 at the far end of the mine. That level will be rehabilitated for allowing production in this part of the mine. It will direct water to the pumping station #1.

Pump setup #9 is located on production Level 0235, which is a bottom level. It will direct water to pumping station #8.

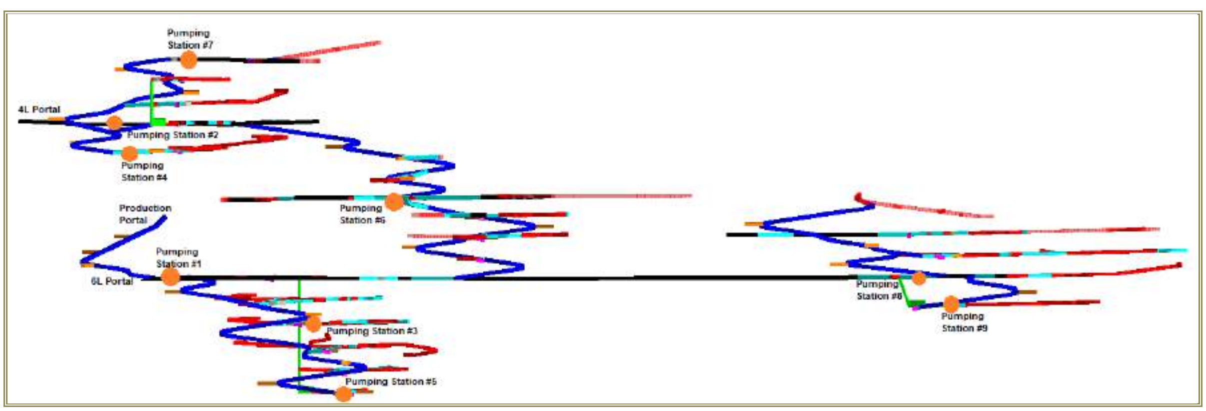


Figure 16-41: Progression of Development at the End of Year 2025, Looking NW

Pump setup #10 is located on production Level 0555, which is an existing level that will be rehabilitated for production. This level will allow ventilation to travel to the furthest end of the mine and allows to connect the TWO upper levels. Setup#10 will collect water from these TWO levels and will direct it to pumping station #7.

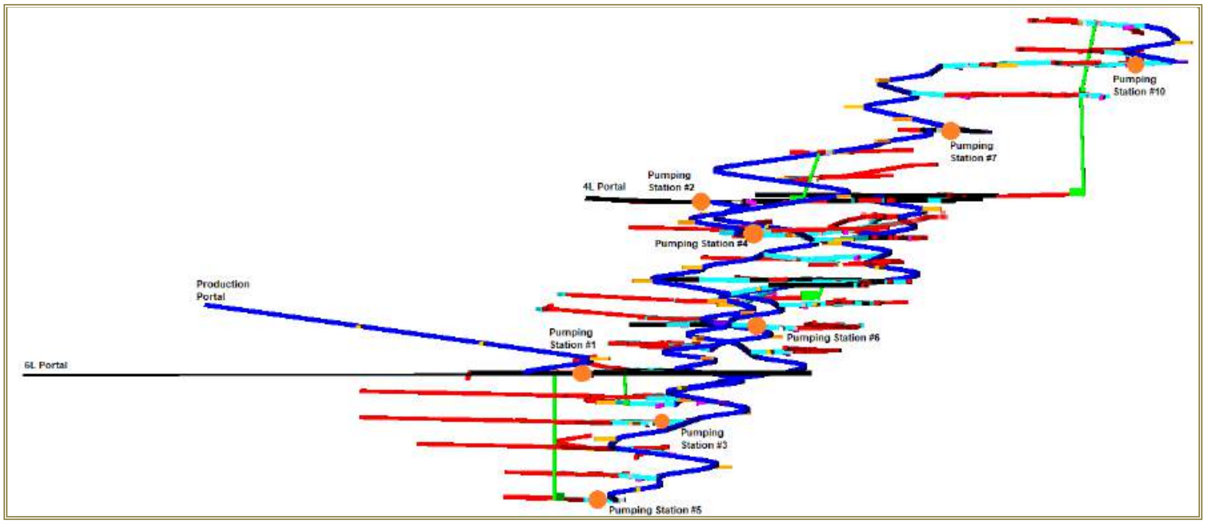


Figure 16-42: Permanent Dewatering Network, Looking NE

Pump setup #11 is located on production Level 0350, which is an existing level that will be rehabilitated for production. This pumping station will direct water to pumping station #8.

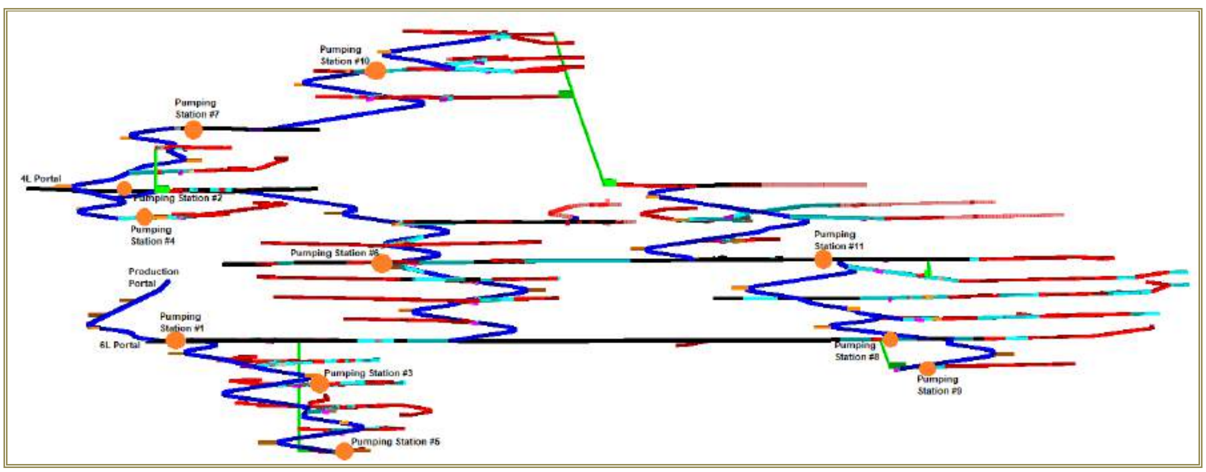


Figure 16-43: Permanent Dewatering Network, Looking NW

Silver Coin and Big Missouri

A total of six pumping stations will be needed at Silver Coin and five Big Missouri to ensure mine production. The location of each pumping station at Silver Coin will follow the development of the ramp while respecting the maximum water head that can be managed by the pumps. There are no defined levels at Big Missouri. The pumping stations will be located in strategic locations.

A temporary setup will channel seepage and production water through the existing portal on Level 0810. An HDPE pipe will connect to the settling ponds located close to Big Missouri. A second temporary setup will be installed in the production portal at the top of Silver Coin and will connect to the first pipe North of the zone. Because this setup is planned to be outside, the pipe will be insulated and heated to avoid freezing. Once the drift connecting Silver Coin and Big Missouri is completed, a permanent pumping network will be installed through that drift and exit through the Big Missouri production portal.

Each station will be equipped with a pump or set of pumps that will ensure the levels are dewatered.

Pump setup #1 will be located on production Level 0810, an existing level that will be rehabilitated to allow production. It will be located near the junction where development meets the new drift to Big Missouri. This station is the main pumping station for the Silver Coin zone.

Pump setup #2 will be located near the Silver Coin production portal to pump water through the temporary pipeline setup until the ramp connects to Level 0810 and Level 0810 connects with Big Missouri. Once the ramp breaks though, the water will be redirected to pumping station #4.

Pump setup #3 will be located on Level 0855 near the connection to the ramp. It will pump water to pumping station #1.

Pump setup #4 will be located on production Level 0920 and will direct water to pumping station #2. The ramp breaks through in the first quarter of Year 2021. After that, the water will be directed to pumping station #3.

Pump setup #5 will be located on production Level 0765 and will direct water up to pumping station #1. It will be located at the entrance of the level.

Pump setup #6 will be located on production Level 0725, which is the lowest level at Silver Coin. It will be located at the entrance of the level and will direct water up to pumping station #5.

For Big Missouri, pump setup #1 will be located near the production portal and will be the main pumping station for the area. All water from Silver Coin and Big Missouri will eventually be pumped to this station where it will be redirected to the settling ponds near Big Missouri.

Pump setup #2 will be located in the connecting drift from Silver Coin, which is production Level 0810. This is an existing level (Level 2850) that will be rehabilitated for production and it connects to an existing portal. The pumping station will be located at the junction of the connecting drift from Silver Coin and the drift leading to the portal, which is at an elevation of 875masl. Water from this pumping station will be directed to pumping station #1.

There are four main ramps at Big Missouri below pumping station #2, each of which will require a pumping station at the bottom. These four pumping stations will direct water to pumping station #2 (Figure 16-44).

Four more pumping stations will be needed in the upper part of Big Missouri. Pumping stations #7, #8, and #9 will collect water from levels having a negative grade and will direct water to pumping station #10. Pumping station #10 will be located at the junction of the two main ramps and will redirect water to pumping station #1.

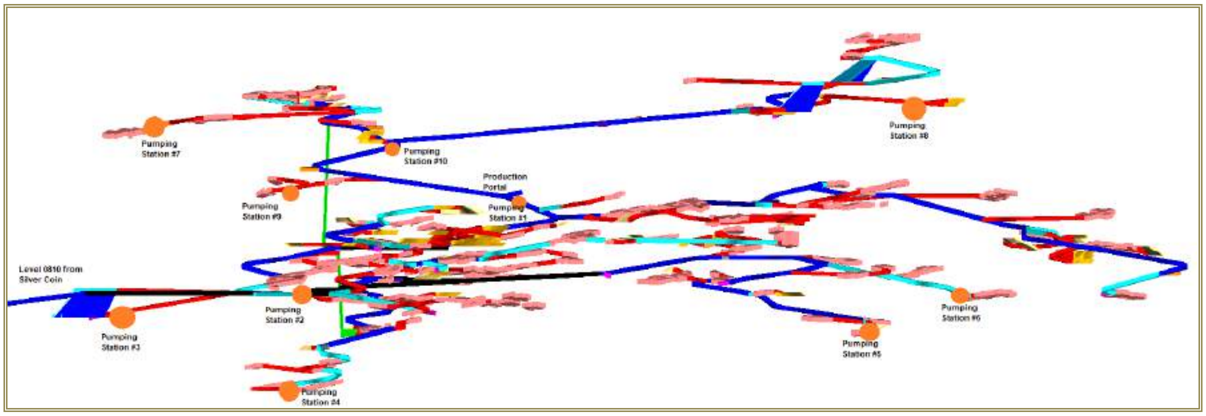


Figure 16-44: Pumping Network for the Big Missouri Zone, Looking NW

The remaining mine services listed below will use common equipment and approaches regardless of site. What is shown is typical of either PGP or RMP, except that PGP will be supplied by grid power while RMP will use gensets.

16.12.3 Compressed Air

At RMP compressed air supply is provided by two 1,476 CFM electric compressors that had been installed temporarily when the underground development work was initiated in July 2016. There are plans to add two additional units that will be installed in the newly built compressor room. The two temporary units will then be relocated and installed alongside the others in the compressor room. Similar installation will be used at the various PGP sites, with equipment moved or re-used as necessary.

The compressed air piping network will be installed along the ramp, the main drifts and the escapeways throughout the mine. Compressed air shall provide power to pump for dewatering development work, handheld drills (for specific and limited use in planned development, and R&P stope), as well as provide an emergency supply of air to the refuge station.

16.12.4 Industrial Water

Industrial water supply for mining will predominantly come from settling ponds where water will be recirculated for use underground. Naturally occurring supplies such as creeks may be used as needed.

16.12.5 Underground Explosive Storage

Each operating site will have two separate underground storage facilities for explosives and accessories. The storage capacity is designed to hold 90,000 kg of explosives material and 80,000 detonators (caps). Based on the typical design at RMP as shown below, similar designs will be used at PGP for PNL and SC/BM.

16.12.6 Underground Power Distribution

One 4,160 V feeder (riser teck cable), through the portal, will be used to supply the first level underground substation (and the future substation to be located on the 400 m level). One 13.8 kV feeder, through dedicated services hole from surface, will supply the lower levels of the mine. Junction boxes will be installed at each level to feed the main underground substations. The 13.8 kV and 4.16 kV feeder will ensure satisfactory voltage regulation throughout the ramp and raise, while minimising the size of conductors.

Underground main substations will be installed, in conformity with the Mine Code and Canadian Electrical Code (CEC), at each main level. Substations will provide 4.16 kV and 600 V power to underground loads such as pumps, ventilation, garages, 4.16 kV mobile substations, etc. All substation equipment will be located inside dedicated electrical room excavations along the main ramp.

A common modular diesel generator station will be installed near surface electrical substation sites to accommodate underground emergency loads (compressors, and pumps).

The total emergency load is estimated (under full mine operation) as indicated in the following Table 16-28.

Table 16-28: Estimated Emergency Load

Description	Power (kW)
Compressors	300
Pumping	200
Ventilation	500
Total	**1000**

The generators will be installed in individual waterproof shelters and the 4,16 kV switchgear will be installed in a separate building along with control, protection, metering, and batteries.

Generator grounding will be connected to the isolated grounding grid in order to keep a constant ground fault level.

One above-ground fuel tank will be required, having a capacity of 10,000 L. At full load, there will be enough fuel to supply all the generators for a period of 24 hours.

16.12.7 Underground Communication

System Overview

The various mining operations will be provided with a fibre optic and "leaky feeder" network to support all required communications to operate the mine and undertake data handling inside the mine and to the surface. The communication infrastructure will have all the capabilities to support voice communications, PLC monitoring and control, video, operation data and to control and monitor the electrical network. The communication network will be based on a communication infrastructure composed of two major technologies: a fibre-optic cable backbone and a radiating cable technology. This communication network will meet all data transmission requirements and ensure redundancy and improved reliability for most of the Ethernet/IP applications. The network equipment will be specified to enable reconfiguration of the network without communication services outage as the exploration and operation shafts are being developed.

Design Standards

Unless otherwise specified, the design standards take into consideration the fact that communication equipment will be installed from the very first stages of the mine construction. The installed equipment must therefore be operational from the construction stage to the mine production stage. A provision has been made for installing temporary communications infrastructure.

Whenever possible, all systems must respect the following criteria:

- Reliability (redundancy implementation)
- Flexibility
- Scalability.

Fibre-Optic Backbone

The fibre-optic cable backbone will be installed in the vent raise in a ring configuration wherever possible. Fibre termination point will be provided for each main electrical room in a communication cabinet.

Radiating Cable Technology (Leaky Feeder)

The radiating cable is primarily used for radio mobile communication services but can also transmit all critical data communications in case of either failure or during maintenance work of the fibre network. This technology also makes it very easy to move mobile, temporary and long-run temporary equipment, without modifying the configuration and programming of existing systems. This system will have the largest extent in the mine and will extend up to wireless sensors at the very end of each working area.

Communication Services

Both the fibre network and the radiating cable technology are expected to facilitate the integration of the services listed in Table 16-29.

Table 16-29: Communication Services and Cable Technology

Communication Services	Fiber Optic	Radiating Cable
Radio communication system	-	X
Interface to the surface communication system	X	
Surveillance and process camera network (High-Bandwidth)	X	X
Clock synchronization of the electrical protection relays	X	-
Monitoring and control of the electrical system	X	-
Control network for stationary and mobile equipment	X	X
Telemetry system	X	
Specific emergency frequency as well as dedicated phone to be used exclusively for mine rescue	X	X
Bridging to the surface radio link	X	-
3-gas detection system (CO, NO_2, O_2) and LEL	X	X
Blasting system		X
Auxiliary fan control	X	X
Control circulation lighting	X	X
Geolocation of personnel and vehicles	X	X

Emergency Power for Communication System

It is essential that communication facilities be supplied by a reliable power source. Generally, communication services are heavily used during power outages, major events, or emergencies. Having access to different power sources to supply the control and communication systems improves reliability during critical operation. Among others, proper power autonomy is quite critical as the mine relies on communications to supervise and control electrical elements of the system.

The primary source is based on independent and isolated battery supply such as a double-conversion uninterruptible power supply (UPS). This equipment will additionally protect the communication systems from electrical grid problems and temporary power failures. Sizing of this power source will be determined by the equipment installed and the charge required to ensure at least one hour of autonomy at full load with no assistance from another source.

A secondary power source should also be considered to improve reliability in case of a localized electrical issue that could affect the communication equipment in the event of extended power outage.

16.12.8 Mine Safety

Fire Prevention

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground refuge stations, electrical substations, pump stations, fueling stations, explosive magazines, and other strategic areas. Every vehicle would carry at least one fire extinguisher; the correct size and type would depend on the type of vehicle. All underground heavy equipment will be equipped with automatic fire suppression systems (Ansul system).

Mine Rescue

A fully trained and equipped Mine Rescue Team is essential to the safe operation of any mine and shall be organized at PGP with its dedicated location and equipment. The mine rescue team will be trained for surface and underground emergencies. A dedicated mine rescue vehicle is planned to be purchased and based at PGP to accommodate both PGP and RMP sites.

Refuge Station

Self-contained portable refuge stations will be installed at specific locations within the underground workings. Those refuge stations are equipped with compressed air, potable water, and first aid equipment. They will also be equipped with a fixed telephone line and emergency lighting. The refuge chambers will be sealable to prevent the entry of gases. The portable refuge stations are planned to be moved to new locations as the work areas advance. Permanent refuge stations have also been incorporated in the mine plan.

Emergency Egress

The main ramp is planned to provide primary egress from the underground workings. The fresh air raise (FAR) system with a dedicated manway would provide the secondary egress in case of emergency. The manway would be equipped with steel ladders and platforms.

Emergency Stench System

A stench gas system will be installed on each fresh air intake that may be triggered to alert underground personnel in the event of an emergency.

17 RECOVERY METHODS

A nominal plant treatment rate of 2,500 t/d has been selected as the basis to recover gold and silver at the Premier Gold Project (PGP) Mill site. The plant will operate 365 d/a to produce gold and silver doré with an overall plant availability of 92%. Ore will be supplied from several underground mines from the following locations:

- Premier Gold Project
  - Silver Coin
  - Big Missouri
  - Premier
- Red Mountain Project
  - Marc
  - AV
  - JW

It is expected that ore will be fed from either one of the PGP or one of the RMP deposits using a method of campaigning as each ore will require subtle adaptations to the processing method and its control. In the initial period after the mine restart, only PGP ore will be fed for approximately the first two years, after which, equal quantities of PGP or RMP ore will be campaigned for the remaining life-of-mine (LOM).

The plant feed will be stored on stockpiles prior to the primary crusher system which will allow for campaigning the ore on a bi-weekly basis. The feed to the plant will be reclaimed from either the PGP or RMP run-of-mine (ROM) stockpiles, and will be introduced to the ROM dump hopper via a haul truck or front-end-loader (FEL) and fed via a reciprocating plate feeder into a jaw crusher for initial size reduction. The primary sized crushed ore will be conveyed and stored in a coarse ore stockpile (COS), the stockpile is sized for approximately one day of capacity. Feed to the mill will be reclaimed by apron feeders, then conveyed to the semi-autogenous grinding (SAG) and ball mills, the latter of which will be operated in a closed circuit with hydrocyclones. While processing the PGP ores, a portion of the ball mill discharge will be pumped to a centrifugal gravity concentrator and the resulting gravity concentrate will discharge into an intensive leach reactor (ILR) for precious metals recovery. The Red Mountain ore types have very low amounts of gravity recoverable gold (GRG) and the gravity recovery circuit will be by-passed during the Red Mountain campaigns. The hydrocyclone overflow with P_{80} of 80 µm will flow into a carbon-in-leach (CIL) circuit for the PGP ores, or will report to a fine grinding mill for further size reduction (to 25 µm) for Red Mountain ores, which then reports to pre-leach thickening and then the CIL circuit.

Gold and silver leached in the CIL circuit is recovered on carbon and eluted in a pressure Zadra-style elution circuit, then electrowon in the gold room to form a precious metal sludge (gold/silver mixture). For PGP ores, an additional electrowinning cell is used to recover gold from the intensive leach solution. Precious metal sludge is then dried in an oven prior to being mixed with fluxes and smelted in a refining furnace to pour doré bars. Carbon is re-activated in a carbon regeneration kiln before being returned to the CIL circuit.

Leached tailings are detoxified in an SO_2/Air cyanide destruction circuit. For PGP ores, the detoxified tailings are thickened, then pumped to a tailings storage facility (TSF). For Red Mountain ores, the detoxified tailings will be pumped to the TSF.

Fresh water, primarily from Cascade Creek is delivered to the plant and is used for potable water, fire water, elution, and reagent mixing. Process water is recovered from the decant water from the TSF and is used for grinding and utility water.

17.1 Process Flowsheet Development

The fundamental basis of design for the original process plant has been retained for PGP, with new technologies and extraction methods employed to bring efficiency to the applied capital and associated operating costs. The original process flowsheet has been re-appraised and updated based upon the 2018 Base Metallurgical Laboratories Ltd. testwork results as outlined in Section 13, and was then confirmed and optimized with the 2020 testwork, completed primarily at SGS Vancouver, with some additional testwork completed at Base Metallurgical Laboratories. A large proportion of existing processing equipment will be re-furbished and re-utilized.

The updated process plant flowsheet consists of the following unit operations:

- ROM ore management
  - PGP ore and RMP ore stockpiles to be used to manage feed into the process plant
- Coarse ore management
  - A primary crushing system that sizes the ore and stockpiles the ore onto a coarse ore stockpile system; the primary crusher is in a standalone building complete with dust collection equipment
  - A semi covered coarse ore stockpile that places ore directly over the mill feed system reclaim tunnel
  - A coarse ore mill feed system that consists of a reclaim tunnel containing two variable rate apron feeders, and a third feed point using a vibrating feeder that can accommodate a front-end loader for returned materials
- Grinding and classification, consisting of a SAG and Ball Mill with hydrocyclones
- Gravity concentration and intensive leaching
- CIL
- Carbon management including carbon regeneration
- Gold room
- Tailings detoxification and deposition.

Following the introduction of the Red Mountain ore in Q4 Year 2, the following additional processing equipment will be installed and commissioned:

- A 3000 kW stirred media fine grinding mill
- A 27-m diameter pre-leach thickener.

The details regarding each of these processing steps are covered Section 17.3. Figure 17-1 illustrates the overall processing flowsheet, while Figure 17-2 shows the overall layout of the process facilities.

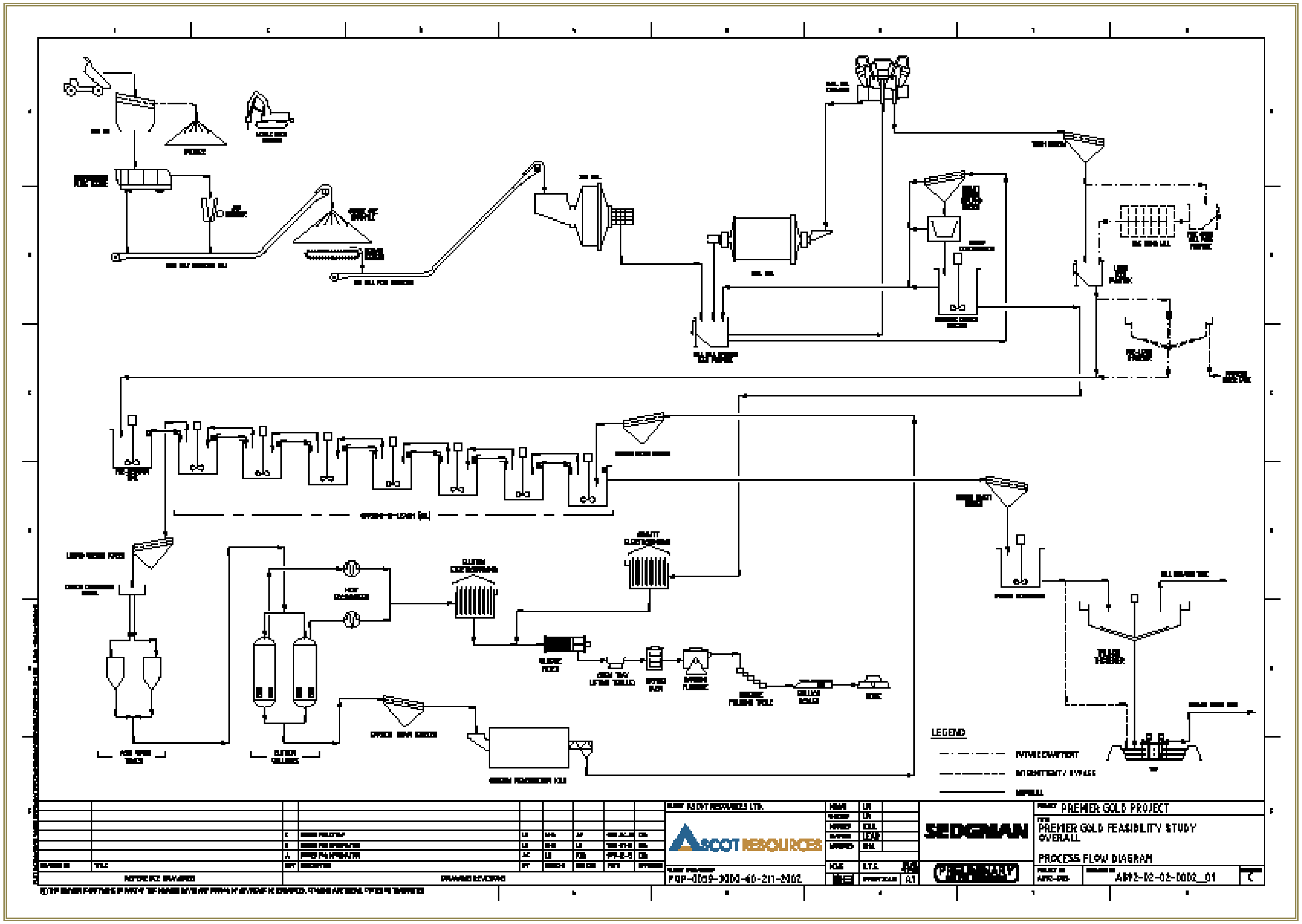


Figure 17-1: Project Overall Process Flowsheet

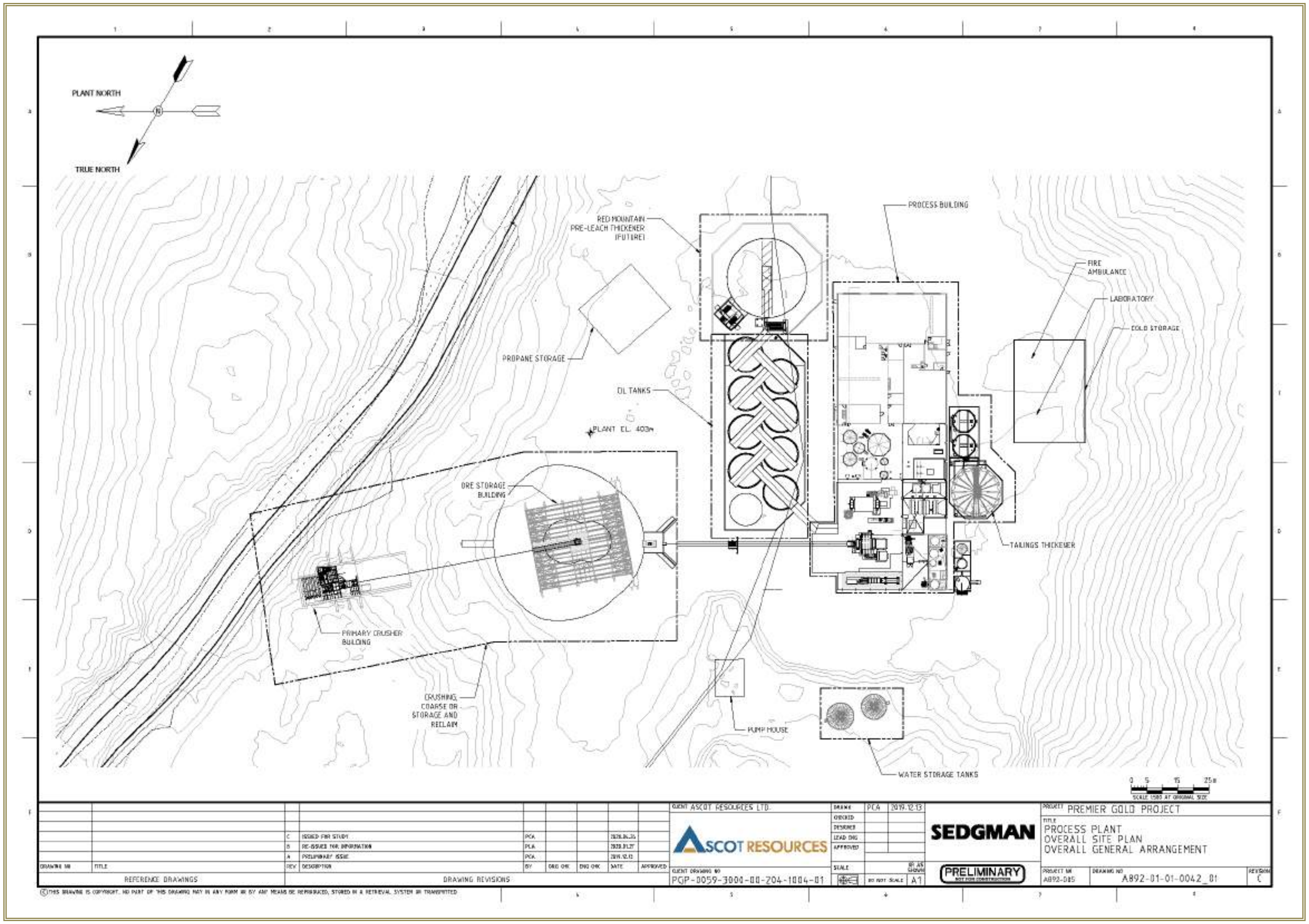


Figure 17-2: Process Plant Overall Layout

17.2 Major Process Design Criteria

Table 17-1 outlines the major process design criteria developed for PGP. For the first two years of plant production, PGP ore will be fed exclusively to the process plant at a throughput of 2,500 t/d. In Q4 Year 2, Red Mountain ore will be introduced through a campaigning process at a rate of 2,500 t/d. The campaign durations will vary but will be typically conducted over two-week intervals. During the period prior to the start of Q4 of Year 2, the additional process equipment items needed to allow for Red Mountain ore processing will be installed and commissioned.

Table 17-1: Major Process Design Criteria

Description	Unit	PGP Ore	RMP Ore (Q4 Year 2)
Ore throughput	t/a	456,250	456,250
Design Grade – Au	g/t	8.50	10.0
Design Grade – Ag	g/t	21.3	31.3
Crusher Availability	%	50	50
Process plant availability	%	92	92
Throughput, daily – average	t/d	2,500	2,500
Crushing plant capacity, hourly	t/h	208	208
ROM feed particle size, F_{80}	mm	600	600
Primary crusher	type	Jaw Crusher	Jaw Crusher
Coarse ore stockpile – live	tonne	2,400	2,400
Grinding circuit capacity	t/h	113	113
Circuit type		SAB	SAB (+fine grind)
Axb – design	-	30.6	32.2
Bond ball mill work index (BWi) – design	kWh/t	17.7	22.5
Abrasion index (Ai) – design	-	0.39	0.30
SAG mill power (drive)	kW	2000	2000
Ball mill power (drive)	kW	2000	2000
Grinding circuit feed particle size, F_{80}	mm	111	111
Product particle size, P_{80}	µm	80	90 (25)
Overall gravity gold recovery	%	50	N/A
Total leach time	h	35	35
Number of tanks	No.	1 pre-aeration + 7 leach / adsorption	1 pre-aeration + 7 leach / adsorption
Cyanide addition	kg/t	1.0	1.4
Lime addition	kg/t	0.4	1.8
Carbon concentration	g/t	25	25
Carbon loading (Au + Ag)	g/t	6,725	7,915
Elution method	-	Modified Zadra	Modified Zadra
Carbon batch size	t	8.0	8.0
Elution CIL strips per week	No.	7	10.5
Gravity strips per week	No.	7	7
Cyanide detox method	-	SO_2/Air	SO_2/Air

The design feed rate of 113 t/h to the SAG mill ensures that the average daily mill production will be 2,500 t/d after allowing for plant availability of 92%. To accommodate anticipated wide fluctuations in mill feed grades from these historically high-grade deposits, the design head grades are higher than LOM averages in order to be aligned with peak grades within the mine plan, ensuring a robust design. During operation such variations are managed by the precious metals loading levels on the activated carbon and the frequency of carbon stripping.

17.3 Process Plant Description

17.3.1 Crushing

The crushing facility will consist of a single stage of crushing that will process the ROM ore at a nominal processing rate of 208 t/h and it is expected that the crushing plant will be operated over a 12-hour shift to meet the 2,500 t/d production requirement. The major equipment and facilities at the ROM receiving and crushing areas include:

- Re-furbished static ROM bin grizzly
- Lined ROM surge bin
- Reciprocating plate feeder
- Re-furbished double-toggle primary jaw crusher
- Mobile rock breaker (FEL attachment)
- Covered coarse ore stockpile
- Two new stockpile-reclaim apron feeders
- A single re-furbished re-entry vibrating pan feeder.

Prior to the introduction of Red Mountain ore in Q4 Year 2, PGP ROM ore will be trucked and dumped directly into the existing ROM surge bin or stockpiled on the ROM storage pad, which can be reclaimed by the FEL for continuous feeding and blending. Particles larger than the bar openings of the stationary inclined ROM bin grizzly will fall to ground and be subsequently removed by the FEL prior to breakage using a mobile rock-breaker attachment. In Q4 Year 2 of operation, Red Mountain ore will be introduced either directly into the ROM bin or stockpiled adjacent to the facility on the ROM storage pad, where it can be reclaimed by the FEL and re-introduced to the process.

The ROM ore from the bin will be withdrawn by the reciprocating plate feeder discharging ore directly into the re-furbished double-toggle primary jaw crusher. A small proportion of fine ore bypasses the jaw crusher and combines with the crushed ore on the re-furbished product conveyor which delivers the crushed material to the covered coarse ore stockpile. This fully enclosed and galleried conveyor is fitted with a weightometer which will be used to monitor the crushing plant throughput and assist with operational accounting. Using the variable speed control of the reciprocating plate feeder, the crushing plant feed rate can also be regulated to maintain a choke fed crusher, while also preventing overfeeding.

The covered coarse ore storage facility consists of a conical crushed ore stockpile with 2,400 tonnes of live capacity.

Coarse ore from the stockpile will be reclaimed by means of two new reclaim apron feeders which discharge crushed ore onto the existing and re-furbished SAG mill feed conveyor. To bring flexibility and supplement normal operation, a re-furbished vibrating feeder is located over the SAG mill feed conveyor, which will allow for ore to be fed directly via a FEL into the SAG mill. This arrangement will allow for ore crushed prior to be stockpiled adjacent to the existing coarse ore stockpile and re-introduced into the milling circuit. The material from the SAG mill feed conveyor discharges directly into the SAG mill through the re-furbished retractable feed spout. The SAG mill feed conveyor is equipped with a weightometer to provide feed rate data for feed rate control to the grinding circuit.

17.3.2 Grinding Circuit

The grinding mills originally installed in the PGP mill building were purchased and removed by other parties in the early part of the 1990s, yet the existing pedestals and pump boxes remain and are in good condition. To suit this arrangement, the new grinding circuit will consist of a new SAG mill (in open circuit) and a new ball mill (SAB) in a closed circuit with new classifying cyclones. To benefit the project capital and construction schedule, the mill dimensions were derived from the original mill geometries and aimed to suit the existing concrete foundations, with one exception: a further 0.3 m of mill inside diameter was applied to the ball mill. The purpose of this minor adjustment was to allow the load between both mills to be managed when harder or softer ores are encountered during operation, particularly when the harder Red Mountain ores are introduced in Q4 Year 2 of plant production. In doing so, the slightly wider ball mill will also reduce the grinding duty requirements and operating costs of the stirred fine grinding mill.

Both mills will share a variable frequency drive (VFD) controller that can be used to start each mill. The VFD will also be used to adjust the SAG Mill speed to meet the optimum power draw and achieve the preferred grinding conditions for each of the different ores.

The SAG and ball milling (SAB) circuit is designed to deliver a product size P_{80} of 80 µm for PGP ores and between 80 µm and 90 µm for Red Mountain ores which are more competent. The nominal grinding circuit circulating load ratio for the ball milling circuit is 300%. When the Red Mountain ores are introduced in Q4 Year 2, the ore will be further ground in a fine grinding circuit, which consists of a high speed stirred fine grinding mill to reduce the product to a P_{80} of 25 µm.

For the initial period of processing PGP ores, the major equipment in the grinding circuit will include:

- One 6.7 m ft diameter by 2.4 m effective grinding length (EGL) single pinion SAG Mill driven by a single 2000 kW low speed synchronous motor
- One 4.4 m diameter by 6.1 m EGL single pinion ball mill driven by a single 2000 kW low-speed synchronous motor
- One cyclone cluster consisting of three 380 mm diameter cyclones (2 operating, 1 standby).

Following the introduction of the Red Mountain ore in Q4 Year 2, the following additional processing equipment will be installed and commissioned:

- A single-stirred fine-grinding mill in open circuit to reduce the P_{80} of Red Mountain ore to 25 µm, driven by a 3,000-kW variable speed drive
- A 27m diameter high rate pre-leach thickener.

Ore addition to the SAG and Ball mills is supplemented with process water to achieve a milling density of approximately 75% and 64% solids weight by weight (w/w) respectively. The SAG mill discharge will flow through a trommel screen and will discharge into the ball mill cyclone feed pump box where it will be diluted with additional process water and pumped by one of the two cyclone feed pumps up to the cyclone distribution manifold. Steel grinding media will be manually added to the SAG and ball mills using a kibble, which will be positioned by the re-furbished bridge crane servicing the mill building.

Initially for PGP ore feeds, the circuit will be equipped with a single new centrifugal gravity concentrator to recover gravity recoverable gold (GRG). For PGP ores, a proportion of the cyclone feed flowrate will be treated while during the Red Mountain ore campaigns, the gravity concentrator will be bypassed.

The overflow from the operating cyclones will flow via gravity to the re-furbished vibrating trash screen which removes trash or fibrous materials into a trash bin using water sprays. The trash screen underflow then flows to the pre-aeration tank feed pump box, which is pumped to the existing pre-aeration tank for the PGP ores. For the finer grinding duty of the Red Mountain ores, the trash screen underflow will instead report to the fine-grinding mill feed pump box and is pumped into the stirred fine-grinding mill. The fine grinding mill discharge will then be pumped to the pre-aeration feed pump box which is then pumped to the new pre-leach thickener, which will be installed in Q4 Year 2.

The ball mill cyclone underflow will flow via gravity into the ball mill feed chute. The ball mill discharge will flow into the ball mill cyclone feed-pump box, where it mixes with the incoming SAG mill discharge to complete the closed circuit.

Maintenance activities in the grinding and classification area will be serviced by the grinding area overhead crane. Spillages in the grinding and classification area will be pumped by the grinding area sump pump to the ball mill cyclone feed pump box.

17.3.3 Gravity Concentration and Intensive Leaching

As highlighted in Section 13, a significant proportion of the PGP ores contain GRG, thereby supporting the inclusion of a GRG gravity circuit. Whereas, in Year 2 when the Red Mountain ores are to be introduced through campaigning, the quantities are insufficient to warrant the use of a GRG circuit, warranting the bypassing strategy mentioned earlier in this section. The original mill facility at PGP contained an inefficient gold jigging circuit which processed a proportion of the cyclone underflow, prior to regrinding and separate cyanidation to the CIL circuit. During operation, it was reported that the grinding circuit encountered process control challenges with water flow balancing, with the jig consuming large quantities of process water, making it impractical to use.

However, since the de-commissioning of the existing circuit, and coupled with the advent of the more efficient centrifugal concentrator, a new gravity gold circuit will be effective in removing GRG. The grinding circuit will be equipped with a new centrifugal gravity concentrator to recover coarse gold from PGP ores. Approximately 30% of the cyclone feed flow will be pumped from the ball mill cyclone feed-pump box to the gravity circuit scalping screen. The scalping screen will remove any coarse particles, wood, or other light materials such as plastics that may have entered the grinding circuit and might cause damage to the centrifugal concentrator.

The centrifugal concentrator is a semi-continuous process and as such, feed is continuously fed for a specified period, then paused to allow the concentrate to be flushed from the concentrator bowl into the new downstream ILR. The combined gravity concentrator tailings and scalping screen oversize streams are pumped via a pumpbox back to the ball mill cyclone feed-pump box. The cycle times are designed to be 45 minutes each and approximately 20 kg to 25 kg of concentrate is expected from each cycle.

As concentrate is accumulated in the ILR feed hopper, excess water from the concentrator flush cycle will be decanted. Once per day, the accumulated concentrate will flow into the ILR where it is dosed with sodium hydroxide and sodium cyanide to leach the gold. Once the leaching process is complete, the pregnant leach solution is separated from the solids, aided by flocculant, and is then subsequently pumped to the gravity electrolyte tank adjacent to the gold room. ILR tailings are then pumped to the gravity-return pump box where it will mix with concentrator tailings and the scalping screen oversize to be pumped back to the grinding circuit.

Access to the gravity area will be restricted to authorized personnel only. Decant water and other spillages in the gravity area will be caught in the gravity area floor sump and pumped to the gravity return pump box. Maintenance for the gravity circuit will be provided by a monorail with additional capacity from hoists and mobile equipment.

17.3.4 Carbon-in-Leach

A pre-aeration tank is included ahead of the CIL circuit to oxidize any sulfides present in the ore, preventing the consumptions of cyanide and lime, thereby reducing overall reagent consumption. With the introduction of Red Mountain ores in Q4 Year 2, a pre-leach thickener will be installed and placed prior to the pre-aeration tank to remove the impact of additional water added in the fine grinding circuit. The pre-leach thickener underflow will be pumped to the pre-aeration tank during Red Mountain ore campaigns through one of the underflow pumps, while the overflow will report to a standpipe and will then pumped back to the mill solution tank for use as process water.

A CIL circuit already exists at the PGP site, and for cost efficiency is planned to be refurbished and re-used. By virtue of the existing agitator, the pre-aeration tank mixes the ground slurry with lime and fresh sodium cyanide while low pressure air is pumped through a sparger located at the base of the tank into the slurry to oxidize minerals which would otherwise consume oxygen during the leaching process. The passivated slurry overflows the pre-aeration tank into the first CIL tank where it is further adjusted with lime to meet the target pH of 10-11. Fresh sodium cyanide is also introduced at this point of the process.

Leached gold and silver are adsorbed onto the granular carbon which is present in all seven of the agitated CIL tanks. Fresh and barren carbon is added to the last CIL tank and travels up through the circuit in the opposite direction to the slurry flow (counter-current). Carbon is advanced once per day with carbon transfer pumps which pump loaded carbon to the next tank in the train. Carbon is retained in the tanks after the transfer with inter-stage flat-panel screens located in the overflow launder to allow slurry to pass through, but not carbon.

Leached tailings overflow the last tank and are pumped to the carbon safety screen which collects carbon that would otherwise be lost to tailings in the event of a hole in one of the inter-stage screens. Loaded carbon is pumped from the first leach tank to the elution circuit via a loaded carbon screen which separates the carbon from slurry and returns the slurry back to the CIL circuit.

17.3.5 Acid Wash, Desorption, and Refining (ADR) Circuits

The original gold recovery circuit at the PGP mill used a Merrill-Crowe sequence, downstream of the elution circuit, utilizing zinc powder to produce a gold sludge which was them smelted. For the re-commissioned circuits, the original acid wash tanks and the two existing elution columns will be re-furbished and re-utilized using new pumping systems and some associated piping. When the mine is restarted, the silver to gold ratio of the reserve is less than 5:1 on average and with the existing equipment already removed, the decision was made to move to a more standard electrowinning circuit, which saved capital, operating cost and removed the need for additional reagents for this particular ore body.

For the newly adapted circuit, loaded carbon from the CIL circuit is loaded into one of the existing acid wash tanks (of four existing, two will be removed and two will be refurbished and used) where it is submerged in a 3% hydrochloric acid solution to dissolve lime scale that would otherwise interfere with the elution process. After acid is recirculated for 30–60 minutes the acid is drained, and several bed volumes of fresh water are circulated through the column to rinse the carbon. Diluted acid from this process is pumped across to the cyanide detox tank, where it is neutralized by the addition of lime. After this process, the washed carbon is pumped to one of the two existing elution columns.

Gold will be recovered from loaded carbon by a modified Zadra style elution circuit. The anticipated batch size is 8 tonnes of carbon per cycle. For PGP ores, a single strip will be conducted once per day using only one elution column per day, whereas for the Red Mountain ores, the strip rate will be 1.5 times per day, utilizing both of the existing elution columns.

The strip cycle will involve making up a strip solution of cyanide and sodium hydroxide in the eluate tank and pre-heating that via an elution heater system which includes a direct-fired solution heater and primary recovery heat exchangers. Once heated, the solution will be circulated through the carbon bed in the elution column, and back to the eluate tank until it is ready for electrowinning to begin.

Once the pregnant eluate is sufficiently heated and loaded, it will begin transferring to the new electrowinning tank rather than back to the strip solution tank. The electrowinning feed pumps will transfer heated pregnant solution through the electrowinning cells which will then gravity flow back to the eluate tank to be recirculated through the elution and heater circuit. The pregnant solution may also be returned to the electrowinning circuit which can be separated from the elution circuit, to reduce final solution tenors in parallel with the final carbon washing steps of the elution process (prior to sending barren carbon for regeneration).

The barren carbon from the stripping circuit will be transferred to the kiln dewatering screen and the overflow carbon from the screen will be regenerated in the horizontal kiln. Carbon from the kiln will be quenched in water in the carbon quench tank before being pumped by the carbon sizing screen feed pump onto the carbon sizing screen which discharges into the reactivated carbon tank for use in the next adsorption cycle.

The gold room will contain all the sludge filtering, drying, and smelting equipment. The gravity electrowinning and elution electrowinning tanks will have immersion heaters to keep the solution hot, which will then be pumped in closed circuit with the electrowinning cells. Once the electrowinning cycle is complete, the barren solution can be re-routed to send the barren eluate to the electrowinning barren tank.

Gold sludge recovered from electrowinning will be washed inside the electrowinning cells and then flow via gravity to the sludge filter feed tank, which will batch filter the sludge via a plate and frame filter. Filtered gold sludge will be further dewatered in the drying oven and then finally smelted with flux to produce gold doré.

The gold room and electrowinning area will have security fencing and close-circuit television (CCTV) cameras and will only be accessed by authorized members of staff. Electrowinning cells will have a monorail system above for maintenance, while the gold room has a vehicle access door which will allow the equipment to be maintained by mobile vehicles.

17.3.6 Cyanide Detoxification Circuit

Tailings from the plant will be screened for carbon on the carbon safety screen and then pumped to the cyanide destruction Tanks No. 1 and No. 2 (duty and standby) to undergo cyanide detoxification prior to thickening and final deposition in the TSF. Cyanide destruction will be a duty stand-by style SO_2/Air system where sodium metabisulfite, lime (for pH control), copper sulfate, and air are injected into the slurry to react with and neutralize residual cyanide.

The residence time is targeted at 120 minutes (in each tank), and the slurry is then pumped via tailings transfer pumps into the tailings thickener (for PGP ores) prior to discharge through the tailings pipeline to the TSF. Return water is pumped from the dam back to a return water tank where it can be reused within the process plant.

17.4 Reagents, Consumables, and Utilities

17.4.1 Reagents

The reagents will be prepared and stored in separate, self-contained areas within the process plant and delivered by individual metering pumps or centrifugal pumps to the required addition points. Acidic and basic reagents are stored and mixed in physically separated areas to ensure no exposure of cyanide to acidic chemicals which would generate hydrogen cyanide gas.

