NI 43-101 Feasibility Study Technical Report for The Red Mountain Project British Columbia, Canada

JDS Energy & Mining Inc.

IDM MINING

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1 Executive Summary

1.1 Introduction

IDM Mining Ltd. (IDM or IDM Mining) commissioned JDS Energy & Mining Inc. (JDS) to complete a feasibility study (FS) for the Red Mountain Gold Project (Red Mountain or Project) located in northwestern British Columbia near the town of Stewart. The purpose of this study is to update and define the project to a FS level and to declare a mineral reserve estimate. This technical report contains a summary of the FS results.

JDS conducted a preliminary economic assessment on the project in 2016 and produced a technical report with an effective date of July 12, 2016 titled "NI 43-101 Preliminary Economic Assessment (PEA) for The Red Mountain Project, British Columbia, Canada" (JDS 2016). The main changes in this FS from the JDS 2016 PEA study are:

- Updated mineral resource estimate – conducted by Gilles Arseneau of ARSENEAU Consulting Services (ACS), which was based on additional drill results and subsequent reinterpretation which changed the overall mineral resources estimates as per Table 1-1.

Table 1-1: 2017 Mineral Resource Estimate with Comparison to the Historical 2016 Estimate

Resource by Classification	Tonnes	In-Situ Gold Grade (g/t)	In-Situ Silver Grade (g/t)	Contained Gold (oz)	Contained Silver (oz)
2016 Historical Mineral Resource Estimate					
Measured and Indicated	1,641,600	8.36	26.0	441,500	1,379,800
Inferred	548,100	6.10	9.0	107,500	153,700
2017 Mineral Resource Estimate					
Measured and Indicated	2,074,700	8.75	24.8	583,700	1,655,700
Inferred	324,700	6.21	10.1	64,800	105,500
2017 to 2016 Mineral Resource Comparison					
Measured and Indicated	433,100	0.39	-1.2	142,200	275,900
Inferred	-223,400	0.11	1.1	-42,700	-48,200

Source: ACS (2016 & 2017)

- Revised process flow sheet – change from flotation and concentrate leach to whole ore leaching to improve gold recoveries across mineralization types;
- Addition of a water treatment plant;
- Addition of a construction camp in Stewart, BC;
- Revised mine production schedule – increased life-of-mine (LOM) tonnes, mine life and recovered metal, based on the updated 2017 mineral resource estimate;

- Revised mine operating schedule – change from a seasonal (8-month) mining operation to continuous year-round mining;
- Updated capital and operating costs based on improved detail, information, designs and quotes;
- Updated metal prices and exchange rate to:
    - US$1,250/oz Au;
    - US$17.00/oz Ag;
    - 0.76 US$:C$ exchange rate; and
- Design and engineering considerations to minimize the environmental footprint and improve operational sustainability and mine closure.

1.2 Project Description & Ownership

1.2.1 Property Description

The Red Mountain gold property is situated in northwestern British Columbia approximately 18 kilometers (km) east-northeast of Stewart (Figure 1-1). The Project is located west of the Cambria Ice Field and north of the Bromley Glacier at elevations ranging between 500 and 2,000 metres (m). On NTS map sheets 103P/13 and 104A/4, the property is centred at 55º59'4"N, 129º45'37"W. The UTM coordinates are 452,450 mE, 6,250,325 mN in Zone 9 (NAD 83).

The area is characterized by rugged, steep terrain with weather conditions typical of the north coastal mountains including significant (+2 m) snow accumulation in the winter. Access to the site is presently by helicopter from Stewart with a flight time of 10 to 15 minutes. A historical road extends for approximately 13 km along Bitter Creek Valley but stops approximately 7 km from the proposed mine site.

The Red Mountain gold property consists of 47 contiguous mineral claims totalling 17,125 hectares (ha). The mineable deposits are located roughly in the middle of the mineral claims. No significant risks are identified which would affect access, title, or the right or ability to perform work on the property.

Figure 1-1: Red Mountain Location Map

Source: IDM (2017)

The property falls within the Nass Area and Nass Wildlife Area as set out in the Nisg̱a'a Final Agreement (NFA). Pursuant to the NFA, Nisg̱a'a Nation, as represented by the Nisg̱a'a Lisims Government (NLG) has Treaty rights to the management and harvesting of fish, wildlife, and migratory birds within the Nass Area and Nass Wildlife Area.

The Project contains four main mineable mineralized zones; Marc, AV, JW, and 141. Access to the deposits will be via two main portals, one existing portal at about 1,850 metres above sea level (masl) and a second planned portal at about 1,650 masl. The deposits range in thickness from < 2 m to 40 m, and average 16 m. Approximately 1,500 m of underground development currently exists and will be utilized for initial mine access.