Hydrated Lime

Hydrated lime is used in leaching and detoxification for pH control, and is expected to be delivered by truck with a pneumatic delivery system into the lime storage silo. Hydrated lime is continuously metered to the lime mixing tank via a rotary valve and screw feeder. Process or fresh water is added at the same time to form a lime slurry with a mixture strength of 20% weight by weight (w/w).

Lime slurry is distributed to the various dosing points using a ring main that provides constant flow to various destinations. Dosing is accomplished with drop lines off the ring main, with automated on–off valves that open when pH is low and close when the operator-specified target is reached.

Sodium Cyanide

Sodium cyanide is used in leaching as a lixiviant and in elution as a carbon-stripping aid. Cyanide is delivered to site in 1-tonne bulk bags contained within wooden boxes, and is stored in a separate area of the plant from the other reagents.

When the storage tank level is low, a cyanide mix batch is started by removing a cyanide bulk bag from its box and dropping it onto a bag breaker, which discharges cyanide into the mix tank. The mix tank has been previously filled with sufficient fresh water and buffered to a basic pH (using lime or NaOH). Once mixing is complete the 28% w/w cyanide solution is pumped to the holding tank by a sodium cyanide transfer pump.

Sodium cyanide is dosed from the storage tank to dosing points (each leach tank, the eluate tank and the ILR), via dedicated positive displacement metering pumps.

Sodium Hydroxide

Sodium hydroxide is delivered to site in 1-tonne bulk bags in pellet form. The bulk bags are broken and mix with water in the sodium hydroxide mixing tank. Once a batch is at the correct mix strength, it is transferred to the sodium hydroxide storage tank.

It is dosed to its various demands at full strength using dedicated positive displacement metering pumps. Sodium Hydroxide is used for pH control in cyanide mixing and the ILR. It is also used as an electrolyte in carbon elution and electrowinning.

Sodium Metabisulfite

Sodium metabisulfite (SMBS) is used in cyanide destruction as a source of SO_2 for the INCO™ process. It is expected to be delivered to site in 1-tonne bulk bags and is stored in the reagents storage area of the warehouse.

When the storage tank level is low, an SMBS mix is started by dropping a bulk bag of SMBS onto a bag breaker, which discharges SMBS into the mix tank. The mix tank has been previously filled with enough process water to produce a mixture strength of 30% weight by volume (w/v). Once mixing is complete, and there is enough room in the holding tank, the mixed SMBS solution is pumped to the SMBS storage tank by the transfer pump.

SMBS is dosed from the storage tank to the detoxification circuit via a dedicated positive displacement metering pump. A standby metering pump is installed to ensure that dosing can occur when required.

Copper Sulfate

Copper sulfate is used in cyanide destruction as a catalyst for the INCO™ process. It is expected to be delivered to site in 1-tonne bulk bags and is stored in the reagents storage area of the warehouse. Copper sulfate is mixed in the same mixing tank as SMBS, as these two reagents are both used in the same locations and have no interaction issues. This set-up reduces the required capital and allows the reagents to remain inside the main mill building, which is an operational and maintenance benefit.

When the storage tank level is low, copper sulfate is added to the mixing tank by dropping a bulk bag onto a bag breaker which discharges copper sulfate into the mix tank. The mix tank has been previously filled with enough process water to produce a mixture strength of 20% w/v. Once mixing is complete, the copper sulfate solution is transferred by gravity to the holding tank.

Copper sulfate is dosed from the storage tank to the detoxification circuit via duty–standby positive displacement metering pumps.

Hydrochloric Acid

Hydrochloric acid is used to remove lime scale from loaded carbon in the acid wash column of the elution circuit. It is delivered to site in 1 m^3 totes at about 32% solution strength and stored in the reagent storage area of the mill building.

Hydrochloric acid is mixed with fresh water in the dilute acid tank to 3% strength and pumped to the acid-wash tanks with a dedicated dilute-acid pump.

17.4.2 Utilities

Air Services

The plant and instrument air will be supplied from air blowers and compressors and will be filtered and dried before distribution to the plant.

Water Services

Fresh Water

Fresh water will be pumped from the Cascade Creek abstraction point to the fresh water tank. Fresh water in the tank is used to supply the following services:

- Fire water
- Gland seal water
- Potable water
- Reagent mixing
- Make-up water for the process water system.

Fresh water is supplied to the plant by two fresh water pumps in a duty-standby configuration.

Potable Water

Potable water will be sourced from the fresh water tank and treated in a treatment plant. The treated water is stored in the potable water storage tank for use by two potable-water pumps in a duty–standby configuration.

Gland Water

The process plant gland water is supplied from the gland-water tank which has water supplied from the fresh-water system and distributed to the plant by two gland-water pumps in a duty–standby configuration.

Process Water

The process water consists mainly of tailings pond reclaim water. The process water is stored in the process-water storage tank and distributed by the two process water pumps, in a duty–standby configuration.

18 INFRASTRUCTURE

This section discusses the Project infrastructure, including the surface mining infrastructure, on-site infrastructure, the tailing storage facility (TSF), water treatment plant (WTP), and water management systems and infrastructure. The existing mine has been in long-term care and maintenance for approximately 20 years, so the existing facilities need to be upgraded or replaced.

18.1 Mining Infrastructure

The PGP includes four underground mining areas: Premier, Big Missouri, Silver Coin, and Red Mountain—a satellite deposit approximately 40 km away that will come on-line in Year 2 of production. The Red Mountain mine ore will be hauled to the PGP mill site to be processed. The mining infrastructure includes:

- Power and energy supply
- Propane storage facility
- Heating and ventilation facilities
- Mine dewatering management
- Haul road construction or upgrade.

Figure 18-1 shows a typical heating and ventilation system for underground mines.

Source: INN, 2020

Figure 18-1: Typical Heating and Ventilation System for an Underground Portal

18.1.1 Premier Surface Portal

Groundwater from the portal will be directed to a collection/settling pond, then pumped through a pipeline incorporated into the upgraded road design to the new WTP.

Electrical power will be distributed from the new main substation to the Premier portal at 4.16 kilovolts (kV). Power will be delivered to the underground operations from an electrical house via a single 4.16 kV underground-tunnel power cable.

Primary mine propane-heated air and ventilation will be provided at surface. The main fans will be located in a building on surface and the main ramps will provide passage for the exhaust. Secondary ventilation with is incorporated into the mine design.

18.1.2 Big Missouri Surface Portal

Groundwater from BM and SC portals will be directed to a collection/settling pond, then pumped through a pipeline incorporated into the berm of the re-worked existing road to the WTP.

Electrical power will be distributed from the new main substation to the BM portal at 4.16 kV. The power line will follow the new upgraded road from the process plant to Big Missouri / Silver Coin. Power will be delivered to the underground operations from an electrical house via a single 4.16 kV underground-tunnel power cable. Electrical power to the SC deposit will be distributed underground.

Mine propane-heated air and ventilation will be provided at surface. The main fans will be located in a building on surface and the main ramps will provide passage for the exhaust.

18.1.3 Red Mountain Project Surface Portal

Groundwater from the RM portal will be directed to a collection/settling pond and then released into the environment.

A small power plant consisting of three diesel generators positioned close to the portal will deliver 4.16 kV to the underground operations via a single 4.16 kV underground-tunnel power cable.

Mine propane-heated air and ventilation will be provided at surface. The main fans will be located in a building on surface and the main ramps will provide passage for the exhaust.

18.1.4 Process Plant Facilities

The process plant consists of ore stockpiling, crushing, conveying, grinding, gravity concentration, leaching, cyanide detoxification, and reagents. Refer to Section 17 for details of the process facilities. The following is a general description of the existing ancillary facilities at the process plant site which needs upgrading or replacing. All the existing facilities require the removal of debris from buildings (such as walls with mould, corroded piping, general garbage and debris, chemical spills, etc.). The infrastructure planned at the process plant to support the mining and processing operations includes:

- Upgraded site and access roads
- Upgrading/replacing the administration facilities, mine dry, truck shop, and maintenance facilities

- Upgrading/replacing elements of the assay laboratory/cold storage building
- Waste water treatment systems
- Solid waste disposal facilities
- Tailings storage facility
- Water management
- Water treatment plant
- Temporary construction camp
- Power supply and distribution system
- Site services
- Fuel
- Propane
- First aid station
- Water supply
- Communication system.

Minor ancillary infrastructure facilities currently in use for the care and maintenance period, such as the bunkhouse, trailers, generators, fuel tanks, trailers, and minor shops are included in the infrastructure on site but require no upgrade or replacement.

The current WTP will be replaced with a new WTP as described in Section 18.6.

Figure 18-2 shows the mill process site infrastructure of the PGP.

18.1.5 Truck Shop and Warehouse

The warehouse/truck shop is located in the ground floor of the existing process building designed for truck maintenance and repair, and storage. The building layout is shown in Figure 18-2.

The warehouse/truck shop includes indoor truck bays, a waste oil system, an exhaust system, lube-oil systems, water systems, coolant systems, a machine shop and equipment, a welding bay, and a tire-change area. The building includes offices for maintenance and warehouse personnel.

The mill building roof drainage will be repaired and heat-traced to prevent further ingress of water into the building.

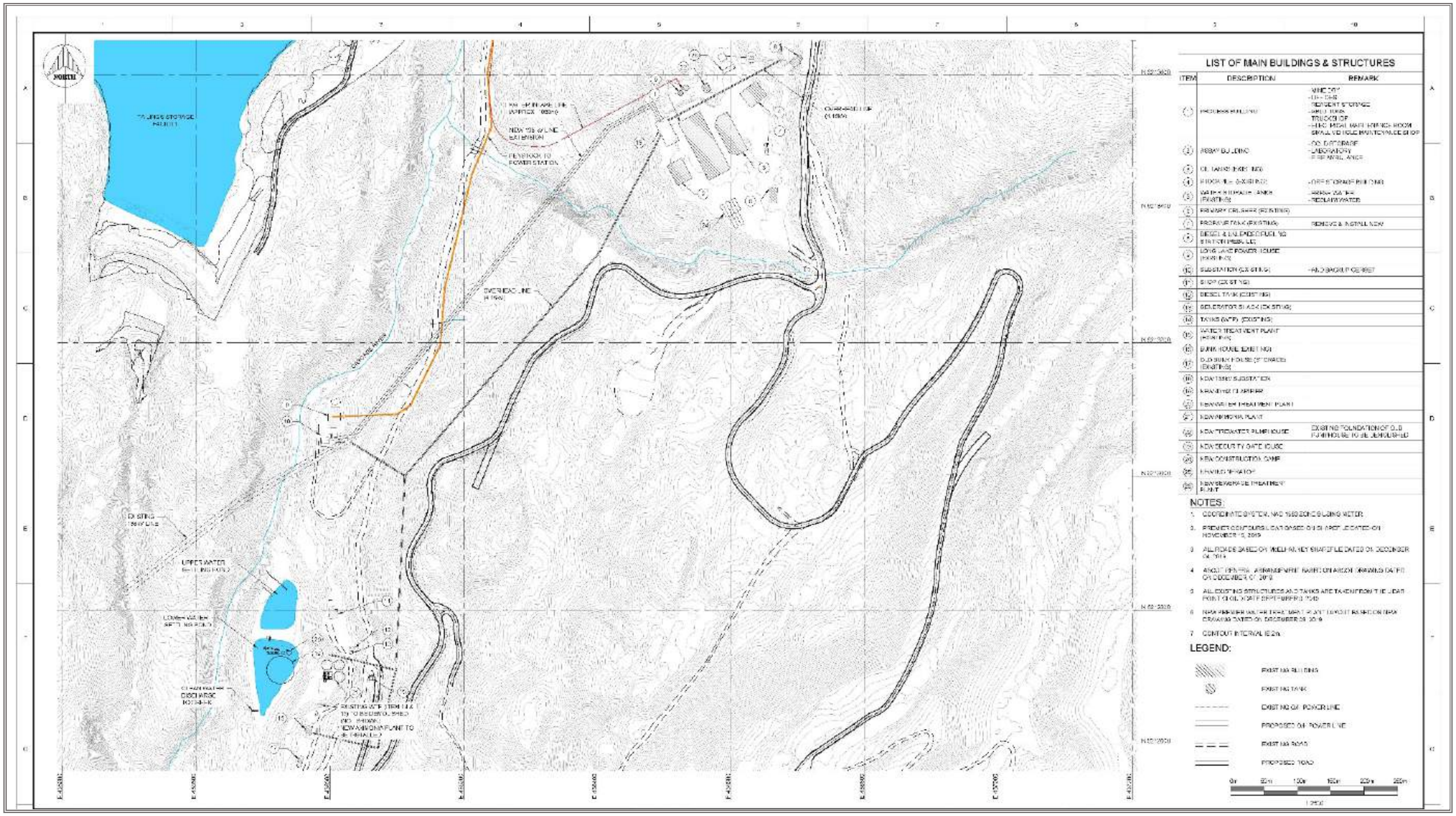


Figure 18-2: Site Layout—Main PGP Facilities

18.1.6 Administration and Mine Dry

The administration offices are located on the first and second floors of the process building. Approximately 35% of the administration offices are considered to be in good condition, with minor repairs required to replace broken interior windows and ceiling tiles. However, approximately 65% of the office area has sustained water damage from leaking roof drains warranting replacement of the roof drains and office refinishing.

The administration building includes working space and offices for engineering, technical, surveying, and administration personnel. The offices of the general manager, mine manager, mill superintendent, mine operations superintendent, maintenance superintendent, and mine supervisors will be located in this building. A first aid safety area, control station, kitchen, and lunchroom facilities are located in the building.

The mine dry area will include washing areas and men's/women's locker rooms.

Mill building inspection of the heating, ventilation, and air conditioning (HVAC) equipment was limited to ducting and superficial inspections of the propane heater units. The ducting was in fair condition, with some sections affected by rust that could not be confirmed as superficial. Further to a site inspection, the HVAC units were assessed and will be replaced by new units.

Additional site grading is needed around the building and the other existing ancillary facilities to provide a positive drainage and ensure a dry surrounding grade for plant operations.

The processing plant roof will be cleaned of vegetation and superficial moss during the upgrade.

The heat-trace power feed to the roof junction box appears to be in good condition; however, the cabling, controller, and terminals from the junction box into the field need to be replaced due to environmental damage.

18.1.7 Cold Storage / Assay Laboratory

The assay laboratory is located adjacent to the main mill building and is divided into two main sections, a core/storage/workshop area and the laboratory and offices area. The laboratory will be upgraded and new equipment installed. The building itself is an A-frame building made of steel with cladding and several vents located along the roof for various exhaust functions.

The ducting and vents in the laboratory are intact. However, the HVAC system will be upgraded, including the fans, and overall ventilation requirements with any new fit-out.

The plumbing/ablutions will be replaced with any new fit-out, as they have been damaged during either the decommissioning process or over time by the elements. The building structure is in good condition, with some repair required to the roof membrane, as well as installation of new overhead door and windows.

The building interior requires a full refurbishment as most of the surfaces have been damaged with mildew and exposure to the elements. The concrete slab floor is in a good condition.

The assay laboratory will be re-equipped to perform analyses of mine and process samples.

The propane pumping enclosure and the associated equipment is in poor condition and will be replaced due to weather-related damage and the removal of essential equipment.

18.1.8 Propane Facility

The propane tank and vaporizer will be upgraded or replaced to provide heat to the main mill building (including the enclosed ancillary facilities) and the Cold Storage/assay laboratory building.

The foundations for the propane storage tank are composed of a standard twin-saddle concrete structure and are in good condition. No repair or upgrade is required.

18.1.9 Bunkhouse

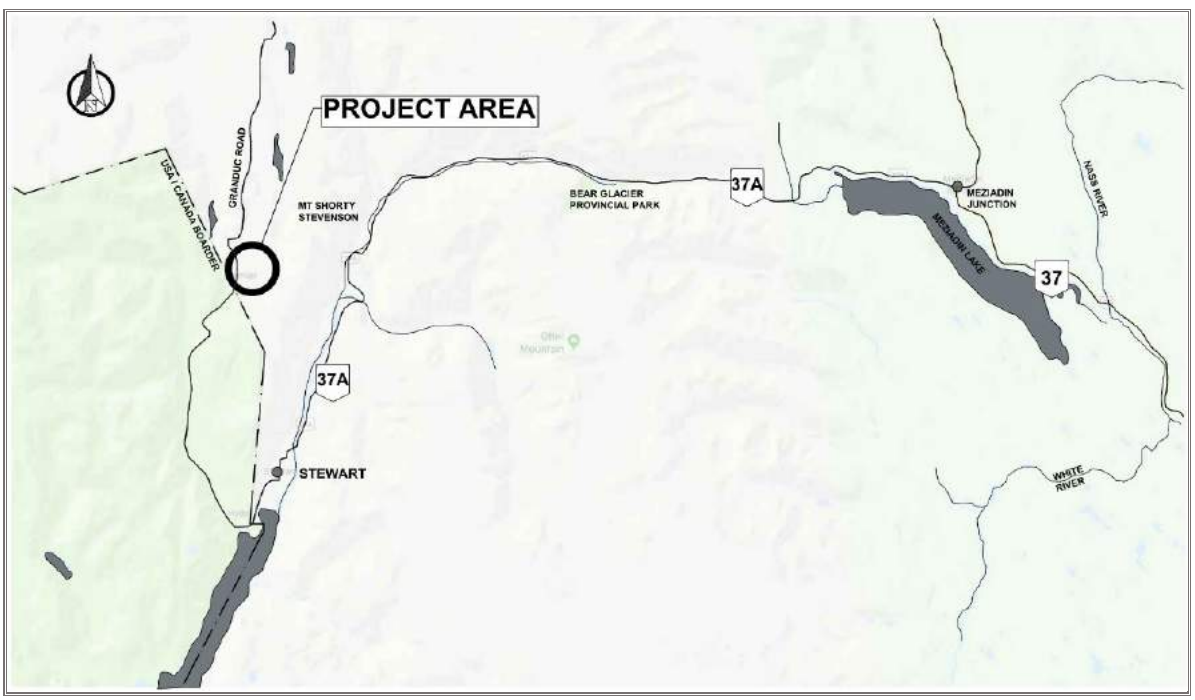
The bunkhouse is located near the entrance to the mill complex and accommodates ten people. No upgrade is required for this building.

18.1.10 Explosive Storage

Explosives storage facilities are described in Section 16 Mining Methods.

18.2 Premier Gold Project Access Roads

Currently, PGP is accessed via Highway 37. Minimal helicopter support is required because the project still has access roads in place from the past when it operated as the Premier Mine. Access to the project site is from the Port of Stewart via the Granduc Road/Highway 37A or Continental North America via Granduc Road, 37A, Highway 37, and points beyond (Figure 18-3).



Source: McElhanney, 2020 – Not to scale

Figure 18-3: Access Roads to the Premier Gold Project

18.2.1 Premier Gold Project Site Access Roads

Highway 37A

Highway 37A runs between the Canada/US Border at Hyder Border Crossing, to Highway 37 at Meziadin Junction. Highway 37A is a two-lane, paved provincial highway that is part of the Provincial Highway network.

Salmon River Road (Granduc Road US Segment)

Salmon River Road is a 17 km, two-lane, public roadway that runs between International Street (in Hyder, AK) north to the Canada/US Border. The roadway between International Street and Fish Creek Bridge was paved in 2012. A bridge spanning 12.5 m (41 ft) is located along the road (Fish Creek Bridge, Alaska DoT Structure ID 1217). The bridge was constructed in 1965 and rehabilitated in 2012. It is a laminated timber stringer bridge with a reinforced concrete deck. Due to the jurisdiction of this roadway, McElhanney did not perform any assessments of this road segment.

Granduc Road (Canadian Segment)

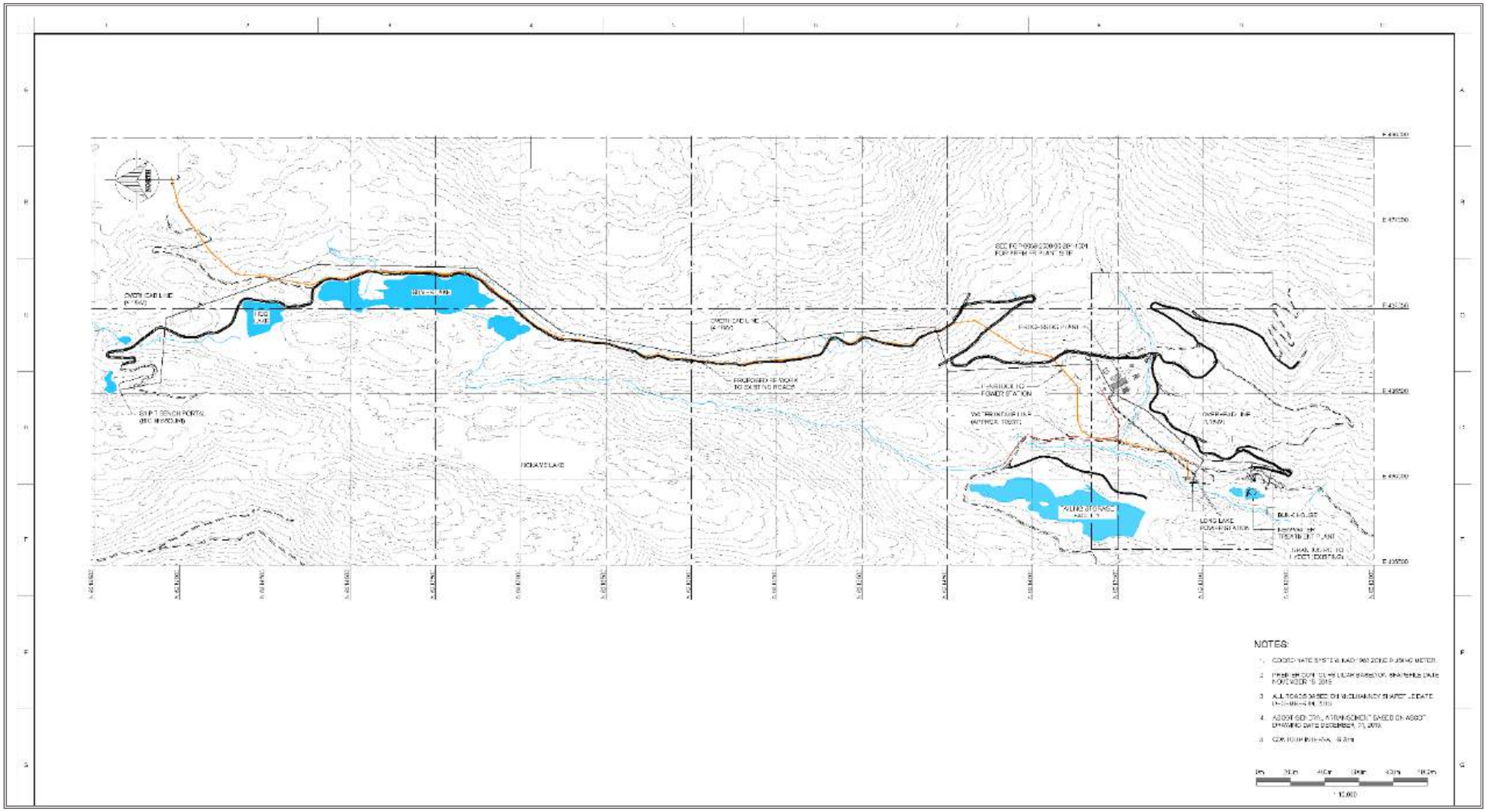
The Granduc Road is a two-way, public roadway that provides access to the PGP site. The roadway has a gravel surface ranging from 7.0 to 10.0 m wide and is predominantly oriented in a north–south direction. The roadway extends north of the PGP site up to the Granduc Mine, while to the south, the roadway travels southwest to the Canada/US Border. An approximately 3.0 m diameter multi-plate arch is located 250 m north of the Canada/US Border.

Premier Gold Project Site Roads

The PGP will have a total of four main site roads, Big Missouri (15.1 km), Granduc Road, Tailing Pond access road (1.0 km), and Silver Coin road (1.0 km). The total length of roads on the PGP site total 17.1 km of haul road, not including the Granduc Road, which is considered a public road and must maintain access to the general public. There is one bridge crossing and one major culvert with a diameter of 3,500 mm. Table 18-1 provides a summary of the haul road characteristics. Figure 18-4 shows the proposed site roads.

Table 18-1: PGP Site Roads Summary

Road	Section	Length (km)	Width	Radio Assisted
Big Missouri Haul Road	Granduc Road to Premier Pit Access Road	2.6	10.5 m (two lanes)	Yes
Big Missouri Haul Road	Premier Pit Access Road to Station 8+400	8.4	10.5 m (two lanes)	Yes
Big Missouri Haul Road	Station 8+400 to North Limit (Station 12+500)	4.1	7.0 m (one lane)	Yes
Tailings Pond Access Road	Granduc Road to Tailings Pond	1.0	7.0 m (one lane)	Yes
Silver Coin Road	Silver Coin to Granduc Road	1.0	Existing	Yes



Source: Sacré-Davey, 2020

Figure 18-4: Site Roads

Big Missouri Haul Road

Big Missouri consists of 17 km of haul road from the existing Granduc Road to the access portal of Big Missouri Mine. The road shall consist of a gravel structure. Several corrugated steel pipe culverts will be installed throughout the road alignment to better manage existing and proposed drainage patterns. The road running parallel to the penstock pipeline shall be equipped with a gabion basket (or equivalent) retaining wall which will have to be constructed before the haul road upgrades. The existing bridge located at KM 12.1 was inspected and found to have suitable capacity to facilitate proposed haul traffic. A design variance has been requested however, as the bridge does not meet the haul road design standard for road width. It is anticipated that special operating procedures will be developed for this crossing.

Granduc Road (Through Project Site)

Given that Granduc Road is an existing public roadway, no changes to the road design or speed limit are proposed. The design of the road was reviewed based on Ministry of Transport and Infrastructure (MoTI)'s *BC Supplement to TAC Geometric Design Guide* (2017) and the *Forest Road Engineering Guidebook* (2002).

Although there are no posted speed limits within the mine site, it is suggested that the speed limit on Granduc Road is 50 km/h based on other similar roadways.

In general, the existing road design is compatible with a 50 km/h design speed. There are two horizontal curves (850 m and 1,400 m north of Big Missouri Haul Road) that have a design speed of 40 km/h. It is recommended that these curves be signed with curve warning signs to inform motorists of the need to slow down.

It has been verified that the section of Granduc Road between the TSF Access Road and Big Missouri Haul Road is currently adequate to accommodate a WB-20 semi-trailer, which will be used for the transport of mine site loads and freight deliveries to and from the site.

Tailings Pond Road

The tailings pond road will provide access to the newly designed tailings pond via Granduc Road. Traffic volumes on this road are anticipated to be minimal and road grades will be low; consequently, only minor improvements are required.

Silver Coin Road

The Silver Coin Road is an existing road in fairly good condition and is expected to need minimal to no upgrades other than some surface gravels to smoothen out the driving surface.

18.2.2 Road Design Requirements

The haul road and other infrastructure was designed for the life span of the mine, approximately 20 years. Minimal maintenance is required to maximize the durability of the road and prevent major interruptions in the mining operation. Where possible, two-lane operations were maintained through the haul road; however, in cases where extensive rock blasting would be required, radio-assisted single lane roads will be constructed.

18.2.3 Design Criteria

Where designated on the engineering plans, appropriate concrete reinforced barriers (CRB) are to be installed along the road edge as a safety measure to reduce the likelihood of vehicles driving down embankments.

A safety berm made up of on-site materials shall run along the entire stretch of the Big Missouri Road on the down-mountain side. The location and height of berms is as per Section 6.9 of the Health, Safety and Reclamation Code for Mines (Gov.bc.ca, 2020).

Signage and reflective posts will be installed after completion of the road to indicate speed limits, steep hills, sharp curves, and others. Where the road exceeds 5% vertical grade, runaway lanes have been proposed as per Section 6.9 of the Health, Safety and Reclamation Code for Mines (Gov.bc.ca, 2020).

18.2.4 Access Road Design

Preliminary Access Engineering

A preliminary concept design was completed in the summer of 2018 by McElhanney and a technical memo was submitted to Ascot. This technical memo along with the conceptual drawings was referenced during the design of the haul roads.

Current State of Design

As of this writing, the design has been progressed to an “Issued for Tender” or 100% complete stage. No further design is required until construction commences at which point an “Issued for Construction” design package will need to be prepared. A review of the existing bridge on the Big Missouri Haul road reveals that it does not meet current haul road width standards. A request for a design exemption has been prepared for future submission to the Ministry of Energy and Mines.

Design Considerations

In order to construct the haul road with minimal effort, the design approach was taken to widen the existing road into the mountain side. This resulted in an excess of material, specifically an increase in rock blasting. Where the haul road runs parallel to the penstock pipe, it was widened away from the pipe to avoid disturbance to the penstock. This resulted in the installation of retaining walls along the haul road where the mountain is too steep to construct proper side slopes. Upon the completion of a field inspection, it was determined the existing culvert by the Mill access had sustained significant structural damage and would not sustain the loading from the loaded haul trucks. A new culvert crossing was designed at this location to replace the existing one.

While the centreline of the road and penstock surveys were provided for this design, a road edge or shoulder has not been surveyed. Our design team derived the existing road width and edges using most-current LiDAR. We estimate the accuracy of this technique to be in the range of ±0.5 m.

After a thorough and detailed design, it has been determined that one-way haul road sections would be required in certain areas to minimize the capital expenditures due to extensive rock cuts required. Where one-way traffic is situated, pull-outs were designed approximately 250 m apart to allow haul trucks to pass one another and minimize the impact to hauling schedule.

The excess rock blasting required to construct the haul road can be used as material for the haul road structure, providing it is crushed to the appropriate size and areas with potential for acid rock drainage (ARD) are dealt with accordingly.

18.2.5 Red Mountain Portal Access Road

The following section has been extracted from the JDS Feasibility Report, Report Date: August 10, 2017. One modification has been made to remove the certainty of a power transmission line to reflect the allowance to include a powerline adjacent to the road structure in the even Ascot elects to replace the currently favoured genset power generation to line power supplied by BC Hydro. This has no material effect on the assumptions, design or cost of the road.

Site Access Road—KM 0 to KM 14

The first segment of the Red Mountain access road will be accessed via a 14.5 km all-season access road that follows the Bitter Creek Valley. The access road follows a pre-existing resource road through the valley bottom for 12.7 km from Highway 37A along the north/northeast side of Bitter Creek to the proposed mill site.

The proposed road is designed for a B-Train truck and trailer combination. It will be a gated single lane road with pullouts supporting two-way, radio-controlled traffic travelling at a maximum speed of 50 km/h. In sections of the alignment where a 50 km/h design speed is unfeasible due to topography, excessive earthworks, and increased cost, the design speed will be reduced to 30 km/h. Speed limits will be imposed for site traffic; dusting is not expected to be problematic.

The design will include regular drainage culverts and road signs. Major stream crossings will have engineered design plans for construction and include clear span bridges and modified fords.

The access road overall right-of-way (ROW) is typically 25 m. In sections that encroach on Bitter Creek, the ROW is 10 m towards the creek and 15 m on the high side for a total of 25 m. The additional 5 m on the high side were designed to accommodate potential power lines running to the mine site. Site specific conditions may necessitate a wider ROW where cut and fill slopes extend beyond the typical ROW. In these locations, the ROW will increase 3 m beyond the typical toe of the fill or crest of the cut.

The specifications to meet these design speeds are summarized in Table 18-2 and have been obtained from the 2012 Standard Specifications for Highway Construction (BC Ministry of Transportation and Infrastructure, 2011), BC Supplement to TAC Geometric Design Guide BC Ministry of Transportation and Infrastructure, 2007), Steep Grade Descent Calculator (Parker, 2016), MFLNRO Engineering Manual (BC Ministry of Forests, Land Resource Operations, 2016), and the Forest Road Engineering Guidebook (BC Ministry of Forests, 2002).

Construction of the road prism will require surficial material earthworks and ripping of rock where the road passes through near surface bedrock. The road subgrade will be tracked in lifts and the subgrade will then be surfaced with 15 cm to 30 cm (as specified in the design) of designated surfacing material. All material to be used for surfacing shall be at the discretion of the on-site engineer. Following placement and grading, the surfacing layer will be track packed.

Table 18-2: Access Road Design Specifications

Components	50 km/h	30 km/h
Maximum Road Grade	12%	18%
Tightest Vertical Curve	1% grade change over 12 m (11 m for crest curves)	1% grade change over 4 m (3 m for crest curves)
Minimum Curve Length	50 m	30 m
Minimum Stopping Sight Distance	135 m	65 m
Minimum Horizontal Curve Radius	80 m	35 m (16 m for switchbacks)
Minimum Cross Drain Culvert Diameter	600 mm	600 mm
Ditch Size	0.6 m deep with a 0.6 m wide base	0.6 m deep with a 0.6 m wide base
Road Width	5 m	5 m
Pullout Size	Additional 4 m width, 30 m long with a 7.5 m long taper at each end	Additional 4 m width, 30 m long with a 7.5 m long taper at each end

Source: Onsite Engineering Ltd. (Onsite), 2017

Site Access Road—KM 14 to KM 22

From mile marker KM 14, there is 11.7 km of private haul road to be newly constructed from the mill site to the upper portal. The haul road follows the valley for a short distance before making numerous switch backs over steep terrain towards the portals.

The proposed haul road is designed as a private, gated, single lane road with pullouts supporting two-way, radio-controlled traffic travelling 30 km/h. The road was designed using a Western Star 4900SB tandem truck paired with an SX3-Tri axle side dump trailer. The haul road will have a berm ¾ the height of the largest size haul truck tire to be used. The haul road is designed with a 25 m ROW in sections of the alignment where a potential power line could run parallel to the road and could be reduced to 20 m where the power lines deviate from the road alignment. Site specific conditions may necessitate a wider ROW where cut and fill slopes extend beyond the typical ROW. In these locations, the ROW will increase 3 m beyond the typical toe of the fill or crest of the cut.

The specifications to meet the design speeds are summarized in Table 18-3 and were obtained from the 2012 Standard Specifications for Highway Construction (BC Ministry of Transportation and Infrastructure, 2011), BC Supplement to TAC Geometric Design Guide (BC Ministry of Transportation and Infrastructure, 2007), Steep Grade Descent Calculator (Parker, 2016), BCMFLNRO Engineering Manual (BC Ministry of Forests, Lands Resource Operations, 2016), Forest Road Engineering Guidebook (BC ministry of Forests, 2002), and the Guidelines for Mine Haul Road Design (Tannant & Regensburg, 2001).

Table 18-3: Haul Road Design Specifications

Components	30 km/h
Maximum Road Grade	18%
Tightest Vertical Curve	1% grade change over 4 m (3 m for crest curves)
Minimum Curve Length	30 m
Minimum Stopping Sight Distance	65 m
Minimum Horizontal Curve Radius	35 m (16 m for switchbacks)

Components	30 km/h
Minimum Cross Drain Culvert Diameter	600 mm
Ditch Size	0.6 m deep with a 0.6 m wide base
Road Width	5 m
Pullout Size	Additional 4 m width, 30 m long with a 7.5 m long taper at each end
Shoulder Barrier (Haul Road Only)	0.9 m tall, with a top width of 0.5 m and fill slopes of 100%

Source: Onsite, 2017

18.2.6 Steep Grade Considerations

There are sections of the haul road between mile marker KM 14 and the lower portal identified as requiring significant grades for short sections. These grades could be up to 18%. In these sections, truck drivers must:

- Not exceed speeds of 15 km/h when descending grades between 5% and 17%
- Not exceed speeds of 10 km/h on grades of 18%
- Limit payload to 36 tonnes
- Make a mandatory stop at a designated location to cool brakes
- Ensure the road surface provides adequate traction in adverse weather.

In all sections with switchbacks, trucks must only travel at a maximum speed of 10 km/h. No hauling is to occur during periods of snow or ice cover on the road surface.

18.2.7 Avalanche Control

The combination of rugged glaciated topography, latitude, and coastal weather systems creates severe winter conditions in the Red Mountain Area. The region receives some of the heaviest snowfall in North America, with settled seasonal snowpack depths ranging from 3 m to 6 m. The alpine portion of the Project area (i.e., the lower to upper portal area) is an especially severe microclimate, similar to that at the Brucejack Mine. This microclimate is characterized by localized recurrent and highly variable strong winds. This compounds the effects of regional weather systems, often contrary to prevailing winds. Local drifting snow creates highly variable snow deposition, often to depths much greater or less than the regional average.

For the purposes of this study, a system of preventative closures and active avalanche control with explosives deployed by helicopter has been assumed to be the primary risk mitigation measures for routine operations. Helicopters are to be considered as the primary means of accessing active avalanche control target locations, but overland snowcat access has been included as a secondary means of control. Severe winds and other weather conditions in the Project alpine microclimate may contribute to a significant number of additional Closure Potential Days if the avalanche program is limited to helicopters. Helicopters cannot be used to access avalanche control targets at night or in adverse weather conditions (such as storm activity that drives cycles of avalanche activity). Snowcat access enables avalanche control missions to be carried out at night and during many weather conditions that would otherwise ground a helicopter. If the only means of active avalanche control is limited to helicopter use for accessing target locations there will likely be a significant number of days per winter when wind conditions prevent active hazard reduction. This in turn dictates prolonged closure periods until the weather clears for flying or the situation stabilizes naturally.

A high-level investigation of the current weather data indicates that the Project area will experience approximately 53 days with avalanche activity having Closure Potential between November and April on an average snow avalanche year. A combined helicopter and snowcat program could result in an average closure time on the order of 4 to 8 hours (average 6 hours) for a given closure day. Based on an average of 53 days of closure potential and an average of 6 hours of closure time this study assumes 15 days per year of lost mine production due to avalanche control and risk mitigation.

Avalanche control and assessments will be conducted by three full time avalanche technicians on the Project site during the winter season. The active winter avalanche season is assumed to be the months of November to April inclusive in the Project region.

18.3 Bridges

It was concluded that the existing bridge located on the north portion of Big Missouri road is in excellent condition and, after load calculations, that it is more than capable of handling the proposed haul truck loads. However, the width of the bridge does not conform to the haul road width standards and therefore McElhanney has written a letter to the mining authority requesting a design exemption. The existing culvert beneath the Granduc Road that passes Cascade Creek will be replaced with a single span 40 m to 45 m long bridge as part of the expansion of the Cascade Creek Diversion Channel (CCDC).

18.4 Waste Storage Facilities

Waste will predominantly be disposed of on site to minimize or eliminate the requirement for off-site waste removal services.

18.4.1 Water and Sewage Systems

Wastewater or sewage generated on site will be treated at the sewage treatment plant. Treated effluent generated at the sewage treatment plant will be compliant with local, provincial, and national regulations.

All potable water that is generated and consumed on site for domestic use is expected to report to the sewage treatment plant for treatment prior to discharging to the environment. Therefore, it is assumed that the volume of sewage generated is equivalent in volume to the amount of potable water produced on site for domestic use.

The total volumetric input to the sewage treatment plant thus comprises all potable water produced on site for domestic use, plus a 15% contingency. Treated sewage effluent will be discharged to the environment, while sludge generated at the sewage treatment plant will be dewatered and incinerated on site.

18.4.2 Waste Disposal Facilities

Waste management for the PGP is focused on the principles of waste reduction, reuse, and recycling, where feasible. Wastes that cannot be eliminated, reused, or recycled will be either landfilled or incinerated at the PGP site, or shipped off site for final disposal. Currently, there are no waste disposal facilities on the site, but waste processing facilities are situated in relatively close proximity to the site, for the handling, storage, treatment, transport, and disposal of wastes generated from the PGP. Several treatment/disposal facilities are proposed to ensure that the waste is managed in a safe, efficient, and environmentally friendly manner.

An incinerator for the disposal of combustible, non-hazardous, and waste oil will be located in the waste management facility located on the southeast corner of the main plant site.

All hazardous waste streams will be shipped off site for final disposal at an approved facility. Wherever economically feasible, recyclable materials will be collected and shipped to a recycling facility offsite.

18.5 Tailings Storage Facility

The principal objectives for the TSF are to provide safe and secure storage of tailings to protect regional groundwater and surface water during operations and in the long term (post-closure), and to achieve effective reclamation at mine closure. The design of the TSF has taken into account the following requirements:

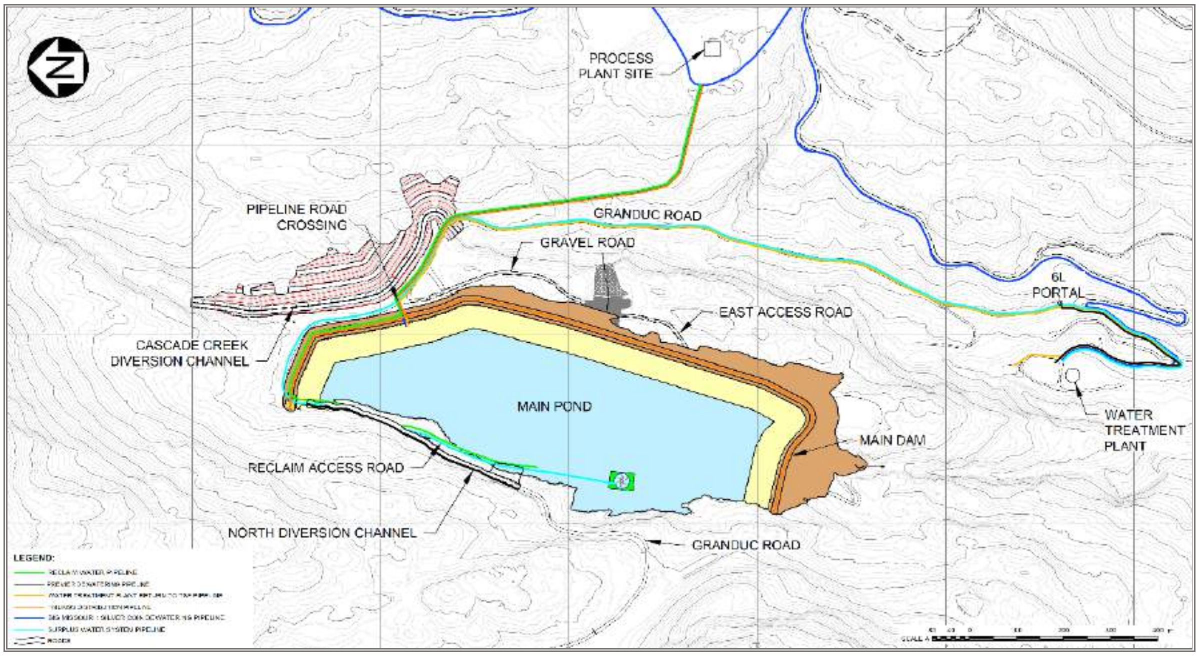
- Permanent, secure, and total confinement of all tailings materials within an engineered disposal facility
- Diversion of non-contact water around the TSF to the maximum extent possible
- Control, collection, and removal of free water from the TSF during operations for recycling as process water to the maximum practical extent
- The inclusion of monitoring features for all aspects of the facility to confirm performance goals are achieved and design criteria and assumptions are met
- Staged development of the facility over the life of the PGP.

The TSF was designed to permanently store tailings generated during the operation of the mine. This will be accomplished by constructing staged embankment raises on the existing TSF, which has been in long-term care and maintenance for over 20 years. The TSF comprises a basin constrained by a rockfill embankment on three sides, and natural topography to the west. The design of the TSF foundations relied on historical site investigation programs from pre-construction and construction periods of the Project, and as-built reports and information. The embankment will be expanded during operations using the centreline method of construction. The final (Stage VI) layout for the TSF is shown on Figure 18-5.

Tailings will be delivered to the TSF in a single stream, in a single overland pipeline. Tailings will be discharged from the embankment crest via spigots spaced along the length of the embankment. Supernatant water will be reclaimed to the Plant Site for use in processing of ore via a floating pump barge and overland pipeline. The supernatant pond volume will be managed by removing surplus water to the WTP for treatment and subsequent discharge to the environment. The surplus water system pumps will be housed on the same barge as the reclaim water system pumps.

The tailings are characterized as potentially acid generating (PAG). Mitigation to prevent oxidation of the tailings includes: continuous deposition of fresh layers of tailings over the above-water beaches adjacent to each dam section, maintaining a supernatant pond over a portion of the tailings during the operating life, and constructing a cover at closure once the operational supernatant pond has been removed.

An alternatives assessment, completed in 2019, assessed a number of tailings disposal locations and technologies and concluded that tailings storage in the existing PGP TSF was the preferred alternative for tailings management for the Project (KCB, 2019).



Source: Knight Piésold, 2020a

Figure 18-5: TSF General Arrangement (Stage VI – Year 8.5)

18.5.1 Design Basis and Operating Criteria

The basic design criteria for the TSF are summarized in Table 18-2. The design nominal throughput for the mill is approximately 2,500 t/d.

Table 18-4: TSF Design Criteria Summary

Parameter	Unit	Value
Nominal Mill Throughput	t/d	2,500
Design Operating Life	years	8.5
Total Tonnes Tailings Produced	Mt	6.5
Tonnes Tailings PGP	Mt	4.0
Tonnes Tailings Red Mountain Project (RMP)	Mt	2.5
PGP Tailings Nominal Solids Content (by weight)	%	60
PGP Tailings Specific Gravity	-	2.72
RMP Tailings Nominal Solids Content (by weight)	%	50
RMP Tailings Specific Gravity	-	3.06
Embankment Crest Width	m	10
Embankment Downstream Slope	-	2H:1V
Minimum Required Static Factor of Safety (Stability)	-	1.5
Inflow Design Flood	m^3/s	68
Embankment Spillway Depth	m	2.5
Embankment Spillway Base Width	m	15

Source: Knight Piésold, 2020a, 2020b; CDA, 2019; EMPR, 2016

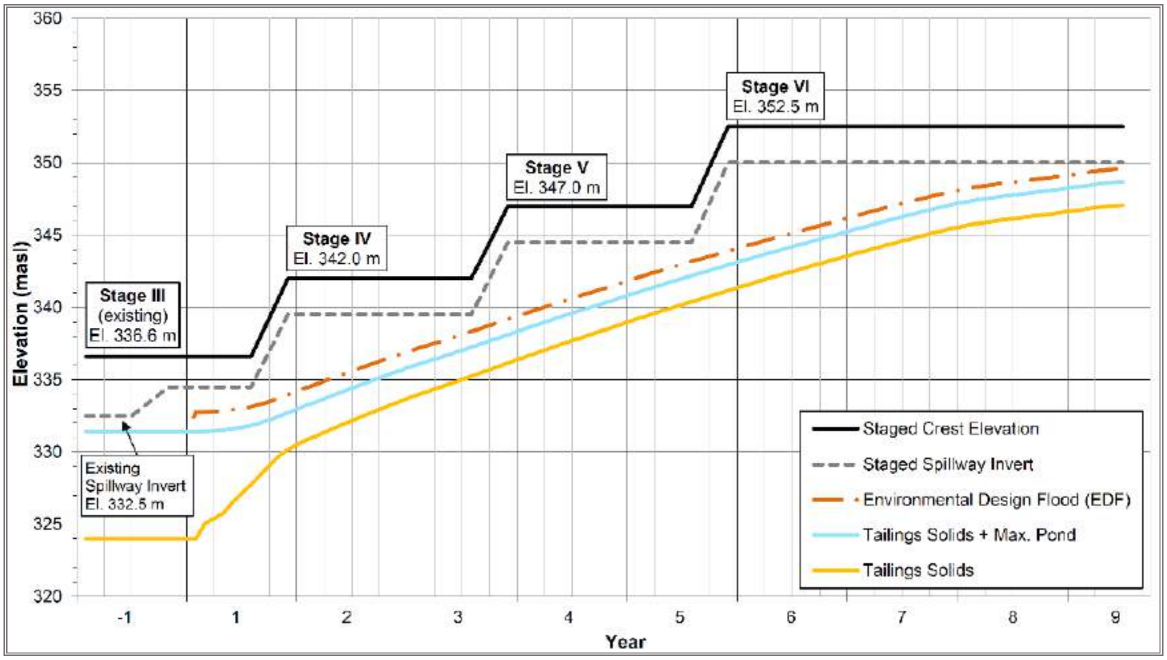
18.5.2 Tailings Storage Facility Features

The TSF feasibility design is based on a mine life of 8.5 years and a total of 6.5 Mt of tailings. The TSF will have a storage capacity of 5.5 Mm^3 above the historical in-situ tailings to contain the design volume of tailings and water. The 5.5 Mm^3 includes 4.6 Mm^3 of tailings (6.5 Mt at an average dry density of 1.4 t/m^3 [estimated based on a combination of laboratory testwork results and historical data]), 500,000 m^3 of process water (equivalent to three months of total process water plus the maximum monthly runoff from the contributing catchment area), and 280,000 m^3 for the environmental design flood (EDF) aligned with CDA Guidelines (CDA, 2013, 2014, and 2016). The EDF was selected as the total runoff from a 1 in 1,000 year, 24-hour precipitation event that reports to the TSF. Flood events exceeding these volumes will be passed through an emergency spillway and report either to Lesley Creek (Stage III to Stage V) or Cascade Creek (Stage VI) (see Figure 18-6 for TSF staging).

The TSF expansion includes:

- A TSF embankment constructed using rockfill from local quarry
- Upslope surface water diversion channels
- A tailings distribution system
- A reclaim water system (Figure 18-5 and Figure 18-10)
- A surplus water system
- Tailings beaches
- A supernatant water pond
- An emergency discharge spillway.

The existing TSF (Stage III) will be prepared for deposition by raising the existing spillway invert by 2 masl to 334.5 masl, and by upgrading the existing embankment to meet current provincial guidelines and regulations. The TSF embankment will be raised in stages over the mine life. The TSF, tailings distribution system, reclaim water system and surplus water system are shown on Figure 18-5. The TSF filling schedule and embankment raise construction schedule is shown on Figure 18-6.



Source: Knight Piésold, 2020a

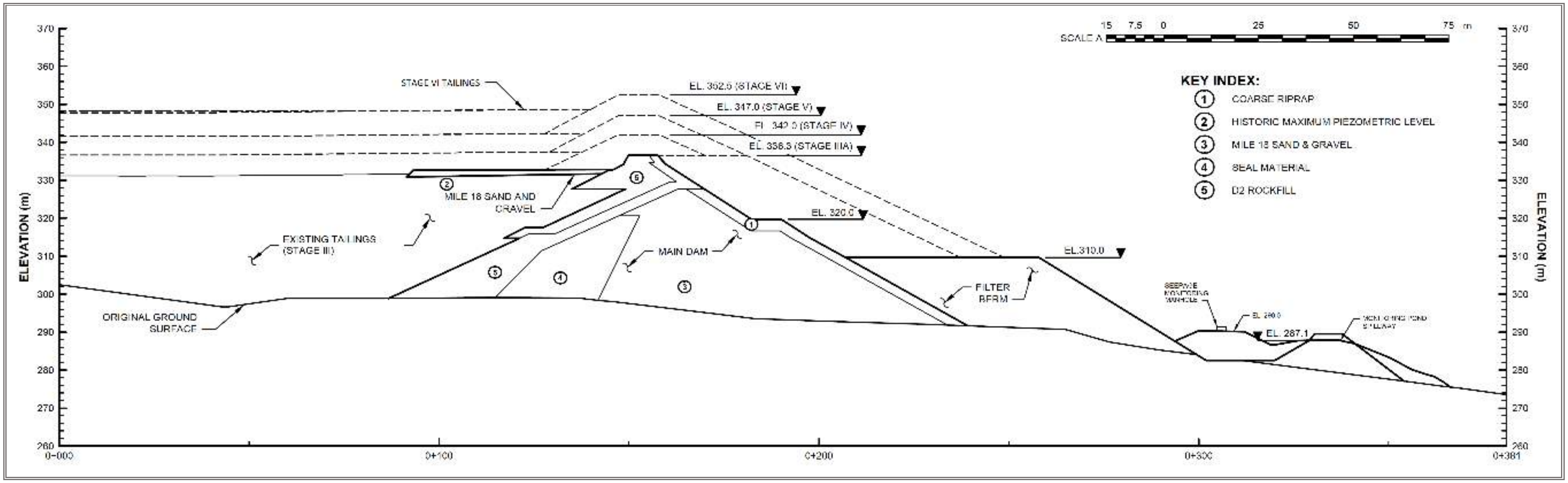
Figure 18-6: TSF Filling Schedule and Raise Construction Schedule

18.5.3 Tailings Storage Facility Embankment Construction Requirements

The TSF embankment impounds the facility on three sides. The TSF embankment has been divided into three zones: the North Dam (the section of the embankment which is aligned parallel to the North Diversion Channel and the Cascade Creek Diversion Channel [CCDC]), the South Dam (the existing TSF embankment section), and the East Dam (along the east side of the facility, linking the North Dam and South Dam sections). All embankments will be constructed using the centreline method.

The South Dam and East Dam will be constructed on a bedrock foundation, while the North Dam will be constructed partially on bedrock and partially (for the upstream toe) on the historical tailings. The North Dam will be constructed as a full-section dam to enhance physical stability in this area.

The embankment raises will be constructed using rockfill generated from the expansion of the CCDC. Filter zones within the embankment will be constructed to limit the migration of fines through the embankment and to control seepage pathways. Rockfill generated from quarrying activities at the CCDC has been characterized as being non-potentially acid generating (NPAG). The embankments will be constructed with 2H:1V slopes (upstream and downstream). The minimum embankment crest width is 10 m to allow working space for pipeline corridors and construction traffic. The final embankment crest elevation is 352.5 masl and the maximum embankment height (crest to toe) is approximately 70 m. A cross-section through the South Dam is shown on Figure 18-7.



Source: Knight Piésold, 2020a

Figure 18-7: TSF South Dam Cross-Section

18.5.4 Tailings Storage Facility Emergency Spillway

An emergency spillway will be constructed adjacent to the East Dam for the initial raises, and finally constructed adjacent to the North Dam for the final stage to align with the closure concept for the TSF. The emergency spillway is designed to safely pass the peak flows from the probable maximum flood (PMF) event throughout operations, closure and post-closure. The spillways for each staged raise of the TSF will be constructed as a rock-cut spillway.

The existing TSF spillway is approximately 4 m deep with a base width of 60 m. It will continue to function as the emergency spillway for the TSF for the restart of operations, but with a 2 m reduction of the invert, which will still facilitate the passing of the PMF.

18.5.5 Closure and Reclamation

The primary objective of the closure and reclamation initiatives will be to return the TSF site to a self-sustaining condition with pre-mining usage and capability. The reclaimed TSF will be required to maintain long-term geochemical and physical stability and protect the downstream environment. Reclamation and closure will involve an active closure period, in which all mine components will be prepared for permanent closure.

TSF closure and reclamation activities will be carried out progressively during the operations phase (where possible) and at the end of economically viable mining. Closure and reclamation activities will be conducted in accordance with best practice for closure of tailings facilities, which is derived from provincial, national and international guidance. Specifically, measures will be taken to ensure that:

- Dust is not emitted from the facility as a result of moisture loss from the TSF surface
- Runoff does not affect surface or groundwater
- The TSF embankments remain stable
- The stored tailings remain physically and chemically stable.

General aspects of closure will include:

- Selective discharge of tailings around the facility prior to closure to establish a final tailings beach that will facilitate surface water drainage and reclamation
- Removal of the supernatant pond
- Dismantling and removal of the tailings, surplus water, and Reclaim Water Systems (RWSs), as well as all pipelines, structures, and equipment not required beyond mine closure
- Placement of a combined rock and soil cover that will achieve long-term chemical stability of the underlying tailings and shed runoff to the final (Stage IV) TSF closure spillway
- Removal and re-grading of all access roads, ponds, ditches, and borrow areas not required beyond mine closure
- Long-term stabilization and vegetation of all exposed erodible materials.

Surface facilities will be removed in stages and mine closure will initiate full reclamation of the TSF. The standpipe piezometers and all other geotechnical instrumentation will be retained for use as long-term TSF safety-monitoring devices. Post-closure requirements will also include annual inspection of the TSF and closure spillway and ongoing evaluation of water quality, flow rates, and instrumentation records to confirm design assumptions for closure.

18.6 Water Management

Site water management involves controlling surface water around the PGP site during the construction, operations, closure, and post-closure phases of the PGP. Water in contact with mine workings or disturbed areas (groundwater inflows from the underground mines; runoff from waste rock, ore stockpiles, quarry areas, tailings, laydown areas, etc.) is considered contact water. Non-contact water is runoff from undisturbed areas, including those areas that are being diverted.

Management of surface water on site will be undertaken by upgrades to existing water diversion structures, construction of the TSF and other infrastructure, selective grading of surfaces, and installation of pump and pipeline systems. The major facilities for contact water management include:

- Tailings storage facility
- Cascade creek diversion channel
- Site diversion ditches
- Water treatment plant
- Water management pump and pipeline systems.

A summary of water management design criteria is provided in Table 18-3. Specific water management features are described in the following sections.

Table 18-5: Water Management Design Criteria Summary

Parameter	Unit	Value
Pipeline Design Criteria		
Tailings Pipeline Design Flowrate (PGP Tailings)	m^3/h	113
Tailings Pipeline Design Flowrate (RMP Tailings)	m^3/h	145
Reclaim Water Pipeline Design Flowrate	m^3/h	47
Surplus Water Pipeline Design Flowrate	m^3/h	365
BM/SC Dewatering Pipeline Design Flowrate	m^3/h	152
Premier Dewatering Pipeline Design Flowrate	m^3/h	425
WTP Contingency Pipeline Design Flowrate	m^3/h	425
Diversion Ditch and TSF Spillway Design Criteria		
Cascade Creek Diversion Channel Inflow Design Flood	Return Period	PMF
Cascade Creek Diversion Channel Peak Design Flow	m^3/s	600
North Diversion Channel Inflow Design Flood	Return Period	1:1000
North Diversion Channel Peak Design Flow	m^3/s	8.7
TSF Spillway Inflow Design Flood	Return Period	PMF
TSF Spillway Peak Design Flow	m^3/s	68

Source: Knight Piésold, 2020a

18.6.1 Tailings Storage Facility

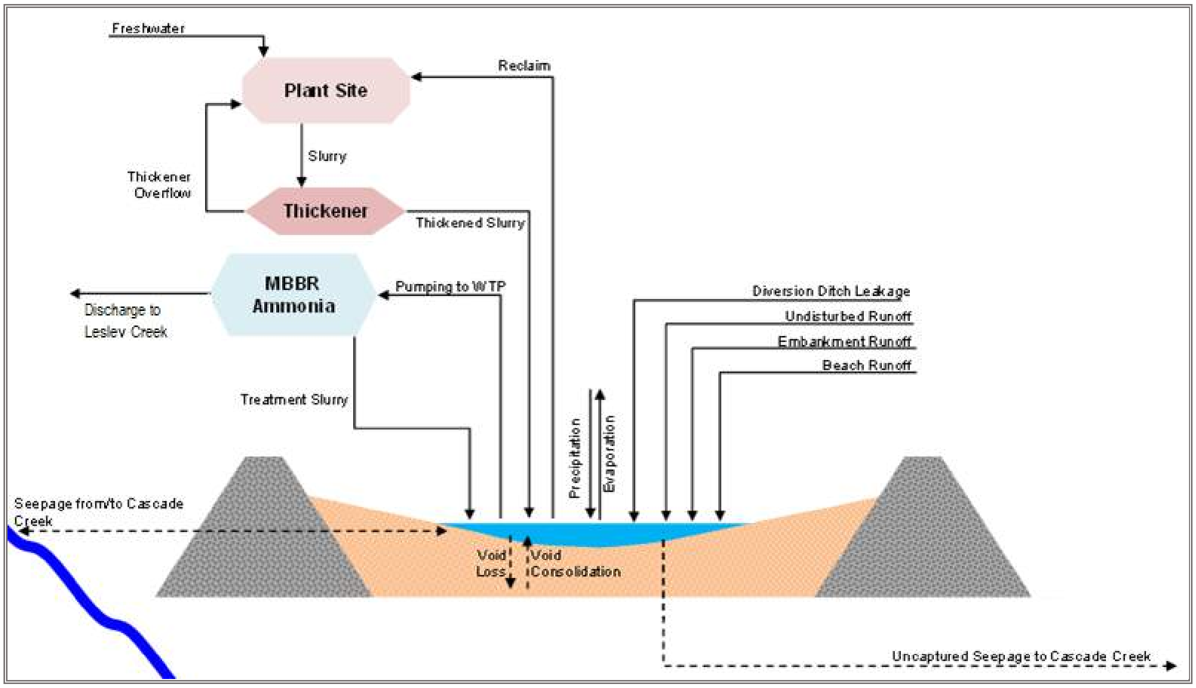
Tailings slurry from the process plant will be discharged into the TSF at a nominal solids content of 60% solids by weight for the PGP tailings, and 50% solids by weight for the RMP tailings (Knight Piésold, 2020b). Water will be reclaimed from the TSF supernatant pond to be used as mill process water. Additional inflows to the TSF supernatant pond include:

- Direct precipitation on the supernatant pond
- Consolidation seepage from the tailings mass as a result of consolidation of the tailings
- Runoff from the tailings beaches, TSF embankment, and undiverted contributing catchment areas
- Dewatering flows from underground mine workings at Big Missouri, Silver Coin, and Premier/Northern Lights (contingency storage in the event of WTP upset conditions and downtime).

Outflows from the TSF supernatant pond includes evaporation from the supernatant pond, reclaim water recycled to the mill, water retained in the tailings voids, seepage, and surplus water removal to the WTP.

The TSF supernatant pond has a maximum operating pond capacity of 500,000 m^3. The TSF water-balance model treats this maximum capacity as a maximum allowable volume to estimate the volume of surplus water that needs to be discharged to maintain water inventory below this volume. A minimum pond volume of 100,000 m^3 was assumed for the TSF from the start to the end of operations. The minimum pond volume was calculated based on TSF reclaim requirements for three months of mill operations. Seepage loss was represented in the water balance model by an outflow of 8 L/s (based on measured seepage rates through the existing TSF embankment).

The water inventory in the TSF was estimated to fluctuate between 100,000 m^3 (minimum) and 500,000 m^3 (maximum) under the water balance flow scenarios. The maximum estimated volume typically occurs during spring freshet and the minimum volume typically occurs during the winter months. A flow schematic for TSF water management is shown on Figure 18-8.



Source: Knight Piésold, 2020a

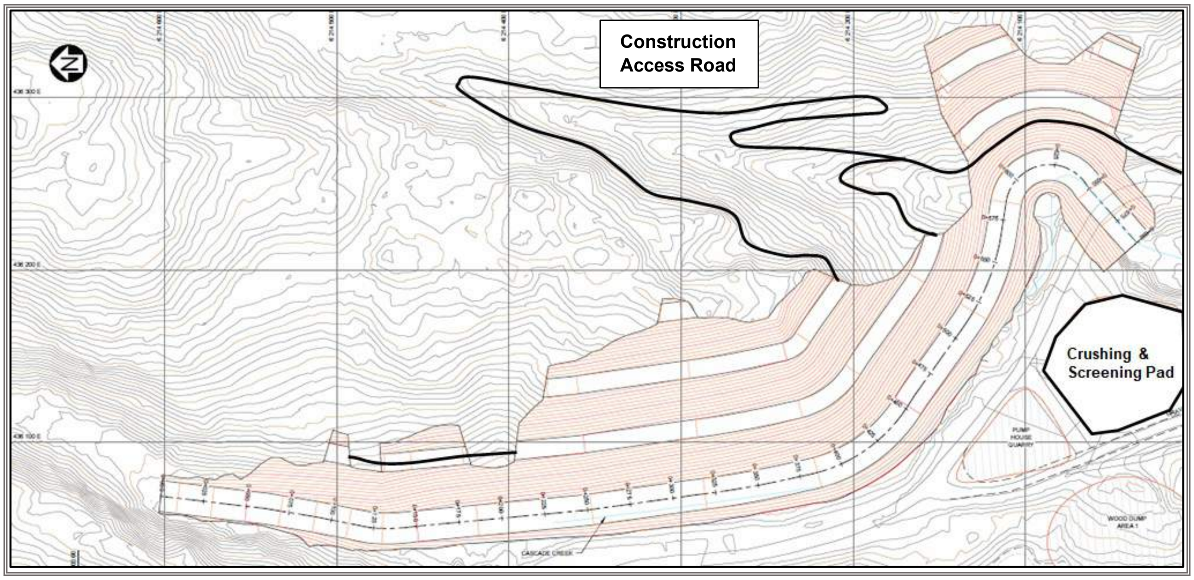
Figure 18-8: TSF Water Management Flow Schematic – Operations

18.6.2 Cascade Creek Diversion Channel

Cascade Creek flow is currently diverted around the east side of the tailings impoundment in the CCDC. The CCDC discharges into Lesley Creek through a 5.5 m diameter culvert beneath Granduc Road. The current diversion berm is made of rockfill and granular soil and was constructed to divert Cascade Creek away from the TSF into the Lesley Creek channel.

As part of the restart of operations, the CCDC will be expanded to pass a peak flow of 600 m^3/s, equivalent to the PMF from the Cascade Creek catchment upstream of the CCDC. The expansion of the CCDC will generate quarried rock for use in construction for the TSF and across the mine site.

The existing culvert beneath Granduc Road that ties the CCDC into Lesley Creek will be removed and replaced with a single-span bridge crossing so that the PMF design flow can be passed without restriction in the expanded CCDC. The CCDC is shown on Figure 18-9.



Source: Knight Piésold, 2020a

Figure 18-9: Cascade Creek Diversion Channel Expansion—Plan

18.6.3 Site Diversion Ditches

The main diversion ditches on site are the North Diversion, which runs along the Granduc Road on the west side of the TSF, and the South Diversion, which is located southwest of the TSF.

The South Diversion (formerly known as the Indian Creek Diversion) diverts Indian Creek away from the TSF to the south.

The North Diversion, on the west side of Granduc Road, collects flow from two seasonally active creeks west of the TSF. Flow from the ditch discharges into Cascade Creek near the north end of the TSF. A berm separates the downstream end of the North Diversion from Cascade Creek. Water passes through the berm via an 800-mm diameter pipe with a downstream flap valve. An 800-mm diameter overflow pipe, located above the first pipe, is also provided as redundancy. The berm and valve are intended to prevent Cascade Creek water from flowing back into the North Diversion.

As part of the restart of operations, the North Diversion will be upgraded to pass the peak runoff flow from a 1 in 1,000-year precipitation event from its contributing catchment area. The majority of the existing channel can pass the new design flow; however, certain sections will need to be expanded to ensure the entire channel can pass this design flow. Additionally, the existing channel will be remediated (i.e., vegetation removed and riprap repaired) as part of the upgrade works. The North Diversion alignment is shown on Figure 18-5.

No additional work or upgrades are planned for the South Diversion channel.

18.6.4 Water Treatment Plant

The existing WTP consists of a low-density sludge (LDS) lime treatment plant and two settling ponds; the Upper Pond and the Lower Pond. Currently the WTP treats underground dewatering flows from the historical Premier underground mine workings via a pipeline from the 6-Level portal to the lime treatment plant. Treated water is discharged into the Upper Pond to settle out suspended solids and flows through a spillway to the Lower Pond for secondary suspended solids settlement. From the Lower Pond, water is discharged to Lesley Creek via a timber-lined spillway.

The WTP will be replaced with a new treatment system as part of the restart of operations, to handle the increased flows and loads from operations. The settling ponds will be drained and backfilled with structural fill to provide a foundation for construction of the new WTP components. The new WTP will consist of a high-density sludge (HDS) lime treatment plant, which will replace the existing LDS WTP; and an ammonia treatment plant.

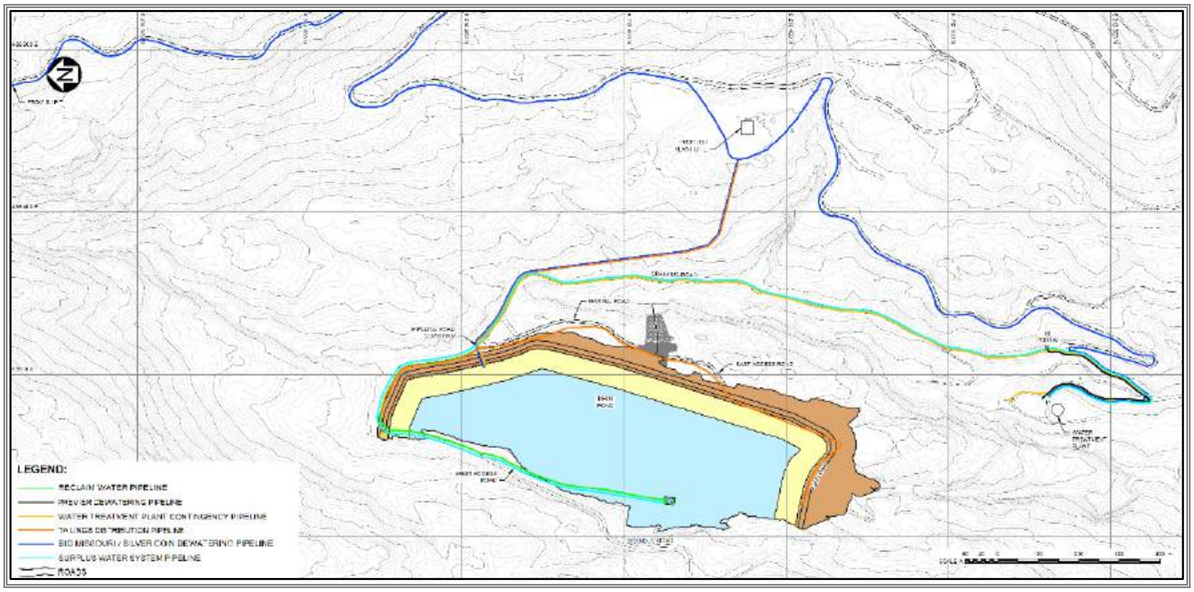
The HDS lime treatment plant uses a clarifier for settling and consolidation of sludge. In the HDS process, sludge collected by the clarifier is pumped back to the treatment process. The sludge recycle improves treatment performance and lime utilization, and reduces the volume of sludge produced. Flows from underground dewatering of the Premier/Northern Lights and the Big Missouri/Silver Coin deposits, as well as surplus water from the TSF Supernatant Pond, will be treated at the WTP and discharge to Lesley Creek.

18.6.5 Water Management Pumps and Pipelines

Contact water will be managed through a number of pump and pipeline systems, including the following:

- Tailings distribution system
- Reclaim water system
- Surplus water system
- Seepage monitoring manhole
- Big Missouri / Silver Coin dewatering system
- Big Missouri / Silver Coin contingency pipeline
- Premier dewatering system
- WTP contingency system.

The water management pipeline layouts are shown on Figure 18-10.



Source: Knight Piésold, 2020a

Figure 18-10: Water Management Pipeline Alignments

Tailings Distribution System

The tailings distribution system is designed to deliver tailings to the TSF and to facilitate the development of tailings beaches along the inside perimeter of the TSF embankments. The system will consist of three primary components: a tailings pump station (located at the mill building), a tailings distribution pipeline to deliver tailings to the TSF, and tailings discharge spigots to deposit the tailings in the TSF. The tailings distribution system and the configuration of discharge spigots will evolve during operations as the TSF embankments develop and operating procedures are refined.

Tailings will be delivered to the TSF through a 175 mm diameter high-density polyethylene (HDPE) pipeline. The pipeline will split at the northeast corner of the TSF to deliver tailings around the perimeter of the TSF. Tailings will be discharged from the TSF embankment crest. Tailings discharge will be rotational, whereby a spigot (or multiple spigots) will be used for a while, then discharge is moved to the next spigot and so on.

Repeating this process will ensure a suitable above-water tailings beach is established and that the pond remains separated from the embankments. Tailings will be selectively discharged to ensure a degree of saturation is maintained within the tailings mass to reduce tailings oxidation.

Reclaim Water System

The RWS allows for the reclaim of supernatant water to the mill for use in ore processing. The RWS consists of a pump barge located on the TSF supernatant pond, centrifugal water pumps, and a 125 mm diameter HDPE pipeline that will extend from the barge to the reclaim water tank at the plant site. From this tank, water will be used in mill processing.

The pipeline will follow the reclaim access road constructed alongside the west side of the TSF, and from there will follow the Granduc Road back to the plant site.

Surplus Water System

The surplus water management system (SWMS) allows for the removal of excess water from the TSF supernatant pond during operations, to maintain target operating pond volumes, tailings beach length, and minimum freeboard requirements. Surplus water will be removed by pumping water to the WTP. The SWMS includes a 300 mm diameter HDPE pipeline for removal of surplus water from the TSF, with centrifugal pumps that are housed on the same barge as the reclaim water pumps. Surplus water will be discharged from the WTP to Lesley Creek.

Seepage Monitoring Manhole

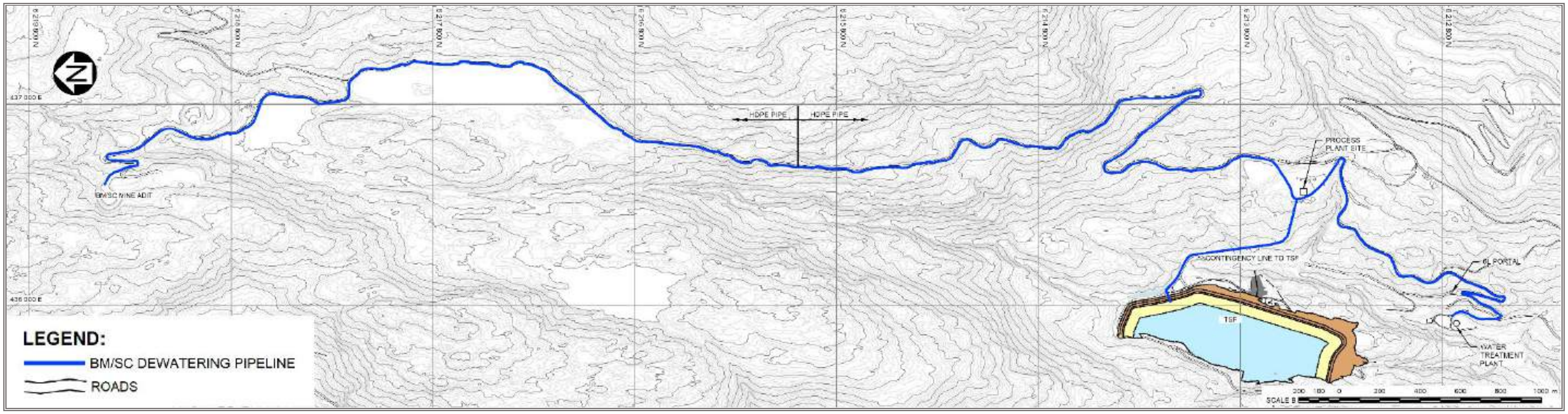
Seepage from the TSF will be monitored for water quality through the existing seepage monitoring manhole. Seepage from the TSF currently feeds through the manhole and discharges to Cascade Creek via two 200-mm diameter HDPE pipelines.

Big Missouri / Silver Coin Dewatering System

Dewatering flows from underground mining activities at the Big Missouri / Silver Coin deposit will be pumped to surface to the S1 Pit mine adit, then delivered to the WTP in a 200-mm diameter HDPE pipeline. The pipeline will be buried in a pipeline corridor that follows the BM Haul Road back to the plant site and from there to the WTP.

A contingency line will be included from the plant site to the TSF with a control valve that will allow for redirection of dewatering flows to the TSF in the event of WTP shutdown.

The Big Missouri / Silver Coin dewatering pipeline alignment is shown on Figure 18-11.



Source: Knight Piésold, 2020a

Figure 18-11: Big Missouri / Silver Coin Dewatering Pipeline Alignment

Premier Dewatering System

Dewatering flows from underground mining activities at the Premier deposit will be pumped to the new 6-Level portal, then delivered to the WTP in a 300-mm diameter HDPE pipeline.

Water Treatment Plant Contingency System

In the event of a WTP system shutdown (i.e., for maintenance) a contingency system consisting of centrifugal water pumps and a 400 mm HDPE pipeline will be used to pump water to the TSF supernatant pond temporarily. The temporary storage of water will be withdrawn from the TSF as soon as the WTP returns to operations.

Additionally, a contingency line will split off from the Big Missouri / Silver Coin Dewatering Pipeline at the Plant Site to divert flow to the TSF in the event of WTP shutdown.

18.7 Water Management, Water Balance, and Water Quality Predictions

A site-wide water balance model and water quality model (WBM/WQM) was developed with GoldSim™ software to simulate the operation of the proposed water management system under the processing demands and environmental conditions that can be expected at the PGP. The estimated rates of pumping, milling, and waste generation were combined with site precipitation, runoff, evaporation, and storage capacity to optimize the operating logic of the proposed water management system. Loading rates for mine-contact water quality were derived from geochemical test work on mine waste samples and from historical water-quality monitoring data. Loading rates and flow rates were combined to estimate water quality in Cascade Creek. The WBM/WQM is illustrated schematically in Figure 18-12.

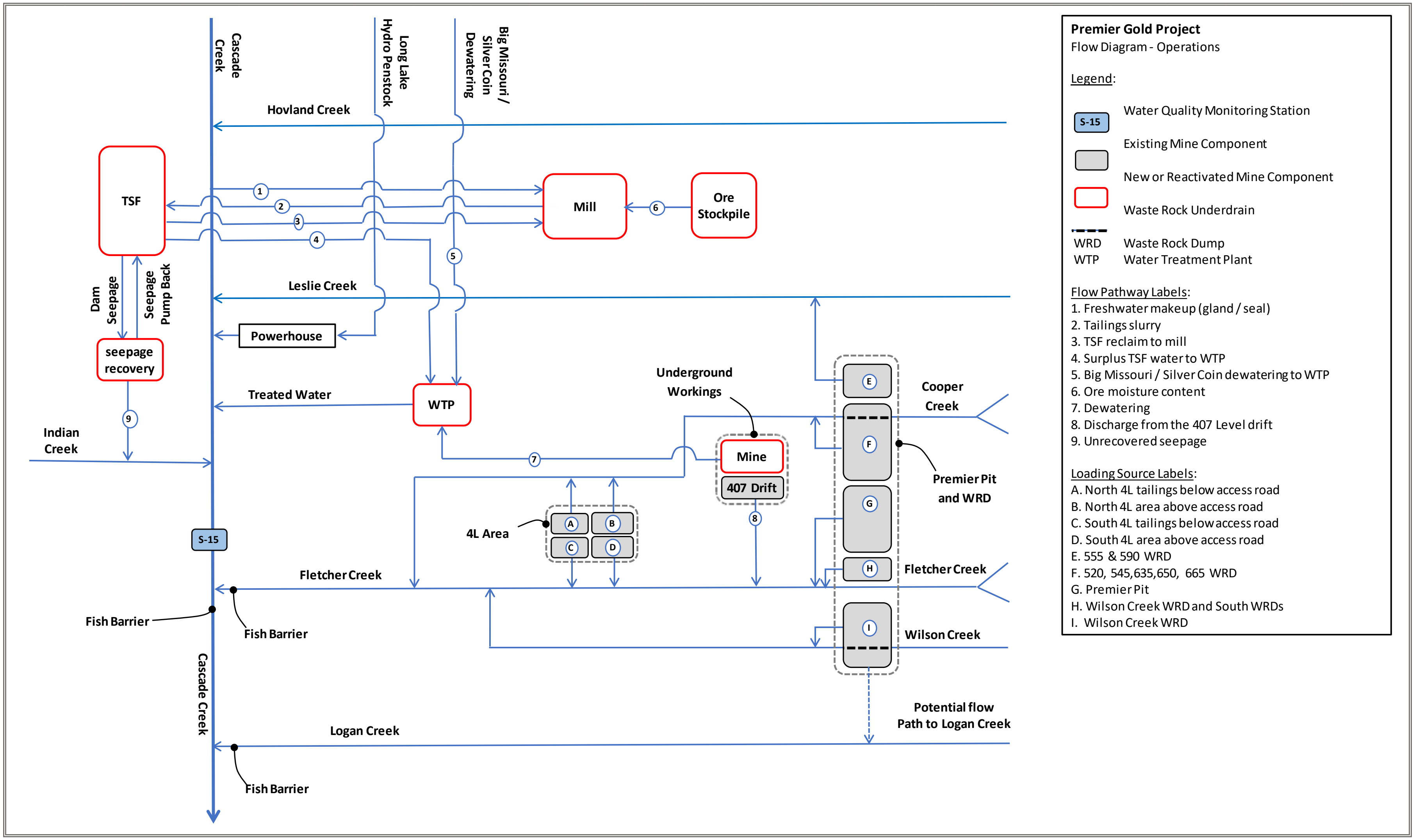


Figure 18-12: Schematic Flow Diagram for PGP Operations

18.7.1 Water and Waste Management System Components

Surplus mine contact water will be treated in the WTP prior to discharge into Cascade Creek. The following components were accounted for in the WBM/WQM.

Mine Dewatering

Dewatering from Premier, Big Missouri, and Silver Coin will be piped to the WTP. WBM input assumptions for dewatering flow rates varied by month, and range as shown in Table 18-6, determined using a hydrogeological model based on flows measured from 2003 to 2018 at the existing 6-L portal at Premier.

Table 18-6: WBM Input Values for Mine Dewatering Flow Rates

Underground Workings	Dewatering Rate (L/s)		
	Average Annual	Maximum Month	Minimum Month
Big Missouri / Silver Coin	39	42	26
Premier	78	126	47

Waste Rock Storage

Waste rock will be backfilled into the underground workings; it will be stockpiled on surface until sufficient void space is available in the underground for waste rock backfill. In the water balance, 50,000 tonnes of waste rock were assumed to be present in the stockpiles at a given time during operations. No waste rock produced by the Project is expected to remain on surface following operations.

Mill

The PGP and RMP ore will be processed in the PGP mill. Fresh water for gland, seal, reagents, and elution will be drawn from Cascade Creek. Make-up water will be reclaimed from the TSF pond. All tailings produced in the mill will be pumped to the TSF as a slurry. Physical and chemical properties of the PGP and RMP tailings streams were combined with the mine plan and the ore process schedule to estimate the slurry water requirements, storage, and slurry water quality.

Tailings Storage Facility

The existing TSF will be expanded to accommodate the tailings produced in the mill. The model accounted for tailings and water storage in the TSF, storage of freshet inflow to the TSF, supernatant reclaimed to the mill, and treatment/release of surplus contact water to Cascade Creek. TSF dam seepage will be discharged to Cascade Creek. TSF seepage water quality will be monitored for discharge water quality and recycled to the TSF, if necessary. Surplus TSF pond water will be pumped to the WTP prior to discharge to Cascade Creek.

WTP Effluent Water Quality

The WTP comprises two treatment components: a moving bed biofilm reactor (MBBR) for removal of nitrogen species, and a HDS lime system for metals removal. Surplus TSF water will flow through both systems and mine dewatering will bypass the MBBR and pass through only the HDS system. The WTP effluent was assumed to have maximum concentrations (Table 18-7). The estimated effluent concentrations were based on typical performances of similar treatment plants in similar settings. Additional details on the WTP design and operation are included in Section 18.

Table 18-7: WTP Effluent Water Quality

Water Quality Parameter	Treatment Process	WTP Effluent (mg/L)
Aluminum	HDS	0.090
Antimony	HDS	0.10
Cadmium	HDS	0.00030
Copper	HDS	0.0180
Iron	HDS	0.10
Lead	HDS	0.0070
Manganese	HDS	0.10
Zinc	HDS	0.025
Ammonia	MBBR	5.0
Nitrate	MBBR	2.0
Nitrite	MBBR	0.010

18.7.2 Cascade Creek Streamflow

Treated WTP effluent will discharge year-round into Cascade Creek. Cascade Creek streamflow is controlled by Regional Power's Long Lake Hydroelectric Project (LLHP). Commissioning of the LLHP began in October 2013 and achieved commercial operation in December 2013. The LLHP headworks, including the intake, is at the outlet of Long Lake upstream of the PGP on Cascade Creek and consists of a main dam and saddle dam, a low-level outlet, penstock intake, and emergency spillway. The Long Lake reservoir provides seasonal storage, and augments winter low flows in Cascade Creek to meet the objectives of the LLHP's operating procedures (Regional Power, 2015). The penstock follows the alignment of the Big Missouri haul road for about 7 km to the powerhouse. The tailrace from the powerhouse discharges into Cascade Creek about 500 m upstream from the existing WTP discharge.

The total drainage area of Cascade Creek is about 80.3 km^2, about half of which (37.8 km^2) is regulated by Regional Power at the Long Lake Dam. Generally, from June to October excess water not used for power production is stored in Long Lake and released at full output from December to January. LLHP tailrace flow rates were based on the average monthly flow projections from the LLHP operating procedures (Regional Power, 2015).

Unregulated watercourses within the Cascade Creek basin (i.e., downstream from the Long Lake Dam) typically experience peak annual flows during spring freshet. Precipitation and streamflow decline throughout the summer. Large precipitation events are common in the fall and may result in dramatic short-term increases in discharge, sometimes triggering peak annual flows. However, aside from short-term increases, flow generally continues to decrease throughout the fall, returning to base flow levels in the winter.

Streamflow in the unregulated portion of the Cascade Creek watershed (tributary streams not affected by LLH) was calculated in the WBM accounting for precipitation, snowpack, snowmelt, and evapotranspiration processes.

The input data set comprised 110 years of monthly precipitation and temperature data collected at Stewart from 1910 to 2019. The Stewart data set was adjusted to reflect the inland conditions experienced at the Premier mine site, including orographic effect; 110 unique model realizations were run to produce a realistic approximation of inter-annual variability in environmental conditions that can be expected at the site.

18.7.3 Water Balance Model Results

On average, more water is expected to enter the water management system than is expected to be lost to processing or to the surrounding environment (i.e., water balance surplus). Surplus TSF water will flow through both the MBBR and HDS treatment systems while underground mine dewatering flows will bypass the MBBR and only pass through the HDS system. The monthly treatment rate for the MBBR system is estimated to be 80 L/s (1,300 USgpm) or less. The monthly treatment rate for the HDS system is estimated to be about 200 L/s (3,200 USgpm) or less.

Table 18-8 shows average monthly flows for each streamflow component in Cascade Creek at S-15 (WTP discharge point on Cascade Creek). Values represent the average results from the 110 model realizations during operations.

Table 18-8: Average Monthly Flow Components in Cascade Creek at S-15 Monitoring Point during Operations

Month	Cascade Creek Downstream From Long Lake Dam (m^3/s)	LLHP Tailrace (m^3/s)	WTP Effluent (m^3/s)	Total Flow in Cascade Creek at S-15 (m^3/s)
Jan	0.42	4.92	0.12	5.47
Feb	0.47	0.48	0.12	1.07
Mar	0.58	0.48	0.13	1.19
Apr	1.12	0.48	0.16	1.76
May	3.85	0.48	0.20	4.54
Jun	9.29	2.90	0.19	12.39
Jul	4.60	4.92	0.15	9.67
Aug	1.75	4.92	0.13	6.80
Sep	2.12	4.92	0.15	7.19
Oct	2.23	3.27	0.18	5.69
Nov	1.06	0.48	0.16	1.71
Dec	0.53	4.92	0.13	5.58
Annual	**2.33**	**2.76**	**0.15**	**5.25**

18.7.4 Water Quality Model Results

Water quality predictions showed that Cascade Creek water quality is expected to remain below the BC Water Quality Guidelines (BCWQG) for the protection of aquatic life below a short mixing zone extending downstream from the WTP discharge point.

18.7.5 Water Balance and Water Quality Model

The WBM/WQM was developed to simulate the proposed site-wide water management system and was run for a realistic range of operational and environmental conditions to assess the performance of the proposed system. Overall, the model results demonstrated that the proposed water management system is sufficiently robust to meet the dynamic conditions that may be experienced during operations. With the proposed treatment system in place, exceedance of BCWQG is not expected in Cascade Creek at monitoring station S-15.

18.8 Water Treatment Plant

The existing water treatment plant (EWTP) consists of an LDS lime treatment system that comprises a lime silo and slaker and two lime reactors (mixing tanks) arranged in series. Currently, the WTP treats underground dewatering flows from the historical Premier underground mine workings. Mine water flows from the 6-Level portal via a pipeline to the lime reactors where it is mixed with slaked lime. The first reactor overflows to the second reactor, which in turn discharges to the Upper Pond where suspended solids (or sludge) settles. From the Upper Pond, the treated mine water flows through a spillway to the Lower Pond for secondary suspended solids settlement. From the Lower Pond, water is discharged to Cascade Creek via a timber-lined spillway.

The EWTP will be replaced with a new water treatment system as part of the restart of operations, to handle the increased flows and loads from operations. The new system will consist of:

- A MBBR water treatment facility for removal of ammonia, cyanide, cyanate and thiocyanate from tailings supernatant (nominal treatment nitrogen capacity of 585 kg/d and 240 m^3/h).
- An HDS lime water treatment plant for removal of dissolved metals and total suspended solids (TSS) (nominal treatment capacity of approximately 720 m^3/h).

Both of the planned treatment processes are commonly implemented to treat mine water produced at underground gold mines. The HDS lime treatment plant uses a clarifier for settling and consolidation of sludge rather than settling ponds. In the HDS process, sludge collected by the clarifier is recycled back to the treatment process. The sludge recycle improves treatment performance, lime utilization and reduces the volume of sludge produced. The planned water treatment processes were selected based on results of a Best Achievable Technology (BAT) assessment of water management and water treatment options. The BAT assessment and water treatment process selection were conducted in collaboration with representatives from Nisga'a Lisims Government.

An amendment to PGP's *Environmental Management Act* (EMA) Permit is required before restart of the operations can commence. Although the completion of a BAT assessment is key to establishing the type of treatment required for a project, unanticipated treatment requirements could be identified during the review of the EMA Permit amendment application.

The MBBR process is required to treat TSF supernatant for ammonia, cyanate and other nitrogen species that are by-products of the cyanide destruction process that will be implemented to remove cyanide in mill tailings as well as minor amounts of residual ammonium-nitrate fuel oil residues associated with the ore. The MBBR process will only treat TSF supernatant. Loadings of ammonia in underground mine water from the new Premier and Big Missouri / Silver Coin underground workings are expected to be minor or negligible compared to loadings contained in the TSF supernatant.

The HDS lime WTP will treat water from underground working at Premier and Big Missouri / Silver Coin as well as TSF supernatant. TSF supernatant will first be treated by the MBBR process before it is routed to the HDS plant. Treated effluent from the HDS WTP will be discharge to Cascade Creek.

The planned WTPs will be implemented in a way that allows the existing LDS WTP to continue to operate. First, the Upper Pond will be reconfigured to improve settling of sludge by relocating the inflow and installing baffle curtains. Improving settling in the Upper Pond will allow for decommissioning of the Lower Pond. A pumping system will be installed to discharge treated water from the upper pond to Cascade Creek. The Lower Settling pond will then be drained and backfilled with structural fill to provide a foundation for construction of the new HDS plant. When construction and commissioning of the new HDS plant has been completed, the underground mine water from the Premier Underground will be treated by the new HDS lime plant and the existing LDS plant will be decommissioned. The new MBBR WTP plant will be constructed at the site of the existing LDS plant. The MBBR process will commence operation when milling and processing of ore commences.

The MBBR WTP is expected to operate during the operations phase of the Project and for five years following the end of operations. At this time, the loadings of ammonia, cyanate and other nitrogen species in the TSF supernatant is expected to be low enough that treatment for nitrogen species will no longer be required. HDS lime water treatment of underground mine water is expected to continue in perpetuity.

18.8.1 Red Mountain Water Treatment Assessment

A water treatment assessment was completed for the Red Mountain Underground Gold Project as part of the Environmental Assessment for that site, which was finalized in 2018. The water treatment assessment concluded that a lime water treatment and a MBBR process would be required to treat tailings supernatant from the (proposed) Red Mountain tailings pond. The assessment also indicated that underground mine water from the Red Mountain mine likely would require removal of TSS in a settling pond but that no other treatment would be required. The water treatment technologies planned for the PGP are therefore consistent with the treatment requirements associated with processing of Red Mountain ore.

18.9 Camps and Accommodation

Ascot will make provision for temporary construction accommodation and catering. However, the PGP labour philosophy is that construction labour will be sourced locally from the Stewart, Terrace, Smithers, and surrounding areas.

18.10 Power and Electrical

18.10.1 Plant Load

The mill throughput is nominally 2,500 t/d. At this production level, the plant load is estimated to be approximately 15 MW ±10%.

18.10.2 Power Source—Long Lake Line

Electrical power will be supplied from a 138 kV tap from the Long Lake Independent Power Producer (IPP) line. Figure 18-2 indicates the location of the Long Lake IPP and the route of the 138 kV line to the New Main Substation.

18.10.3 Service to PGP

Service to PGP will take power from the existing 138 kV transmission line currently running to Long Lake Power Station. The line to PGP will be approximately 1 km long and terminate at the proposed PGP New Main Substation at the PGP mill site.

There will be one main transformer feeding the mill site. Each transformer will be base rated at 15 MVA, with additional fan cooled ratings of 20 MVA. Transformers of this size are in the range of 40-tonnes and will be one of the largest loads to transport into the site.

The transformer will feed a secondary bus at the 4.16 kV level. Large motor loads (e.g., ball mills) will be served at 4.16 kV level. Power will be distributed at 4.16 kV around the site using cables and overhead lines, and additional step-down transformers will be located near remaining loads. Medium-sized motor loads (250 hp to 5,000 hp) will be served at 4.16 kV. Smaller motor loads will be served at 600 V.

Electrical rooms (housed within the heating and ventilation structure) will be provided at the Premier Portals (Big Missouri, Silver Coin, and Premier). These electrical rooms will include motor control, lighting panels, and other electrical equipment necessary for facility operation. The power supplied at the BM portal will be reticulated underground to the SC deposit.

The portals will be fed with 4.16 kV, a suitable voltage to feed via cable through the portals to the underground workings, where it will be further stepped down to 600 V to feed the Jumbos and drills.

About 1 MW of power will be reticulated to each portal.

Power at the RMP portal will be provided at 4.16 kV by a generator system. Power will be distributed throughout the RMP mine site at this voltage for large electrical loads. A number of centrally located electrical rooms (by others) will transform the 4.16 kV power to 600 V as necessary.

Electrical power will be distributed at 4.16 kV by overhead power line to locations such as the TSF, reclaim and booster pump stations, fresh water intake, the WTP and the existing infrastructure facilities, such as the bunk house.

Single-line diagrams and layout drawings can be found in Appendix D.

Underground electrical power distribution equipment will feed the underground ventilation and miscellaneous loads. Power will be delivered to underground operations via a single 4.16 kV underground power cable.

18.11 Fuel Facilities

Diesel and gasoline for the mining, process, and ancillary facilities will be supplied from above ground diesel fuel storage tanks located near the process plant. The diesel fuel storage tank will have a capacity sufficient for approximately seven days of operation. Diesel storage will consist of above-ground tanks and will include loading and dispensing equipment. A dedicated service truck will transport diesel to the mining equipment operating in the pit.

The diesel tanks were used to supply the plant generators before a permanent power supply was connected to the process plant. They are housed within a concrete secondary containment that is in poor condition and will be upgraded to contain spilled fuel.

18.12 Water Supply

Fresh/firewater will be pumped from the Cascade Creek Diversion to the fresh/firewater tank adjacent to the mill site by a surface pipeline. The pipeline will be heat-traced to prevent freezing. Fire water piping, equipment, and alarms will be installed to meet current codes/standards.

Fresh and Firewater Supply

Fresh water intake will be from Cascade Creek and pumped at ground level to the fresh/firewater tank. It will be stored in the refurbished fresh/firewater tank. Firewater will be distributed from the new Fire/freshwater pump house and reticulated via underground pipelines to the process and non-process facilities. A new fire water protection systems will be installed in the process and non-process buildings at the mill site.

Potable Water

Potable water will be stored in carbon steel, double-walled potable water distribution tank located in the new Firewater Pump house next to the fresh/firewater tank and will be distributed by pumping. A hypo-chlorinator will disinfect the water prior to use.

18.13 Communication

The telecommunications system will be provided by Ascot.

18.14 Closure Plan

It is recognized that an effective and robust closure plan is critical to the permitting and approval of the PGP, and that the closure plan will receive scrutiny from regulators, affected First Nations, and stakeholders.