The underground mine is envisioned to produce at an average rate of approximately 1,000 tonnes per day (t/d) and will operate year-round. Planned mining methods are sublevel longhole stoping for steeper dipping zones, and cut and fill (C&F) for shallower dipping areas. Both methods will use waste rock backfill, cemented when necessary. Ore will be trucked from the mine located near the top of Red Mountain to the process plant, about 11 km away located in the Bitter Creek Valley.

The processing plant and tailings management facility (TMF), a fully lined thickened tailings facility, will be located adjacent to each other at about 500 masl at the Bromley Humps location, approximately 15 km from Highway 37A.

The processing plant will operate year-round at an average rate of 1,000 t/d. The processing plant will consist of a run-of-mine (ROM) stockpile, three stages of crushing, a ball mill, verti-mill, carbon-in-leach (CIL) tanks and an adsorption, desorption and refining circuit (ADR). Red Mountain mineralized material is considered hard (19.3 kWhr/t Bond Ball mill index) and a fine grind (P_{80} 20µ) will be required to attain the projected recoveries averaging 90.9% for gold and 86.3% for silver.

Approximately 430,000 oz of gold and 1,185,000 oz of silver will be recovered over the six-year mine life from 1.95 million tonnes (Mt) of feed. The average mine grades are 7.53 grams per tonne (g/t) gold and 21.86 g/t silver.

1.2.2 Ownership

On April 15, 2014, IDM entered into an option agreement for the Red Mountain gold Project with Seabridge. In May 2017, IDM completed the acquisition of the property and acquired 100% interest, subject to certain underlying agreements and royalties, pursuant to the option agreement.

The property has two royalties; Franco Nevada Corporation (Franco) holds a 1% Net Smelter Return (NSR) royalty and a 2.5% NSR royalty is payable to Wotan Resources Corp. A $50,000 advance royalty is payable to Wotan annually.

Upon the commencement of commercial production, IDM will make an additional one-time $1.5 million cash payment to Seabridge and Seabridge will also retain a gold metal stream on the Red Mountain Gold Project to acquire 10% of the annual gold production from the property at a cost of US$1,000 per ounce up to a maximum of 500,000 ounces produced (50,000 ounces 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.

1.3 Geology & Mineralization

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt. There are three primary stratigraphic elements in Stikinia and all are present in the Stewart 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. Many primary textures are preserved in rocks from all of these groups, and mineralogy suggests that the regional metamorphic grade is probably lower greenschist facies.

Mineralized zones consist of crudely tabular, northwesterly trending and moderately to steeply southwesterly dipping gold and silver bearing iron sulphide stockworks. Pyrite is the predominant sulphide; however, locally pyrrhotite is important. The stockworks zones are developed primarily within the Hillside porphyry and to a lesser extent in rafts of sedimentary and volcaniclastic rocks.

The stockwork zones consist of pyrite microveins, coarse-grained pyrite veins, irregular coarse-grained pyrite masses and breccia matrix pyrite hosted in a pale, strongly sericite-altered porphyry. Vein widths vary from 0.1 cm to approximately 80 cm but widths of 1 to 3 cm are most common. The veins are variably spaced and average 2 to 10 per metre. 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.

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 sulphosalts. The stockwork zones are surrounded by more widespread zone of disseminated pyrite and pyrrhotite alteration.

1.4 History, Exploration & Drilling

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. staked claims in 1988 and optioned the property to Bond Gold Canada Inc. (Bond) in 1989.

In that year, gold mineralization in the Marc and Brad zones were discovered by drilling. LAC Minerals Ltd. (LAC) acquired Bond in 1991. Surface drilling on the Marc, AV, and JW zones continued in 1991, 1992, 1993, and 1994. Underground exploration of the Marc zone was conducted in 1993 and 1994. In 1995, LAC was acquired by Barrick who subsequently optioned the property to Royal Oak Mines Ltd. (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 who 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 in 2014.

Table 1-2 provides a chronological summary of recent exploration efforts on Red Mountain.

Table 1-2: Red Mountain 1988-2017 Chronological Exploration Summary

Year	Description of Exploration
1988-89	Staking of Red Mountain by Wotan Resources Inc.
1989	Red Mountain and Wotan properties optioned to Bond. Discovery of gold-silver mineralization by drilling in the Marc zone (3,623 m); airborne EM and magnetic survey.
1990	Exploration of Marc zone and adjacent area (11,615 m of drilling) by Bond.
1991	LAC acquired 100% of Bond. 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 and 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	Red Mountain Gold Project acquired by Barrick following Barrick's take-over of LAC. No exploration work completed by 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	NAMC purchased the property and Project assets from Price Waterhouse Coopers, conducts detailed relogging of existing drill core and constructs a geological model for resource estimation purposes and exploration modelling.
2002-2012	Seabridge purchases property, completes two Preliminary Assessment Studies (PEA)
2012-2013	Banks options property, two surface drill holes completed, completes PEA study.
2014	IDM options property and drilled 12 diamond drill holes
2016	IDM drilled 11 surface diamond drill holes and 51 underground infill and resource expansion drill holes and conducted geotechnical and metallurgical investigations from the holes.
2017	Continued metallurgical test work was conducted on the 2016 drilling samples and a new UG definition and expansion drilling campaign was started however, no results from the 2017 resource drilling were used in this report.