The following are objectives for the PGP:

- Protect public health and safety
- Mitigate and prevent environmental damage
- Ensure chemical and physical stability of permanent structures
- Restore natural drainages to the extent possible
- Restore the site to conditions commensurate with appropriate post-closure land use requirements
- Promote long-term social and economic benefits.

All the existing process facilities and non-process facility buildings/structures will be torn down and equipment removed from site. All concrete pedestals will be broken down to grade level and the original footprints covered with crushed rock.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies have been undertaken regarding the sale of gold and silver produced from the mining of gold ore from the project and its processing into doré. The final product will gold and silver doré. The gold and silver doré will likely be transported to a North American based previous metals refinery or sold to precious metals traders. Indicative doré refining terms were obtained from a Canadian refinery in early 2020. Doré can be refined by several refineries throughout North America, and the gold and silver produced sold either domestically or abroad in order to realize the highest available price.

Doré refining charges were estimated at US$0.76/oz. Transportation costs for doré produced were estimated at US$0.54/oz shipped, based on bi-weekly shipping to eastern Canada for refining and sale.

Table 19-1: Refining and Transportation Assumptions

	Unit	LOM
Gold (Au) Payable	%	99
Silver (Ag) Payable	%	99
Refining Charges	US$/oz doré	0.76
Transportation	US$/oz doré	0.54

19.2 Commodity Price Projections

Precious metals are freely traded globally in many markets and are highly liquid commodities. Metals prices and foreign exchange rates used in the economic analysis are in line with recently published technical reports and are within a few percentage points of both the 24 month and 36-month trailing averages.

A gold price of US$1,400/oz and a silver price of US$17.00/oz have been used. A flat US$:C$ exchange rate of 0.76 was also used in the analyses. The metal prices and foreign exchange rates used in the analyses are only estimates and future values cannot be reliably forecast.

19.3 Contracts

There are no material contracts in place with respect to the marketing, sale, or transport of doré produced at the PGP.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

In 2018, Ascot Resources Limited (Ascot, the Owner) initiated a baseline gap analysis to reflect current environmental conditions, to address refinement of the Premier Gold Project (PGP) design, and to incorporate regulatory requirements for a permit amendment application needed to restart the PGP mine. Baseline study objectives were established to develop an understanding of species that occupy or move through the PGP area; gather data to inform an assessment of potential adverse project effects; inform the development of environmental management plans; and to inform assessments of Nisg̱a'a Treaty rights and interests. This analysis found gaps in the following subject areas: fish and aquatic habitat; climate and hydrology; hydrogeology; geochemistry; terrain, soils, and natural hazards; water and sediment quality; vegetation and ecosystems; and wildlife and wildlife habitat.

Baseline data collection and reporting programs were then prepared to fulfill all data requirements identified by the gap analysis. These programs were shared with the Nisg̱a'a Lisims Government (NLG) for their review and input. Fieldwork was initiated in June 2018, and sufficient baseline data have now been collected and analyzed in baseline reports to support ongoing permitting and assessment efforts.

For the Red Mountain property, comprehensive studies were conducted between 2015 and 2018 with a focus on the following topic areas: climate, surface hydrology, groundwater, aquatic resources, water quality, sediment quality, terrestrial ecology, wildlife, and fish habitat. Comprehensive studies also focussed on rock geochemistry, archaeology, heritage resources, land use, cultural, and socioeconomic conditions to characterize the regional human environment. These studies were key to building the contents and requirements of an environmental assessment (EA) application report.

20.2 Premier Gold Project

20.2.1 Land Capability and Use

The PGP mine is located on provincial Crown lands within the Regional District of Kitimat-Stikine (RDKS) and the Nass South Sustainable Resource Management Plan. The PGP is located within the Nass Area defined in the Nisg̱a’a Final Agreement (NFA, 2000). The mine is not located within a designated water management unit, special habitat for general wildlife, nor has there been an ecosystem network unit identified. Ascot holds mining leases and mineral tenure claims for the mine. There are no applicable zoning, Agricultural Land Reserve designations, or other land use designations (e.g., parks or protected areas) relevant to the mine. The nearest park is the Bear Glacier Provincial Park, which is approximately 60 km from PGP by road. The mine, as well as the other nearby mines, have been developed, operated, and closed since the gold rushes in the late 19^{th} and early 20^{th} centuries.

Placer mining began in 1898 near Stewart, BC, the closest Canadian municipality to the mine, and developed into hard-rock mining by the end of 1910 at the Red Cliff Mine (McLeod & McNeil, 2004). The ore at the Premier mine was first discovered in 1919 and was followed by the neighbouring Big Missouri Mine. During the 1930s, Premier was an isolated and self-contained town of approximately 200 employees living in a bunkhouse, and in about 50 houses on the hill near the mine. The mine was purchased by Westmin Resources in 1981, and a 2,000 to

3,000 t/d mill produced 260,000 oz of gold and 5 Moz of silver from the Premier and Big Missouri mines. The mine was placed into care and maintenance in the mid-1990s (McLeod & McNeil, 2004).

Forestry production in the mine area is limited by steep terrain, climatic conditions, and thin, infertile soil. Poor regional forestry values, low timber quality, and long haulage distances combine to limit the economic viability of timber harvesting in the Stewart–Portland Canal area. Agricultural potential in the PGP area is also limited by poor soil conditions, marketing restrictions, and a short growing season.

Other resource interests overlapping with the mine include a trapline (TR0616T012), a guide outfitting license (601084) held by Nisg̱a'a Guide Outfitters, and a commercial recreation license (File No. 6406136) held by Last Frontier Heliskiing.

20.2.2 Vegetation

According to the Terrestrial Ecosystem Mapping (TEM) in the Premier mine, one parkland unit (Mountain Hemlock–Moist Maritime parkland [MHmmp]) and one alpine unit (Coastal Mountain heather Alpine–Undifferentiated [CMAun]) comprise most of the local study area. Field verification of bioterrain and TEM were completed in August and October 2018 and July 2019, resulting in collection of ecological, terrain, and soil data from 198 sample plots. Sensitive ecosystems were identified for the PGP based on a review of existing information and adjacent mine proposals. Sensitive ecosystems have high ecological values or sensitivities to human disturbance, including high biodiversity, habitat values for keystone species and at-risk species, and are generally limited in distribution. Identified sensitive ecosystems included: wetlands; freshwater and floodplains; mature and old forests; parkland and alpine ecosystems; and at-risk ecological communities.

The surveys for at-risk species identified a total of 273 species of vascular plant, moss, liverwort, and lichen from field studies in August 2018 and July 2019. A total of 27 at-risk species were identified in the local study area (LSA) in 30 locations. The identified species includes 10 species of lichens (including three Red-listed species), four liverworts, eight mosses (including three Red-listed species), and one vascular plant.

Surveys for invasive plants identified no invasive and three exotic plants: perennial ryegrass (*Lolium perenne*); common timothy (*Phleum pratense*); and annual bluegrass (*Poa annua*).

Botanical forest products were mapped using the TEM based on occurrence of potential habitat. A list of 10 desirable species was provided by NLG that included nine vascular plants and one mushroom. Of the 10 species, five were short-listed based on presence and abundance: Alaska blueberry (*Vaccinium ovalifolium*), black huckleberry (*Gaylussacia baccata*), devil's club (*Oplopanax horridus*), and Indian hellebore (*Veratrum viride*). Each species was assessed for potential occurrence, and the analysis indicated that Alaska blueberry is abundant in the forested portions of the study area, and black huckleberry is also common, with widespread occurrence in mid- to upper-elevation forested areas. Potential occurrence of medicinal plants included Indian hellebore, which has the potential to be widespread in alpine meadows and avalanche slopes, and devil's club, which is limited to uncommon wet, organics-rich, seepage sites.

Sample plots for analytical chemistry were completed in two sub-areas in proximity to the proposed mining infrastructure. Plant and soil samples were extracted from the plots and submitted for laboratory analysis of select elements (metals and metalloids). A total of 11 of 13 analyzed metal/metalloid trace elements exceeded Canadian Council of Ministers of the Environment (CCME) soil guidelines for industrial use at 69% of the LSA sample sites.

The majority of the exceedances involved arsenic (65%) and zinc (13%). Multiple factors are implicated in the trace element concentrations that were measured, and the pattern is complex. Site disturbance, the pH and texture of soil, type of vegetation that was present, local area, and depth within the rooting zone where the soil sample was taken also influenced the statistical likelihood of the trace element concentrations that were reported.

20.2.3 Wildlife

Wildlife and wildlife habitat baseline studies for the PGP area were undertaken during the summer of 2018 for grizzly bear (Ursus arctos), mountain goat (Oreamnos americanus), moose (Alces alces), and northern goshawk (Accipiter gentilis). Field studies included ground surveys for selected species and focused on grizzly bear, bats, breeding birds, western screech owl (Megascops kennecottii), northern goshawk, marbled murrelet (Brachyramphus marmoratus), common nighthawk (Chordeiles minor), and western toad (Anaxyrus boreas). There was recorded presence of two bat species, little brown myotis (Myotis lucifugus) and northern myotis (Myotis septentrionalis) that indicated usable habitat. Based on the wildlife suitability model developed for marbled murrelet, no high-suitability habitat, but some moderate habitat is present. For reproducing bird habitat, northern goshawk, western screech owl, common nighthawk, and sooty grouse summer and winter habitat were identified in the study area.

Western toads were detected in most surveyed locations, most notably in the vicinity of the tailings storage facility (TSF). Cascade Creek valley is likely a key overland corridor linking habitats and dispersal patterns from near the mine to those in the upper elevations near Silver Lakes.

20.2.4 Fisheries and Aquatic Resources

Environmental baseline investigations initiated in 2018 at the PGP covered surface water, groundwater, and fish and aquatic resources. Historical studies were integrated into the current studies to understand and characterize environmental conditions at the site. Fish presence within the PGP area is limited to the lowermost 800 m of Cascade Creek, approximately 500 m downstream from the existing water treatment plant (WTP) discharge point on Cascade Creek.

Surface water and sediment in the PGP area lakes and streams frequently have naturally occurring, low-magnitude guideline exceedances for the protection of aquatic life; however, there is no indication that this is affecting the health of biological communities, including fish in lower Cascade Creek.

Prior studies identified a waterfall 1 km upstream from the Salmon River as the upstream limit of fish distribution in Cascade Creek watershed. A barrier to fish passage (a 50 m-long 20% gradient section) approximately 200 m below the waterfall (800 m upstream of the Salmon River) was identified in 2019 and is now considered the upstream limit of fish distribution in the Cascade Creek watershed. The location marking the upstream limit of fish presence (i.e., the downstream end of the 50 m-long, 20% gradient section that is impassable for fish) is approximately 500 m downstream from the existing WTP discharge point on Cascade Creek. Fletcher Creek is a non-fish-bearing tributary of Cascade Creek (entering Cascade Creek above the barrier) carrying loading from the PGP waste dumps. Logan Creek is a background tributary that enters the fish-bearing section of Cascade Creek; however, it is non-fish bearing owing to a high-gradient barrier at its confluence with Cascade Creek. Within the fish-bearing section of Cascade Creek, and downstream in the Salmon River, most species of Pacific salmon are present. Dolly Varden char (*Salvelinus malma*), Coastrange sculpin (*Cottus aleuticus*), and coho salmon

(*Oncorhynchus kisutch*) have been captured in lower Cascade Creek at a sampling site located approximately 400 m upstream from the Salmon River. The Salmon River is known to support chum (*O. keta*), pink (*O. gorbuscha*), Chinook (*O. tshawytscha*), and coho salmon. Benthic macroinvertebrates are present in relatively high abundances in the project area drainages, likely because of the low predation given the absence of fish.

20.3 Red Mountain Property

20.3.1 Land Capability and Use

The Project is located within the Nass Wildlife Area, as defined in the Nisg̱a'a Final Agreement (NFA, 2000). Gold was first discovered at Red Mountain in 1965 and mineral exploration in the area dates to the late 19th century. Forestry production in the area is limited by steep terrain, climatic conditions, and thin, infertile soil. Poor regional forestry values, low timber quality, and long haulage distances combine to limit the economic viability of timber harvesting in the Stewart–Portland Canal area. Agriculture potential in the study area is also limited by poor soil conditions and a short growing season.

Other resource interests overlapping with the PGP area include one guide outfitter concession (601036) and two traplines (TR0614T101, TR0614T094), as well as one commercial recreation licence (File No. 910116) for a heli-ski operation.

20.3.2 Vegetation

The Bitter Creek valley contains two major biogeoclimatic zones, namely the Coastal Western Hemlock (CWH) along the valley floor and the Mountain Hemlock (MH) at mid-elevations. Most of the land within the alpine area is occupied by glaciers or recently exposed bare rock. Trees near the treeline are mostly mountain hemlock, yellow cedar, and subalpine fir. In the alpine, vegetation is made up of low-growing, evergreen dwarf.

The CWH landscape, at low elevations in the Bitter Creek valley, is dominated by shallow organic and morainal surficial materials. Characteristic vegetation includes coastal muskeg and stunted coniferous forests of western hemlock, western red cedar, yellow cedar, Amabilis fir, and shore pine (Red Mountain EA, 2017).

The MH zone is considered subalpine lands and is present at mid-elevations in the Bitter Creek valley. This landscape is characterized by dense, closed-canopy forest at lower elevations, transitioning to open parkland, heath, and meadow at higher elevations. The dominant tree species include mountain hemlock, Amabilis fir, and yellow cedar. The understory is characterized by interspersed sedge and mountain-heather shrubs (Red Mountain EA, 2017).

20.3.3 Wildlife

Wildlife species present near the Red Mountain property or in adjacent habitats include: black bear, grizzly bear, wolf, and mountain goat. Smaller furbearers present in the region may include marten, red squirrel, and the hoary marmot. In the alpine area surrounding the immediate mine site, the presence of these furbearers is limited. In addition to smaller passerines, bird species that inhabit the area include rock ptarmigan, blue grouse, and ruffed grouse.

20.3.4 Fisheries and Aquatic Resources

Red Mountain lies within the Bitter Creek drainage basin, a tributary of Bear River, with the confluence near Highway 37A. Bitter Creek is a confined, heavily turbid mainstem comprising predominantly strong riffle habitat through steep valleys. Bitter Creek is fish-bearing up to the first of seven physical barriers, located 13.8 km upstream from the confluence with Bear River. The 2014 to 2017 baseline studies indicated that Dolly Varden were present in Bitter Creek. Their distribution covers the entire fish-bearing section of Bitter Creek. Coastrange sculpin have been documented in Bitter Creek near the mouth. Bear River has a more diverse fish community, with salmon species (coho, Chinook, chum, and pink), rainbow trout, Dolly Varden, eulachon (*Thaleichthys pacificus*), and Coastrange sculpin. Bitter Creek has multiple tributaries, two of which are fish-bearing within the lower reaches: Hartley Gulch and Roosevelt Creek.

20.4 Environmental Liabilities

The Study has been designed to minimize short- and long-term environmental impacts, and to maximize lasting benefits to local communities, employees, and shareholders. Ascot's goal is to create a sustainable operation that employs best available technology and practices in all aspects of the design and operation and considers both the short- and longer-term effects on the environment.

For the PGP, a *Mines Act Permit* was issued to Ascot in October 2018 with a bond totalling $14.65 million as of October 15, 2020.

At Red Mountain, a $1,000,000 cash reclamation bond has been posted with the provincial government against the property and can be recovered pending closure and remediation of all environmental requirements listed in the permit.

Ascot is not aware of any significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the Study properties.

20.5 Permitting and Required Approvals

Depending on the scope of a given project, assessment and permitting of major mines in BC may proceed through the BC EA process pursuant to the *British Columbia Environmental Assessment Act* (BC EAA) and the federal environmental impact statement (EIS) process, pursuant to the *Impact Assessment Act* (2019).

Red Mountain formally entered both the provincial and federal assessment processes in October 2015, with the filing of a Project Description Report. Since that time, an EA application was submitted and reviewed, then received provincial approval and issuance of an EAC on October 5, 2018. Federal approval of the EIS was received on January 14, 2019.

PGP is an existing mine in Care and Maintenance, and does not require an EAC; however, it will require multiple permits and amendments to existing permits, including a provincial *Mines Act* (1996) permit and *Environmental Management Act* (2003) permit.

All provincial permit applications for the Study will be coordinated through the Major Mines Office of the BC Ministry of Energy Mines and Petroleum Resources (EMPR).

Ascot has, or will acquire, all required permits for ongoing exploration and future project work on all of the properties. Ascot is currently in possession of the following permits:

- Amended MX-1-743 granted by EMPR that allows Ascot to conduct exploration on the Property. The permit allows for 800 drill sites to be completed by March 31, 2023.
- A Free Use Permit (FUP) for timber cutting, issued for a term of January 8, 2018 to March 31, 2023 for a maximum 50 m^3 volume of timber to be cut.
- Permit MX-1-643, for exploration work at Silver Coin. The current permit expires on March 31, 2022 and allows 40 ground-supported drill sites and 2.35 km of new trail. A bond held by EMPR in the amount of $71,300 will be held until reclamation of these drill sites is completed to the satisfaction of EMPR.
- Permit PE-8044, granted by the Ministry of Environment & Climate Change Strategy (ENV) under the Environmental Management Act (EMA), which allows for two effluent discharges: 6-Level adit WTP effluent to Cascade Creek, and supernatant from the TSF and monitoring pond to Cascade Creek.
- An amendment to Permit 109154, issued under the provisions of the EMA in January 2020 for effluent discharge of adit water at Red Mountain.
- Permit MX-1-422 for exploration work at Red Mountain, issued by EMPR in June 1993 and amended May 2002.

Table 20-1 presents the principal provincial authorizations, licenses, and permits still anticipated for project construction and operations. The list will be refined to reflect feedback from regulatory agencies throughout the permitting process. All permits listed below are assumed to apply to both the Premier Gold Project and the Red Mountain property, unless specifically detailed herein.

A list of anticipated federal permits and authorizations is shown in Table 20-2. This list will be refined through working with the appropriate government agencies.

Table 20-1: List of Anticipated Provincial Permits and Authorizations

Permits	Agency	Legislation	Description
Mines Act Permit M-179 amendment	EMPR	Government of BC, *Mines Act* (1996)	PGP facilities and infrastructure within the Mine Lease Authority for the construction, operation, and closure of the mine site
Mines Act Permit	EMPR	Government of BC, *Mines Act* (1996)	Authorization to construct, operate, close/decommission, and reclaim a mine at Red Mountain
Environmental Assessment Certificate (# M18-03) amendment	EAO	*Environmental Assessment Act* (2019)	Under Section 19(1) of the *Environmental Assessment Act*, EA Certificate to be amended to reflect changes to the Red Mountain mine plan
Amendment of PE-8044 Waste Discharge Permit -Effluent	ENV	Government of BC, *Environmental Management Act* (2003)	PGP WTP currently treating adit discharge water will require modification to accommodate operations and closure effluent discharge
Air Emissions Discharge Permit	ENV	Government of BC, *Environmental Management Act* (2003)	Authorization for air emissions discharge

Permits	Agency	Legislation	Description
Mining Lease	EMPR	Government of BC, *Mineral Tenure Act* (1996)	Authorization for the exploration or development of the mineral resource
Licence of Occupation and Statutory Right of way	FLNRORD	Government of BC, *Land Act* (1996)	Authorization to occupy crown land for construction of freshwater pipeline
Temporary Use/Work Permits	FLNRORD	Government of BC, *Land Act* (1996)	Temporary (short-term) use of Crown land portions for construction of fresh water pipeline
Investigative Use Permit	FLNRORD	Government of BC, *Land Act* (1996)	
Hazardous Waste Registration	ENV	Government of BC, *Environmental Management Act* (2003) - Petroleum Storage and Distribution Facilities Storm Water Regulation	Authorization for temporary storage of hazardous waste
Explosives Storage and Use Permit	EMPR	Government of BC, *Mines Act* (1996)	Approval for surface and underground explosive storage and use
Section 9 Approval or Authorization for Changes In and About a Stream	FLNRORD	Government of BC, *Water Sustainability Act* (2014)	Approval for changes in and about a stream that are of a complex nature
Section 8 Approval, Water Use License	FLNRORD	Government of BC, *Water Sustainability Act* (2014)	Authorization to divert and use surface water
Construction Permit	Northern Health Authority	Government of BC, *Drinking Water Protection Act* (2001)	
Special Use Permit	FLNRORD	Government of BC, *Forest Act* (1996)	Approval to build an access road and gravel pits on unencumbered crown land (non-tenure)
Occupant Licence to Cut	FLNRORD	Government of BC, *Forest Act* (1996)	Authorizes timber harvesting consistent with approved road upgrade and road design at Red Mountain
Permit to connect a Powerline	BC Hydro	Government of BC, *Safety Standards Act* (2003) - Electrical Satefy Regulation	Approval of plans to connect a private powerline to the BC Hydro grid
Road Use Permit	FLNRORD	Government of BC, *Forest Act* (1996)	Approval for the use of forest service roads at Red Mountain
Industrial Access Permit	Ministry of Transportation and Infrastructure	Industrial Roads Act (1996)	Access improvements to Red Mountain Access Road
Highway Access Permit/Provincial Public Highway Permit Application	Ministry of Transportation and Infrastructure	Government of BC, *Transportation Act* (2004), *Motor Vehicle Act* (1996)	Approval for industrial access to Highway 37A; and Highway 37A to Alaska border

Notes: EMPR = Energy, Mines and Petroleum Resources; EAO = Environmental Assessment Office; ENV = Environment & Climate Change Strategy; FLNRORD = Forests, Lands, Natural Resource Operations & Rural Development.

Table 20-2: List of Anticipated Federal Permits and Authorizations

Permits	Agency	Legislation	Approval Requirement
Fisheries Act Authorization	Fisheries and Oceans Canada	*Fisheries Act*	Authorization required for impacts to Bitter Creek fish habitat along the Red Mountain Access Road
Explosives Factory Licence	Natural Resources Canada (NRCan)	Government of Canada, *Explosives Act* (1985)	Authorization for explosives storage magazines, and for explosives manufacture/mixing
Radio Licences	Industry Canada	Government of Canada, *Radiocommunication Act* (1985)	Establish and operate radio frequencies and related infrastructure
Radio-isotope Licences	NRCan	Government of Canada, *Nuclear Safety and Control Act* (1997)	Authorization for nuclear devices such as slurry density flow meters

Figure 20-1 outlines the anticipated scenario for an approval schedule, subject to ongoing discussions with NLG and relevant provincial agencies.

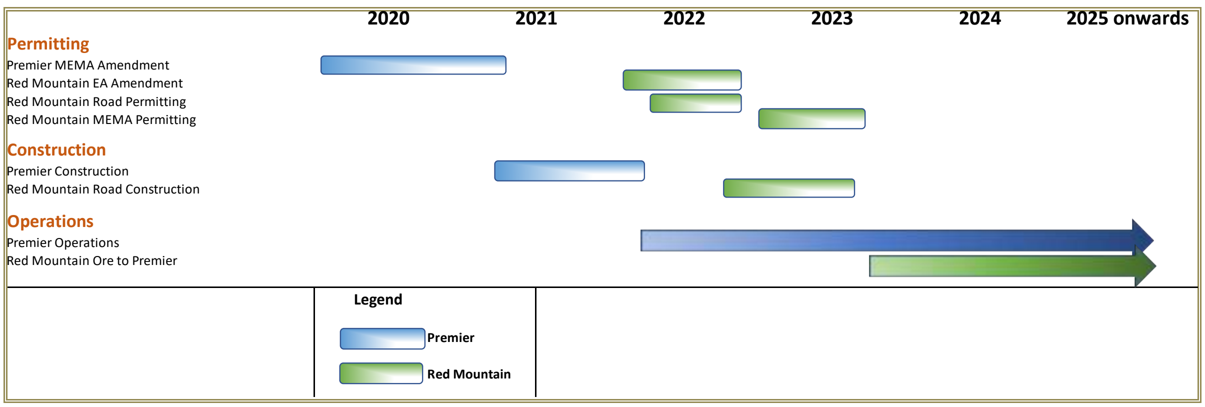


Figure 20-1: Permitting Schedule

20.6 Considerations of Social and Community Impacts

Ascot is committed to meaningful, timely, and transparent engagement and consultation with Aboriginal Groups, community members, stakeholders, and the public. Ascot will maintain this commitment throughout the proposed development, construction, operation, and closure of the project.

The nearest community is the town of Stewart, a town of approximately 400 people, according to the 2016 census. Other stakeholders include the town of Hyder, Alaska, tenure holders (such as trapline holders, guide outfitters, and independent power producers), local and regional governments, and government regulatory agencies.

20.6.1 Consultation

PGP is located in the Nass Area and RMP is located in the Nass Wildlife Area, as defined in the Nisga'a Final Agreement (2000), a modern treaty between the federal government, provincial government, and Nisga'a Nation, which sets out Nisga'a Nation's rights under Section 35 of the Canadian Constitution Act. Nisga'a Nation's Treaty rights under the Nisga'a Final Agreement include: establishing the boundaries and the Nisga'a Nation's ownership of Nisga'a Lands and Nisga'a Fee Simple Lands; water allocations; the right of Nisga'a citizens to harvest fish, wildlife, plants and migratory birds; and the legislative jurisdiction of the NLG. Nisga'a citizens have Treaty rights to manage and harvest wildlife in the Nass Wildlife Area and to harvest fish, aquatic plants, and migratory birds within the Nass Area. The clarity and certainty provided by the Nisga'a Final Agreement, including Chapter 10, which sets out the required processes for the assessment of environmental effects on Nisga'a Nation Treaty rights from projects such as this one, is a major advantage to development.

Ascot and the Nisga'a Nation signed a Benefits Agreement for the Red Mountain Project in March 2019.

The Project is also within the asserted traditional territories of Tsetsaut Skii km Lax Ha (TSKLH) and Métis Nation BC (MNBC).

20.6.2 Economic Impacts

The Project is expected to provide economic benefits to the local communities as a result of direct training and employment opportunities. It is anticipated that the Project will employ 100 workers during construction, and 300 full-time workers during operations. The overall economic impacts to the District of Stewart, as well as nearby communities and the province, are expected to be beneficial due to an increase in employee and company expenditures.

Additional indirect employment opportunities, such as goods and services contracts, will increase, creating growth in the local and regional economies. The Project will also generate annual revenues associated with property tax, licensing fees, royalties, and income tax for local, provincial, and federal governments.

Through the implementation of an Impact Benefits Agreement with the Nisga'a Nation, Ascot is committed to maximizing employment of Nisga'a citizens. The hiring process will be open, merit-based, and competitive; however, within this context Ascot will take a proactive stance for the hiring of local and Nisga'a Nation jobseekers. Recruitment will include early communication activities and development of employment policies and programs to encourage recruitment.

Ascot will provide competitive compensation and benefit packages for all workers consistent with mining industry standards in BC.

The goal is to encourage non-resident hires to relocate to Stewart on a full-time basis. Ascot will work closely with the District of Stewart, service providers, and local businesses to ensure adequate housing, amenities, and recreational facilities are available to help attract and retain new workers to the community.

20.7 Environmental Management

Ascot has developed an Environmental Management System (EMS) that defines the processes by which compliance will be met and demonstrated. The EMS will include ongoing monitoring and reporting to relevant parties at the various stages of the Project.

In gathering data to achieve scientific consensus, Ascot conducted extensive research to establish baseline data and, where data were not available, incorporated examples from other similar, established operations. Ongoing consultation with Aboriginal Groups, communities, and local stakeholders ensures that local knowledge will be fully evaluated and incorporated to support the goal of achieving scientific consensus.

Where there is uncertainty or some plausible risk, conservative approaches, together with a dynamic process of adaptive management, will be implemented. A flexible approach will be supported by the design of monitoring programs to address all areas of uncertainty, provide a process for mitigation, and to further support the ongoing collection of scientific data.

Ascot's commitments to environmental protection and sustainable development will be achieved through the following actions. Ascot will:

- Seek to minimize the potential adverse effects of Ascot's operations on the environment using technology and ecologically conscious decision-making and best practices. This includes respectful consultation with Aboriginal Groups, local communities, government regulators, and stakeholders.
- Comply with and abide by all applicable environmental legislation and best practices, with legal compliance considered the minimum standard of business operation.
- Support the development, implementation, and continuous improvement of a comprehensive EMS and provide all Ascot employees, consultants, contractors, and subcontractors with the information, training, and equipment they need to fully understand and implement the EMS.
- Incorporate the principles of sustainable development into all levels of planning, decision-making, and implementation.
- Choose environmentally preferable products and services as much as possible.
- Support initiatives that promote the long-term socioeconomic, heritage, human health, and ecological well-being of the communities where Ascot is active.

All Ascot employees, as well as anyone acting on behalf of Ascot, are responsible for acting in accordance with this policy, regardless of the jurisdiction in which they are working.

20.8 Water Management

Water management will be a critical component of the Study. As such, Ascot has developed site water-management plans that address mining activities undertaken during all project phases. Management plan goals include: providing and retaining water for mine operations; providing a basis for managing freshwater on the site; avoiding harmful impacts to fish and wildlife habit; and managing water to ensure that discharges comply with the applicable water quality levels and guidelines.

Tailings and waste rock were characterized as having potential for metal leaching/acid rock drainage (ML/ARD), under certain conditions, over extended periods of time. Tailings process water is expected to contain residual metals and ammonia from destruction of cyanide solutions. As further detailed in Chapter 18, the Project incorporates appropriate design features and mitigation measures consistent with best practices for waste and water management to address these issues, including:

- A WTP at PGP to treat effluent from underground workings and the TSF.

- At RMP, water management infrastructure (i.e., ditches, etc.) to collect runoff associated with the temporary development waste rock stockpile, and to manage the groundwater discharged from the mine.
- Backfilling of all underground development rock into the underground mines as part of the mining process.
- Appropriate handling and management practices for all water and waste generated at the site.

Potential contact water sources consist of underground mine development drainage and reclaim from the TSF. Non-contact water from the upstream catchment above the TSF will be routed around the TSF through several diversion channels.

Water stored in the TSF will be reclaimed to the mill for reuse as process water through a reclaim barge in the pond. The results of the water balance indicate that the TSF will operate in surplus conditions, which will be managed by removing surplus supernatant water to the WTP and then discharging to Cascade Creek after treatment. Seepage will be monitored for water quality through the Seepage Monitoring Manhole and will be discharged to Cascade Creek. Seepage can be pumped back to the TSF if required or if it fails to meet discharge water quality requirements.

Site contact water will either be routed to the TSF via a system of diversion ditches or conveyed to the WTP. Underground dewatering will be pumped to a collection system reporting to the WTP and then discharged to Cascade Creek.

20.9 Mine Closure

A detailed Closure and Reclamation Plan has been developed and will be refined as part of the provincial permitting process. Four broad closure objectives have been developed for this Project. They are: design the mine for closure; achieve long-term physical stability; achieve long-term chemical stability; and consider future land-use objectives and aesthetics.

During the Closure Phase, each of the underground portals will be evaluated for individual amenability to portal closure. Any potentially acid-generating mine tailings left on surface will be stored in the TSF. The supernatant pond will be drained, and a cover consisting of rock and soil will be placed over the facility to achieve long-term chemical stability of the underlying tailings and shed runoff to the final closure spillway.

The structures will be decommissioned and removed from the site upon completion of mining. All explosives, explosive magazines, fuel, and fuel containers will also be removed from the site at closure. Footings and retaining walls will be broken down to grade and taken apart and any concrete fragments will be buried. After removal of the process building, equipment, and foundations, a soil-sampling program will be conducted to determine if there are any contaminants in the immediate vicinity. Building footprints (whether bedrock or pads) will be re-contoured to allow for restoration of natural drainage to the receiving environment.

The underground mines will be backfilled using unconsolidated waste rock in the secondary stopes and cemented rock fill in the primary stopes. Backfilling operations will consume all new waste rock brought to surface. Existing, historical, waste rock will remain in place and will be revegetated as applicable.

Bridges will be removed from deactivated mine roads. Additionally, all culverts will be removed from these roads and cross-ditched for drainage. Growth media will be spread on the road surface and the roads will be revegetated as required.

Groundwater monitoring wells and all other geotechnical instrumentation will be retained for use as long-term dam-safety monitoring devices. Post-closure requirements will also include annual inspection of the former TSF, and ongoing evaluation of water quality, flow rates, and instrumentation records to confirm design assumptions for closure. The WTP will remain in operation in perpetuity, or until such time as water quality is deemed acceptable by the BC Ministry of Environment.

The cost of closure and reclamation has been estimated at approximately $23.5 million and is detailed in the Capital and Operating Cost section of this study (Section 21).

21 CAPITAL AND OPERATING COSTS

To complete this estimate Ascot Resources Limited (Ascot) engaged a team of consultants led by Sacré-Davey Engineering Inc. (SDE). SDE was responsible for overall coordination, infrastructure and the economic evaluation; INNovexplo Inc.(INN) and Mine Paste Ltd. (Mine Paste) for mining; Sedgman Canada Limited (Sedgman) for metallurgy and processing; Knight Piésold Ltd. (KP) for tailings and surface water management; SRK Consulting (Canada) Inc. (SRK) for the water treatment plant; Paul Hughes Consulting Ltd. (PHC) for site geotechnical; McElhanney Ltd. (MEL) for access roads; Prime Engineering (PE) for the Electrical substation; Palmer Environmental Consulting Group Inc. (Palmer) for geochemistry, hydrology and water quality modelling; and Marsland Environmental Associates Limited for environmental studies.

21.1 Capital Costs

The Project is focused on developing the existing Premier Gold Project (PGP) (placed in care and maintenance in 1996) and the Red Mountain Project (RMP). PGP is located in northwestern BC, 25 km east of the port community of Stewart in the Golden Triangle area of British Columbia. The study contemplates refurbishing existing infrastructure to current regulatory standards where possible or installing new infrastructure where needed. The mine was declared closed in 2001.

This Feasibility Study considers development of underground mines and repair or upgrade of existing infrastructure and the PGP processing facilities to process the resources from four key deposits: Premier, Big Missouri (BM), Silver Coin (SC), and RMP. Three of the deposits—Big Missouri, Silver Coin, and Premier—are at PGP, and the fourth deposit is located at the RMP, approximately 23 km to the southeast of the PGP mill. RMP will feed the PGP mill approximately two years post start-up of Silver Coin and Big Missouri mill feed.

A consolidated capital cost estimate was prepared for the Project based on underground mine operations with a process plant with a capacity of 2,500 tonne per day (t/d); supporting infrastructure, new primary 138 kV substation, site power distribution, tailings storage facility (TSF), and water management; and a new water treatment plant (WTP). The estimate describes the methodologies and sources of information applied to prepare an estimate that meets the American Association of Cost Engineers (AACE) Class 3 requirement of an accuracy range between -15% to +15%. The purpose of this section is to summarize the capital costs and describe the methods, assumptions and exclusions used in developing the estimate.

These cost estimates are expressed in Q1 2020 Canadian dollars ($), with no allowances for price escalation or currency fluctuations. All values are shown in Canadian dollars unless otherwise noted.

The capital cost estimate was divided into initial and sustaining capital. For the estimate, capital expenditures (CAPEX) were identified as initial capital for all costs before production. All capital expenditure incurred on commencement of production through to the end of the mine was categorized as sustaining capital. The mine closure cost is included in the sustaining capital.

21.1.1 Capital Cost Estimate

Total project costs are presented in Table 21-1, indicating the area of responsibility by consultant for each work breakdown structure (WBS).

The initial capital cost distribution and the initial and sustaining direct cost distributions are shown in Figure 21-1 and Figure 21-2, respectively.

Table 21-1: Total Project Capital Cost Summary by Area

WBS	Description	Total Cost ($ '000s)		
		Initial	Sustaining	LOM Total
1 Direct Costs		**100,036**	**161,229**	**261,311**
1000	Mining/Dewatering	14,019	110,183	124,202
2000	Overall Site Development	8,187	1,378	9,565
3000	Mineral Processing	35,637	10,266	45,903
4000	TSF	8,659	4,580	13,240
4500	Site Wide Surface Water Management	7,016	4,695	11,711
4900	Closure and Reclamation	0	20,500	20,500
5000	On-Site Infrastructure	14,038	0	14,038
5800	WTP	12,480	0	12,480
6000	Off-Site Infrastructure	0	9,672	9,672
2 Indirect Costs		**30,457**	**9,392**	**39,849**
9000	Project Indirect Costs	30,457	9,392	39,849
3 Owner's Costs		**3,663**	**204**	**3,867**
9800	Owner's Costs	3,663	204	3,867
4 Contingency		**12,443**	**6,690**	**19,133**
9900	Contingency	12,443	6,690	19,133
Total Project Costs		**146,600**	**177,515**	**324,160**

Detailed reports of the capital and sustaining cost estimates are included in Appendix E-1.

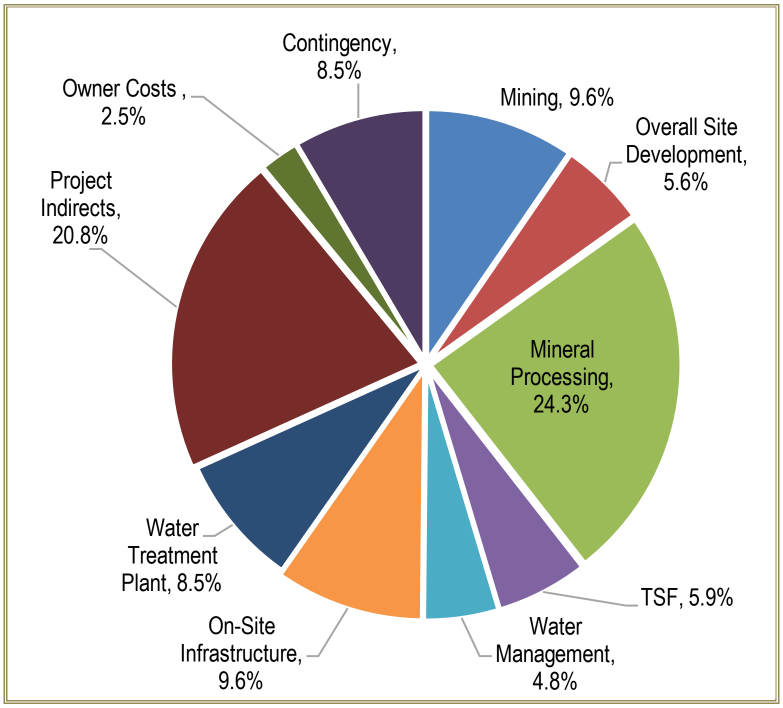


Figure 21-1: Initial Capital Cost Distribution

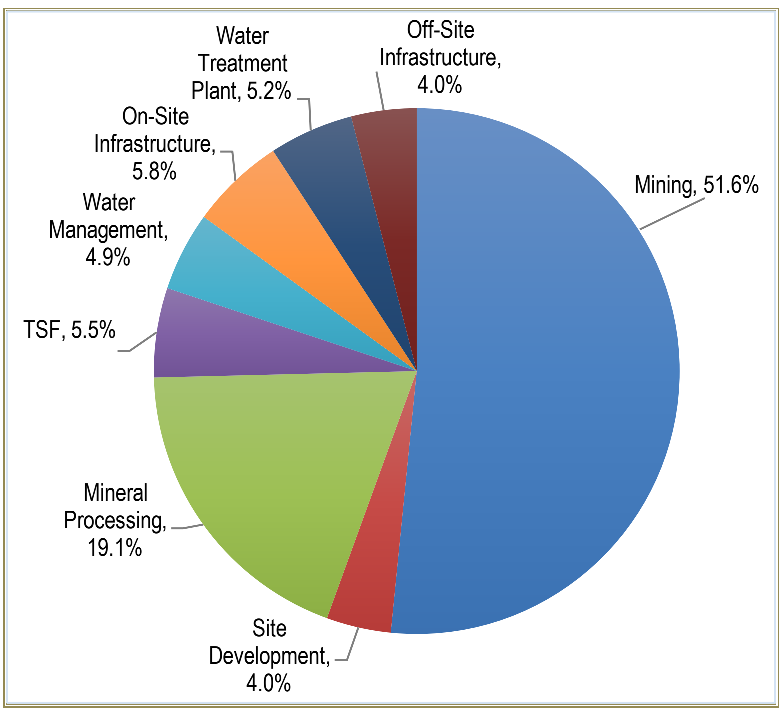


Figure 21-2: Initial and Sustaining Direct Cost Distribution

21.1.2 Basis of Estimate

The consultants' basis of estimates (BOEs) are presented in Appendix E-2. The following is a high-level summary of information contained in detail in Appendix E-2.

The consultants who assisted in contributing to the estimated capital costs responsibilities are as follows:

- Mine Paste and INN—initial capital, and sustaining underground mining at PGP and RMP, including surface infrastructure related to heating and ventilation for all the deposits, and fuel for generated electrical power at RMP.
- Sedgman—metallurgy and processing layout, and general arrangements, PGP plant upgrade, including process and mechanical equipment, process plant electrical distribution, instrumentation and controls, and piping.
- Knight Piésold—tailings and surface water management, incorporating upgrades to the TSF including modification of water management structures, additional material added to the embankment to flatten the slopes to meet current codes, and installation of new tailings distribution and reclaim water systems. Non-contact water diversion structures located upstream of the TSF will be upgraded to minimize flood routing through the TSF. Site surplus water and underground dewatering will be directed to the new WTP for treatment as required prior to release.
- McElhanney—upgrades to existing access roads and stream crossing structures to the PGP site, including access roads to the Big Missouri, Silver Coin, and Premier underground portals.
- SRK—WTP, clarifier, and associated moving bed biofilm reactor (MBBR) facilities.
- SDE—upgrading the on-site infrastructure, including the non-process ancillary facilities and buildings; site services; building services; on-site electrical distribution to the concentrator building; Big Missouri, Silver Coin and Premier underground portal. Off-site infrastructure constructing the access road to RMP.

The overall capital cost estimate developed in this Feasibility Study meets the American Association of Cost Engineers (AACE) Class 3 requirement of an accuracy range between -15% and +15% of the final project cost:

- The estimate was produced using Excel and units of measurement are in the metric system
- A WBS structure developed by Ascot for the Project was used to structure the Feasibility Study estimate
- Cost information was assigned to all the WBS areas and each area was broken down further into subsequent levels detail.

To allow for proper cost breakdown and reporting, each estimate line item has an associated commodity code.

Commodity codes are used to collect the estimate items into groups of work of a similar nature or discipline (Figure 21-3).

The estimate is categorized into five cost types represented by five columns in the estimate. The cost types are presented in Table 21-2.

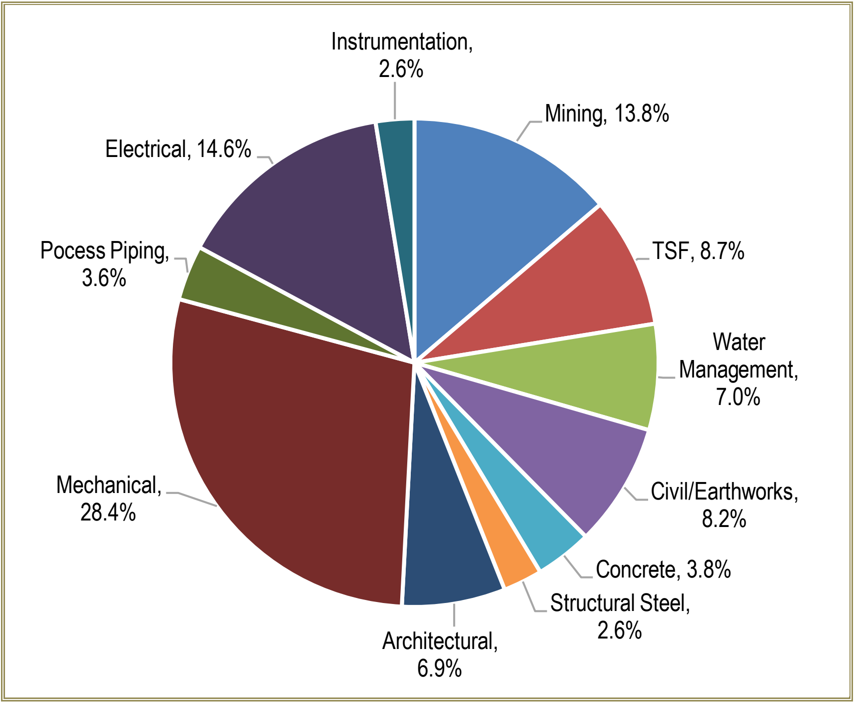


Figure 21-3: Initial Capital Commodity Cost Distribution

Table 21-2: Cost Types Initial Capital

Description	Initial Estimate ($ '000s)
Labour	22,659
Material	21,972
Construction Equipment	8,089
Equipment	32,317
Sub-Contract	61,561
Total	**146,598**

The cost type definitions are as follows:

- Labour—on-site installation hours to perform the task
- Material—material required for the task (e.g., engineering fill, concrete, structural steel, bulk items)
- Construction Equipment—the minor construction equipment required to install or complete the task (e.g., excavators, man lifts)
- Equipment—all equipment costs related to the mining, process (e.g., jumbos, LHDs, haul trucks, mine ventilation, crushers, ball mill, and transformers)
- Sub-Contract—catch all category to pick up costs that cannot be split into labour and material (e.g., sub-contractor quotations and lump sum quotations).

21.1.3 Major Cost Types

Direct Costs—includes for the supply and construction of all permanent project facilities

Indirect Costs—includes costs that are not readily allocated to individual direct cost items. These costs include both, time-based (driven by the Project schedule) and fixed costs. The following are typical items to be included with indirect costs:

- Common distributable costs (e.g., temporary construction facilities and services, construction equipment, commissioning spares, freight, insurance, and EPCM services)
- Owner's costs.

Owner's Costs—Costs to be incurred by the Owner (this cost is developed by Ascot) during the pre-production phase of the Project.

Contingency—A cost allowance, to cover necessary work within the defined scope of the Project that cannot be identified or itemized at this stage of the Project development.

The cost estimate base date is Q1 2020.

BOEs were prepared by SDE, Sedgman, KP, and SRK, and are presented in in Appendix E-2. Except for SRK all the consultants adhered to the Project specific labour rates and productivity presented in Section 21.2.1.

The SRK BOE is presented in Appendix E-2, and is an AACE Class 4 estimate. Their estimate is based on in-house experience and similar projects.

Work Breakdown Structure

The WBS BOE was compiled based on the WBS compiled by SDE, presented in Appendix E-3.

The WBS was used by all contributing parties; it is a hierarchical system based on primary tags (major, area, sub-area) and secondary tags (commodities).

Currency Exchange Rate

The estimate is prepared in Canadian dollars (C$). Any quotes received in foreign currency are converted to Canadian dollars based on the following exchange rates shown in Table 21-3.

Table 21-3: Foreign Currency Exchange Rates

Currency	Currency Code	Exchange Rate
United States Dollar	US$	US$1 = C$1.3179
Australian Dollar	AU$	AU$1= C$0.8899
China Yuan Renminbi (CNY)	¥	¥1= C$0.1900
Euro Member Countries (Euro)	€	€1= C$1.4509
South African Rand (ZAR)	R	R1= C$0.0903

Estimating Methodology

The starting point of the Feasibility Study cost estimate for the PGP site was the engineering assessment conducted by CGT Industrial Ltd. (CGT) and Sedgman on the existing processing facility to establish its working condition. New or revised scope was provided with an updated mechanical equipment list in Q4 2019.

The 2017 Feasibility Study Technical Report for RMP was used for the access road and the RMP mining estimate.

The consultants prepared their estimates in line with their BOE.

Direct Costs

The direct costs in the estimate have been grouped by commodity as shown in Figure 21-3. Quantities and costs were developed from a combination of budget quotes, database pricing, engineering take-offs, and factoring. Budget quotes for major equipment (>70% equipment value) were obtained from vendors using project-specific performance specifications.

Labour Rates, Hours, and Productivity

Table 21-4 presents the trade or craft labour all-in crew rates used in the capital cost estimate.

Table 21-4: Trade All-In Crew Rates

Trade	All-In Crew Rate ($/h)
Civil/earthworks	97.50
Concrete	101.50
Structural Steel	109.00
Mechanical and piping	109.50
Electrical and Instrumentation	110.00

The direct labour rate components include:

- Taxable wage rate
- Holiday pay
- Statutory holidays
- WCB
- CPP
- Insurance
- Health, safety, and PPE
- Pensions
- Small tools
- Consumables
- Supervision
- Overhead and profit.

The details of the above calculation can be found in Appendix E-4.

Transportation, catering, construction camp, and operations costs are not included in the all-in labour rates. These costs were calculated separately and included in project indirect/Owner's costs outlined in this section.

Labour is assumed and anticipated to be readily available from the surrounding area. Predominantly from Stewart and Terrace.

Generally, a productivity factor of 1.15 is applied to the base hours. Typical productivity factors include project, site, labour, and weather specific factors, as well as number of shifts, amount of mechanization, and complexity of the work.

For the processing plant scope a brownfield productivity factor of 1.25 was applied to items tagged as brownfield installation to reflect the higher complexity of these items. A lower brownfield productivity factor of 1.125 was applied to items tagged with brownfield removal.

Area 1000—Mining

The mining cost estimate for the underground mining was developed by INN and handed over to Mine Paste for final approval and inclusion in this NI 43-101.

- The total capital and operating costs were estimated from first principles using a standard cost model constructed by INN.
- In addition to information in INN databases, quotes were obtained for all major equipment and consumables.
- Contingencies were calculated based on the type of cost (multiple quotes, single quote, or factored).
- After discussion with Ascot, it was decided the 10% contingency, based on development meterage, would only be applied to the preproduction development capital cost.

Area 3000—Processing Facilities

The PGP process costs are based on:

- Capital costs assume that construction will occur in two stages: the first stage will be to repair and upgrade the existing processing plant to process PGP ore, and the second stage will be to modify the Stage 1 plant to process both PGP and RMP ores. The second stage costs are included in the sustaining capital.
- Contingency assessed for the processing plant was developed using a Monte Carlo analysis via a risk review. Through the risk review, individual aspects of the plant design, the layouts, the sourcing of materials and all other key aspects for the capital cost build-up were reviewed by the team involved as well as independent technical experts to identify key risk ranges; then those were compiled to form a single contingency value.

Area 4000—Tailings and Surface Water Management

The basis for the PGP tailings and surface water management costs are:

- Labour rates and productivity were in line with the Project rates
- Contractor fleet was employed for all earthworks

- All equipment productivities and capacities from the *Caterpillar Performance Handbook* (Caterpillar, 2018)
- All rental rates of equipment referenced from the BC Road Builders and Heavy Construction Association's Equipment Rental Rate Guide, the *2019–2020 Blue Book*, unless otherwise stated.

Civil costs were excluded from the estimate (e.g., pads for transformers and motor control centres [MCC]), as they are assumed covered within the overall civil scope for adjacent features where the transformers and MCCs would be situated.

Unit rate data was obtained from the following, in order of preference:

- Recent vendor quotes
- In-house resources
- General cost textbooks/catalogue (RSMeans *2020 Heavy Construction Costs Book* (RS Means, 2019)).

Quantities for mechanical unit rates were developed by measuring lengths and head differentials for all pipelines using AutoDesk AutoCAD 2019 design software. Base topography was provided by Ascot (2 m contour intervals) and was used for all pipeline and water management systems estimates (KP, 2020c).

Area 5000—On-Site Infrastructure and Area 6000—Off-Site Infrastructure

The BOE for the Project on-site and off-site infrastructure costs are presented in Appendix E-2.

Area 5700—Site Roads

Cost estimates for the Granduc Road, Tailings Pond Road, and Silver Coin Roads are based on standard construction practices in NE BC. For the purposes of the estimate a unit price costing methodology was utilized. The BOE for the road infrastructure costs are:

- Quantity take off from Autodesk Civil 3D 2019 models
- Recent vendor quotes for specified unit prices
- When vendor quotes were not provided, unit prices for other nearby projects with similar scope were utilized
- It was assumed that all work would be completed by single contractor in the summer on two construction headings
- The scope of each unit item was considered in costing to prevent scope overlap.

21.1.4 Mine Capital Costs

The major components of the mine capital costs consist of mobile and stationary equipment and mine development. Multiple quotes were obtained from equipment lessors quotes were obtained for the rest. Construction estimates were made for the ventilation, power and dewatering infrastructure and included quotes for major systems.

21.1.5 Underground Capital Cost Summary

Table 21-5 shows a summary of the capital cost over the life of the Project for each site and includes both lateral and vertical development, other mine infrastructure as well as non-leased equipment. A total $6.1 million of operating costs incurred at PGP in the preproduction period have been capitalized, the remaining $114.7 million at both sites is considered sustaining capital.

Table 21-5: Mine Capital Expenditures

Capital Expenditures	LOM Capital C$M
PGP	
Mine Infrastructure (C$)	1.1
Lateral/Vertical Development (C$)	57.1
Mobile & Stationary Equipment - non lease (C$)	28.5
Capitalized Operating Cost (C$)	6.1
Subtotal Premier Site	92.9
RMP	
Mine Infrastructure (C$)	1.6
Lateral/Vertical Development (C$)	15.5
Mobile & Stationary Equipment - non lease (C$)	10.9
Subtotal Red Mountain Site	28.0
Total PGP & RMP Capital Expenditures	120.8

Mobile and Stationary Equipment Costs

The quantity, unit cost, and total cost of the mobile equipment required for the Project was based on a multipurpose fleet that will be transferred as required between the various deposits based on the mining requirements. Quantities of were calculated based on productivity factors for equipment and activities to achieve the required daily production targets supply tonnage to the mill at 2,500 t/d.

Table 21-5 shows the major mobile and fixed equipment required by the mine. The majority of the mobile fleet is assumed to be leased-to-own over a five-year term, with a lease rate of 7.5% and monthly payments included in the operating costs.

A small existing fleet consisting of a 2-boom jumbo, two 10-tonne LHDs and three 15 tonne mine haul trucks is currently situated at RMP. Tier 4 engines have been purchased for this fleet to meet current mine standards.

Major equipment such as mine haul trucks and LHDs that are subject to typical intense underground duty, have varying operating life. Table 21-6 shows typical operating hours for the mine equipment. Regular maintenance costs have been incorporated into the operating costs including minor overhauls. Due to the relatively short project operating life, there has been no allowance for major overhauls or rebuilds. If the mine expanded some of the earlier equipment would likely be overhauled, but for the study case, it was assumed that the equipment would be managed to the end of its useful life.

Table 21-6: Major Mobile and Stationary Equipment

ITEM	PGP Max Purchased	RMP Max Purchased	Project Total		VALUE
10t LHD	5	2	7	Lease	1,150,000
30t Minetruck	5	3	8	Lease	1,110,000
2 Booms Jumbo	3	0	3	Lease	1,212,000
Bolter	2	1	3	Lease	1,135,000
ITH Drill - Long Hole	2	0	2	Lease	912,000
Explosives Truck	2	2	4	Lease	505,000
Scissor Lift	2	2	4	Lease	345,000
Shotcrete Sprayer	1	1	2	Lease	26,000
Personnel Carrier	1	1	2	Lease	412,000
Lube Service Truck	1	1	2	Lease	405,000
Boom Truck	1	1	2	Lease	385,000
Motor Grader	1	1	2	Lease	290,000
Utility Vehicle	1	1	2	Lease	115,000
Backhoe with Rockbreaker	1	1	2	Lease	155,000
Telehandler	1	1	2	Lease	131,000
Mechanics Truck	2	2	4	Lease	75,000
Toyota PC	5	5	10	Lease	61,000
Jumbo drifter - spare	1	1	2	Purchase	130,000
Electrical & Comm Equipment	0.25	0.75	1	Purchase	25,964,899
Silver Coin temporary dewatering	1		1	Purchase	347,000
Main Pumping System (lump sum)	1	1	2	Purchase	944,501
Drop Raise Equipment	1	1	2	Purchase	89,500
Jackleg	8	8	16	Purchase	8,250
Stoper	8	8	16	Purchase	8,250
Plugger	4	4	8	Purchase	7,000
Chipper	4	4	8	Purchase	3,400
Main Ventilation System	0.17	0.83	1	Purchase	3,337,330
Fan 15 HP & accessories	3	3	6	Purchase	10,000
Fan 60 HP & accessories	6	3	9	Purchase	26,000
Fan 100 HP & accessories	6	4	10	Purchase	47,000

Equipment purchases were scheduled in the mine plan to account for the planned equipment life and the potential to use equipment at the different mining deposits during the eight-year mining period. Equipment was standardized across the sites to minimize spares inventories.

Contractor haulage from RMP to the PGP mill was assumed.

Table 21-7: Equipment Operating Hours

Major Production / Development Equipment	Average Operating Hours
10t LHD	5500
30t Minetruck	5500
2 Boom Jumbo	3500
Bolter	2500
ITH Drill - Long Hole	5000
Explosives Truck	2000
Scissor Lift	2000
Shotcrete Sprayer	2000
Personnel Carrier	2000
Lube Service Truck	2000
Boom Truck	2000
Motor Grader	3000
Utility Vehicle	2000
Backhoe with Rockbreaker	1999
Telehandler	2000
Mechanics Truck	2000
Toyota PC	2000

21.1.6 Process Plant Capital Costs

Figure 21-5 is a 3-D rendering of the PGP process site layout, indicating both existing infrastructure (in blue) and new construction (in yellow) to receive mill feed from RMP in Q4 Year 2.

Source: Sedgman (2020a)

Figure 21-4: 3-D Rendering of the PGP Process Site Layout

The capital costs developed for the refurbishment and upgrades required to bring the processing facilities back into an operating state were developed from first principles utilizing a detailed 3-D scan of the existing processing facility and a field condition assessment of the plant which was carried out in September 2019 involving both a local constructor and Sedgman.

The capital cost estimate for the process plant is based on process flow diagrams (PFD), drawings, and scope definition, and includes all the areas as laid out in Section 17.

The processing plant initial capital costs are $55.5 million with a sustaining capital cost of $15.4 million (excluding Closure Costs) for a planned expansion in operating Year 2.

Table 21-8 shows the itemized breakdown of capital cost.

Table 21-8: Itemized Process Plant Initial Capital Cost Estimate (Sedgman's Scope, only)

Area	Initial Capital Cost ($ million)	SDE Scope ($ million)	Total Initial Capital Cost ($ million)
Crushing area	1.30	0.03	1.33
Coarse ore storage and reclaim	0.95		0.95
Grinding	9.76		9.76
Gravity circuit	1.28		1.28
Carbon-in leach	2.83		2.83
Gold recovery and carbon regeneration	5.04	0.13	5.17
Tailings thickener	1.83		1.83
Cyanide destruction and tailings pumping	1.21		1.21
Plant building and services	10.88	0.40	11.28
Directs Subtotal	**35.08**	**0.56**	**35.64**
Construction indirect costs	6.84	SDE indirect costs not shown.	
Spares	0.26		
Initial fills	0.15		
Freight and logistics	1.42		
Commissioning and start-up	1.10		
EPCM	6.50		
Indirect Costs Subtotal	**16.27**		
Contingency	**4.20**		
Total Estimate	**55.5**		

21.1.7 PGP Infrastructure Capital Costs

Area 5000 on-site infrastructure direct capital cost is presented in Table 21-9.

Table 21-9: Infrastructure Direct Cost Breakdown

	Initial CAPEX ($ '000s)
Upgrade of existing non-process facilities and ancillary building	4,481
Process water, fire/freshwater/potable storage and distribution	1,399
On-site propane and fuel storage and site utilities	976
Sewage collection and treatment and solid waste management	1,323
Site roads	5,859
Total	**14,038**

The on-site infrastructure includes:

- Upgraded site and access roads (Big Missouri, Tailing Pond, and Silver Coin)
- Upgrading/replacing the administration facilities, mine dry, truck shop, and maintenance facilities
- Upgrading/replacing elements of the assay laboratory/cold storage building
- Waste water treatment systems
- Sewage treatment plant
- Solid waste disposal facilities
- Site services
- Fuel storage
- Propane
- Medical facilities
- Water supply.

21.1.8 Area 2000 New Substation, 138 kV Powerline and Electrical Site Distribution

Electrical power will be supplied from a 138 kV tap from the Long Lake Independent Power Producer (IPP) line to the proposed new substation. Electrical power will be stepped down to 4.16 kV and distributed to the mill and mine portals (Table 21-10).

Table 21-10: Electrical Power Supply and Distribution Direct Cost

	Initial CAPEX ($ '000s)
Power supply and distribution	2,871
138 kV Main Substation	5,316
Total	**8,187**

This area includes:

- 138 kV power supply from Long Lake to the main substation
- Construction of a new main substation
- 4.16 kV power distribution to the mill and the mine portals.

Budgetary quotations were received for the new substation and associated 138 kV powerline from Long Lake IPP to the mill site.

The 4.16 kV electrical distribution was estimated from in-house data.

21.1.9 Tailings and Surface Water Management

Initial capital cost estimated for tailings and surface water management is summarized in Table 21-11 for the following areas:

- TSF earthworks
- TSF tailings distribution system
- TSF reclaim water system
- TSF surplus water system
- WTP settling ponds earthworks
- Cascade Creek Diversion Channel expansion
- Site diversion ditches construction
- Underground mine portal surface water management
- Construction water management
- Site water management pumps and pipelines
- Construction indirect costs.

Table 21-11: Summary of Tailing / Water Management Estimates

Area	Initial Capital ($ '000s)
TSF earthworks	5,730
TSF tailings distribution and reclaim systems	2,930
Site water management	7,016
Project indirect costs	3,160
Contingency	2,107
Total	**20,943**

21.1.10 Water Treatment Plant

The planned water treatment facilities for the PGP includes the design, construction, and commissioning of a WTP, including a MBBR process for removing ammonia, cyanate, and other nitrogen species and an HDS lime plant for removal of dissolved metals and TSS. The capital cost estimate for the water treatment facilities for this Feasibility Study was developed according to the following basis:

- Estimate accuracy—+30%/-15% (AACE Class 4)
- Base currency—Canadian Dollars and assumes an exchange rate of C$0.75 to US$1 for US vendor equipment pricing
- Estimate base date—Q1 2020
- All freight for equipment, modularized skids, and commodity materials are assumed to be shippable via existing waterways and roads.
- Labour rates include general contractor profit.

The estimate is developed using installation factors against identified process equipment costs to predict the overall direct field installation cost (DFC). Indirect costs are calculated as a percentage of DFC. Equipment is priced from vendor budgetary quotes and has an accuracy range of ±30%. The estimate was prepared for a British Columbia project location using a historical location factor of 1.15 as measured by the operator through execution of projects in this location.

The estimated capital costs for the SRK water treatment facilities (excluding concrete foundations estimated by SDE) are $18.8 million, which includes an overall contingency of 16.3% of the whole estimate. Maintenance and repairs are included in the operating cost estimate, but long-term sustaining capital estimates were not developed and are not included in the estimate. Additional details concerning the BOE is included in Appendix E-2.

21.1.11 Indirect Costs

Table 21-12 shows a summary of indirect costs of the initial capital cost estimate, which includes temporary construction facilities, construction equipment, construction camp, personnel flight costs, freight, vendor representatives, first fills, initial spares, and general engineering, procurement, and construction management (EPCM) costs. The indirect costs were estimated by each individual consultant.

Table 21-12: Summary Initial Indirect Costs (excluding contingency)

	CAPEX ($ ‘000s)
EPCM	11,911
Construction indirect costs	12,858
Spares	325
Initial fills	433
Freight and logistics	2,637
Commissioning and start-up	2,137
Vendors assistance	156
Total	**30,457**

Temporary Construction Facilities

Temporary construction facilities, allowance was included in the estimate (e.g., water management, construction office, perimeter fencing, and portable power and heating).

Construction Equipment

Construction equipment was estimated to include mobile equipment (e.g., man lifts, forklifts, scaffolding, and rental of major crane equipment).

Construction Services

Construction services include vendor representatives, medical services, surveying, geotechnical services, snow removal and others.

Construction Camp

The construction camp estimate includes an allowance for 100 beds. Camp costs were estimated at $100/person/day based on recent quotes and Ascot experience. Costs were agreed upon and communicated to each consultant supplying estimate inputs as part of their indirect costs. This estimate carries the cost accommodations, catering, ablution, power, and mobilization. An allocation for demobilization was made under Owner's costs.

Flights for Construction Labour Workforce

Personnel flights are based on a three-week turnaround and estimated by each consultant's estimation of labour sources.

Freight

Freight costs have been calculated as a percentage of equipment and material supply cost. It is assumed that all leased equipment costs are inclusive of freight costs, and divided into imported and locally purchased items.

Major equipment vendors used barging to Stewart and then transported the equipment to site. This was the primary method of transportation with several exceptions.

Vendor Representative Assistance

Vendor representative assistance is required during construction to verify that the equipment installation is in compliance with Vendors technical documentation.

An allowance for these costs was estimated based on the equipment costs.

It is assumed that the cost of Vender assistance is included in any equipment leasing agreement.

First Fills

An allowance for the first fills of lubrication and transformer oils was included for the start-up of operations.

Engineering, Procurement, and Construction Management

Process EPCM costs were assessed on the number of deliverables. Other EPCM costs were also factored.

Spare Parts

An allowance of $230,190 was made for an initial spares inventory for process plant, which was based on 20% of the estimated annual spares.

21.1.12 Owner's Initial Capital Costs

The Owner's costs for implementing the Project scope have been prepared and estimated by Ascot at $3.7 million. A reclamation bond of $15 million was posted for the Project in 2019. Table 21-13 shows what is included as Owner's costs.

Table 21-13: Owner's Costs

Description	Y-2	Y-1	Total ($ '000s)
PP Owner's project team labour	242	1,427	1,669
HR recruiting and training	-	78	78
Camp and catering	-	300	300
Insurance	-	409	409
IT and communications	-	20	20
Legal and audit fees (external)	-	100	100
Community relations	-	22	22
Environmental and permitting	-	125	125
Site care and maintenance	11	415	426
Office and miscellaneous expenses	-	56	56
Operational readiness	-	140	140
Operation and maintenance manuals	-	80	80
Expenses (flight and accommodation)	6	64	70
Transportation light vehicles	-	50	50
Commissioning and start-up	-	10	10
Health and safety	-	9	9
Consulting fees	60	40	100
Total	**319**	**3,344**	**3,663**

21.1.13 Contingency

The contingency was estimated by each individual consultant. The combined contingency is $12,4 million for the Project.

21.1.14 Qualifications

Crushed rock backfill is assumed to be free issue (excluding crushing) for road and civil platform construction. However, each contractor has included for haulage from the aggregate crushing plant to their work area. (The aggregate crushing plant will be resident on site for the blasting activities and subsequent crushing of the rock required for the Cascade Creek diversion. The QP determined that sufficient crushed rock will be available to meet all construction requirements). Costs for crushing all required material were including in the tailings pond costing.

21.1.15 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- Land acquisition and right-of-way
- Definition drilling, assaying, and related reports and models
- All studies prior to notice to proceed
- Fees or royalties relating to use of certain technologies or processes
- Project financing and interest charges
- Force Majeure
- Sunk or deferred costs
- Working capital (included in financial model)
- Venture capital (included in financial model)
- Permits or fees associated with construction
- Performance bond premiums
- All taxes and duties
- Removal of hazardous material
- All facilities not included in the capital cost estimate
- Cost of environment and ecology related items including:
  - Environmental and ecological considerations other than those incorporated in the current design
  - Impact caused by modifications directed by governing authorities, including schedule
  - Environmental testing and monitoring after mechanical completion
- Inclement weather conditions
- Schedule acceleration
- Local workforce competitive markets
- Credits for salvage value of any demolition, modification work, residual construction materials, vehicles, and temporary buildings
- Currency fluctuations
- Cost to identify, locate, remove, or relocate existing underground obstructions or utilities
- Escalation beyond Q1 2020.

Any additional exclusions are specified in their respective BOEs.

21.2 Sustaining Capital

Table 21-14 presents the sustaining capital over the 8-year life-of-mine (LOM), mill, tailings, and water management.

Table 21-14: Sustaining Capital Summary, Years 1–11

Year	Sustaining Capital ($ million)											Total Sustaining Capital ($ million)
	LOM (8 years)										Closure	
	1	2	3	4	5	6	7	8	9	10	11	
Mining	23.45	23.41	9.69	8.85	19.61	18.22	6.93	0.03	-	-	-	110.18
Overall site development	-	1.38	-	-	-	-	-	-	-	-	-	1.38
Mineral processing	10.27	-	-	-	-	-	-	-	-	-	-	10.27
TSF	0.79	-	0.96	-	2.83	-	-	-	-	-	-	4.58
Site wide water management	1.81	-	0.76	-	2.13	-	-	-	-	-	-	4.69
Closure and reclamation	-	-	-	-	-	-	-	-	-	-	20.46	20.46
On-site infrastructure	-	-	-	-	-	-	-	-	-	-	-	-
WTP	-	-	-	-	-	-	-	-	-	-	-	-
Off-site infrastructure	2.86	6.81	-	-	-	-	-	-	-	-	-	9.67
Project indirect costs	5.15	0.83	0.94	0.40	1.02	-	-	-	-	-	1.05	9.39
Owner's costs	-	-	-	-	-	-	-	-	-	-	0.20	0.20
Contingency	1.70	0.72	0.26	-	0.72	-	-	-	-	-	3.30	6.69
Total Sustaining	**46.02**	**33.14**	**12.61**	**9.25**	**26.31**	**18.22**	**6.93**	**0.03**	-	-	**25.01**	**177.52**

21.2.1 Mining

The majority of the underground project will be developed with company crews using leased equipment, with planned development start up three months prior to the mill commissioning to meet stockpiling and commissioning tonnage requirements.

Short-term development directly related to stope production is considered operating cost, while main ramps or development for infrastructure such as for ventilation, pumping or electrical reticulation is considered sustaining capital. The LOM sustaining capital is distributed as follows:

- Lateral and Vertical Development – 63.3%
- Mine infrastructure (refuge, vent, electrical, pumping) – 2.4%
- Non-leased mobile equipment and stationary equipment – 34.4%.

21.2.2 Process Plant

The mine plan is based on processing of RMP ore through the PGP processing facilities starting Q4 Y2.

To process this different ore body as described in Section 17, there are a number of PGP process plant upgrades needed to address the additional grinding requirements for the harder RMP ore. As defined in Section 17, the present design and capital cost estimate takes into account these requirements to avoid installing equipment that would be later deemed not suitable.

Section 24 presents the Project's milestones outlining detailed design, procurement, construction, and commissioning milestones of the process facility and infrastructure. The key changes to the facilities include the installation of a regrind mill, as well as modifications to the leaching and final gold processing circuits.

Sustaining capital breakdown by area is presented in Table 21-14.

Table 21-15: Sustaining Capital Breakdown by Area

Area	Capital Cost ($ million)
Crushing	-
Ore storage and reclaim	-
Grinding	7.44
Recovery circuits	2.66
Cyanide detoxification and tailings	-
Plant building and services	0.17
Directs Subtotal	**10.27**
Construction indirect costs	1.81
Spares	0.01
Initial fills	0.04
Freight and logistics	0.21
commissioning and start-up	0.32
EP	1.67
Indirect Costs Subtotal	**4.06**
Contingency	**1.05**
Total	**15.38**

The expected duration of the planned upgrade is approximately 18 months on the basis that the re-grind mill (12 months lead time) is procured upfront in the planning phases of the upgrade.

The costs are built up on the basis that the construction activities would be completed as much as practical in the working plant requiring a brief shutdown late in the upgrade program to tie-in the additional equipment and processing circuits.

21.2.3 Tailings Storage Facility and Surface Water Management

Sustaining capital costs for tailings and surface water management include the following items:

- Earthworks for ongoing TSF embankment raises (staged raises in Years 1, 3, and 5 of operations)
- Additional quarrying at the Cascade Creek Diversion Channel to generate material for embankment construction

- Underground mine portal surface water management
- Construction water management
- Site water management pumps and pipelines
- Construction indirect costs.

21.2.4 Off-Site Infrastructure

Sustaining for off-site infrastructure include costs for the RMP access road.

21.3 Closure Costs

Mill, TSF, water management, and infrastructure closure estimates were prepared as of the date of this report.

The closure cost estimate based on these estimates is summarized in Table 21-16.

Table 21-16: Closure Cost

Description	Closure Cost ($ '000s)
Mining (surface infrastructure)	150
Process building	7,334
TSF and TSF embankment seepage collection pond	11,845
Access roads	655
On-site infrastructure	476
Directs Subtotal	**20,500**
Owner's costs for road closure	204
Project indirect costs	1,051
Contingency	3,300

Detailed reports of the Closure Cost Estimate are included in Appendix E-1

21.3.1 Mining

Specific closure details for the mines have not been discussed with the appropriate regulatory bodies. Guidance on closure requirements are expected during the permitting phase. Additionally, it is the experience of Ascot, INN, and Mine Paste that the sale of equipment from the mine would likely offset the closure costs of the mine.

The estimate shown in Table 21-13 covers the demolition and removal of the propane facility and the mine heating and ventilation systems.

21.3.2 Processing Plant

The closure costs for the processing plant facility have been based on the dismantling, or disposing of the equipment, structures, and electrical components that exist within the processing plant envelope. The estimate was developed by assessing the dismantling of the facilities and disposal where appropriate, the costs developed were assessed using a factor of installation costs for each item used to determine dismantling costs. A similar approach was applied to determine the delivery charges for off-site disposal of scrap materials post the equipment removal.

Table 21-17 provides the summary for the processing facility closure costs (processing plant scope only).

Table 21-17: Closure Cost

Item	Estimate Cost ($ '000s)
Process facility dismantling and delivery to disposal sites	7,334
Management of the works	587
Contingency for the works	721
Total	**8,642**

21.3.3 TSF and Surface Water Management Infrastructure

Closure costs for the TSF and surface water management infrastructure will include for:

- Selective discharge of tailings around the facility prior to closure to establish a final tailings beach that will facilitate surface water drainage and reclamation
- Removal of the supernatant pond
- Dismantling and removal of the tailings, surplus water and reclaim water systems, as well as all pipelines, structures, and equipment not required beyond mine closure
- Placement of a combined rock and soil cover that will achieve long-term chemical stability of the underlying tailings and shed runoff to the final (Stage IV) TSF closure spillway
- Decommissioning of all access roads, ponds, ditches, and borrow areas not required beyond mine closure
- Long-term stabilization and vegetation of all exposed erodible materials.

21.3.4 On-Site Infrastructure

Closure costs were estimated based on:

- New security gatehouse
- Fresh water tank
- Truck wash bay
- Buried services
- Propane storage
- WTP
- Fresh, fire-water pump house
- Intake freshwater pump house
- Incinerator

- Fuel storage
- Ancillary facilities inside the concentrator building
- Cold storage / assay building
- Electrical distribution and 138 kV Substation
- Access roads (by McElhanney)
- General plant site coverage with crushed rock.

21.3.5 Off-Site Infrastructure

The access road to the RMP facilities Upper Portal will be rehabilitated but the Lower Portal road will not be rehabilitated. No cost impact for closure was considered.

21.4 Operating Costs

Table 21-18 presents the unit operating cost summary.

Table 21-18: Unit Operating Costs

	Operating Costs ($/t milled)
Mining	97.00
Processing	31.05
General and Administration (G&A)	7.93
Site Services	3.36
Total Cost	**139.34**

Detailed Operating Cost Estimate is presented in Appendix E-1.

21.4.1 Operating Cost Input Parameters

Table 21-19 presents the operating unit costs.

Table 21-19: Operating Unit Costs

Parameter	Unit	Cost
Diesel Price	$/L	1.00
Power Fuel	$/L	1.00
Power Cost	$/kWh	0.066
Propane Cost	$/L	0.495

21.4.2 Mine Operating Costs

Underground Operating Cost Summary

The LOM mine operating cost for the combined operations was approximately $97/t milled as shown in Table 21-19. A. At PGP direct and indirect costs were about $74/t and $17/t, respectively, for a total site cost of approximately $91/t. At the RMP site the total cost was $106/t, consisting of $84/t of direct costs and $22/t of indirect. RMP costs are higher for two reasons: Diesel genset power, and additional cost for ore haulage from RMP to the PGP mill.

Mine infrastructure operating costs were estimated by INN, comprising:

- Ventilation power and equipment
- Dewatering power and equipment
- Mobile equipment power supply
- Mobile equipment diesel fuel
- Mine air heat, propane fuel.

Table 21-20: Underground Operating Cost by Site

Operating Cost	LOM OpCost C$M	OpCost C$/t Processed
PGP Direct		
Development	50.4	13.88
Production	62.2	17.14
Surface Haulage	40.5	11.14
Backfill	34.9	9.61
Equipment Lease	21.8	6.00
Maintenance	29.4	8.08
General Expenses	29.0	7.99
Direct Total	268.2	73.85
PGP Indirect		
Manpower	48.3	13.31
Equipment	8.0	2.19
Fuel	5.0	1.38
Indirect Total	61.3	16.87
PGP Total Opex	329.5	90.72
RMP Direct		
Development	12.2	4.80
Production	32.6	12.82
Surface Haulage	47.6	18.70
Backfill	37.3	14.67
Equipment Lease	10.7	4.20
Maintenance	25.4	9.99
General Expenses	47.4	18.61
Direct Total	213.2	83.79
RMP Indirect		
Manpower	44.1	17.34
Equipment	7.6	2.97
Fuel	4.8	1.87
Indirect Total	56.4	22.18
RMP Total Opex	269.7	105.97
PGP & RMP Direct		
Development	62.6	10.14
Production	94.9	15.36
Surface Haulage	88.1	14.26
Backfill	72.2	11.69
Equipment Lease	32.5	5.26
Maintenance	54.8	8.87
General Expenses	76.4	12.37
Direct Total	481.4	77.94
PGP & RMP Indirect		
Manpower	92.4	14.97
Equipment	15.5	2.51
Fuel	9.8	1.58
Indirect Total	117.7	19.06
PGP & RMP Opex	599.2	97.00

Equipment Operating Costs

The equipment operating costs for major equipment were obtained from supplier quotes and INN databases. Operating costs include wear and maintenance parts, lube and oil, and tires. The total costs included fuel and power as appropriate to each piece of equipment.

Underground mining equipment usage costs are based on the equipment operating hours required to meet the LOM plan. Equipment usage costs are shown in Table 21-3.

Table 21-21: Equipment Operating Costs

Unit	Operating C$/hr	Fuel C$/hr	Power C$/hr	Total cost C$/hr
10t LHD	91.83	32.00	-	123.83
30t Minetruck	74.77	47.00	-	121.77
2 Boom Jumbo	105.78	2.75	6.18	114.71
Bolter	109.16	2.25	1.69	113.10
ITH Drill - Long Hole	92.68	1.50	2.75	96.93
Explosives Truck	46.76	2.25	2.06	51.07
Scissor Lift	42.99	9.00	-	51.99
Shotcrete Sprayer	57.89	2.25	2.52	62.66
Personnel Carrier	45.56	17.00	-	62.56
Lube Service Truck	39.22	17.00	-	56.22
Boom Truck	39.02	17.00	-	56.02
Motor Grader	12.60	23.00	-	35.60
Utility Vehicle	5.92	14.00	-	19.92
Backhoe with Rockbreaker	10.80	10.00	-	20.80
Telehandler	9.70	18.00	-	27.70
Mechanics Truck	8.75	8.00	-	16.75
Toyota PC	8.75	8.00	-	16.75

Underground Mine Workforce

The workforce was estimated based on the requirements of mine activities, shift needs and mobile equipment fleets, utilizing industry standard ratios for maintenance personnel, and input from INN databases and Ascot benchmarks against other underground gold mines in Canada. The workforce consisted of both staff and hourly labour, with costs for each position shown in Table 21-22. The annual costs were calculated on base salaries and hourly rates, with markups for bonus, overtime, and standard benefits.

Two rotation schedules were utilized, five days on and two days off (5/2) for management positions, and four days on four days off (4/4) for most technical, maintenance and operating personnel. It has been assumed that the workforce will reside in the town of Stewart and will be company employees.

Underground mining staffing levels are directly related to production activities and were built up based on the productivities (work-hours) required for mining activities occurring within a given time period based on two shifts per day.

Table 21-22: Mine Personnel Cost

Position	Schedule	Yearly Hours hrs	Total cost C$/yr
Superintendant	5/2	2,086	263,250
Captain	4/4	2,190	227,500
Supervisor	4/4	2,190	179,400
Jumbo/Drill/Bolter Operator	4/4	2,190	169,752
Blaster	4/4	2,190	139,503
LHD Operator	4/4	2,190	132,528
Truck Operator	4/4	2,190	122,065
Services operator	4/4	2,190	122,065
Miner	4/4	2,190	129,040
Maintenance Manager	5/2	2,086	227,500
Maintenance Planner	4/4	2,190	149,500
Maintenance supervisor	4/4	2,190	149,500
Electrician	4/4	2,190	146,478
Mechanic	4/4	2,190	146,478
Helper	4/4	2,190	104,627
Technical Services Manager	5/2	2,086	260,000
Mine Engineer	5/2	2,086	164,450
Mine Technician	4/4	2,190	104,650
Surveyor	4/4	2,190	149,500
Production Geologist	5/2	2,086	149,500
Geology technician	4/4	2,190	134,550
Health and Safety Coordinator	5/2	2,086	179,400
Training Coordinator	5/2	2,086	149,500

Consumables

The unit costs for the main consumables used in the study are shown in Table 21-23. Costs of explosives, diesel, cement, and power were determined through supplier quotes and were provided by Ascot. Additional consumables including ground support, additional blasting supplies, drilling wear parts, pipes, and ventilation ducting were generated from the INN database for Canadian operations.

Table 21-23: Supply Cost

SUPPLIES DATA Item	Cost C$/unit	Unit
Fuel	1.00	liter
Propane	0.60	liter
Power - Grid	0.06	kWh
Power - Genset	0.24	kWh
Cement	285.00	t
2 Boom Jumbo	57.56	hrs
Bolter	18.34	hrs
ITH Drill - Long Hole	54.57	hrs
10T Lhd	-	hrs
30T Minetruck	-	hrs
Explosives Truck	-	hrs
Scissor Lift	-	hrs
Hand Drill Consummables & Maintenance	3.20	drilled m
Ground Support 4.0X4.0	89.09	m of advance
Ground Support 4.5X4.5	157.17	m of advance
Ground Support 4.5X6.0	292.44	m of advance
Ground Support Drop Raises	198.94	m of advance
Installed Cables	7.31	drilled m
Emulsion	1.85	kg
Detonator Nonel	3.32	unit
Detonator	6.10	unit
Development Explosives Accessories	6.15	unit (ls)
Accessories	0.05	t
Bit Grinding Wheels	84.50	day
Flexible Vent Tubing And Accessories	35.00	m of advance
Dewatering Line & Accessories	80.00	m of advance
Water Line & Accessories	60.00	m of advance
Compressed Air Line & Accessories	60.00	m of advance
Mechanics Supplies	442.00	day
Electrical Supplies	26.00	day
Survey Supplies	97.50	day
Manway Furnishing	1,040.00	m

Lateral and Vertical Development

The majority of the underground project will be developed with company crews using leased equipment. Table 21-24 shows the average cost per metre for the various heading sizes at each site. It should be noted that although heading sizes are standardized for the same equipment at both sites, the RMP site uses diesel gensets while PGP is on the grid, contributing to slightly higher costs at RMP.

Short-term development directly related to stope production is considered operating cost, while main ramps or development for infrastructure such as for ventilation, pumping or electrical reticulation is considered capital.

Table 21-24: Unit Development Costs at Both Sites

Development Cost	Direct C$/m	Total C$/m
PGP		
4.0x4.0	1,613.01	3,837.01
4.5x4.5	1,878.49	4,102.49
4.5x6.0	2,157.54	4,381.54
Dropraises 3.5x3.5	1,508.08	3,732.08
RMP		
4.0x4.0	1,636.38	4,280.38
4.5x4.5	1,907.57	4,551.57
4.5x6.0	2,183.57	4,827.57
Dropraises 3.5x3.5	1,508.08	4,152.08

Table 21-25 details the breakdown of direct costs per metre for a typical 4.5 m x 4.5 m development heading at PGP, costs at RMP are marginally higher based on genset power. Approximately 26% of the cost is for equipment operation, while 44% represents consumables and the remaining 30% is labour. To come to the total cost per metre shown previously in Table 21.6 the indirect costs of the mine covering items such as supervision, technical, ventilation, pumping and other services must be added. These mine indirect costs are distributed over all mine activities.

Table 21-25: Detailed Cost Breakdown for Development

PGP 4.5 x 4.5 heading	C$/m	
Equipment		26%
2 Boom Jumbo	151.91	
Explosives Truck	24.78	
Bolter	108.37	
Scissor Lift	36.11	
10t LHD	171.72	
Subtotal	492.89	
Consumables		44%
Emulsion	209.79	
Detonator Nonel LatDev	51.46	
Development Explosives accessories	95.33	
Ground support 4.5x4.5	157.17	
Flexible Vent Tubing And Accessories	35.00	
Dewatering Line & Accessories	80.00	
Water Line & Accessories	60.00	
Compressed Air Line & Accessories	60.00	
Fuel	67.39	
Power - Grid	8.27	
Subtotal	824.41	
Labour		30%
Operators	561.19	
TOTAL	1,878.49	

21.4.3 Processing Plant Operating Costs

The process operating costs were developed from first principles using operating experience across the world in combination with local execution strategies. The target accuracy of the estimate meets the AACE Class 3 -15%/+15% requirements and was broken into the following major components:

- Labour
- Consumables (including mill/crusher liners and chemical reagents)
- Power
- Maintenance.

The operating costs BOE for the processing facility presents the operating costs in $/t of ore feed to the processing facility. The operating costs presented are split to reflect processing exclusively the PGP ore for the first two years, and, once introduced, processing a combination of RMP ore and PGP ore, which will be campaigned through the plant in 2-week campaigns of each ore to meet the average 2,500 t/d production target on a 365 d/a basis. The operating costs for each type of ore is presented in Table 21-26.

Table 21-26: Operating Cost for Each Different Ore Campaign Type

Area	Years 1 & 2 PGP Ore Only ($/t)	Q4 Year 2 + RMP and PGP Ores ($/t)
Labour	7.41	7.41
Power	3.13	3.83
Consumables		
Reagents	13.15	15.03
Liners	0.74	0.96
Media	1.90	2.82
Maintenance	0.88	1.15
Total	**27.22**	**31.47**

Due to the different processing requirements of each of the ore types, in Year 3 the overall operating costs will change due to the increased grinding and consumable requirements for processing the RMP ore.

Plant Labour

The manning schedule for the Project is developed in conjunction with Ascot to ensure the plant will be operated and maintained in a safe and operable manner. The manning levels reflect the following:

- Senior roles (e.g., Superintendent, Metallurgists) are based on 5-days-on, 2-days-off roster, 40-h weeks
- Trades and operators are based on 14-days-on, 14-days-off, 12-h shifts.

Table 21-27: Operating Cost – Labour Roles Defined

Position	Number of Personnel
Mill Superintendent	1
Chief Metallurgist	1
Plant Metallurgist	2
Operations General Foreman	0
Chief Assayer	0
Metallurgical Technician	1
Operations Shift Foreman	4
Total Mill Salaried	**9**
Mill Operations	
Roaming Operator	0
Crusher Operator	2
Grinding / Gravity Operator	8
Leach / CIP Operator	4
Strip / Regeneration Operator	2
Gold Refining Operator	2
Labourers	4
Total Mill Operations	**22**
Laboratory	
Lab Managers	2
Laboratory Assayer	4
Lab Sample Prep/Trainees	4
Total Laboratory	**10**
Shared Maintenance	
Maintenance Superintendent	1
Total Shared Maintenance	**1**
Mill Maintenance	
Maintenance Planner	1
Millwrights	6
Electricians	6
Process Control Technician	0
Welders	2
Instrument	2
Apprentice	4
Total Mill Maintenance	**21**
Total Head Count	**63**

Operating Consumables

Operating consumables constitute grinding media, mill and crusher liners, and reagents. No allowance is made for minor consumables like lubrication oils and greases.

Consumable cost rates were developed from quotations.

Consumables usage were developed from the testwork and process flowsheet information.

Mill liner costs have been based on recent liner pricing received from mill vendors during package pricing, applied at estimated intervals expected for mills in this application. Grinding media costs based on recent prices received from media suppliers.

Table 21-28: Operating Cost – Grinding Media Breakdown

Grinding Media	Consumption (g/t)	Annual Media Consumption (t/a)	Media Size (mm)
PGP Ore Treatment			
SAG Mill	400	183	75
Ball Mill	800	365	50
IsaMill	0	0	3.5
Subtotal – PGP		548	
RMP ore treatment			
SAG Mill	400	183	75
Ball Mill	800	365	50
IsaMill	300	137	3.5
Subtotal – RMP		684	
Total Media Costs		**1231.88**	

Reagents

The breakdown shown in Table 21-29 includes the expected reagents and consumptions.

Table 21-29: Operating Cost – Reagents Usage

Reagents	Dosage (g/t)	Annual Use (t/a)
PGP Ore Treatment		
Description		
Sulfur	0	0
Sodium Metabisulfite (SMBS)	2,267	1,034
Lead Nitrate ($PbNO_3$)	0	0
Sodium Cyanide (NaCN)	1,780	812
Flocculant (Vanfloc 10)	25	11
Copper Sulfate ($CuSO_4$)	191	87
Lime ($Ca(OH)_2$)	3,220	1,469
Sodium Hydroxide (NaOH)	730	333
Antiscalant	0	0
Carbon	50	23
Hydrochloric Acid (HCl)	409	187
Diesel	0.323	295
	*L/t	*m^3/a
Subtotal PGP		
RMP Ore Treatment		
Description	g/t	t/a
Sulfur	0	0
Sodium Metabisulfite (SMBS)	2,167	989
Lead Nitrate ($PbNO_3$)	100	46
Sodium Cyanide (NaCN)	2,420	1,104
Flocculant (Vanfloc 10)	65	30
Copper Sulfate ($CuSO_4$)	381	174
Lime ($Ca(OH)_2$)	3,010	1,373
Sodium Hydroxide (NaOH)	1,020	465
Antiscalant	0	0
Carbon	50	23
Hydrochloric Acid (HCl)	613	280
Diesel	0.424	387
	*L/t	*m^3/a
Subtotal RMP		
Total Reagent Costs		

Power Costs

Power for the processing plant will be derived from the electrical grid supply. The quoted unit cost for the produced power is provided by Ascot at $0.06 /kWh and is based on initial indicated pricing from BC Hydro. The installed operating equipment power demand is determined from vendor data and equipment lists. Utilisation and load factors are then applied to the total to obtain power usage.

Table 21-30: Estimated Power Consumption

Description	Power Consumption	
	PGP Ore Estimated Power Consumption (MWh/a)	RMP and PGP Ore Estimated Power Consumption (MWh/a)
Crushing and Reclaim	983	983
Grinding	15,373	26,615
Gravity	527	0
Leaching	1,631	1,631
Elution	1,638	1,638
Gold Room	893	893
Cyanide Destruction and Tailings	935	935
Reagents	355	355
Water and Air Services	1,457	1,457
Total	**23,791**	**34,507**

Table 21-31: Operating Cost Basis of Estimate for the Processing Facility

Item	Basis
Labour	Labour cost rates were taken from recent project experience across several nearby or in-province projects. Labour quantities were developed from the scope of the facility and skills required. The labour levels reflect the following: • Senior roles (e.g., Superintendent, Metallurgist) are based on 5-days-on, 2-days-off roster, 40-h weeks • Trades and operators are based on 14-days-on, 14-days-off, 12-h shifts. The following staffing levels were assessed • 9 salaried positions • 22 operators • 10 lab staff • 1 maintenance superintendent • 21 maintenance trades.
Consumables	Consumable cost rates were developed from quotations, and usage was developed from the testwork and process flowsheet information. Mill liner costs have been based on recent liner pricing received from mill vendors during package pricing, applied at estimated intervals expected for mills in this application. Grinding media costs based on recent prices received from media suppliers.
Power	Power consumption is derived from the average power utilisation of all process equipment and was developed from the master equipment list.
Maintenance	Maintenance costs were derived from an assumed % of the direct capital costs. The assessed costs were then reviewed against experience from other Sedgman operations.
Reagents	Reagent operating costs have been priced based on design dosage rates and recent prices received from suppliers.
Mobile Equipment	The mobile equipment assessed includes the fleet for the following key tasks: • Managing the re-handle of the ores on the stockpile pad • General site vehicles for plant and maintenance purposes. Mobile equipment includes: • A front-end loader (CAT 980 or equivalent) • A skid steer loader • An integrated tool handler • Light vehicles.

21.4.4 Infrastructure Operating Costs

Table 21-19 presents the site infrastructure costs.

Table 21-32: Site Infrastructure Costs

Description	Total LOM Operating Costs ($ '000s)	Unit Cost $/t milled LOM Average
RMP road avalanche control (Y2+)	8,307	1.22
Infra power	699	0.10
Infra maintenance	578	0.09
Infra fuel (excludes mobile equipment)	80	0.01
Infra propane	1,605	0.24
Infra waste streams	825	0.12
Water treatment plant (by SRK)	9,537	1.40
TSF and surface water management (by KP)	1,168	0.17
Total Operating Cost	**22,800**	**3.36**
Total Operating Cost Average (LOM)	**2,850**	

Details of the Infrastructure Operating Costs are included in Appendix E-5. The annual site services are presented in Table 21-20.

Table 21-33: Annual Site Services Operating Costs*

	Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	LOM Total
RMP road avalanche control	$ '000s	-	-	0	616	1,282	1,282	1,282	1,282	1,282	1,282	8,307
Infrastructure power	$ '000s	-	-	108	108	108	108	108	108	54	0	699
Infrastructure maintenance	$ '000s	-	-	80	83	87	87	87	87	56	12	578
Infrastructure fuel (excluding mobile equipment)	$ '000s	-	-	10	10	10	10	10	10	10	10	80
Infrastructure propane	$ '000s	-	-	321	321	321	321	321	0	0	0	1,605
Infrastructure waste streams	$ '000s	-	-	103	103	103	103	103	103	103	103	825
Water treatment plant	$ '000s	-	-	1,192	1,192	1,192	1,192	1,192	1,192	1,192	1,192	9,537
TSF and water management	$ '000s	-	-	144	144	144	144	148	148	148	148	1,168
Infrastructure Total	**$ '000s**	**-**	**-**	**1,957**	**2,577**	**3,246**	**3,246**	**3,250**	**2,929**	**2,845**	**2,747**	**22,797**

Note: *Includes TSF, Water Management, and WTP. Totals do not add up due to rounding.

21.4.5 RMP Road Avalanche Control

The avalanche control operating costs were assumed to be expended annually, between October and April.

21.4.6 Power

The unit operating power costs is $0.066/kWh. The operating costs are based on the usage from the following facilities:

- Main substation and transmission lines
- Non-process office facilities, based on a percentage power cost per square metre (m^2) of each building

- Fresh, fire-water pumping
- Sewage treatment plant
- Site services based on energy usage per annum.

21.4.7 Maintenance

As shown in Table 21-21, maintenance costs per annum have been included based on a percentage of the initial capital cost. Refer to Appendix E-5 for details.

Table 21-34: Maintenance Costs per Annum

System	Initial Capital Cost ($ '000s)	Allowance per Year (%)	Maintenance Cost per Year ($ '000s)
Incinerator	495	4	22
Plant and instrument air	129	3	4
Fire/freshwater storage and distribution	696	2	14
Potable water system	4	2	0.1
Site utilities	42	2	1

21.4.8 Diesel Fuel

Diesel fuel costs at $1.00/L are included in the operating costs for the following areas:

- Incinerator
- Gen sets at RMP (included in the mining operating costs)
- Mining mobile fleet (included in the mining operating costs)
- Plant and site mobile equipment (included in G&A costs).

21.4.9 Propane

Unit Propane costs is $0.495/L. The Propane costs have been included for the process building, the cold storage/assay building (including $12,000/a for propane equipment rentals), as shown in Table 21-35.