Source: ACS (2017)

1.5 Metallurgical Testing & Mineral Processing

Multiple test programs were completed between 1991 and 2015. The most recent test program was completed in 2016-2017 by Base Metallurgical Laboratories Ltd. (Base Met) located in Kamloops, BC. The feasibility-level metallurgical test program was completed on variability and composite samples for Marc, AV, JW and 141 zones. Initially the test work focused on the 2016 PEA flowsheet, which included rougher flotation followed by concentrate leach. Pyrrhotite levels varied significantly in the deposit and were found to affect flotation performance due the reactivity and oxidation of the material. As a result, whole ore leach (WOL) became the focus of the program. Optimization continued primarily on the Marc zone composite and was confirmed with the AV, JW and 141 samples. The final flowsheet included two stages of grinding to target a product size of 80% passing (P_{80}) 25 µm, followed by CIL, and acid wash, stripping and electrowinning for the recovery of gold and silver doré.

Table 1-3 presents the estimated metallurgical recoveries based on the correlation between cyanide concentration and gold or silver recovery. The 141 zone recoveries are a weighted average of test work results from four composites. The overall projected recovery is a weighted average of recovery by zone and projected tonnages based on the mine plan.

Table 1-3: Estimated Metallurgical Recoveries

Recovery by Zone	Au (%)	Ag (%)
Marc Zone	92.8	90.1
AV Zone	88.1	78.3
JW Zone	92.1	90.3
141 Zone	89.9	84.9
Overall Recovery Based on the Projected Mine Plan	**90.9**	**86.3**

Source: JDS and Basemet Laboratories (2017)

1.6 Mineral Resource Estimates

1.6.1 Drill Hole Database

Numerous resource estimates were completed from 1989 to present. During 2000, NAMC conducted a detailed review of all data, re-logged all core within a 20 m envelope of the Marc, AV, and JW mineralized zones and reviewed all exploration holes for potential inclusion into the resource. An extensive quality control and quality assurance (QA/QC) review was completed on all exploration work, and a comparative analysis was performed on drill hole data, underground bulk sampling, and geology. The 2000 NAMC resource was reviewed, cross checked, and verified for accuracy in May 2014. IDM drilled 12 core holes on the property in 2014 and 62 holes in 2016. The 74 IDM drill holes combined with the past historical drilling on the property form the basis for the latest resource estimate in Table 1-4.

Table 1-4: Mineral Resource Statement, Effective January 23, 2017 (3 Au g/t Cut-off Grade)

Zone	Tonnage (tonnes)	In-situ Gold Grade (g/t)	In-situ Silver Grade (g/t)	Contained Gold (troy ounces)	Contained Silver (troy ounces)
Marc Zone					
Measured	682,000	10.62	38.3	232,800	840,500
Indicated	32,300	9.69	32.6	10,100	33,800
Inferred	4,500	10.43	43,4	1,500	6,200
AV Zone					
Measured	519,400	7.73	20.0	129,100	334,500
Indicated	236,300	9.07	19.2	60,700	146,300
Inferred	43,300	8.13	15.4	20,400	21,400
JW Zone					
Measured	44,600	10.11	13.2	14,500	18,900
Indicated	314,200	8.54	18.0	86,300	181,600
Inferred	111,700	6.78	7.4	24,400	26,500
141 Zone					
Indicated	188,600	4.91	11.1	29,700	67,300
Inferred	15,100	4.67	4.7	2,300	2,300
Marc Footwall					
Indicated	18,100	6.15	12.1	3,600	7,000
Inferred	12,600	5.12	6.4	2,100	2,600
Marc Outlier Zone					
Indicated	4,200	3.43	16.8	500	2,300
Inferred	7,300	6.54	27.4	1,500	6,400
Marc NK Zone					
Indicated	10,700	5.58	7.6	1,900	2,600
Inferred	7,300	5.98	9.0	1,400	2,100
JW Lower Zone					
Indicated	24,300	8.15	26.6	6,400	20,800
Inferred	2,000	13.94	9.3	900	600
AV Lower Zone					
Inferred	42,500	5.55	6.6	7,600	8,300
132 Zone					
Inferred	78,700	4.73	11.5	12,000	29,100
Total Measured & Indicated	**2,074,700**	**8.75**	**24.8**	**583,700**	**1,655,700**
Total Inferred	**324,700**	**6.21**	**10.1**	**64,800**	**105,500**

Source: ACS (2017)

Note: 3 g/t Au is calculated as the cut-off grade for underground longhole stoping.