Table 21-35: Annual Propane Costs

Location	Propane Consumption (Imp gal/a)	Cost ($/gal)	Annual Cost ($/a)
Concentrator building (no process heat)	113,984	2.25	256,464
Laboratory Building	23,331	2.25	52,495
Subtotal HVAC Load	-	-	308,959
Propane Tank Rental	-	-	-
Propane Tank Rental (1 No. 30,000 USWG)	-	-	12,000
Propane Costs	-	-	320,959

21.4.10 Waste Streams

Annual costs have been included for solid and liquid waste, such as grey/black water (STP), food, and waste oil incinerator).

The operating cost for snow removal and leasing mobile equipment is included in Ascot's G&A costs.

21.4.11 Tailing Storage Facility and Surface Water Management Operating Costs

Operating costs for tailings and water management include electrical power costs to operate pump and pipeline systems and maintenance costs for the pump and pipeline systems.

21.4.12 Water Treatment Plant

Ongoing operating costs for the water treatment facilities are detailed in Appendix E-2. Annual operating costs, excluding labour, are estimated at $770,000 for the operations phase and the first five years of closure, and $460,000 starting in Year 6 after closure and in-perpetuity in post-closure.

Operating costs are based on current costs for quicklime delivered to site and current power costs. Cost for maintenance and repairs, as well as for operating supplies were factored based on equipment costs.

Operating cost assumptions include:

- Operating days are 365 d/a
- Mechanical availability is 92%
- Operating time is 24 h/d
- One full time operator equivalent for 24/7 operation
- Operating supervision costs are by the Owner
- Solids removed from the system will be disposed at the tailings facility
- Flocculant type and dose will be determined during bench-scale tests
- The WTP will be connected to electrical grid power, supplying electricity at a cost of $0.06/kWh
- Chemical reagent costs are based on current cost of quicklime delivered to site and on recent reagent quotes
- Annual maintenance and repair costs are estimated at 4% of the fixed capital investment
- Operating supplies is estimated at 15% of maintenance and repairs
- Insurance is estimated at 1% of fixed capital investment.

21.4.13 General and Administrative Operating Costs

General and Administrative (G&A) costs are expenses not directly related to the production of gold. These costs are based on Ascot's recommendation and experience.

The G&A costs for the Project are estimated to be $7.93/t. Yearly G&A costs are included for the following:

- Labour
- Health and safety, medical, and first aid
- Environmental
- Human resources
- Insurance and legal
- External consulting
- IT and communications
- Site maintenance
- Office and miscellaneous
- Mobile equipment (lease/fuel/maintenance).

Table 21-36 presents a breakdown summary of overall G&A costs.

Table 21-36: G&A Costs Summary

Area	LOM Cost ($ million)	LOM Unit Cost $/t milled
Labour	26.42	4.27
Health & Safety, Medical, & First Aid	1.69	0.27
Environmental	1.50	0.24
Human Resources	1.73	0.28
Insurance & Legal	7.21	1.16
External Consulting	0.40	0.06
IT & Communications	1.42	0.23
Site Maintenance	3.11	0.51
Office & Miscellaneous Costs	1.57	0.26
Mobile Equipment	3.95	0.64
Total	**49.00**	**7.93**

22 ECONOMIC ANALYSIS

22.1 Cautionary Statement

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates
- Assumed commodity prices and exchange rates
- Mine production plans
- Projected recovery rates
- Sustaining and operating cost estimates
- Assumptions as to closure costs and closure requirements
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed
- Unrecognized environmental risks
- Unanticipated reclamation expenses
- Unexpected variations in quantity of mineralized material, grade, or recovery rates
- Geotechnical and hydrogeological considerations during mining being different from what was assumed
- Failure of plant, equipment, or processes to operate as anticipated
- Accidents, labour disputes, and other risks of the mining industry.

22.2 Assumptions

The economic analysis assumes the Project is a 100% equity financed project. All dollar amounts in this analysis are expressed in Canadian dollars (C$), unless otherwise specified.

The economic analysis includes the entire Project life. The valuation date on which the net present value (NPV) and internal rate of return (IRR) are measured is the start of Year −2.

Details of the capital and operating cost estimates are described in Section 21. The production schedule used for the economic analysis is described in Section 16.

Base-case prices are derived from recent common peer usage discussed in this Section.

22.3 Introduction

The economic evaluation of the Project was carried out using a financial model developed by Ascot based on production schedule, application of operating, capital, and sustaining costs, as discussed earlier in this document. The financial model generated an estimate of annual pre-tax and post-tax cash flows and Ascot has undertaken an internal review of both pre-tax and post-tax economic evaluation models.

Key economic indicators were developed for both pre-tax and post-tax, and include the Project's NPV, IRR, and payback period (time in years to recover the initial capital investment once operations commence). The base-case results apply the following key inputs:

- Gold – US$1,400/oz
- Silver – US$17.00/oz
- Discount rate – 5.0%
- Exchange Rate – C$1.00 to US$0.76.

A discount rate of 5% has been applied to determine the base-case NPV of the Project.

Table 22-1 summarizes the base case results of the analyses performed.

Table 22-1: Analyses Base Case Results

Description	Unit	Value
Average Milling Rate	t/d	2,500
Mining	$/t milled	97.00
Processing	$/t milled	31.05
Site Services	$/t milled	3.36
G&A	$/t milled	7.93
Transportation and Refining	$/t milled	0.90
Royalties	$/t milled	15.30
Gold Recovery	%	91.4
Silver Recovery	%	76.5
Pre-Tax NPV 5%	$ million	516
Pre-Tax IRR	%	62
Pre-Tax Payback Period	years	1.7
Post-Tax NPV 5%	$ million	341
Post-Tax IRR	%	51
Post-Tax Payback Period	years	1.8

Sensitivity analyses were performed for variations in metal prices, gold grade, the Canadian to US dollar exchange rate, operating costs, and capital costs to determine the economic impact of changes in these variables on Project economics.

22.4 Methodology Used

Analyses were performed using a discounted cash-flow model prepared by Ascot. This model uses mid-year discounting at a base-case discount rate of 5%. Pre-production period is estimated to be two years. NPV and IRR are calculated based on the start of this two-year period. No provisions are made for inflation and/or future increases in costs, and analysis is presented in constant dollars. The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered production by calculating them from production head grades and tonnage, and mining and processing recoveries.

Metal revenues, principally gold, were calculated based on each scenario's prices and rates. Operating costs and offsite charges (refining, transportation, insurance, and royalties) were deducted from the gross revenues to calculate annual operating cash flow.

Initial and sustaining capital costs, as well as closure and reclamation costs have been incorporated on an annual basis over the life-of-mine (LOM) and deducted from the operating cash flow to determine the net pre-tax cash flow. Initial capital costs include costs accumulated prior to first production of concentrate, including all pre-production mining development costs.

Working capital costs are estimated based on two months of necessary capital needed (2 months of operating expenses) to fund operating costs prior to revenue starts in the first year of operation, and are constant every year.

Costs in the analyses exclude sunk costs (e.g., drilling costs, corporate overheads, permitting, government reclamation bond) prior to the commencement of Project construction. Future bonding estimates are not included in the analysis.

22.5 Pre-Tax Model

Table 22-2 shows the primary assumptions used in the base-case economic analyses.

Table 22-2: Primary Assumptions

Assumptions	Unit	Combined
Gold Price	US$/oz	1,400
Silver Price	US$/oz	17.00
Exchange Rate	C$/US$	0.76
Effective Royalty Rate	%	4.7
Payable Metals		
Gold Production	'000 oz	1,059
Silver Production Ounces	'000 oz	2,964
Mining		
Mine Life	years	8
Total Ore Tonnage Mined	**'000 t**	**6,177**
Processing		
Gold Recovery	%	91.4
Silver Recovery	%	76.5

Assumptions	Unit	Combined
Processing Throughput	t/d	2,500
Average Gold Grade	g/t	5.89
Average Silver Grade	g/t	19.72
Capital Expenditure (CAPEX) Costs		
Initial CAPEX	$ million	147
Sustaining Capital	$ million	157.1
Closure Costs	$ million	25
Operating Costs		
UG Mining Cost	$/t milled	97.00
Processing Cost	$/t milled	31.05
G&A Cost	$/t milled	7.93
Site Services	$/t milled	3.36
Total Operating Costs	**$ million**	**139.34**

Table 22-3: Base Case PGP and Red Mountain Combined Operation

Metric	Unit	Total
Ore Mined		
PGP	‘000 t	3,632
Red Mountain	‘000 t	2,545
Total	**‘000 t**	**6,177**
Payable Gold		
PGP	‘000 oz	601
Red Mountain	‘000 oz	459
Total	**‘000 oz**	**1,059**
Payable Silver		
PGP	‘000 oz	1,569
Red Mountain	‘000 oz	1,395
Total	**‘000 oz**	**2,964**
Average C1 Cash Cost	US$/oz Au	642
AISC	US$/oz Au	769
Combined		
EBITDA	$ million	1,034
Pre-Tax FCF	$ million	710
After-Tax FCF	$ million	472
Initial CAPEX	$ million	147
Sustaining CAPEX	$ million	178

Metric	Unit	Total
Pre-Tax NPV @5%	$ million	516
Pre-Tax IRR	%	62
After-Tax NPV @5%	$ million	341
After-Tax IRR	%	51

22.5.1 Smelting and Refining Terms

The Project will produce a doré that will be refined into high-purity gold and silver. A marketing study was not completed on the sale of doré from the Project. Refineries and secure transportation companies were contacted in an effort to estimate current market rates. Preliminary refining terms were informed by these discussions; in addition, an estimate of transportation costs was provided by a national armored transportation firm. More detail can be found in Section 19.

The study recommends that a marketing and logistics report be completed to confirm the accuracy of the terms as the Project progresses towards production.

22.5.2 Metal Prices

Metal prices and foreign exchange rates that Ascot uses are in line with both current technical reports, market rates, and long-term consensus estimates by large banks and brokerages. Flat real prices were assumed for the life of the Project. A sensitivity analysis was also performed by varying multiple financial and operating variables to see the effect that changes in these inputs have on the overall economics of the Project. The base case uses US$1,400/oz and the exchange rate of C$0.76 to US$1.

22.5.3 Operating Costs

The Project operating costs are summarized in Table 22-4 and can be described in detail in Section 21.

Table 22-4: Operating Costs

	Operating Costs ($/t milled)
Mining	97.00
Processing	31.05
General and Administration (G&A)	7.93
Site Services	3.36
Total Cost	**139.34**

22.5.4 Capital Costs

The Project capital costs are summarized in Table 22-5 and described in detail in Section 21.

Table 22-5: Capital Operating Costs

WBS	Description	Total Cost ($ '000s)		
		Initial	Sustaining	LOM Total
1 Direct Costs		**100,036**	**161,229**	**261,311**
1000	Mining/Dewatering	14,019	110,183	124,202
2000	Overall Site Development	8,187	1,378	9,565
3000	Mineral Processing	35,637	10,266	45,903
4000	TSF	8,659	4,580	13,240
4500	Site-Wide Water Management	7,016	4,695	11,711
4900	Closure and Reclamation	0	20,500	20,500
5000	On-Site Infrastructure	14,038	0	14,038
5800	WTP	12,480	0	12,480
6000	Off-Site Infrastructure	0	9,672	9,672
2 Indirect Costs		**30,457**	**9,392**	**39,849**
9000	Indirect Costs	30,457	9,392	39,849
3 Owner's Costs		**3,663**	**204**	**3,867**
9800	Owner's Cost	3,663	204	3,867
4 Contingency		**12,443**	**6,690**	**19,133**
9900	Contingency	12,443	6,690	19,133
Total Costs		**146,600**	**177,515**	**324,160**

22.5.5 Royalties

The Project is subject to various royalties on different portions/claims constituting the Project. For this economic analysis, it is assumed that Ascot buys back some of the existing royalties (as per the underlying legal agreements) after the completion of construction and in the first operating year.

The royalties purchased by Ascot are as follows:

- 5% net smelter return (NSR) on the Boliden Claims (Premier Option), repurchased for $9.55 million
- 5% NSR on the Boliden Claims (Dilworth Option) repurchased for $2.075 million
- 5% NSR on the Kasum Claims (Dilworth Option) repurchased for $2.075 million
- 1% NSR on the Kasum Claims (to R. Kasum and the estate of J. Wang) repurchased for $1.0 million.

After repurchasing these royalties, a 1% NSR and 5% net profit interest (NPI) royalty will remain and will apply to a majority of production; these costs have been incorporated in the economic analyses. On the Red Mountain Project (RMP) site, royalties consist of a 3.5% NSR (consisting of two separate royalties) and an obligation to deliver 10% of annual gold production from the Red Mountain property to Seabridge Gold Inc., up to a maximum LOM total of 50,000 oz in exchange for US$1,000/oz of refined gold.

On four claims that comprise a small portion of the Silver Coin deposit, there is an NSR royalty payable to Nanika Resources; however, an immaterial amount of production comes from those claims and this cost has been excluded from the analyses.

Over the LOM, approximately $94 million in royalties are payable, which includes the cost of 46,000 oz of payable gold produced at RMP, then sold to Seabridge at a contractual price of US$1,000/oz.

22.5.6 Working Capital

Ascot has included a general working capital provision of $13.7 million, which is built up in the first year of operations. This figure represents approximately two months of on-site operating expenditures in Year 1 (mining, processing, site services, and general and administrative [G&A] costs), and is fully recovered at the end of mine life.

22.5.7 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate, value of the Project economics. Ascot has developed a tax model, and assumptions of the tax model include:

- British Columbia provincial tax rate of 12% and a federal tax rate of 15%.
- British Columbia net current proceeds tax rate of 2%, while capital investment is being recovered and a net revenue tax rate of 13% applied against net revenue once the cumulative expenditure account (CEA) balance reaches zero.
- Tax pools and allowances applied:
  - Canadian exploration expense (CEE) and Canadian development expense (CDE) tax pools were used, as appropriate, in the calculation of taxes payable.
  - Capital cost allowance rates and associated claims were used, as appropriate, in the calculation of taxes payable.
- Total cash taxes for the Project are $238.2 million.

22.5.8 Closure Costs and Salvage Value

Closure costs are discussed in Section 21.4 and include mining costs. Zero allowance for salvage value was included in the cash flow model.

22.5.9 Financing

Cash flows were estimated on an annual basis and are shown on an unlevered 100% project basis. Certain operating equipment is assumed to be leased and this expense is included in the operating costs.

22.5.10 Inflation

The figures provided in this section are presented on a real basis and have not been adjusted for inflation.

22.6 Financial Results

The Project has an after-tax NPV at 5% (NPV 5%) of $341 million and an after-tax IRR 51%. The pre-tax payback period is 1.7 years, and the after-tax payback period is 1.8 years. Table 22-6 shows a summary of the annual cash flows and the cash flow model.

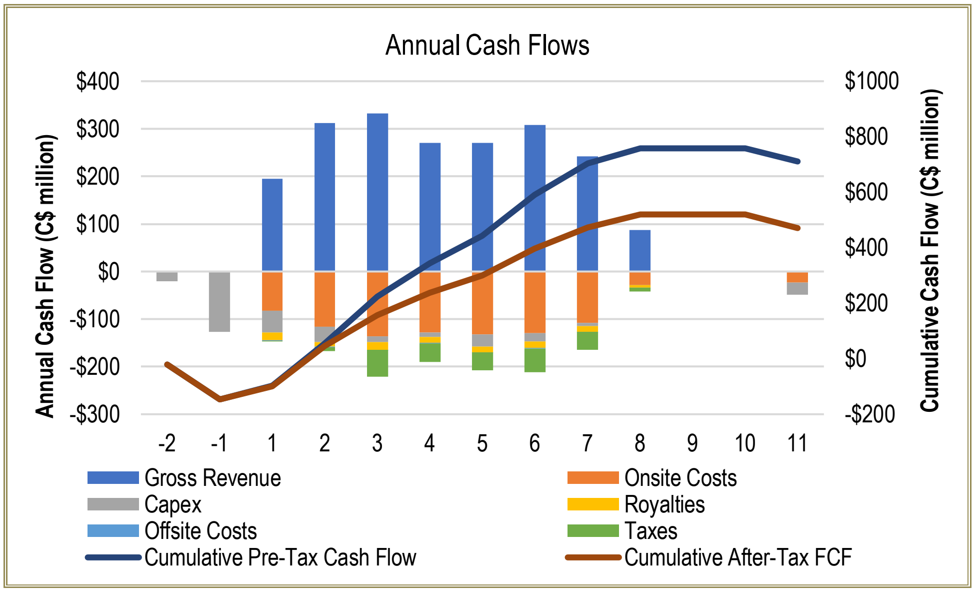


Figure 22-1: Annual Cash Flows

Table 22-6: Annual Cash Flows and Cash Flow Model Summary

ASCOT PREMIER PROJECT	Unit	Pre-Production Total	Production Total	Life Of Mine Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
METAL PRICES & EXCHANGE RATE																		
Metal Prices																		
Gold (Au)	US$/oz	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400
Silver (Ag)	US$/oz	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Exchange Rate	CAD/USD	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76
PRODUCTION SCHEDULE																		
PGP Ore	kt	9.0	3622.9	3631.9	0.0	9.0	610.9	810.3	478.2	462.6	459.9	461.5	339.5	0.0	0.0	0.0	0.0	0.0
Au Grade	gpt	6.79	5.45	5.45	0.00	6.79	5.60	5.87	6.20	4.89	4.80	5.30	4.98	0.00	0.00	0.00	0.00	0.00
Contained Au	koz	2.0	634.8	636.8	0.0	2.0	110.0	152.9	95.3	72.7	71.0	78.6	54.4	0.0	0.0	0.0	0.0	0.0
Ag Grade	gpt	10.14	19.13	19.11	0.00	10.14	11.60	12.17	16.83	13.62	26.20	23.70	44.24	0.00	0.00	0.00	0.00	0.00
Contained Ag	koz	2.9	2228.0	2231.0	0.0	2.9	227.8	317.1	258.8	202.5	387.3	351.6	482.9	0.0	0.0	0.0	0.0	0.0
RMP Ore	kt	0.0	2544.8	2544.8	0.0	0.0	0.0	86.2	419.6	437.3	438.7	435.6	434.2	293.2	0.0	0.0	0.0	0.0
Au Grade	gpt	0.00	6.52	6.52	0.00	0.00	0.00	8.94	7.24	6.19	6.19	6.98	6.02	5.83	0.00	0.00	0.00	0.00
Contained Au	koz	0.0	533.6	533.6	0.0	0.0	0.0	24.8	97.7	87.1	87.4	97.7	84.0	54.9	0.0	0.0	0.0	0.0
Ag Grade	gpt	0.00	20.60	20.60	0.00	0.00	0.00	42.44	25.81	21.96	16.53	21.50	16.33	15.75	0.00	0.00	0.00	0.00
Contained Ag	koz	0.0	1685.4	1685.4	0.0	0.0	0.0	117.6	348.2	308.8	233.2	301.1	228.0	148.5	0.0	0.0	0.0	0.0
PROCESSING SCHEDULE																		
Ore Processed	kt	0.0	6176.7	6176.7	0.0	0.0	615.0	901.4	897.8	899.9	898.6	897.1	773.7	293.2	0.0	0.0	0.0	0.0
Au Grade	gpt	0.00	5.89	5.89	0.00	0.00	5.63	6.16	6.69	5.52	5.48	6.11	5.57	5.83	0.00	0.00	0.00	0.00
Contained Au	koz	0.0	1170.4	1170.4	0.0	0.0	111.2	178.4	193.0	159.7	158.4	176.3	138.4	54.9	0.0	0.0	0.0	0.0
Ag Grade	gpt	0.00	19.72	19.72	0.00	0.00	11.59	15.05	21.03	17.67	21.48	22.63	28.58	15.75	0.00	0.00	0.00	0.00
Contained Ag	koz	0.0	3916.4	3916.4	0.0	0.0	229.1	436.3	607.0	511.3	620.5	652.8	710.9	148.5	0.0	0.0	0.0	0.0
RECOVERED METAL																		
Au	koz	0.0	1070.2	1070.2	0.0	0.0	104.6	167.4	176.4	143.3	142.7	162.7	126.4	46.7	0.0	0.0	0.0	0.0
Ag	koz	0.0	2994.4	2994.4	0.0	0.0	163.6	335.2	485.1	403.3	466.7	496.2	523.8	120.5	0.0	0.0	0.0	0.0
PAYABLE METAL																		
Au	koz	0.0	1059.5	1059.5	0.0	0.0	103.5	165.7	174.6	141.9	141.3	161.1	125.1	46.3	0.0	0.0	0.0	0.0
Ag	**koz**	0.0	2964.5	2964.5	0.0	0.0	162.0	331.8	480.2	399.3	462.1	491.2	518.5	119.3	0.0	0.0	0.0	0.0
GROSS REVENUE																		
Au	C$M	0.0	1951.6	1951.6	0.0	0.0	190.7	305.3	321.7	261.4	260.2	296.7	230.4	85.2	0.0	0.0	0.0	0.0
Ag	C$M	0.0	66.3	66.3	0.0	0.0	3.6	7.4	10.7	8.9	10.3	11.0	11.6	2.7	0.0	0.0	0.0	0.0
Total	**C$M**	**0.0**	**2017.9**	**2017.9**	**0.0**	**0.0**	**194.3**	**312.7**	**332.4**	**270.4**	**270.6**	**307.7**	**242.0**	**87.9**	**0.0**	**0.0**	**0.0**	**0.0**

ASCOT PREMIER PROJECT	Unit	Pre-Production Total	Production Total	Life Of Mine Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
OPERATING & CAPITAL COSTS																		
OPERATING COSTS																		
Mining	C$M	0.0	-599.2	-599.2	0.0	0.0	-56.7	-78.3	-98.9	-91.1	-94.6	-91.6	-75.6	-12.5	0.0	0.0	0.0	0.0
Processing	C$M	0.0	-191.8	-191.8	0.0	0.0	-16.7	-28.4	-28.3	-28.3	-28.3	-28.2	-24.3	-9.2	0.0	0.0	0.0	0.0
Site Services	C$M	0.0	-20.8	-20.8	0.0	0.0	-2.1	-3.0	-3.0	-3.0	-3.0	-3.0	-2.6	-1.0	0.0	0.0	0.0	0.0
Closure	C$M	0.0	-23.2	-23.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-23.2	0.0
General & Administration	C$M	0.0	-49.0	-49.0	0.0	0.0	-6.6	-6.2	-6.2	-6.2	-6.2	-6.2	-5.8	-5.8	0.0	0.0	0.0	0.0
Total Onsite Operating Costs	**C$M**	**0.0**	**-883.9**	**-883.9**	**0.0**	**0.0**	**-82.1**	**-115.8**	**-136.4**	**-128.6**	**-132.0**	**-129.0**	**-108.3**	**-28.5**	**0.0**	**0.0**	**-23.2**	**0.0**
Refining Charges	C$M	0.0	-4.0	-4.0	0.0	0.0	-0.3	-0.5	-0.7	-0.5	-0.6	-0.7	-0.6	-0.2	0.0	0.0	0.0	0.0
Transportation Charges	C$M	0.0	-1.5	-1.5	0.0	0.0	-0.2	-0.3	-0.2	-0.2	-0.2	-0.2	-0.3	0.0	0.0	0.0	0.0	0.0
ROYALTIES																		
PGP Royalty Repurchase	C$M	0.0	-14.7	-14.7	0.0	0.0	-14.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Premier Royalties	C$M	0.0	-23.6	-23.6	0.0	0.0	-1.9	-5.0	-5.0	-3.2	-2.2	-3.6	-2.6	0.0	0.0	0.0	0.0	0.0
Red Mountain Royalties	C$M	0.0	-56.2	-56.2	0.0	0.0	0.0	-4.2	-10.3	-8.8	-8.7	-10.1	-8.6	-5.5	0.0	0.0	0.0	0.0
Total Royalties	**C$M**	**0.0**	**-94.5**	**-94.5**	**0.0**	**0.0**	**-16.6**	**-9.2**	**-15.3**	**-12.0**	**-11.0**	**-13.7**	**-11.2**	**-5.5**	**0.0**	**0.0**	**0.0**	**0.0**
CAPITAL COSTS																		
Mining		-14.0	-110.2	-124.2	0.0	-14.0	-23.4	-23.4	-9.7	-8.8	-19.6	-18.2	-6.9	0.0	0.0	0.0	0.0	0.0
Overall Site Development		-8.2	-1.4	-9.6	0.0	-8.2	0.0	-1.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mineral Processing		-35.6	-10.3	-45.9	0.0	-35.6	-10.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TSF		-8.7	-4.6	-13.2	-6.7	-2.0	-0.8	0.0	-1.0	0.0	-2.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site Wide Water Management		-7.0	-4.7	-11.7	-5.5	-1.6	-1.8	0.0	-0.8	0.0	-2.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Closure and Reclamation		0.0	-20.5	-20.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-20.5	0.0
On-Site Infrastructure		-14.0	0.0	-14.0	0.0	-14.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Water treatment Plant		-12.5	0.0	-12.5	0.0	-12.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Off-Site Infrastructure		0.0	-9.7	-9.7	0.0	0.0	-2.9	-6.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Project Indirects		-30.5	-9.4	-39.8	-5.9	-24.6	-5.2	-0.8	-0.9	-0.4	-1.0	0.0	0.0	0.0	0.0	0.0	-1.1	0.0
Owner Costs		-3.7	-0.2	-3.9	-0.3	-3.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-0.2	0.0
Contingency		-12.4	-6.7	-19.1	-1.7	-10.8	-1.7	-0.7	-0.3	0.0	-0.7	0.0	0.0	0.0	0.0	0.0	-3.3	0.0
Total Capital Costs		**-146.6**	**-177.5**	**-324.1**	**-20.0**	**-126.6**	**-46.0**	**-33.1**	**-12.6**	**-9.2**	**-26.3**	**-18.2**	**-6.9**	**0.0**	**0.0**	**0.0**	**-25.0**	**0.0**
Working Capital		0.0	0.0	0.0	0.0	0.0	-13.7	0.0	0.0	0.0	0.0	0.0	0.0	8.9	4.7	0.0	0.0	0.0

ASCOT PREMIER PROJECT	Unit	Pre-Production Total	Production Total	Life Of Mine Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
CASH FLOWS & ECONOMIC INDICATORS																		
Pre-Tax																		
Pre-Tax Cash Flow	C$M	-146.6	856.4	709.8	-20.0	-126.6	35.5	153.8	167.3	119.8	100.4	145.9	114.7	62.7	4.7	0.0	-48.3	0.0
Cumulative Pre Tax Cash Flow	C$M			709.8	-20.0	-146.6	-111.1	42.6	209.9	329.7	430.1	576.0	690.7	753.4	758.1	758.1	709.8	709.8
After-Tax																		
BC Mineral Taxes	C$M	0.0	-110.0	-110.0	0.0	0.0	-1.3	-8.5	-23.7	-17.1	-14.5	-20.7	-16.4	-7.7	0.0	0.0	0.0	0.0
Income Taxes	C$M	0.0	-128.2	-128.2	0.0	0.0	0.0	-0.2	-32.6	-22.6	-22.6	-29.1	-21.0	0.0	0.0	0.0	0.0	0.0
After-Tax Cash Flow	C$M	-146.6	618.2	471.6	-20.0	-126.6	34.1	145.0	110.9	80.0	63.3	96.1	77.4	55.0	4.7	0.0	-48.3	0.0
Cumulative After-Tax Cash Flow	C$M			471.6	-20.0	-146.6	-112.5	32.6	143.5	223.5	286.8	382.8	460.2	515.2	519.9	519.9	471.6	471.6

Pre-Tax NPV 5%	**C$M**	**516**
Pre-Tax IRR	**%**	**62%**
Pre-Tax Payback	**Years**	**1.7**
After-Tax NPV 5%	**C$M**	**341**
After-Tax IRR	**%**	**51%**
After-Tax Payback	**Years**	**1.8**

22.7 Sensitivity Analysis

A sensitivity analysis was performed on the base-case model to determine the variables that have the greatest impact on the Project value, indicated by the after-tax NPV 5%.

As most of the Project's costs are in Canadian dollars and its revenues in US dollars, the Project is most sensitive to metal prices, gold grade, and exchange rates. NPV was less sensitive to operating and capital expenses, and least sensitive to silver price.

Table 22-7 shows the impact of changes in metal prices and the exchange rate, relative to their base-case values, on after-tax NPV. For example, a 5% increase in the base case Au price would result in an after-tax NPV 5% of $383 million.

Table 22-7: NPV 5% Sensitivity to Base Case Assumptions

	−25%	−20%	−15%	−10%	−5%	0%	5%	10%	15%	20%	25%
Au Price ($)	123	167	211	254	297	**341**	383	425	468	510	552
Ag Price ($)	333	335	336	338	339	**341**	342	344	345	346	348
Mining ($)	414	399	384	370	355	**341**	326	311	296	281	266
Process ($)	363	358	354	349	345	**341**	336	332	327	323	318
CAPEX ($)	386	377	368	359	350	**341**	331	322	312	303	294
C$/US$	643	567	501	441	388	**341**	296	256	220	186	155
Au Grade ($)	114	161	206	251	296	**341**	385	429	473	518	562

Figure 22-2 illustrates the sensitivity of the Project NPV 5% to changes in additional variables, including operating and capital costs.

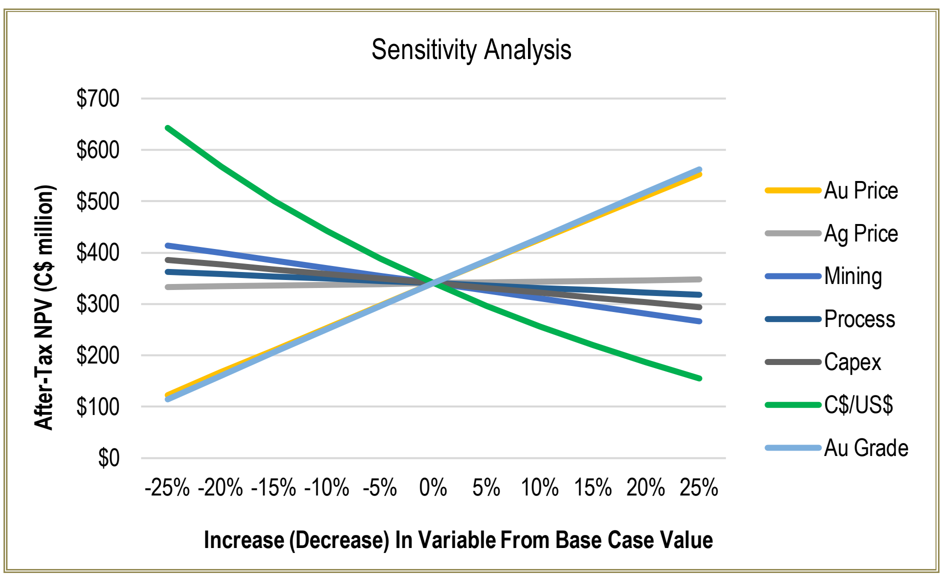


Figure 22-2: Sensitivity Analysis

22.8 *Post-Tax Financial Evaluation*

Ascot internally reviewed inputs and guidance on the post-tax component of the model for the post-tax economic evaluation. The following general tax regimes were recognized as applicable at the time of report writing.

22.8.1 Canadian Federal and British Columbia Provincial Income Tax Consideration

The federal and British Columbia provincial corporate income taxes are 15% and 12%, respectively. For both federal and provincial income tax calculations, capital expenditures are accumulated in tax pools that can be deducted against mine income at different prescribed rates. The prescribed rates depend on the class and type of capital expenditure.

It is common practice that pre-production mining expenditures are accumulated in the CEE pool. The CEE pool is generally amortized at 100%, to the extent of taxable income from the current net revenues of the mine. A phase-out rule was enacted in 2013 to phase out the treatment of pre-production expenses for mine development as CEE to CDE from 2015 to 2017. Effective 2017, all the pre-production development expenses are considered as CDE.

Acquisition costs for resource properties and costs for main mine infrastructure items (mine shafts, main haulage ways, and other underground workings) are considered CDE and are accumulated in the CDE pool. The Ascot financial model reflects those costs as CDE.

Based on current tax regulations, those fixed assets that are acquired for the Project are accumulated in an undepreciated capital cost (UCC) pool (Class 41) and are in consideration to be amortized at 25% on a declining-balance basis. Certain fixed assets (acquired after March 20, 2013 and before 2021) may qualify to be accumulated in a Class 41.1 pool that can be amortized at an accelerated rate of up to 100%. However, as a substantive portion of the fixed assets are expected to be acquired post-2020 (after the phase-out of the Class 41.1 pool), the Ascot financial model assumes that the accelerated depreciation will not be available.

22.8.2 British Columbia Mineral Tax Consideration

The British Columbia mineral tax is a two-stage tax system: a 2% tax for first stage, and a 13% tax for second stage. The 2% tax is assessed on “net current proceeds,” which are defined as gross revenue from the mine after reduction of mine operating expenditures. Hedging income, losses, royalties, and financing costs are excluded from operating expenditures. The 2% tax is accumulated in a cumulative tax credit account (CTCA) and is fully creditable against the 13% tax.

The 13% tax is assessed on “net revenue,” which is defined as mine gross revenues less any cumulative expenditures account balance, to the extent of the gross revenue from the mine for the respective year. All capital expenditures, both mine development costs and fixed asset purchases, and mine operating expenditures are accumulated in the cumulative expenditures account, which is amortized at 100% against the 13% tax.

A “new mine allowance” is available in respect of development of new mines, or capital costs incurred in connection with expansion of an existing mine commencing production with reasonable commercial quantities. Generally, this allowance provides that 133% of eligible capital expenditures incurred prior to commencement of production may be used to offset net revenue for British Columbia mining tax purposes. Once the new mine has begun commercial operations for the first time, or the expansion is complete, this amount is added to CEA. Under current legislation, the provision for the new mine allowance is scheduled to expire on December 31, 2020.

22.8.3 Taxes and Post-Tax Financial Results

For the base case long-term metal prices and exchange rate used for this Feasibility Study, the total estimated and undiscounted taxes payable on Ascot profits are $238.2 million. Post-tax financial results are summarized in Table 22-8.

Table 22-8: Post-Tax Financial Results

	Unit	Lower Case	Base Case	Upper Case
Gold Price	US$/oz	1,200	1,400	1,600
Exchange Rate	C$/US$	0.80	0.76	0.71
After-Tax NPV (5%)	C$ million	177	341	534
After-Tax IRR	%	31	51	71
After-Tax Payback	years	2.5	1.8	1.4
Average Annual After-Tax Free Cash Flow (Years 1-8)	C$ million	56.1	82.7	114.4

23 ADJACENT PROPERTIES

The Project is located at the southern tip of British Columbia's Golden Triangle. This area is host to a large number of deposits of different styles of mineralization such as epithermal, volcanogenic massive sulfides, and porphyry. The mineralization at the Project is interpreted to be hydrothermal in nature and numerous showings and deposits are known in the proximity. The PGP is the largest project in terms of size and contained metal in the Stewart area.

The Scottie Gold Mine is approximately 20 km north of the PGP mill, and is accessed by the Granduc Road along the Salmon Glacier (Figure 23-1). Gold and silver mineralization occurs as bodies of massive pyrite and pyrrhotite with accessory sphalerite, chalcopyrite, galena, arsenopyrite, and tetrahedrite in epithermal quartz-carbonate veins. From 1981 to 1984, the mine produced 160,264 tonnes, containing 2,984 kg Au and 1,625 kg Ag (BC Geological Survey, 2020). The property is currently held by Scottie Resources Corporation.

Five kilometres further north lies the Electrum property, which is 60% owned by Tudor Gold Corp. Gold and silver mineralization occurs in epithermal quartz-carbonate veins, stockworks, and breccias hosted in island-arc volcanic rocks (Tudor Gold, 2016–2019) .Sulfide minerals include pyrite, sphalerite, galena, and chalcopyrite.

The Red Cliff project is a former producing copper and gold property 6 km east of the PGP mill buildings, in the adjacent valley. It is owned 65% by Decade Resources and 35% by Mountain Boy Minerals. Gold is associated with abundant chalcopyrite and pyrite, most commonly in sulfide-bearing veins within a 30 m to 40 m-wide shear that can be traced over 2 km. There are also gold-bearing stockwork zones outside of the vein. Details can be found on the MBM website (see Section 27).

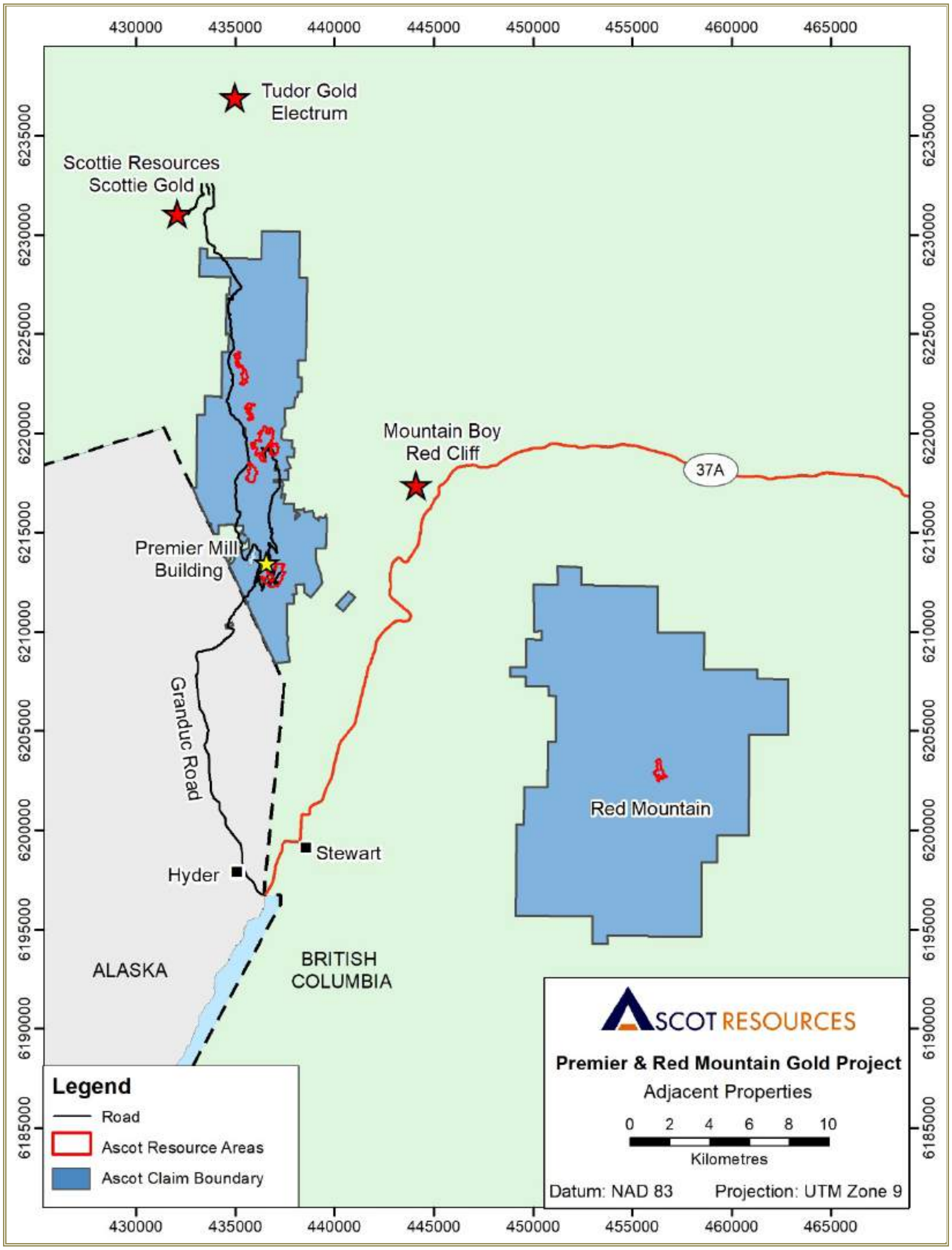


Figure 23-1: Properties Adjacent to the Project

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

24.1.1 Project Milestones

Table 24-1 presents the milestones for the Project

Table 24-1: PGP Milestones

Milestones (Completion Dates)	Milestone by Calendar Date	Milestone Completion Date
BC Hydro System Impact Study (SIS)		
Ascot Initiates SIS with BC Hydro	12-May-20	Q2 Y-2
BCH SIS Review and Approval	27-Oct-20	Q4 Y-2
PGP Powerline Design/Drawings Issued for BCH Approval	8-Nov-20	Q4 Y-2
BCH Approval of PGP Powerline Design/Drawings	3-Dec-20	Q4 Y-2
Environmental Permits		
Submit Final MAPA Documents	16-Jun-20	Q2 Y-2
MAPA Approved	12-Jan-21	Q1 Y-1
Construction Permits Approved	12-Jan-21	Q1 Y-1
Early works Contract Awarded	26-Jan-21	Q1 Y-1
Civil Geotechnical Investigation		
Complete Civil Geotechnical Investigation	14-Jul-20	Q3 Y-2
Mining		
Commence Silver Coin Development and Mining	16-Aug-21	Q3 Y-1
Commence Big Missouri Development and Mining	16-Aug-21	Q3 Y-1
Commence Premier Development and Mining	4-May-25	Q2 Y4
Commence Mining Red Mountain Development	1-Jul-23	Q2 Y2
Complete Mining Activities	31-Mar-29	Q1 Y8
Process Engineering and Procurement		
EP Contract Award	01-Jun-20	Q2 Y-2
Complete Process Design and Drafting	30-Nov-20	Q4 Y-2
Equipment Procurement (General Equipment	07-Jul-21	Q3 Y-1
Long Lead SAG Mill – Award	13-Aug-20	Q3 Y-2
Long Lead SAG Mill – Delivery to Site	07-Jul-21	Q3 Y-1
Long Lead Ball Mill – Award	13-Aug-20	Q3 Y-2
Long Lead Ball Mill – Delivery to Site	07-Jul-21	Q3 Y-1
Fabrication – Structural Steel and Platework	30-Aug-21	Q3 Y-1
Construction – Process Plant and Infrastructure		
Site Establishment	16-Apr-21	Q2 Y-1
Complete Concrete Works	7-Oct-21	Q4 Y-1

Milestones (Completion Dates)	Milestone by Calendar Date	Milestone Completion Date
Complete SMP Installation	5-Dec-21	Q4 Y-1
Complete Site Run/Small Bore Piping	4-Nov-21	Q4 Y-1
Complete LV Electrical and Instrumentation	28-Jan-22	Q1 Y1
Construct Main Substation and 138 kV powerline	15-Jun-21	Q2 Y-1
Construct 4.16 kV powerlines to process plant BM/SC, TSF, and WTP	24-Aug-21	Q3 Y-1
Construct Surface Infrastructure	8-Aug-21	Q3 Y-1
Construction – Cascade Creek Diversion and Tailing Storage Facility		
Decommission WTP Settling Ponds	20-Sep-21	Q3 Y-1
Upgrade Cascade Creek Diversion Channel	21-Sep-21	Q3 Y-1
Construct TSF Embankments	1-Nov-21	Q4 Y-1
Install Tailings Distribution System	13-Jul-21	Q3 Y-1
Install Reclaim and Surplus Water Systems	11-Sep-21	Q3 Y-1
Install Surface Water Management and Water Treatment Infrastructure	23-Sep-21	Q3 Y-1
Construction – Water Treatment Plant		
Construct 40 m Diameter Clarifier	1-Sep-21	Q3 Y-1
Erect WTP, MBBR facilities and tanks	26-Sep-21	Q3 Y-1
Construction – Access Roads		
Construct Access road from Mill to BM/SC	12-Jun-21	Q2 Y-1
Construct Access road from the TSF and SC	5-Jun-21	Q2 Y-1
Construct Access Road to the Red Mountain property	14-Nov-21	Q4 Y-1
Commissioning – Process Plant		
General Commissioning	27-Feb-22	Q1 Y1
Complete Process Plant Pre-Commissioning/Mechanical Completion	28-Jan-22	Q1 Y1
Complete Process Plant Construction	28-Jan-22	Q1 Y1
Commence Process Plant – Introduce 1st Ore and commissioning	29-Mar-22	Q1 Y1
Complete Acceptance Testing and Achieve Provisional Acceptance	29-Mar-22	Q1 Y1
Complete Process Plant Dry and Wet Commissioning	18-Apr-22	Q2 Y1
Safety Guarantees and Achieve Practical Completion	28-Apr-22	Q2 Y1
First Gold-Silver Doré	29-Mar-22	Q1 Y1
Complete Closure Activities	30-Jun-32	Q2 Y11
Project Completion Date	30-Jun-32	Q2 Y11

24.1.2 Scope and Project Approach

The scope of the study includes all the work required for the PGP and Red Mountain mine sites, including the access road to RMP.

The strategy is to refurbish and rebuild the existing mill facilities, ancillary facilities, and buildings, and design/construct the tailings storage facility (TSF) to accommodate the mill throughput tailings from the PGP process plant and reclaim water system. Design and construction of a new water treatment plant (WTP) and associated moving bed biofilm reactor (MBBR) plant is required.

The schedule was developed with input from the mining team, the processing and surface infrastructure teams in order to synchronize the production ore and interactions with the surface infrastructure. This also involved input from the Cascade Creek water diversions, TSF, and WTP installations.

The process schedule was developed in conjunction with a regional contractor to assess the upgrading, and installation or refurbished and new process equipment.

The execution plan at this stage should be considered developmental and will undergo several optimization phases prior to the approved execution of the Project.

The main aspects of the Project include the following:

- Underground mine development work at BM/SC and Premier.
- RMP underground mine development will commence in Q2 Year 2.
- A pipeline will be installed from the BM/SC portals along the BM access road to the WTP. A bypass pipeline will be installed close of the mill building in case of upset conditions at the WTP. This bypass line will direct the underground mine water to decant into the TSF.
- Construct the major water diversion at Cascade Creek, and upgrade/rebuild the other water diversion channels/ponds on site.
- Design an expansion of the existing TSF to handle the resulting tailings from the upgraded PGP facilities (including tailing from the RMP).
- Upgrade and rebuild on-site access/haul roads to the existing mining portals at BM/SC and Premier.
- Upgrade the existing mill facilities.
- Upgrade non-process ancillary facilities and site services
- Construct a new access/haul road to the RMP portal from Highway 37.
- Demolish the existing WTP and construct a new WTP (including a MBBR plant, 40 m diameter clarifier and associated tanks).
- A new 138 kV power supply from the existing Long Lake Power Station to a new Main Substation at the PGP site, located close to the existing mill building.
- Integrate the new power supply (power will be stepped down to 4.16 kV into the existing plant facilities and infrastructure.
- Upgrade the existing power distribution and install new powerlines to service the mining activities at BM/SC and Premier.
- Power at Red Mountain will be supplied by three diesel generators.
- Heating and Ventilation has been included for all facilities. Mine Heating and Ventilations Systems, are also included.

General Principles

The following are a list of general principles to consider for the PGP:

- Safety will be a top priority in the execution of the PGP. The PGP will ensure that safety, health and environmental management is at the forefront of the Project execution.
- Early works will consist of removing all non-essential materials and redundant equipment from the existing mill building and adjacent Cold Storage/Laboratory building.
- Upgrading of the existing Upper and Lower ponds (draining the Lower pond and constructing a platform to facilitate the construction of the new WTP and the concrete clarifier.
- Establish permanent infrastructure early, to the extent practical, to minimize costs of temporary construction facilities.
- Negotiate contracts with Suppliers, Contractors, and Engineering Procurement and Construction Management (EPCM) services with a proven track records in mine developments and brownfield construction projects.

Access—Construction Materials

All construction materials are planned to be shipped to site via existing all-weather roads and via the Port of Stewart. Bridge loading is limited both in size and weight. This will be taken into consideration for the Project.

The Red Mountain site construction materials will be transported via the new access road which will be completed in Q1 of Year 2.

Personnel

The PGP assumes that local labour is readily available from the surrounding area. However, Ascot Resources (Ascot) has made provision for a temporary construction camp close to the existing mill facility. Some construction personnel will be accommodated in a camp environment. Access for the personnel will be surrounding local and national roads.

No camp facilities are envisaged at the Red Mountain property; however, it is assumed that Ascot will provide transportation to and from the Red Mountain property.

Construction Strategy

The construction strategy assumes the following main construction packages:

- Earthworks and access roads
- Early works to clean up and commence upgrading of the mill and cold storage building
- Camp facilities erection and operations
- Concrete installation
- Structural, mechanical, and piping (SMP) – including Vendor’s assistance
- Electrical and instrumentation (E&I)

- Power line installation
- Field erect tanks (new and existing)
- Overland piping – for mine dewatering, fresh/firewater.

Where possible, the contracting strategy will specify several smaller work packages to allow local contractors participation in the Project.

As a special note for major construction fills, crushed waste rock will be free issued to the construction areas from the Crushing Plant at the Cascade Creek diversion area. Costing allowances have only been included for hauling, placement, and compaction costs.

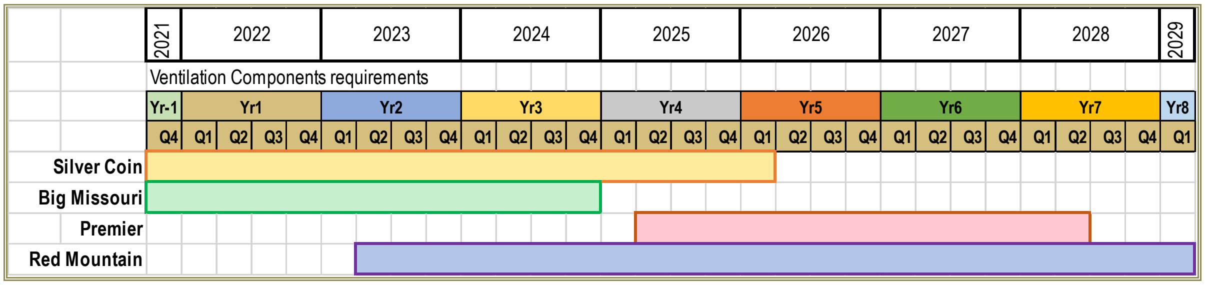
Refurbished existing ancillary facilities will be utilized by the incoming geotechnical personnel and the selected EPCM team. The execution strategy will provide support to the Owner's team.

The general approach to construction execution strategy includes the following key elements:

- Extensive front-end construction planning to identify priorities and increase efficiency in engineering, procurement, and construction.
- Construction will be planned and executed by sub-area and trade according to construction work package philosophy.
- Constructability analysis and input in the design phase to influence engineering, minimizing construction risks and optimizing project costs and schedule.
- Major construction equipment such as crushing plants and large cranes shall be supplied by the Owner.
- It is assumed that the majority of the construction labour force will come from the local region.
- The construction labour is expected to work 10 hours per day and 7 days per week for the duration of the construction and commissioning phase, while abiding within the limits of British Columbia labour law.
- Mine early works include:
  - Site establishment at the BM/SC mines, thereafter Red Mountain and Premier mines
  - Site roads and earthworks
  - Water management to support mine development.

The schedule shown in Figure 24-1 outlines the mining production sequence for the PGP:

- Costs of the Project are heavily dependent on performing work in the appropriate season.
- All earthworks are best executed in spring/summer.
- Concrete work is best done in the spring/summer to avoid winter heating and hoarding costs and improve outside work productivity. Concrete will be supplied by a local concrete supplier from Stewart. Project concrete supply rates were issued to the consultants.
- Pre-assemble as much equipment as possible, e.g., air compressor skids or modules.



Source: InnovExplo Inc. (2020)

Figure 24-1: Mining Production Sequence

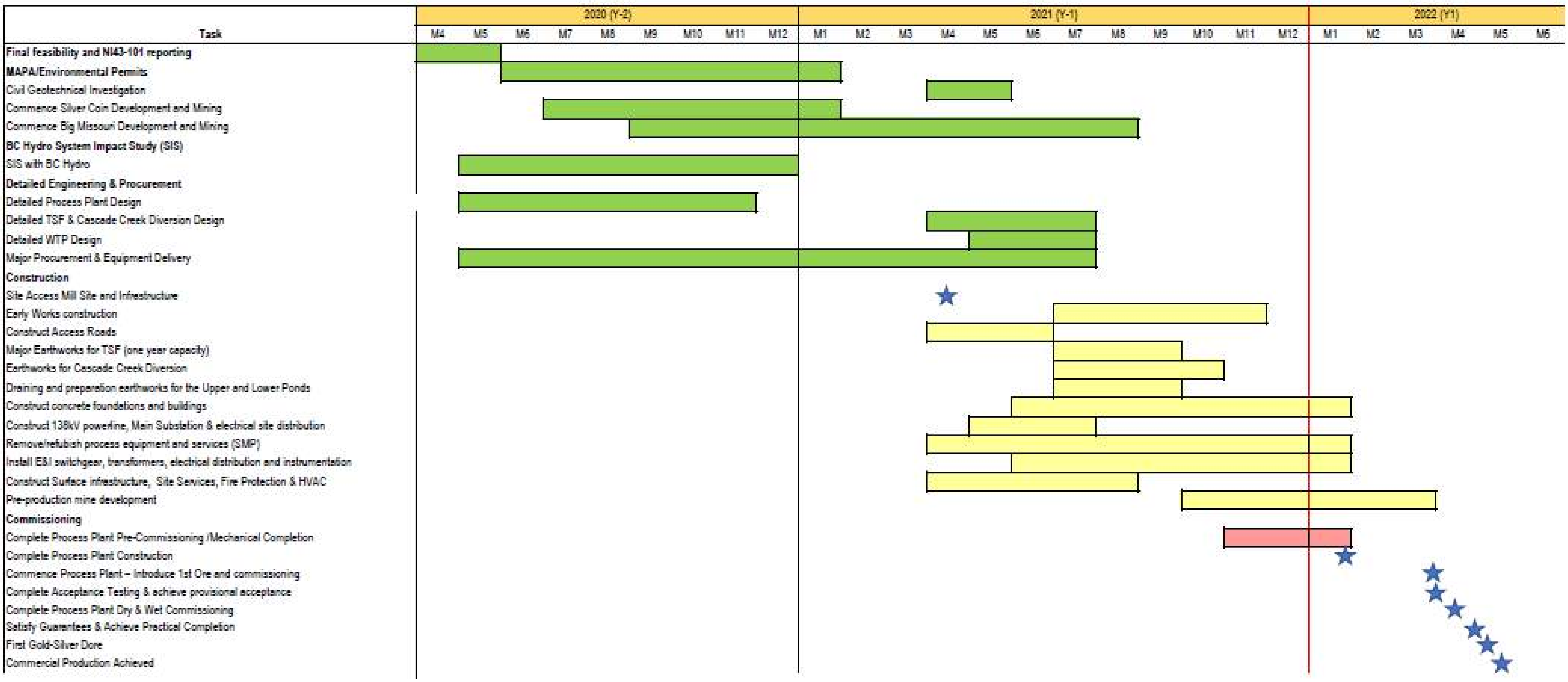
24.1.3 Planning and Scheduling

The project execution schedule was developed for the PGP, as shown in Figure 24-2. The schedule was developed with interaction between the mine development team, the processing and surface infrastructure development team to check for timing of the production ore and interactions with the surface infrastructure and tailings facilities upgrade scopes of work. The schedule was also shared with regional contractors to assess the installation duration and planned sequence of work to develop the works forward from engineering to construction and performance testing of the plant.

The schedule considered the early ordering of equipment using the designs developed for the feasibility study in which key long lead items were tendered. These long lead items were reviewed to the point of a technical recommendation for Ascot.

The critical path of the Project execution which includes the following stages:

The schedule presented in Figure 24-2 outlines the PGP timeline. Construction activities are dependent on receiving an approved *Mines Act* and *Environmental Management Act* permit application (MAPA).



Source: Sacré-Davey (2020)

Figure 24-2: PGP Schedule

Activities allowing the production and delivery of RMP ore also include the access road permit. Construction of the RMP Road will commence in Year 1 to facilitate road access to the mine portal and the installation of generator power, heating, and ventilation facilities. Red Mountain ore is planned to be delivered to the PGP mill by Q2 Year 2.

The current plan will be to complete the Feasibility Study by Q2 2020 (Year -2) and to proceed to detailed engineering. This will allow Ascot to be able to start early works at site in Q1 Year -1. This work will include preparing the mill and ancillary facilities as an accessible workspace/office, early construction of the earthworks for the new water treatment plant and installation of 138 kV powerline from Long Lake to the new Main Substation.

It is expected that the following early works activities will be performed:

- Mill site preparation
- Permanent power installation to reduce the costs of temporary construction power installation
- Site preparation to facilitate the construction of the new WTP
- Access road development – specifically to the mining areas of BM/SC; the permanent powerline will follow the route of the access road
- Cascade Creek Diversion site preparation and construction
- Drawdown and decommissioning of WTP settling ponds
- Aggregate Plant to produce aggregates for the road construction and the fill at the Upper and Lower ponds at the existing WTP
- Main Substation
- WTP, MBBR facility and clarifier
- Fuel Storage refurbishment at PGP mill site
- Temporary accommodation at PGP mill site.

Upon Project approval starting in Q1 of Year -1, the balance of the facilities will be constructed. The total Project is expected to require 52 weeks, resulting in Project start-up in Q2 of Year 1.

24.2 Project Management

24.2.1 Organization and Responsibilities

The Project Management Team (PM Team) will be an integrated team comprising the Owner's personnel, the EPCM contractor, and various engineering sub-contractors. The PM Team will oversee and direct all engineering, procurement, and construction activities for the Project. Figure 24-3 presents a representative preliminary organization chart.

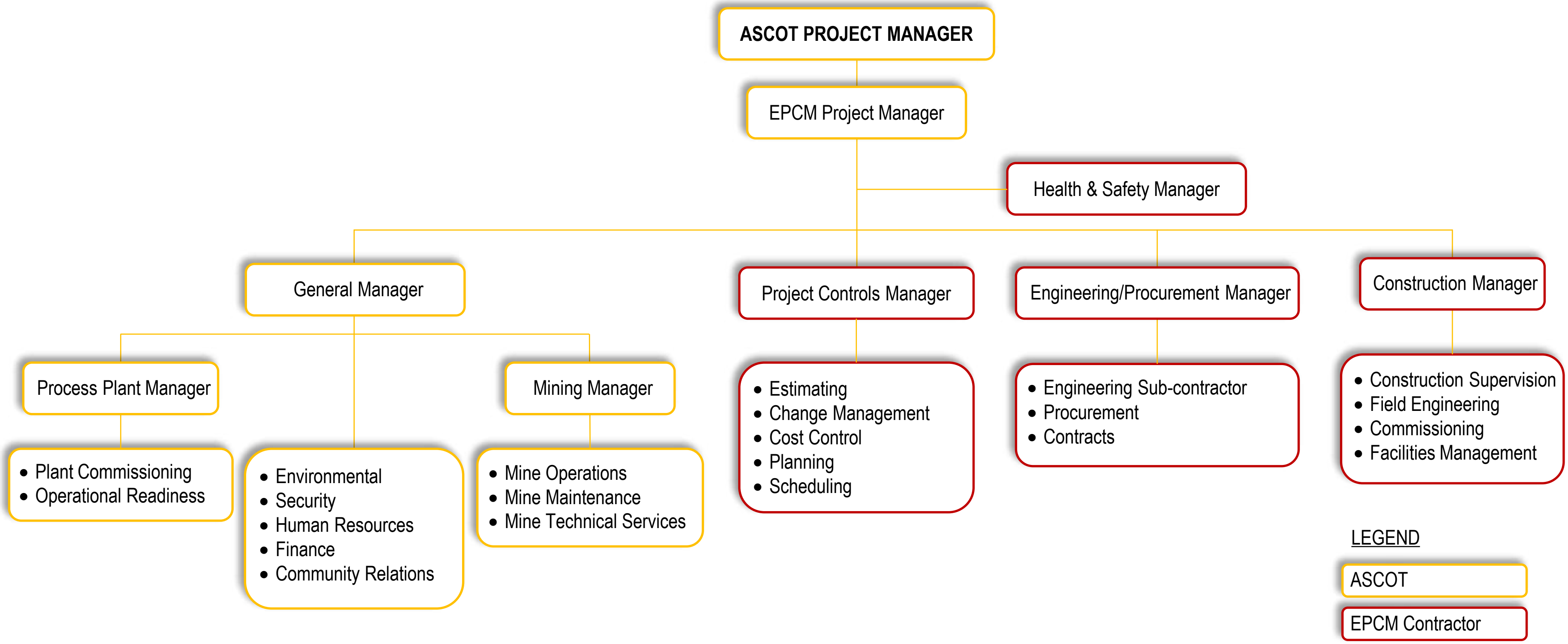


Figure 24-3: Project Management Team and Organizational Chart

Overall delivery of the Project to the defined metrics will be the responsibility of the Ascot Project Manager (APM). The APM will provide high level direction to the EPCM PM Team, with support from Ascot's Pre-Operational Team to manage Project activities.

The APM will be responsible for the execution of PGP activities, including detailed engineering, procurement, logistics, construction, commissioning, and Project controls.

The EPCM scope of services for the Project scope of services includes engineering, procurement, construction management, dry, wet and ore commissioning with associated project management and project controls. The EPCM consultant will do what is necessary to deliver the required facilities in a manner which meets budget and schedule, within the agreed scope.

The EPCM Project Manager will be responsible to the APM and coordinate the engineering scopes of several engineering companies with specialized knowledge of their assigned scope. The following major engineering packages for the PGP are:

- Detailed engineering and procurement of process facilities, and select on-site infrastructure and field engineering support
- Detailed engineering of the TSF and associated water management structures
- Detailed engineering of the site access and haul roads
- Site water management design
- Water treatment plant design
- Substation design
- Power distribution
- Power transmission line detailed design.

Engineering data from the FS including (but not limited to) design criteria, flow sheets, material take-offs, and drawings are considered the engineering baseline data, and form the basis for the capital cost estimate and schedule. Deviations from these baseline engineering inputs, beyond clarifying and finalizing scope, and detailing of designs, will be subject to the PGP change management processes.

An engineering design approvals procedure will be developed to ensure project efficiency, and each engineering consultant will be assessed by earned value analysis (EVA) on a submitted deliverables list as part of their services proposal. Deliverables (and their associated budgets) will be grouped into Work Packages (WPs). A project responsibility matrix will be developed with responsibility by areas and work packages. The Work Packages and functions will be assigned by skill sets to ensure efficiency and that engineering progress is maintained.

Owners Operations Team

A portion of the Owner's Operations Team will be mobilized during the Project development phase for functions required over the life-of-mine (LOM) (i.e., not limited to construction support):

- Safety
- Expediting and material control

- Mining operations, including maintenance
- Environmental
- Security
- Accounting
- Community relations
- Human resources
- Site services (EPCM Construction Manager will be responsible during construction).

Engineering Team

The Engineering Manager will oversee, coordinate, and integrate engineering activities. The Engineering Team will consist of various engineering companies, who will develop the detailed designs and specifications for the Project, and then transition to the field to provide construction support, quality assurance (QA), field engineering, and commissioning support.

This team will contribute to the development of project plans and procedures, the master project schedule, and initial project set-up. In line with the schedule a Construction Manager will be mobilized to site.

Process engineering will be supplied by the EPCM consultant with input from Ascot.

Procurement Team

The Procurement Manager will oversee and manage procurement and contract activities undertaken by engineering (formation and administration of engineering and construction contracts will be overseen and managed by EPCM contracts personnel). The procurement team will use the prepared engineering design packages to obtain competitive tenders, and secure Suppliers and construction contractors to provide the appropriate goods and services.

Factory Approval and Test (FAT) procedures will be incorporated into the procurement and contract procedures.

The general procurement execution strategy for the PGP will involve utilizing known Suppliers, with a preference for First Nation, local and regional suppliers and construction contractors. The Procurement Manager (under the direction of the Project Manager) will have overall responsibility for the majority of pre-purchased procurement and contract formation activities. Contract administration will be the responsibility of the procurement and contracts team on site.

Construction Management Team

The Construction Manager will be responsible for construction safety, progress, and quality. The Construction Management Team will coordinate and manage all site activities to ensure construction progress is on schedule and within budget.

Commissioning Team

The Construction Manager will oversee the Commissioning Team, and be responsible for the timely handover of process and infrastructure systems to Ascot once construction activities have been substantially completed. The Commissioning Team will be supported by disciplined engineering resources to complete dry, wet, and ore commissioning activities, to final handover, custody, and control of completed systems to Ascot's team.

Project Controls Team

The Project Manager will oversee the Project Controls team, and be responsible for the development, implementation, and administration of the processes and tools (EVA) for Project estimating, cost control, planning, scheduling, change management, progressing, and forecasting.

Operational Readiness

Operational Readiness will be overseen by Ascot, which includes but is not limited to the following:

- On-site training—including a series of activities, measures, checklists, and toolkits that facilitate the transition from completion of project construction (mechanical completion), commissioning (pre-operational testing and start-up), and first production, through to continuous operation at 100% throughput.
- As the Project evolves to a functional operation, Operational Readiness is an ongoing exercise directed at assisting safe and reliable handover to Ascot of a functioning plant capable of performing to the targets it was designed and financed.
- Operational Readiness also outlines the contributions expected from EPCM Contractors, and Ascot's project management team.
- The Project Execution Plan, and therefore Operational Readiness, begins at the Feasibility Study stage, and is required at Detailed Engineering through to continuous operation at 100% throughput.

24.3 Transport and Logistics

24.3.1 Route Survey Report

To date a proposed transportation route has not been developed for the PGP. However, this survey will be initiated in the next phase of the Project prior to the construction phase and will include:

- Minimizing the number of trucks to site
- Minimizing the number of shipments to the local Port of Stewart
- Avoiding all unknown/un-surveyed roads to site, and minimize the routes as far as possible
- Understanding limiting width, height, and load limits for access roads—this includes the need for overload assessments for key structures on the transportation route
- Understanding all possible transportation risks, e.g., low hanging power lines
- Ensuring a risk registry is complete that includes potential mitigation strategies / back-up plans.

24.4 Opportunities

24.4.1 SAMS™ Mining Method

Ascot is currently assessing the potential integration of the Shallow Angle Mining System (SAMS™) mining approach developed by Québec based manufacturing company, Minrail Inc. (Minrail). To that end, an opportunity for Ascot is to work collaboratively with Minrail and InnovExplo Inc both from Val-d'Or to complete a trade-off study on the application of the SAMS™ mining method. The trade-off study would be led by Minrail's president and chief executive officer, Mr. Marc R. Beauvais, P.Eng. who was also involved in the mine planning for this Feasibility Study.

At this time, the SAMS™ is experimental in nature without a track record in operational application and therefore couldn't be incorporated into the current study. It is presently being tested at the Lamaque Mine on a full-scale production basis. At present, SAMS™ performance and productivity rates are estimates that have been derived from past test work programs. These estimates are useful in assessing the potential benefits to the Project. The Lamaque program will provide current estimates in the summer of 2020.

Although experimental, it should be noted that the SAMS™ mining concept has been demonstrated during an extensive research and development (R&D) and evaluation program to which the National Research Council of Canada and the *Ministère de l'Économie et de l'Innovation of Quebec* have contributed financially. In addition, several other government agencies in both Quebec and Ontario (Québec's *Commision de la Santé et de la Sécurité du Travail* (CSST), Ontario's Ministry of Labour, and Groupe MISA) have also contributed to Minrail's R&D efforts, as well as several major suppliers of mining equipment (Orica Canada, Mansour Mining Technologies, and Montabert).

Both the SAMS™ mining process and its proprietary double overhead railing system is protected by Canadian (#2,912,261 & #2,912,264), American (#14/890,710 & #14/890,687), and European (#14797763.1 & # 14797349.9) patents.

Description of the Shallow Angle Mining System (SAMS™)

Minrail's Shallow Angle Mining System (SAMS™) is designed to safely and economically develop and mine stopes in the 10- to 45-degree dip range by mechanizing the rock work activity along the true slope angle of the ore body. Shallow angle stopes in this dip range have historically been difficult and costly to mine primarily because theses dips are difficult for miners and equipment to operate on productively; broken rock usually does not run naturally on the footwall and a slusher or similar device is often needed to recover the material, or excessive dilution is incorporated to mine with conventional equipment.

SAMS™ is a fully integrated mining method that efficiently operates through the mining cycle from development, to production, to backfilling, using several easily interchangeable modules. It is a highly selective extraction method, with a typical mine design based on SAMS™ requiring significantly less underground lateral development (between 40% to 50% less) compared to conventional methods which translates into reduced waste rock production. The method has the capacity to operate with up to 60 m vertical separation between levels or 90 m on dip (Figure 24-4), reducing the development work required to access the ore body. In some instances, it can be used without a top sill (blind), leading to further time and capital cost reductions.

There are many variants of the SAMS™ application, however the following is a description of the mining cycle in a typical SAMS™ mining scenario:

- A cut-out or alcove (similar to an Alimak nest) of an appropriate size is excavated on the opposite side of the stope axis perpendicular to the access drift and facing the planned stope (Figure 24-4).
- A centre raise (also called a *draise*) is driven along the hanging wall contact into the mineralization (Figure 24-4). As development progresses, the back and upper walls of the draise are supported with standard ground support materials. Cable bolts are installed in the back of the draise prior to production blasting.
- A remotely controlled production drill module is used to drill off 76 mm diameter nominal blast holes typically drilled at a 15 degree angle from the draise axis, in the mineralized zone on both walls. The projected minimum thickness that can be drilled and blasted (measured perpendicular to the footwall) could be as small as 1.2 m depending on strike extents.
- Blast holes are loaded with emulsion explosive and blasting is initiated remotely using an electronic blasting system. The angled holes are blasted to slash the ore material down the centre of the draise.
- Blasted ore that is not cast down initially, is removed to the sill drift using the SAMS™ mucking module, which can pull material laterally into and down the draise (Figure 24-4).

Once the stope has been mined out, the footwall will be washed down using a camera-guided remotely operated high-pressure hose to help recover fines. Rib pillars may be left between adjacent stopes if there are no plans to backfill mined-out openings.

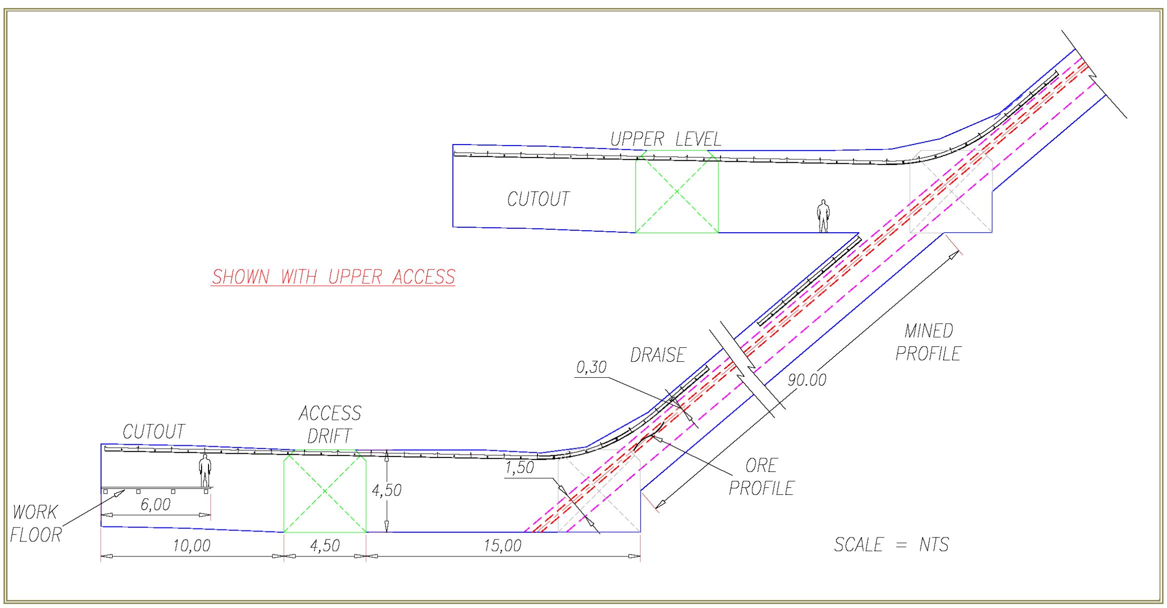


Figure 24-4: Typical Multi-Level Extraction Layout

SAMS™ modules travel on a proprietary overhead double rail system, with electric-powered modules that have multiple safety devices, including a failsafe braking system, and a proposed safety factor between six and ten. Minrail has reviewed its safety devices with mine safety regulators in Québec and Ontario and reports that SAMS™

meets the respective mine safety requirements in both jurisdictions. If a test stope program is contemplated at either PGP or RMP mine sites, the work would be reviewed with occupational health and safety authorities in British Columbia prior to a mining test.

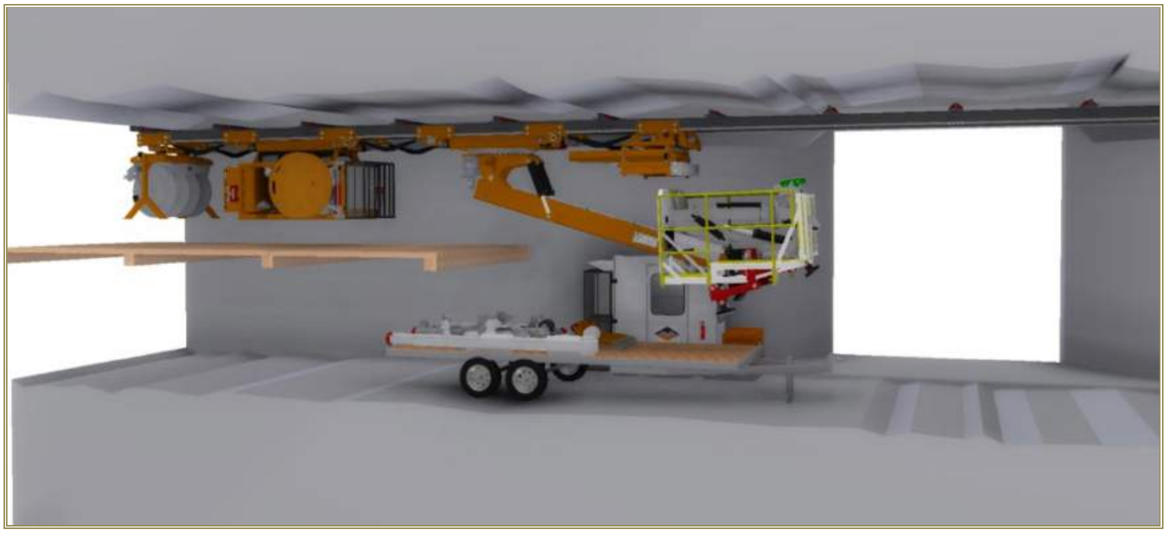


Figure 24-5: Typical System Installation into the Alcove

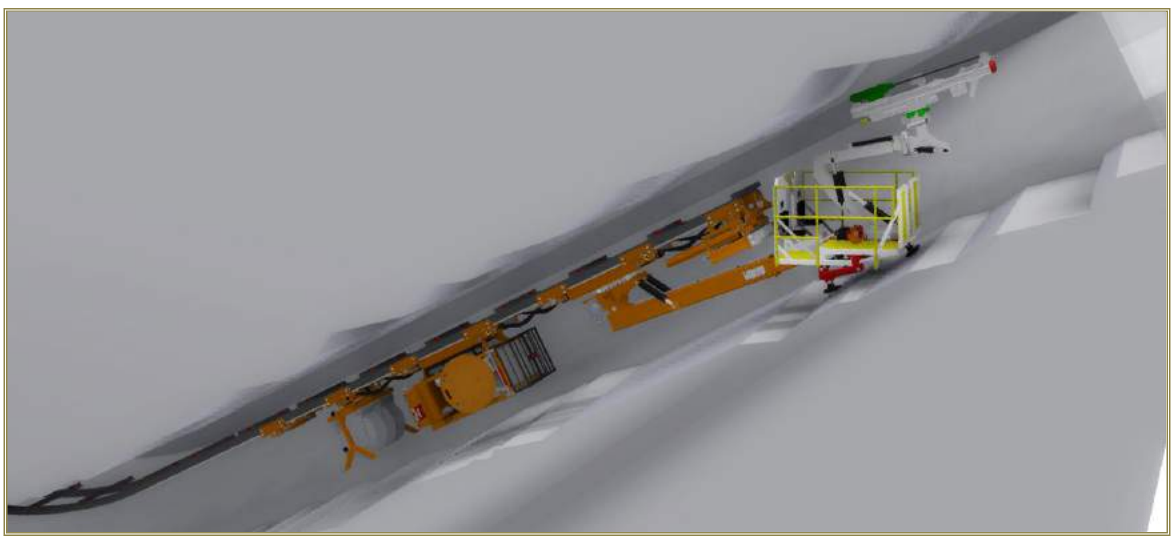


Figure 24-6: Drilling Module

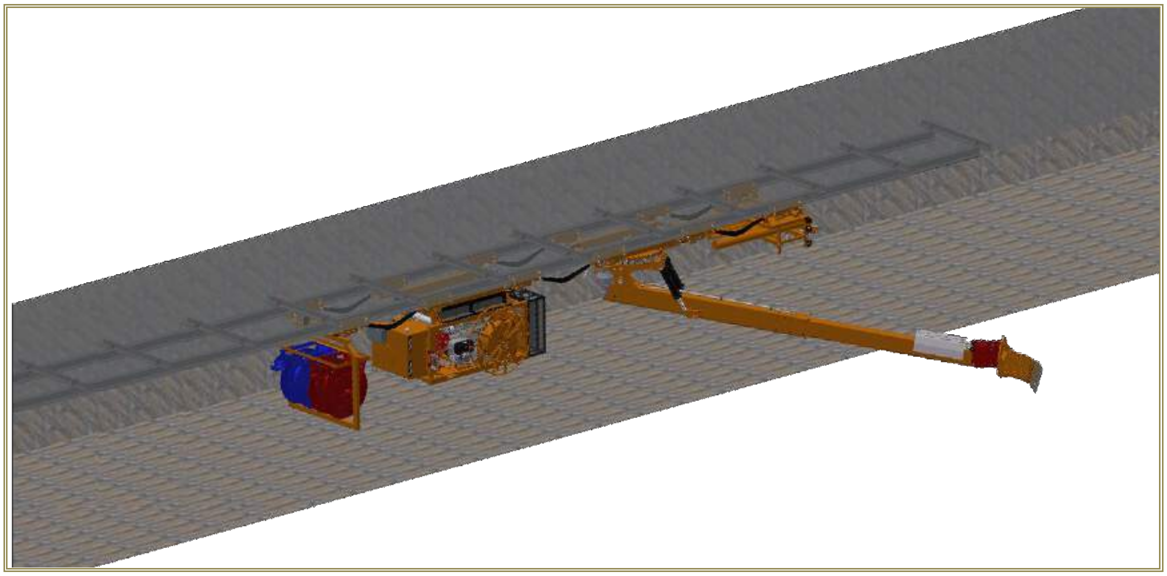


Figure 24-7: Mucking Module

Stope Layout Comparison between SAMS™ and Traditional Mechanized Mining Methods

Both PGP and RMP mine sites often exhibit narrow, shallow-dipping ore zones that are believed to be compatible with the SAMS™ mining technique. Many ore wireframes exhibit dip angles in the range of 30 to 40 degrees with narrow widths down to 2.5 m and less which fall well within the specifications of this mining method. Typical scenarios illustrating the differences are shown in Figure 24-8 and Figure 24-9.

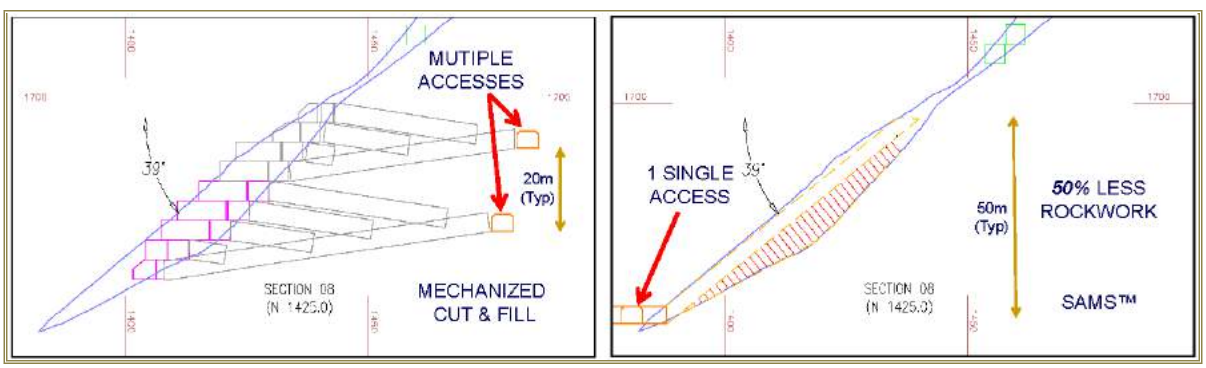


Figure 24-8: Mechanized Cut-and-Fill vs. SAMS™

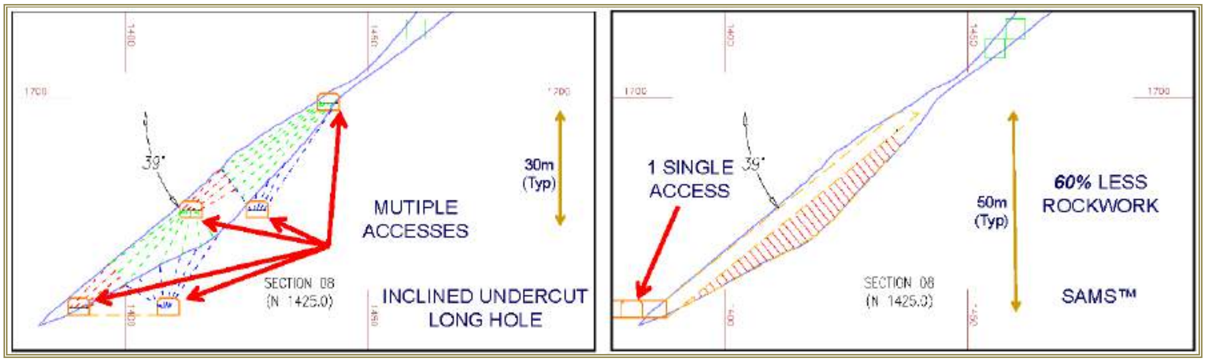


***Figure 24-9:	Inclined Undercut Long-Hole vs. SAMS™***

SAMS™ Conclusion

Replacing either the inclined undercut long-hole (IULH) or the mechanized cut-and-fill (MCF) stopes with the SAMS™ could reduce lateral capital development and improve the economics of the Project by reducing capital costs. MinRail's operating test experience shows that the SAMS™ mining approach has the potential for significant stope productivity gains, and reduced operational stope development. In conjunction these factors have the potential to significantly lower OPEX costs when compared to traditional methods.

A modified mine plan with SAMS™ could benefit from a lower COG and increase the contained ounces in the Indicated category available for mine planning. This in combination with conversion of Resources in the Inferred category (Section 24.4.2) could increase the current LOM. An additional benefit from the method is the ability to mine smaller stope headings which provides the opportunity to reduce dilution and improve mined grades, but also aids in higher recovery of ore reserves.

Given the time horizon to mining, there is adequate time to study the benefits of SAMS™, and incorporate the method as appropriate to an updated detailed mine plan.

### *24.4.2	Conversion of Inferred Resources at the Premier Gold Project*

Drill hole spacing is a critical parameter considered in the classification of geological resources. Many areas of the PGP deposits are challenging to drill from surface, either because of their depth or orientation that makes it difficult to accurately aim for a targeted pierce point in a modeled zone of mineralization. Much of the historical drilling was widely spaced based on an open pit model, and due to the mountainous terrain, placing drill pads for infill drilling from surface is difficult with longer holes required. The underground mine development planned as part of this Feasibility Study will improve access to many areas for exploration and definition drilling, presenting an opportunity to convert inferred material to indicated at minimal cost, while also potentially extending the LOM.

The resource inventory at the Premier, Silver Coin and Big Missouri deposits currently contains 4.173 Mt in the Inferred Category. Approximately 2.2 Mt of this material (about 53%) are located within 100 m of existing or planned underground development. The Company will focus on converting these resources to the Indicated Category and make them available for conversion to reserves and integration into a future mine plan. Inferred blocks within 100 m of development are shown in Figure 24-10 to Figure 24-12.

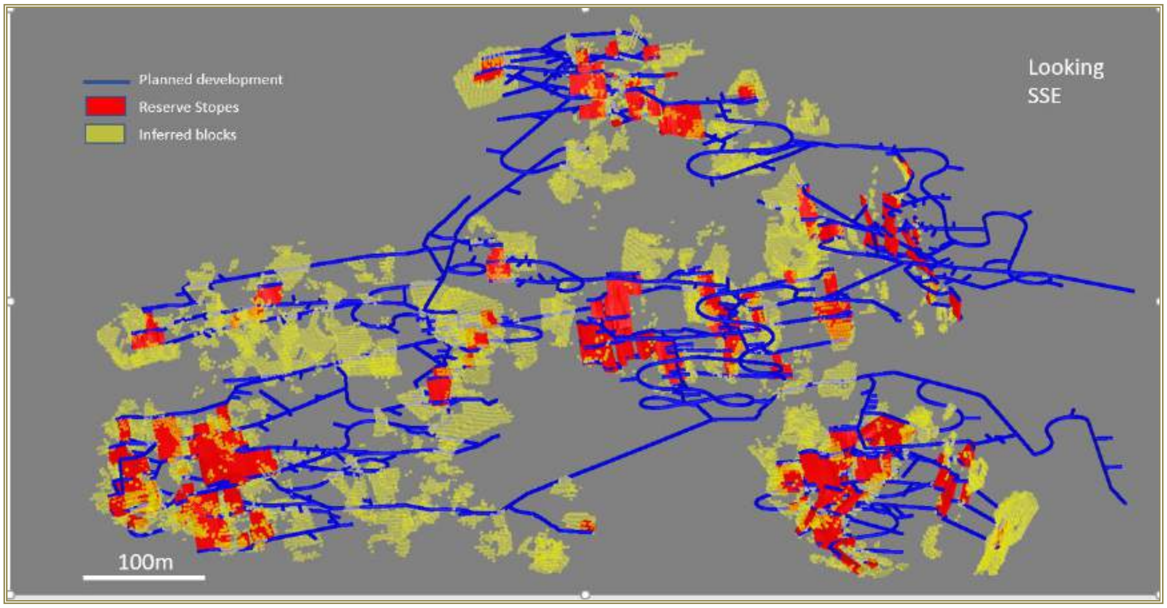


Figure 24-10: Inferred Blocks within 100 m at Premier/Northern Light

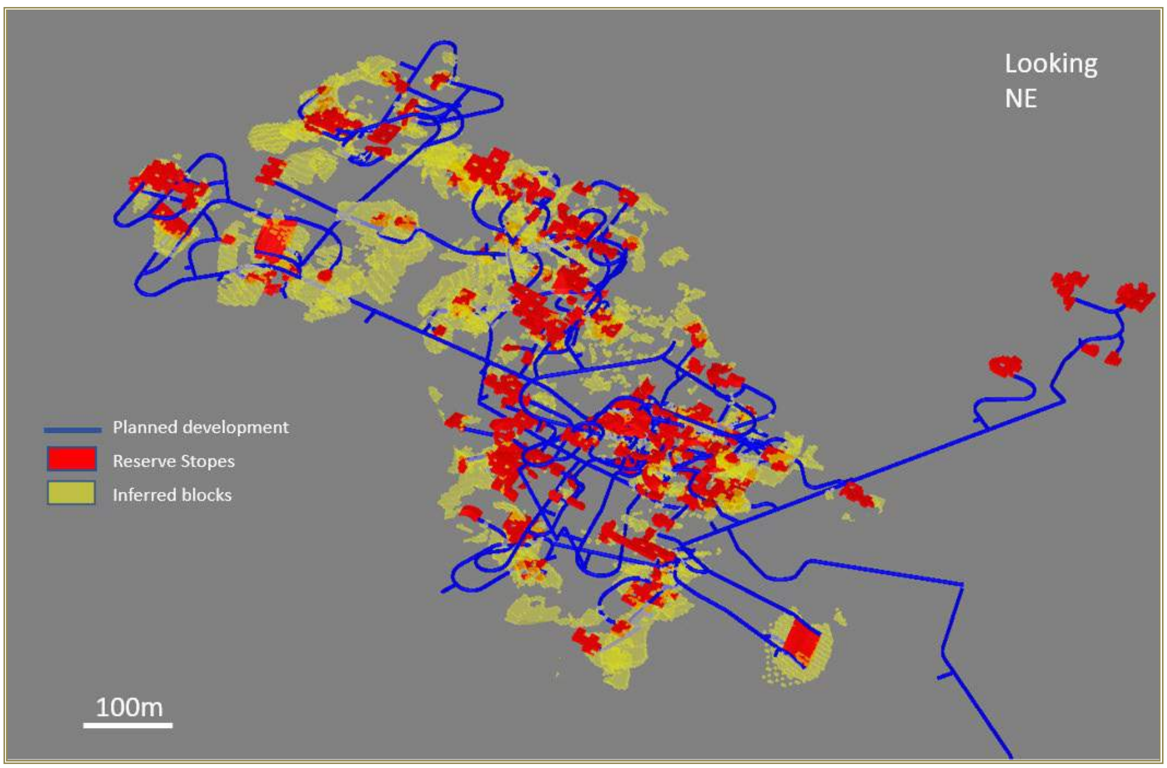


Figure 24-11: Inferred Blocks within 100 m at Big Missouri

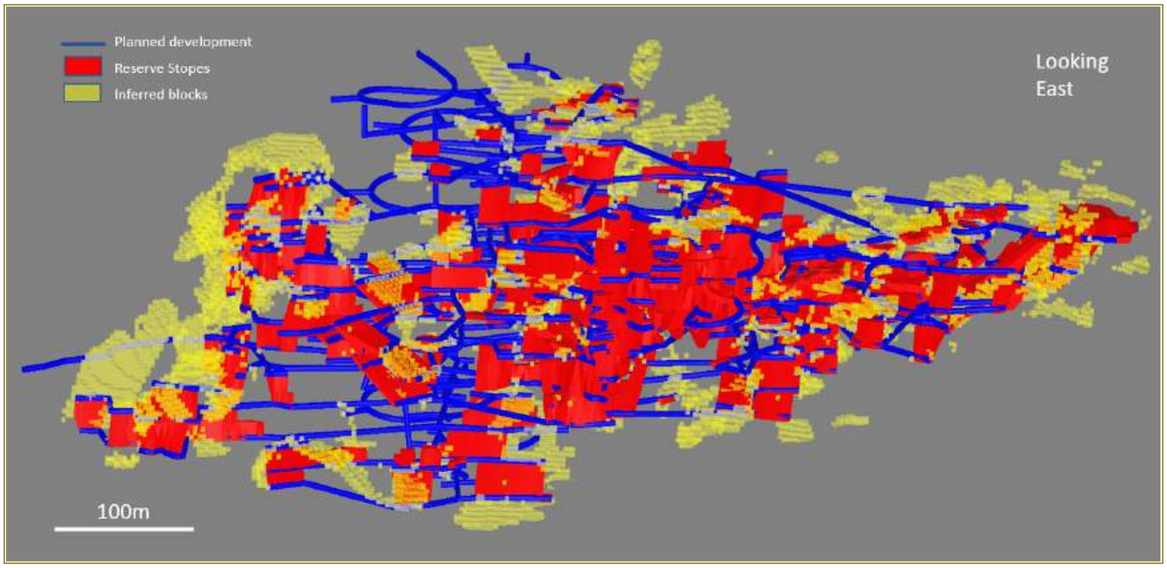


Figure 24-12: Inferred Blocks within 100 m at Silver Coin

25 INTERPRETATION AND CONCLUSIONS

This Feasibility Study represents an economically viable, technically credible, and environmentally sound mine development plan for the Project. This Feasibility Study–level eight-year life-of-mine (LOM) plan has positive economics. Industry-standard mining and processing methods were used in the study, and the Qualified Persons (QPs) are not aware of any fatal flaws that encumber the PGP from undergoing further economic studies, permitting, financing, and ultimately development. The QPs recommend that the Project proceed to permitting and detailed engineering design.

The most significant potential risks associated with the Project are: uncontrolled mining dilution, operating and capital cost escalation, permit acquisition, reduced metallurgical recoveries, unforeseen schedule delays, changes in regulatory requirements, the ability to raise financing, exchange rate, and metal prices. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

Further conclusions and interpretations regarding the next phase of the PGP are given below.

25.1 Geology, Mineral Resources, and Mineral Reserves

25.1.1 Premier Gold Project Resources

- Modelled grades for all deposits were validated and compared to the declustered composited data, suggesting that there is no global bias, and the overall tonnage and grade of the deposits are reasonable. However, due to the highly skewed nature of the gold and silver deposition (even after capping and outlier restrictions have been applied), local block grades should be further validated by definition drilling prior to underground mining.
- The exploration potential for additional underground resources is extensive, particularly in the Premier, Big Missouri, and Silver Coin deposit areas.
- The gold grades of the legacy assay data were validated for grades above the cut-off grades used for the underground resource estimate in this report.
- Sample preparation, analysis, and security is acceptable for all drilling used in the Resource. Legacy drilling was verified by re-assaying of core and coarse rejects. Portions of Indicated blocks have been downgraded to Inferred in some areas of Silver Coin, Dilworth, and Martha Ellen, due to lack of quality assurance/quality control (QA/QC) for some legacy assays.
- True widths were used for the Resource Estimate, therefore eliminating down-dip drilling bias in the results.
- Data collection was updated in 2019 to now constitute a comprehensive property-wide database.
- Gold and silver grade distributions are observed to be moderately to extremely positively skewed, which indicates that capping and outlier restriction of high grades is warranted.
- Definition drilling and drifting is warranted in order to better model local variations in grade.

25.1.2 Red Mountain Resources

A high degree of drilling and quality control work was performed on the Project by previous operators. Relogging the core to create a geological model has created confidence in the understanding of mineralized zone controls.

The Marc, AV, JW, 141, and Smit zones form the main portion of the mineralized deposit. The Marc and AV zones require no further drilling.

The 141 and Smit zones are drilled at nearly a 25 m to 50 m grid spacing, and show reasonably good geological and grade continuity, converting a large portion of the deposits to the Indicated category. The zones will require infill drilling to confirm the geological continuity prior to mining. The infill drilling carried out in 2017 and 2018 confirmed the geological and grade continuity of the AV and JW zones, and was successful in upgrading the Inferred resources in these zones to Indicated classification, and upgrading some blocks in the Marc zone from Indicated to Measured.

The Cambria and GOP zones, and low-grade areas of the Marc, AV, and JW zones are currently classified as Inferred due to wider-spaced drill density. The existing drill holes display reasonable geological continuity, and it is reasonable to assume that most of the Inferred resources could be converted to Indicated with additional drilling.

Exploration potential on the property has been greatly enhanced since 1994 by glacial recession surrounding the deposit. A considerable area that was previously under ice is now exposed for the first time and available for exploration proximal to the Red Mountain gold/silver-bearing sulfidation system.

The underground mine plan for the PGP was evaluated. The QPs note the following interpretations and conclusions in their respective areas of expertise.

25.2 Mineral Processing and Metallurgical Testing

25.2.1 Premier Gold Project Deposits

The site has a long history of operations and supporting testwork. For the restart of the operation, repeated and updated testwork has been conducted that aligns to the updated mine plans, has been reviewed as being suitable for this Feasibility Study, and is referenced below:

- The QP reviewed the sample locations and sampling procedures employed by Ascot Resources (Ascot) for the Silver Coin, Big Missouri, and Premier deposits for the purpose of the 2019 metallurgical confirmation testwork campaigns at the SGS Mineral Services (SGS) and Base Metallurgical Laboratories Ltd. (BML) facilities. It is concluded that they meet industry standards for representative samples, styles of mineralogy, lithology, and mineral deposits as a whole.
- The sample preparation procedure of the composite and variability samples meets industry standards and is consistent with requirements and best practices for use while preparing representative metallurgical testwork samples.
- Based on all available Abrasive index (Ai) testwork data, we can conclude that PGP ore samples are moderately abrasive to abrasive.

- Based on all available SMC, RWi, and BWi testwork data we can conclude that ores from the PGP deposits are moderately hard to hard from a milling perspective.
- Based on the above comminution characterization, a SAB milling circuit configuration is selected for processing ores from the PGP deposits.
- PGP ores have a considerable amount of free gold and are amenable to gravity concentration.
- A gravity concentration/intensive leach followed by CIL is the recommended process plant configuration, which is suitable for smaller throughputs and can yield better project economics than WOL.
- A leachable grind size of 80 µm is recommended as the basis of design, as it provides high recoveries without overcapitalizing in the grinding circuit equipment selection.
- The overall combined (gravity with leach) estimated gold recoveries are 94.5% for Silver Coin (head grade (HG) of 6.3 g/t Au); 93.5% for Big Missouri (HG 7.2 g/t Au), and 98.4% for the PGP ores (HG 5.6 g/t Au).
- The overall estimated gold recovery for design is 95.4%.
- The overall combined (gravity with leach) estimated silver recoveries are 74.2% for Silver Coin (HG 14.4 g/t Ag), 68.6% for Big Missouri (HG 10.5 g/t Ag), and 69.2% for the Premier ores (HG 14.7 g/t Ag).
- The overall estimated silver recovery for design is 71.5%.
- It is estimated that a flocculant addition of 30 g/t could produce a tailings thickener density of 60% solids w/w.
- Based on all available testwork and the PGP plant throughput, an 18 m-diameter tailings thickener is required for the processing of the Prem PGP ores.
- The SO_2/Air CN destruction process was selected for the design, as it proved successful in reducing the CN_{WAD} levels to below 1 ppm for all the PGP deposits.

25.2.2 Red Mountain Deposits

The testwork performed in support of this report is deem suitable for a feasibility study level of effort.

- The QP concludes that samples collected by Ascot from the AV, JW, and Marc deposits for the purpose of the 2019 metallurgical confirmation testwork campaigns at the BML and ALS testwork facilities meet industry standards for representative samples, styles of mineralogy, lithology, and mineral deposits collectively.
- Sample preparation procedures for the composite and variability samples meet industry standards and are consistent with requirements and best practices for use while preparing representative metallurgical testwork samples.
- Based on all available comminution data, we can conclude that ores from the Red Mountain deposits are moderately hard to hard from a milling perspective.
- Precious metal recoveries are very sensitive to the particle grind size, so therefore a particle grind size of 25 µm is recommended as the basis of design, as it provides acceptable recoveries without overcapitalizing in the grinding circuit equipment selection.

- Based on the above comminution characterization, a SAB milling circuit configuration, followed by a tertiary fine grinding stage, was selected for processing ores from the Red Mountain deposits.
- Red Mountain ores do not contain significant amounts of free gold and are therefore not amenable to gravity concentration.
- CIL is the recommended process plant configuration, which is suitable for smaller throughputs and can yield better project economics than CIP.
- The estimated gold recoveries are 91.9% for Marc (head grade (HG) of 7.5 g/t Au), 80.6% for AV (HG 6.4 g/t Au), and 90.1% for JW (HG 5.1 g/t Au).
- The estimated silver recoveries are 89.7% for Marc (HG 29.2 g/t Ag); 75.5% for AV (HG 16.9 g/t Ag), and 87.5% for JW (HG 14.3 g/t Ag).
- The overall estimated gold and silver design recoveries are 86.8% and 83.6%, respectively.
- It is estimated that a flocculant addition of 60 g/t can produce a pre-leach thickener density of 50% solids w/w.
- Based on all available testwork and the PGP plant throughput, a 27 m-diameter pre-leach thickener is required for the processing of the Red Mountain ore.
- The SO_2/Air CN destruction process was selected for the design as it proved successful in reducing the CN_{WAD} levels to below 5 ppm for all Red Mountain deposits; however, more testing is required, particularly on the JW deposits, to improve detoxification performance and lower plant operating costs.

25.3 Mining Methods and Reserves

- Mineral Reserves were determined and calculated using appropriate mining methods and cost estimates to industry standards
- Appropriate cut-off grades were determined based on the selected methods
- PGP is a brownfield site, and as such there is an abundance of information from which a conservative and operationally robust mining plan has been developed. Historical information includes:
  - Stable stope spans
  - Grade continuity
  - Existing available underground access
  - Knowledge of groundwater flow characteristics.
- At RMP:
  - There is existing underground exploration development which can provide an advantage in the initial start up of the mine plan and also provides advanced geotechnical information.
  - Easy access for a bulk sampling campaign for RMP ore processing and tailings optimization.

25.4 Recovery Methods

The PGP mill facility will process ores from the PGP and Red Mountain deposits at the designed feed rate of 2,500 t/d. The SAB and SAB circuit, with additional tertiary grinding for the Red Mountain ores, followed by CIL gold recovery, is selected as the most suitable process flowsheet for the processing of the abovementioned ores.

The plant design, completed at a feasibility study-level of engineering, is supported by the following testwork conclusions:

- A SAG/Ball (SAB) milling circuit configuration is selected for processing ores from the PGP and Red Mountain deposits, with the addition of a fine grinding circuit when processing the Red Mountain ore.
- The PGP ores have considerable amount of free gold and are amenable to gravity concentration; therefore, gravity concentration/intensive leach followed by CIL is the recommended process plant configuration.
- The Red Mountain ores do not contain significant amounts of free gold, and are therefore not amenable to gravity concentration. CIL is the recommended process plant configuration, which is more suitable for smaller throughputs and can yield better project economics than CIP.
- The average LOM design estimated gold and silver recoveries are:
  - For PGP ores, 95.4% and 71.5%, respectively
  - For Red Mountain deposits, 86.8% and 83.6%, respectively.

25.5 Infrastructure

25.5.1 Tailings and Water Management

Potential risks associated with tailings and water management for the PGP include:

- Groundwater Inflows—Although a very detailed hydrogeological model was developed, there is always a risk that groundwater inflows to underground mine workings are larger than currently estimated. This may impact sizing of pipelines and water treatment plant design. This risk can be managed by additional data collection and modelling.
- Seepage from TSF—Seepage values from the TSF may be greater than currently estimated, or future seepage water quality may not meet criteria for direct discharge to the environment. Seepage volumes and/or quality can be managed by mitigation measures present within the design.
- Geotechnical Conditions—Historical geotechnical studies and as-built construction reports were completed for the Cascade Creek Diversion Channel expansion cut. There is a risk that adverse ground conditions may be encountered during construction that may impact project schedule or pose a risk to workers. This risk can be managed by completing additional geotechnical investigations and studies.

This Feasibility Study has highlighted several opportunities to increase mine profitability and Project economics, as well as reduce identified risks.

Potential opportunities associated with tailings and water management for the PGP include:

- Potential for some expansion of the TSF to store more than the planned 6.5 Mt of tailings, however the expansion potential will be limited by the downstream toe of the TSF embankment in the SE corner of the facility; further trade-off studies are required to define the extent of the limits of expansion of the TSF.
- Potential for reduced dewatering inflows from the underground workings allowing for a reduction in the capacity and size of the pipelines and WTP system.
- There is the potential for reduction in size/capacity of the WTP system if water quality characterization based on additional studies and data collection result in an improvement in water quality predictions.
- Potential for remediation of existing infrastructure (such as flattening of the TSF downstream slope, decommissioning of the WTP settling ponds, etc.) to be completed prior to full construction to beneficially affect the construction schedule.
- Potential to remove the reclaim waterline if the flow volume and quality of water coming from Big Missouri meets requirements for process water usage, additional characterization and trade-off studies are required to clarify if this water is suitable.
- Potential to combine the surplus waterline and the WTP contingency line (i.e., the system to recycle water to the TSF from the WTP in case of WTP shutdown conditions) into a single pipeline with reverse flow capabilities.

25.5.2 Water Treatment Process

The water treatment processes planned for the PGP are conventional and proven technologies that are commonly implemented for treatment of mine water. The processes were selected based on a Best Achievable Technology (BAT) assessment, and as such can be considered best achievable technologies for treating mine for discharge to the receiving environment for the PGP.

Potential risks associated with water treatment processes at the PGP include:

- Requirements for additional, unanticipated water treatment processes or water quality mitigation measures. Both, of the planned treatment processes are commonly implemented to treat mine water produced at underground gold mines. The planned water treatment processes were selected based on results of a BAT assessment of water management and water treatment options, which is key for defining treatment requirements in the permit amendment process. Nonetheless, unanticipated treatment requirements could be identified during the review of the *Environmental Management Act* (EMA) Permit amendment application.
- Higher-than-expected loadings in mine contact water. Water quality model results were relied upon as a basis for designing the water treatment processes. Although water quality model results are expected to be conservative it is possible that unanticipated factors could influence water quality and thereby ability of the operation to meet discharge quality limits.
- Poorer-than-expected treatment performance. The assumed treatment performances by the selected water treatment processes were estimated based on typical performances observed at similar mine water treatment operations. Although not expected, unanticipated site-specific factors could affect treatment, which could necessitate changes to the treatment system design.

25.6 Capital and Operating Cost Estimates

The initial capital costs can be optimized by using one Engineering and Procurement (EP) consultant incorporating the process and infrastructure designs. Construction costs can be optimized by using an experienced local Construction Management (CM) contractor to refurbish the process building and equipment, and required infrastructure.

25.7 Economic Analysis

The PGP has positive economics with a base case after-tax net present value (NPV) at 5% of $341 million, an after-tax internal rate of return (IRR) of 51%, and all-in sustaining costs of US$769/oz of payable gold.

26 RECOMMENDATIONS

The Project is well suited for a potential mining operation. The Feasibility Study-level eight-year life-of-mine (LOM) plan has positive economics, and it is recommended that the Project proceed to permitting and detailed design. Further recommendations regarding the next phase of the PGP are discussed below.

26.1 Geology, Mineral Resources, and Mineral Reserves

26.1.1 Premier Gold Project Resources

The QP recommends the exploration work proposed by Ascot Resources (Ascot or the Company) for 2020 be carried out as detailed below.

Definition drilling should be conducted to upgrade the current Mineral Resource classification where possible.

In future, as much exploration drilling as possible should be carried out from underground. Access to the mine and services should be re-established to facilitate this.

In areas where the mineralized zones merge and become difficult to distinguish, a probabilistic modelling method, such as multiple indicator kriging (MIK), may better model the grade distribution. It is recommended to test this at the main mineralized zone in Silver Coin.

The bulk density of a suite of intact core specimens should be measured using a water immersion method to check the pycnometer measurements in the database. The specimens should be selected from a representative group of rock types, and should be of a number sufficient to provide statistically significant results. Approximately 300 to 400 determinations should be sufficient, provided no marked differences between the methods are detected.

In 2020, Ascot is planning to complete 10,000 m of diamond drilling from surface at the western extension of Premier, following up encouraging results from 2019.

The Company also plans to conduct induced polarization (IP) ground geophysical surveys in various parts of the property. Grassroots mapping and sampling are planned for the northern and eastern parts of the property, aiming to identify new zones of mineralization away from the known resource areas.

Additional drilling is budgeted to follow up existing and new IP anomalies on the property.

The budget for the planned 2020 exploration program is discussed in the Resource Estimate Update by Bird (2019). It is recommended that the planned exploration program with a budget of $4.0 million be carried out.

26.1.2 Red Mountain Project Resources

There are no further specific recommendations, RMP will proceed with the recommendations of the Project.

26.2 Mining

The PGP and RMP sections of this NI 43-101 Feasibility Study, including the mine design, stope planning and optimization, mining dilution, and fleet selection were carried out according to industry standards and material risks and concerns have been identified and addressed. The total estimated cost for the recommended activities is approximately $1,000,000.

Based on the above the QP recommends the completion of the following:

- Undertake definition drill for the Big Missouri area to convert inferred to indicated; refer to Section 25
- Optimization of stope design in detailed mine planning
- Geotechnical drilling and analysis of rock mass adjacent to planned stoping areas
- Monitor and address Premier Pit stability prior to mining at the Premier deposit
- Collect samples and undertake uniaxial compressive strength (UCS) testing for cemented rock fill (CRF) optimization of cement usage
- Further investigations of shallow angle mining methods and technologies; refer to Section 24
- Conduct a program on a small test stope to determine the spacing required for underground stope definition drilling and to reconcile to resources as determined by exploration drill holes.

26.3 Processing Facilities and Metallurgical Testwork

The thorough and complete course of testwork described below fully supports the start-up and operation of the PGP mine and processing the ore through the PGP processing facilities.

To ensure that the full effects on processing Red Mountain ore through the PGP processing facilities are understood at an operational level, it is recommended that, during the period of processing the PGP ores prior to Year 2, trials be conducted to examine the process plant response to the introduction of Red Mountain ore. These trials will help confirm the specific energy requirements of the grinding circuit, and the operating conditions for the tertiary grinding mill and other downstream processes, prior to the upgrades.

The following solid/liquid separation and cyanide detoxification testwork recommendations could potentially lead to reduced ore processing costs over the LOM: testwork costs will range between $35,000 and $45,000. A breakdown of the proposed testwork activities for the PGP and Red Mountain deposits is listed below:

- In the next Project phase, additional solid/liquid separation testwork should be conducted on the PGP ores, with the aim of improving thickener overflow clarity while optimizing Project operating costs. From the tested flocculants there is a room for improvement as to what type of flocculant should yield the best results on the PGP ores; therefore, further testwork is recommended Completing an in-house trade-off study between flocculant addition rate and pH in order to improve clarity in the full-scale operation is also recommended.
- The SO_2/Air CN destruction process is recommended as the primary CN destruction method for the PGP; however, copper sulfate consumption during the CN destruction process could be reduced by reducing the level of free CN in solution. Recycling the free CN or optimization of the leaching process to reduce the free CN levels should be investigated further.

- Additional CN detoxification testing should be conducted, particularly associated with the JW deposit, to improve detoxification performance and potentially lower the plant operating costs.
- Additional solid/liquid separation testwork should be conducted on Red Mountain ore with the aim of improving thickener overflow clarity, focusing on investigation of higher pH values effect and use of a coagulant to improve settling of the solids.

26.4 Infrastructure

The QP recommends a site-wide civil geotechnical investigation to verify ground bearing pressures, soil parameters, ground dynamic modulus of elasticity, and settlement parameters: the approximate cost of this work is $250,000.

A route survey report is recommended for the next phase, prior to construction, at an approximate cost of $30,000.

26.4.1 Tailings Storage Facility and Surface Water Management

The QP recommends the following for the advancement of the PGP:

- Designs for TSF and water management should be advanced to support the submission of a *Mines Act* permit amendment (including an updated Mine Waste Disposal Alternatives Assessment) for restart of the mine: the cost for this design work is approximately $0.5 million.
- A project-wide risk assessment be completed as part of the *Mines Act* permit amendment process to identify potential risks and hazards to the PGP.
- Geotechnical investigations should be completed, particularly for the Cascade Creek Diversion Channel upgrade, to confirm assumptions on foundation conditions, and design of drilling and blasting programs for the quarrying activities: the cost for this work is approximately $0.4 million.
- Initiate trade-off study to optimize pipeline usage.
- Initiate trade-off study to optimize pipeline placement throughout the site.
- Initiate trade-off study to determine extent of TSF expansion potential.

26.4.2 Water Treatment Process

The water treatment processes planned for the Project are conventional, proven technologies that are commonly implemented to treat mine and tailings water produced at underground gold mine operations. The high-density sludge (HDS) lime treatment technology for removal of dissolved metals and total suspended solids (TSS) is similar to the existing water treatment process. The MBBR process for removal of ammonia cyanide and other nitrogen nutrients has not previously been implemented on site. Accordingly, it is recommended that a MBBR pilot trial be completed on site prior to completion of detailed design and construction of the plant. The purpose of the trial would be to confirm that the unit ammonia removal rates assumed for the reactor sizing and design can be achieved at the PGP site.

26.5 Marketing Studies and Contracts

The QP recommends that a marketing and logistics report be completed to confirm the accuracy of the terms as the Project progresses towards production, at an approximate cost of $50,000.

26.6 Financial Analysis

The QP recommends the completion of a pre- and post-tax evaluation of the economic model by an external tax consultant, at an approximate cost of $50,000.

27 REFERENCES

A.R. MacPherson Consultant Ltd. (1994). Project No. 8300: Proposed Grinding System for Red Mountain Project. Kelowna: A.R. MacPherson.

Alldrick, D.J. (1993). Geology and Metallogeny of the Stewart Mining Camp, Northwestern British Columbia, Mineral Resources Division, Geological Survey Branch, British Columbia Ministry of Energy, Mines, and Petroleum Resources, Bulletin 85, 105 p.

ALS Kamloops (2019). KM6080-Isa #1 Test Report 2019. Kamloops: ALS.

Anderson, R.G. (1993). A Mesozoic stratigraphic and plutonic framework for the northwestern Stikinia (Iskut River area), northwestern British Columbia. In: Dunne, G.C. and McDougall, K.A., (Eds.). Mesozoic paleogeography of the western United States II Society of Economics Paleontologists and Mineralogists, Pacific Section, Los Angeles, California, pp477-494.

Anderson, T. (2000). Quality Control Measures and Statistical Evaluation of Sampling for the Red Mountain Project (1989-2000). Internal report of North American Metals Corp., 32p and Appendices.

Arseneau, G.J. and Hamilton, A. (2016). Mineral Resource Update for the Red Mountain Gold Project Northwestern BC, Canada.

Arseneau, G.J. and Hamilton, A. (2017). Mineral Resource Update for the Red Mountain Gold Project Northwestern BC, Canada.

Arseneau, G.J. and Hamilton, A. (2018). Mineral Resource Update for the Red Mountain Gold Project, Northwestern BC, Canada, Technical Report. Prepared for IDM Mining Ltd., 161p.

B.C. Road Builders & Heavy Construction Association. 2018. 2018-2019 Equipment Rental Guide: The Blue Book, Authorized by the Government of British Columbia, B.C. Hydro and Power Authority & B.C. Rail Ltd.

Baldwin R. and Jones, L. (February 2013). Technical Report and Preliminary Economic Assessment on the Red Mountain Gold Property. Banks Island Gold Corporation.

Barclay, W.A. (2000). Preliminary Observations Concerning Petrology, Structural Style, and Controls on Sulphide-Gold Emplacement at the Marc Zone, Red Mountain Deposit, Stewart, British Columbia. Private report to North American Metals Corp., 48p.

Barnett, R.L. (1991). Petrographic-Metallurgical Electron Microprobe Investigation of Selected Samples from the Red Mountain Prospect, British Columbia. Private report to Lac Minerals Ltd., 65p.

Barry W. Smee Consulting (1993). Analytical Quality Control Procedures in Mineral Exploration. 28p.

Base Metallurgical Laboratories Ltd. (2017a). Project No. BL 0084: Metallurgical Testing Red Mountain Project. Kamloops: BML.

Base Metallurgical Laboratories Ltd. (2017b). Project No. BL 0184: Additional Metallurgical Testing Red Mountain. Kamloops: BML.

Base Metallurgical Laboratories Ltd. (2018). BL0366 Report Base Met Labs_Nov_2018. Kamloops: BML.

Base Metallurgical Laboratories Ltd. (2020). Project No. BL 0482: Metallurgical Testing of the Red Mountain Project Samples, February 2020. Kamloops: BML.

Base Metallurgical Laboratories Ltd. (2020b). BML-Dewatering Assessment of the Samples from the Premier Gold Project-BL0553, Kamloops: BML.

BC Geological Survey. (2020). MinFile Detail Report: Scottie Gold. Available at https://minfile.gov.bc.ca/report.aspx?f=PDF&r=Minfile_Detail.rpt&minfilno=104B++034, accessed April 4, 2020.

BC Geological Survey. MinFile number104B 034 Scottie Gold. Available at https://minfile.gov.bc.ca/report.aspx?f=PDF&r=Minfile_Detail.rpt&minfilno=104B++034, accessed May 15, 2020.

BC Ministry of Energy, Mines and Petroleum Resources (EMPR). (2017). Health, Safety and Reclamation Code for Mines in British Columbia. Victoria, BC. EMPR.

BC Ministry of Energy, Mines and Petroleum Resources (EMPR). (2016). Guidance Document on Part 10 of the Health, Safety and Reclamation Code for Mines in British Columbia. Victoria, BC. EMPR.

Beattie Consulting Ltd. (2001). Red Mountain Project Flotation Study (Summarizes the Testwork completed by Process Research Associate). Vancouver: Beattie Consulting Ltd.

BitterRoot Group LLC (2009). Pre-scoping Study, Silver Coin Project, Stewart, British Columbia, (for Pinnacle Mines Ltd), May 29, 2009, 93 p.

Bjornson, L. (2011). 2010 Diamond Drilling, Prospecting and Surface Sampling Report on the Premier Gold Property, Stewart BC., Assessment Report prepared for Ascot Resources Ltd.

Bjornson, L. and Deane, S. (2010). 2009 Diamond Drilling, Prospecting and Surface Sampling Report on the Premier Gold Property, Stewart BC., Assessment Report prepared for Ascot Resources Ltd., 892 p.

Bjornson, L. and Tsang, L. (2012). 2011 Diamond Drilling, Prospecting and Surface Sampling Report on the Premier Gold Property, Stewart BC., Assessment Report prepared for Ascot Resources Ltd.

Blake, Cassels and Graydon, LLP (2019). Ascot Resource Ltd. – Premier, Dilworth, big Missouri, Silver Coin, and Red Mountain Properties – Mineral Title Opinion.

Bray, A. (2000). Data Validation Report, Red Mountain Project. Prepared for North American Metals Corp, September 2000. 15p.

Brenda Process Technology (BPT). (1994). Final Report on Metallurgical Testing of Red Mountain Gold-Silver Ores., Peachland: BPT.

Bridgestone Corporation. 2016. Data Book – Off-the-Road Tires. Available at: https://commercial.bridgestone.com/content/dam/bcs-sites/bridgestone-ex/products/Databooks/Bridgestone_OTR_Databook_July_2016.pdf. Accessed March 2020.

Brown, D.A. (1987). Geological Setting of the Volcanic-Hosted Silbak Premier Mine, Northwestern British Columbia (104 A/4, B/1), University of British Columbia Master of Science Thesis, October 1987, 231p.

Butler, S., Mroczek, M., and Collins, J. (2013). NI 43-101 Technical Report on the Silver Coin Project, Technical Report Prepared for Jayden Resources Inc., August 23, 2013, 133 p.

Canadian Dam Association (CDA). (2013). 2007 Dam Safety Guidelines – 2013 Revision. Toronto, ON: Canadian Dam Association.

Canadian Dam Association (CDA). (2014). Guidance Document – Application of Dam Safety Guidelines to Mining Dams. Toronto, ON: Canadian Dam Association.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2019). CIM Estimation of Mineral Resources and Mineral Reserves – Best Practices Guidelines, November 2019.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2014). CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by CIM Council on May 10, 2014.

Caterpillar (2018). Caterpillar Performance Handbook. Peoria, Illinois, USA. 48th Edition.

Christopher, P.A. (2009). Technical Report on the Premier Gold Project Cassiar Mining District, Stewart, British Columbia, NI 43-101 Technical Report for Ascot Resources, August 3, 2009, 58p.

Clark, J. (2010). Silver Coin Gold Project, Stewart, British Columbia, Canada, NI 43-101 Technical Report for Jayden Resources Inc. Minarco-MinConsult, September 15, 2010, 76p.

Clark, J. (2011). Silver Coin Gold Deposit, Stewart, British Columbia, Canada, NI 43-101 Technical Report for Jayden Resources Inc. Minarco-MinConsult, April 13, 2011, 118p.

Craig, D. (2001). Red Mountain Project, Stewart, BC Resource Estimate May 2001 Volumes 1 & 2.

Craig, D. (2002). Red Mountain Project, Stewart, BC. Technical Report March 2002. Prepared for Seabridge Resources Inc.

Craig, D., Doerksen, G., Klimgmann, S., and Shouldice, T. (2014). Preliminary Economic Assessment NI 43-101 Technical Report Red Mountain Gold Project Northwestern BC, Canada. JDS Energy & Mining. 207p.

Deane, S. and Gruenwald, W. (2009). Diamond Drilling, Prospecting, and Surface Sampling Assessment Report on the Dilworth Property, Stewart, BC, Volume I. Unpublished draft report to Ascot Resources Ltd., 2009, 33p.

Gekko Systems (2015). Project No. T1306: Red Mountain Python Amenability Testwork. Vancouver: Gekko Systems.

Government of British Columbia (2020). Health, Safety and Reclamation Code for Mines in British Columbia. Available at https://www2.gov.bc.ca/gov/content/industry/mineral-exploration-mining/health-safety/health-safety-and-reclamation-code-for-mines-in-british-columbia, accessed May 13, 2020.

Government of British Columbia (2020). Mineral Titles Online. Available at https://www.mtonline.gov.bc.ca/mtov/home.do, accessed April 13, 2020.

Greig, C.J. (2000). Review of the structural and stratigraphic setting for gold mineralization at Red Mountain, Stewart District, Northwestern British Columbia. Private report to North American Metals Corp., 21p.

Greig, C.J., Anderson R.G., Daubeny, P.H., Bull, K.F., and Hinderman, T.K. (1994). Geology of the Cambria Icefield Area: Regional setting for Red Mountain Gold Deposit, Northwestern British Columbia. In Current Research 1994-A. Geological Survey of Canada, p. 45-56.

IDM Mining Ltd. (2017). Avalanche Study, August 10, 2017. Vancouver: IDM.

International Metallurgical and Environmental Inc. (1997). A Metallurgical Investigation into the Reduction of the Acid Generation Potential of Red Mountain Tailings. Kelowna: IME.

JDS Energy and Mining Inc. (2016). NI 43-101 Preliminary Economical Assessment Technical Report for the Red Mountain Project, August 26, 2016. Vancouver: JDS Energy and Mining Inc.

JDS Energy and Mining Inc. (2017). NI 43-101 Feasibility Study Technical Report for the Red Mountain Project, British Columbia, Canada, August 20, 2017. Vancouver: JDS Energy and Mining Inc.

Kemetco Research Inc. (2020). L03012 Testwork Report – Evaluation of Cyanide Destruction Technologies for the Premier Gold Project. Richmond: Kemetco.

Kirkham, G. and Bjornson, L. (2012). Revised Technical Report on the Resource Estimate for the Premier Gold Property, Stewart, British Columbia, NI 43-101 Technical Report. Prepared for Ascot Resources Ltd., August 20, 2012.

Klohn Crippen Berger (2019). Premier TSF For MAPA – Alternatives Assessment. Vancouver, BC. KCB.

Knight Piésold Ltd. (2020a). Premier Gold Project – Tailings and Water Management Feasibility Design Report. Vancouver, BC: Knight Piésold Ltd.

Knight Piésold Ltd. (2020b). Premier Gold Project – Tailings Laboratory Testwork Summary Report. Vancouver, BC: Knight Piésold Ltd.

Lakefield Research (1991). Project No. L.R. 4048: The Recovery of Gold from Red Mountain Project Samples.

Lang J.R. (2000a). Petrographic Relationships Between Pyrite, Pyrrhotite and Precious Metal Minerals in Nine Samples from Red Mountain, British Columbia. Private report to North American Metals Corp., 45p.

Lang J.R. (2000b). Selective Assay Analysis of the Distribution of Au and Other Metals in the Red Mountain System, British Columbia. Private report to North American Metals Corp., 34p.

Lang, J.R. (2000c). Geologic Controls on Gold Ores in the Marc, AV and JW Zones, Red Mountain, British Columbia. Private report to North American Metals Corp., 53p.

Lewis, P.D., Macdonald, A.J., and Bartsch, R.D. (2001). Hazelton Group/Bowser Lake Group stratigraphy in the Iskut River area progress and problems, Chapter 2. In: Lewis, P.D., Toma, A., and Tosdal, R.M., (Eds.), Metallogenesis of the Iskut River Area, Northwestern British Columbia. Mineral Deposit Research Unit, Special Publication Number 1, pp. 9-30.

Lhotka, P. (1995). Explore B.C. Report Grant No. 94-95/A-7 Report on Exploration Activities on the Lesley Flats Project 1994, Assessment report for Westmin Resources Limited, February 24, 1995, 68p.

McConnell, R.G. (1913). Portions of Portland Canal and Skeena Mining Divisions, Skeena District, B.C., Canada Department of Mines Geological Survey. Memoir 32, 101p.

McDonald, D.W.A. (1990). The Silbak-Premier Silver-Gold Deposit: A Structurally-Controlled, Base Metal-Rich Cordilleran Epithermal Deposit, Stewart, BC. Western University Doctor of Philosophy Thesis, February 1990, 444p.

McLeod, R. (2014). Surface Mapping, Geochemistry and Diamond Drilling Report, Red Mountain Gold Project, B.C. Assessment Report. 22 pp and appendices.

Mountain Boy Minerals Ltd. Available at https://www.mountainboyminerals.ca, accessed May 15, 2020.

Nelson, J., Waldron, J., van Straaten, B., Zagorevski, A., and Rees, C. (2018). Revised Stratigraphy of the Hazelton Group in the Iskut River region, northwestern British Columbia. BCGS_P2018-01-02.

Panteleyev, A. (1986). Ore Deposits 10, A Canadian Cordilleran Model for Epithermal Gold-Silver Deposits, Geoscience Canada, vol. 13, No. 2, pp. 101-111.

Puritch, E., Sutcliffe, R., Brown, F., Armstrong, T., and Hayden, A. (2013). Technical Report and Resource Estimate for the Big Missouri and Martha Ellen Deposits, Premier Gold Property Skeena Mining Division, British Columbia, NI 43-101 Technical Report. Prepared for Ascot Resources Ltd., February 13, 2013.

Ray, G.E. (2008). Geology and Zn-Pb-Ag-Au Mineralization at the Mount Dilworth Property, Stewart Area, Northern BC. Internal report prepared for Ascot Resources Ltd. 36p.

Ray, G.E. (2011). Geology & Hydrothermal Alteration on Parts of the Silver Coin Gold Property, Salmon River, Stewart Mining Camp, Northern BC. Prepared for Jayden Resources Inc., December 31, 2011, 84p.

Rennie, D.W. and Simpson, R.G. (2018). Technical Report and the Premier-Dilworth Project, Stewart, British Columbia, Canada, NI 43-101 Technical Report. Prepared for Ascot Resources Ltd., June 22, 2018, 299p.

Rennie, D.W., Bird, S., and Butler (2019). Technical Report and the Premier Project, Stewart, British Columbia, Canada, NI 43-101 Technical Report. Prepared for Ascot Resources Ltd., January 17, 2019.

Rescan Consultants Inc. (1995). Red Mountain Project, Application for Mine Development Certificate.

Rescan Engineering Limited (1994). Red Mountain Project, Feasibility Study (Final Draft).

Rhys, D.A., Seib, M., Frostad, S.R., Swanson, C.L., Prefontaine, M.A., Mortenson, J.K., and Smit, H.Q. (1995). Geology and Setting of the Red Mountain Gold-Silver Deposits, Northwestern British Columbia. In Schroeter, T.G. (ed) Porphyry deposits of the northwestern Cordillera of North America, Canadian Institute of Mining, Metallurgy and Petroleum Special Volume 46: p. 811-828.

Roscoe Postle Associates, Inc. (2019). Technical Report on the Premier Project, Stewart, British Colombia.

RS Means Company Inc. (2019). 2020 Heavy Construction Costs with RS Means. Rockland, Massachusetts, USA.

Sanderson, D. (1994). Report on the Preparation of Gold Standard Pulps. Prepared by CDN Resource Laboratories Ltd. for Lac Minerals, April 13, 1994. 5p.

SGS Canada. An Investigation into Supplementary Metallurgical Testwork for the Premier Gold Project SGS 17709-01M – Final Report, Burnaby: SGS Canada.

SGS Lakefield Research Limited (1991). Project No. L.R. 4048: The Recovery of Gold from Red Mountain Project Samples. Lakefield: SGS Lakefield.

Shives, R.B.K. (2009). 2008 Helicopter-Borne Magnetic/Electromagnetic and Gamma Ray Spectrometric Surveys Mount Dilworth Property, Stewart, British Columbia, Canada. An internal interpretation report prepared for Ascot Resources Ltd. 35 p.

Sillitoe, R.H. (2017). Comments on the Geological Model for the Red Mountain Gold-Silver Deposit, British Columbia. Report prepared for Talisker Exploration Services Inc.

Simpson, R.G. (2014). Premier-Dilworth Gold-Silver Project, British Columbia, NI 43-101 Technical Report. Prepared for Ascot Resources Ltd., March 31, 2014, 109p.

Smit, H. (2000). Sampling and QA/QC Procedures, 1993 and 1994, Red Mountain Project, Stewart, British Columbia (Draft). Prepared for Wheaton River Minerals Ltd. 13p.

Smit, H. and Sieb, M. (1995). Red Mountain Project, Underground Sampling and Marc Zone Sampling. Lac Minerals Ltd. Internal Report. February 25, 1995. 11p.

Stanley, J.E. (1994a). Eco-Tech Contract Laboratory Inspection #1. Prepared for Lac Minerals, May 9, 1994. 5p.

Stanley, J.E. (1994b). Contract Laboratory Inspection #2 – July 18, 19. Prepared for Lac Minerals, August 19, 1994. 8p.

Stanley, J.E. (1994c). Contract Laboratory Inspection #2 Follow-up Report – Laboratory Sub-Sampling and Preparation Evaluation. Prepared for Lac Minerals Ltd., October 25, 1994. 8p.

Stone, D.M.R., and Godden, S.J. (2007). Updated Technical Report and Preliminary Economic Assessment on the Silver Coin Property, Stewart, British Columbia, Minefill Services, Inc. for Pinnacle Mines, Ltd., Vancouver, B.C., April 27, 2007, 160 p.

Tetra Tech (2010). Silver Coin Gold Project NI 43-101 Preliminary Economic Assessment Report, Stewart, British Columbia. Prepared for Pinnacle Mines Ltd., Vancouver, B.C. 207 p.

Tsang, L. (2013). 2012 Diamond Drilling Assessment Report on the Premier and Dilworth Properties, Stewart, B.C., Assessment Report Prepared for Ascot Resources Ltd., May 31, 2013.

Tuba, T. (2011). Internal: Diamond Drilling Summary Report on the Silver Coin Property, November 2011. 1,338p.

Tudor Gold (2016-2019). Electrum Project. Available at http://tudor-gold.com, accessed May 15, 2020.

Vincent, J. (2018). Premier-Dilworth Project: August 2018 QAQC Summary of Diamond Drilling Results. Internal memo to Ascot Resources Ltd., September 30, 2018, 25p.

Vincent, J. (2018). Premier-Dilworth Project: July 2018 QAQC Summary of Diamond Drilling Results. Internal memo to Ascot Resources Ltd., September 24, 2018, 19p.

Vincent, J. (2018). Premier-Dilworth Project: June 2018 QAQC Summary of Diamond Drilling Results. Internal memo to Ascot Resources Ltd., September 16, 2018, 16p.

Vincent, J. (2018). Premier-Dilworth Project: May 2018 QAQC Summary of Diamond Drilling Results. Internal memo to Ascot Resources Ltd., June 21, 2018, 13p.

Vincent, J. (2019). Premier-Dilworth Project: August 2019 QAQC Summary of Diamond Drilling Results, Internal memorandum to Ascot Resources, Inc.

Walus, A. (2007). Letter Report titled: Petrographic Report Silver Coin Property, January 4, 2007. 2p.

Westmin Resources Ltd. (1997). Confidential Information Memorandum Premier Gold Operations – Internal Report. Vancouver: Westmin Resources Ltd. and Midland Walwyn.

Wright (2009). Metallurgical Study on the Silver Coin Gold Project. Prepared for Pinnacle Mines Ltd., January 8, 2009, 247p.

Wright (2011). Silver Coin Gold Project Metallurgical Response. Prepared for Jayden Resources, November 7, 2001 (Draft), 55p.

28 CERTIFICATE OF QUALIFIED PERSON

28.1 Sue Bird, P.Eng.

I, Sue Bird, of Victoria, British Columbia, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I have a business address of 1752 Armstrong Ave., Victoria, British Columbia, V8R 5S6.
- I graduated with a Geologic Engineering degree (B.Sc.) from the Queen's University in 1989 and a M.Sc. in Mining from Queen's University in 1993.
- I am a member of the Association of Professional Engineers and Geoscientists of B.C. (No. 25007).
- I have worked as an engineering geologist for over 25 years since my graduation from university. My relevant experience includes:
  - acting as qualified person (QP) for the Resource Estimate on a number of deposits of various types including porphyry copper, skarns, epithermal Au, MVT, banded iron, coal, and laterite bauxite
  - due diligence and project evaluation for numerous projects throughout the world at various stages of development from exploration to operating mines
  - consultant for resource and reserve estimation and mine planning work for many metals and complex coal projects throughout British Columbia.
- I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that because of education, experience, independence and affiliation with a professional organization, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
- I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
- I am independent of Ascot Resources Inc. as well as the Vendor of Silver Coin deposit as defined in Item 1.5 of National Instrument 43-101.
- I visited the property from September 4 to 6, 2018 and June 17 to 20 2019.
- I am responsible for Sections 1 through 12, Section 14 and Sections 23, 25, and 26 pertaining to the PGP deposit.
- I have had no previous involvement with the property that is the subject of the Technical Report other than QP for Silver Coin, big Missouri, Martha Ellen, and Dilworth for the January 2019 NI 43-101 report.
- As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 22nd day of May, 2020 at Victoria, British Columbia.

"Original Signed and Sealed"

Sue Bird, M.Sc., P.Eng.
Owner, BRCC

28.2 Dr. Gilles Arseneau, P.Geo.

I, Dr. Gilles Arseneau, P. Geo., of North Vancouver, British Columbia, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am president of ARSENEAU Consulting Services Inc. with a business address at 900 – 999 West Hastings Street, Vancouver, British Columbia.
- I am a graduate of the University of New Brunswick with a B.Sc. (Geology) degree obtained in 1979, the University of Western Ontario with an M.Sc. (Geology) degree obtained in 1984 and the Colorado School of Mines with a Ph.D. (Geology) obtained in 1995.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#23474).
- I have practiced my profession continuously since 1995.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Red Mountain Deposit includes extensive experience modelling gold mineralization similar to the Red Mountain deposit for projects in North and South America.
- I am responsible for the preparation of the parts of Sections 1, 4 to 12, 14, and 25 to 27 of this Technical Report that are pertaining to the Red Mountain deposit.
- I have visited the property during the period of October 23 and 24, 2018.
- I have had prior involvement with the subject property having authored a previous technical report titled "2019 Mineral Resource Update for the Red Mountain Gold Project, Northwestern BC, Canada" dated November 22, 2019 with an effective date of August 30, 2019 prepared for Ascot Resources Ltd.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at North Vancouver, British Columbia.

"Original Signed and Sealed"

Dr. Gilles Arseneau, P.Geo.
President
ARSENEAU Consulting Services Inc.

28.3 Aleksandar Petrovic, P.Eng.

I, Aleksandar Petrovic, P.Eng. of Vancouver, British Columbia, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020 do hereby certify that:

- I am a Senior Process Engineer with Sedgman Canada Limited with a business address at Unit 1630 – 505 Burrard Street, Vancouver, BC, Canada, V6B 4N9.
- I am a graduate of Belgrade University, Department of Mining and Geology in Belgrade, Serbia (M.Sc. Degree in Mineral Processing, 1991).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License # 31787).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to Premier & Red Mountain gold project includes more than 28 years experience in gold and base metals mining operations, consulting engineering and applied metallurgical research.
- I am responsible for the preparation of Sections 13, 17, 21.1.4, 21.2.2, 21.3.2, 21.4.3 and have provided input into Sections 1, 25, 26, and 27 of this Technical Report.
- I have not visited the Premier & Red Mountain Gold Project property.
- I have had prior involvement with the property that is the subject of the Technical Report, while working on NI 43-101 Red Mountain Gold Project Feasibility Study and Premier Gold Project Scoping Study which has been effectively replaced by this Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Premier & Red Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May 2020 at Vancouver, British Columbia.

"Original Signed and Sealed"

Aleksandar Petrovic, P.Eng.
Senior Process Engineer
Sedgman Canada Limited

28.4 Frank Palkovits, P.Eng.

I, Frank Palkovits, of Sudbury, Ontario, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 15, 2020, do hereby certify that:

- I am a President of Mine Paste Ltd., with a business address at 26 Windsor Crescent, Sudbury, ON. Canada.
- I am a graduate of Laurentian University, (B.Eng., Mining, 1988) and Cambrian College (Geological Tech., 1981).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of (License No. 90276379)
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to mining includes 20 years in operations (engineering/mine design, geology, underground miner and front-line supervisor) as well as 20 years in mine design and mining plants consulting and design engineering for Canadian and global interests.
- I am responsible for the preparation of Sections 15, 16, and 21.1.2; 21.2.1; 21.4.3 of this Technical Report, as well as contributing to the Section 1 (Summary) and Sections 25 and 26.
- I have visited the property during the period June 29 and 30, 2018.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Premier Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at Sudbury, Ontario.

"Original Signed and Sealed"

Frank Palkovits, (P.Eng., Mining)
President
Mine Paste Ltd.

28.5 Jim Fogarty P.Eng.

I, Jim Fogarty, of Vancouver, British Columbia, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am currently employed as a Senior Engineer with Knight Piésold Ltd. with a business address at 1400-750 West Pender Street, Vancouver, British Columbia, V6C 2T8.
- I am a graduate of the National University of Ireland, Galway with a Bachelor of Engineering (B.Eng.) in Civil Engineering (2010).
- I am a Professional Engineer in good standing of the Engineers and Geoscientists of British Columbia in the area of civil engineering (License No. 44041).
- I have practiced my profession continuously since 2011. My experience includes tailings, waste and water management designs, mine planning and permitting, cost estimates and technical report writing for mine developments in Canada, USA, Europe and South America.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43101.
- I am responsible for the preparation of Sections 1.18.3, 1.18.4, 18.3, 18.5, 18.6, 21.1.9, 21.2.3, 21.3.3, 21.4.11, 25.5.1, and 26.4.1 of this Technical Report.
- I have visited the property during the period of June 17 to 19, 2019.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Premier Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020, at Vancouver, British Columbia.

"Original Signed and Sealed"

Jim Fogarty, B.Eng. (Civil), P.Eng.
Senior Engineer
Knight Piésold Ltd.

28.6 Robert Marsland, M.Sc., P.Eng.

I, Robert Marsland, of Nelson, BC, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am an Environmental Engineer with Marsland Environmental Associates Ltd. with a business address at Nelson, British Columbia.
- I am a graduate of McGill University (B.Eng. (Chemical), 1986) and the University of Alberta (M.Sc., Environmental Engineering (1991)).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License # 25110).
- I have practiced my profession continuously since 1990, working on the environmental aspects of mining projects, primarily in Canada and Perú.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I am responsible for the preparation of Sections 1.20, 18.7, 20, and 25.5 of this Technical Report.
- I have visited the property during the period, in July 2019.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Premier Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at Nelson, British Columbia.

"Original Signed and Sealed"

Rob Marsland, M.Sc., P.Eng.
Senior Environmental Engineer
Marsland Environmental Associates Ltd.

28.7 Soren Jensen, P.Eng.

I, Soren Jensen, of Vancouver, British Columbia, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am a Senior Environmental Engineer with a business address at 2200 – 1066 West Hastings Street, Vancouver BC, V6E 3X2.
- I am a graduate of University of British Columbia, B.A.Sc.,2002.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #144012).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to water treatment for mine operations includes 18 years of professional experience and post-graduate research in the water treatment and water management as well as design, commissioning and operation of water treatment plants at mine operations.
- I am responsible for the preparation of Sections 18.6.4; 18.8; 21.1.10; 21.4.12; 25.5.2; and 26.4.2 of this Technical Report.
- I have visited the property during the period of Jul 16 and 17, 2019.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May 2020 at Vancouver, British Columbia.

"Original Signed and Sealed"

Soren Jensen, PEng.
Senior Environmental Engineer
SRK Consulting (Canada) Inc.

28.8 Brendon Masson, P.Eng.

I, Brendon Masson, of Calgary, Alberta, as a co-author of the "*Premier and Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am a Senior Engineer at McElhanney Ltd. with a business address at 100 402 11th Avenue SE Calgary, Alberta, T2G 0Y4.
- I am a graduate of the University of New Brunswick, (Civil Engineering, 2007)
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #36610).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to road design construction includes over 10 years practicing in as a Professional Engineer in the mining and oil and gas industries.
- I am responsible for the preparation of Sections 18.2, 18.3, 21.1.3, and portions of 25, 26, and 27 of this Technical Report.
- I have visited the property during the period June 25, 2019.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Premier Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at Calgary, Alberta.

"Original Signed and Sealed"

Brendon Masson, PEng
Engineer
McElhanney Ltd.

28.9 Shervin Teymouri, P.Eng.

I, Shervin Teymouri, of Vancouver, BC, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am a Senior Engineer with Sacré-Davey Engineering Inc. with a business address at 15 Mountain Hwy, North Vancouver, BC V7J 2K7
- I am a graduate of University of British Columbia, (Geological Engineering 2005).
- I am a member in good standing of the Engineers and Geoscientists of BC (License #35469).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to Mine Project Economic Analysis includes 14 years of Economic Analysis.
- I am responsible for the preparation and review of Sections 19 and 22 of this Technical Report.
- I am responsible for partial preparation and review of Sections 1, 25, and 26 of this Technical Report.
- I have not conducted a site visit to the property.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Premier Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at Vancouver, BC.

"Original Signed and Sealed"

Shervin Teymouri, P.Eng.
Senior Engineer
Sacré-Davey Engineering Inc.

28.10 Frank Grills, P.Eng.

I, Frank Grills, of North Vancouver, BC, as a co-author of the "*Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada*" (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am a Senior Project Manager with a business address at Sacré-Davey Engineering Inc., 315 Mountain Highway, North Vancouver, BC, V7J 2K7.
- I am a graduate of University of Surrey (B. Sc. (Hons), 1979 and the University of Witwatersrand (M.Sc. (Eng), 1985).
- I am a member in good standing of the Engineers and Geoscientists of British Columbia (Registration #29591).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to studies include 33 years' experience in engineering, estimating, construction management and project management of major process plants and studies. Recent studies I have worked on include the Ixtaca Project and Kerr Sulphurets Mitchell Project.
- I am responsible for the preparation of Sections 1.1, 1.2, 1.3, 1.24, 2.1 to 2.7, 3,18.1,18.4, 18.9, 18.10, 18.11, 18.12, 18.13, 18.14, 1.21 (for non-process infrastructure and Site Services only, excluding access roads, TSF and WTP), 21.1.1; 21.1.2; 21.1.3; 21.1.7; 21.1.8; 21.1.9; 21.1.10; 21.1.11; 21.1.14; 21.1.15; 21.2.5; 21.3.4; 21.4.1; 21.4.4 to 21.4.10, 24.1 to 24.3, and portions of Section 25 and 26 of this Technical Report.
- I have visited the property on November 26-27, 2019.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have not been involved with the Premier & Red Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at North Vancouver, BC.

"Original Signed and Sealed"

Frank Grills, P.Eng.
Project Manager
Sacré-Davey Engineering Inc.

28.11 Ken Savage, P.Eng.

I, Ken Savage, of North Vancouver, British Columbia, as a co-author of the *"Premier Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia, Canada"* (the Technical Report), with an effective date of April 15, 2020, prepared for Ascot Resources Ltd., and dated May 22, 2020, do hereby certify that:

- I am a Senior Civil Engineer with Sacré-Davey Engineering Inc. with a business address at 315 Mountain Highway, North Vancouver, British Columbia.
- I am a graduate of the University of British Columbia, (BASc (Civil), 1987).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #18878).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to studies includes 33 years of Civil Engineering Design.
- I am responsible for the preparation of Sections 18.2.5, 18.2.6, 18.2.7, 21.2.4, 21.3.5, 21.4.4 of this Technical Report.
- I have not visited the property.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have not been involved with the Premier Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101 F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of May, 2020 at North Vancouver, British Columbia.

"Original Signed and Sealed"

Ken Savage, P.Eng.
Senior Civil Engineer
Sacré-Davey Engineering Inc.

29 DATE AND SIGNATURE PAGE

The undersigned prepared this Technical Report, titled "Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report, British Columbia," and dated May 22, 2020, in support of the public disclosure for public listing. The format and content of this report conforms to National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

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