1.7 Mineral Reserve Estimate

The effective date for the Mineral Reserve estimate contained in this report is June 26, 2017 and was prepared by JDS. All Mineral Reserves in Table 1-5 are Proven and Probable Mineral Reserves. The Mineral Reserves are not in addition to the Mineral Resources, but are a subset thereof.

The Qualified Person (QP) has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves development.

Table 1-5: Red Mountain Mineral Reserve Estimate

Category	Diluted Tonnes (kt)	Diluted Au Grade (g/t)	Contained Au Ounces (kOz)	Diluted Ag Grade (g/t)	Contained Ag Ounces (kOz)
Proven	1,308	7.82	329	25.09	1,055
Probable	645	6.93	144	15.32	318
Total	**1,953**	**7.53**	**473**	**21.86**	**1,373**

Source: JDS (2017)

Notes:

The Qualified Person for the Mineral Reserve estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.

Mineral Reserves were estimated using a US$1,200/oz gold price and gold cut-off grade of 3.55 g/t for longhole mining and 4.10 g/t for development and cut and fill mining.

Other costs (all C$) and factors used for gold cut-off grade determination were mining, process and other costs of $128 /t for longhole mining and $148 /t for cut and fill mining, transport and treatment charges of $6.00 /oz Au. A royalty of $53.10 /oz Au and a gold metallurgical recovery of 89.3% were assumed.

Silver was not used in the estimation of cut-off grades but is recovered and contributes to the revenue stream.

Tonnages are rounded to the nearest 1,000 t, gold grades and silver grades are rounded to two decimal places. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces. Rounding as required by reporting guidelines may result in summation differences.

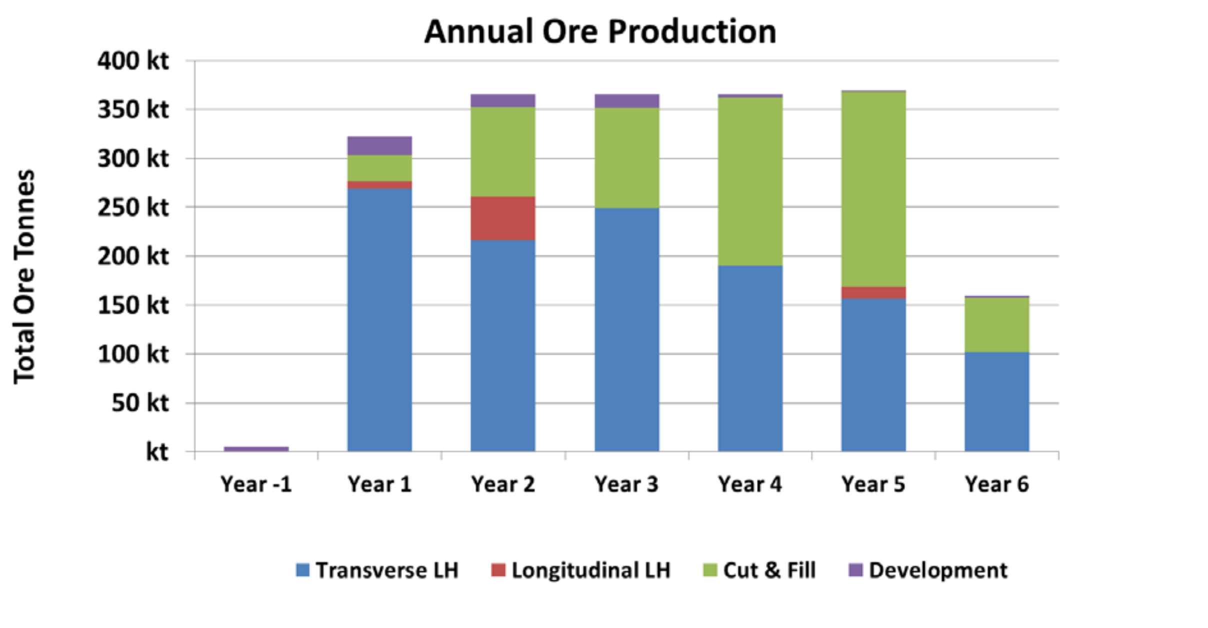
1.8 Mining

The mine plan is based on a ramp access underground mining operation, producing an average annual rate of 1,000 t/d of ore from a blend of mining methods:

- A combination of transverse and longitudinal longhole stoping for mining blocks dipping steeper than 55°, which represents 63% of the reserves. This is the preferred mining method from a productivity and operating cost perspective;
- Cut and fill for mining areas with dips of less than 55° and zones not amenable to longhole stoping, is more selective and represents 33% of reserves; and
- The remaining 4% of the potentially mineable tonnage comes from access and stope cross-cut development.

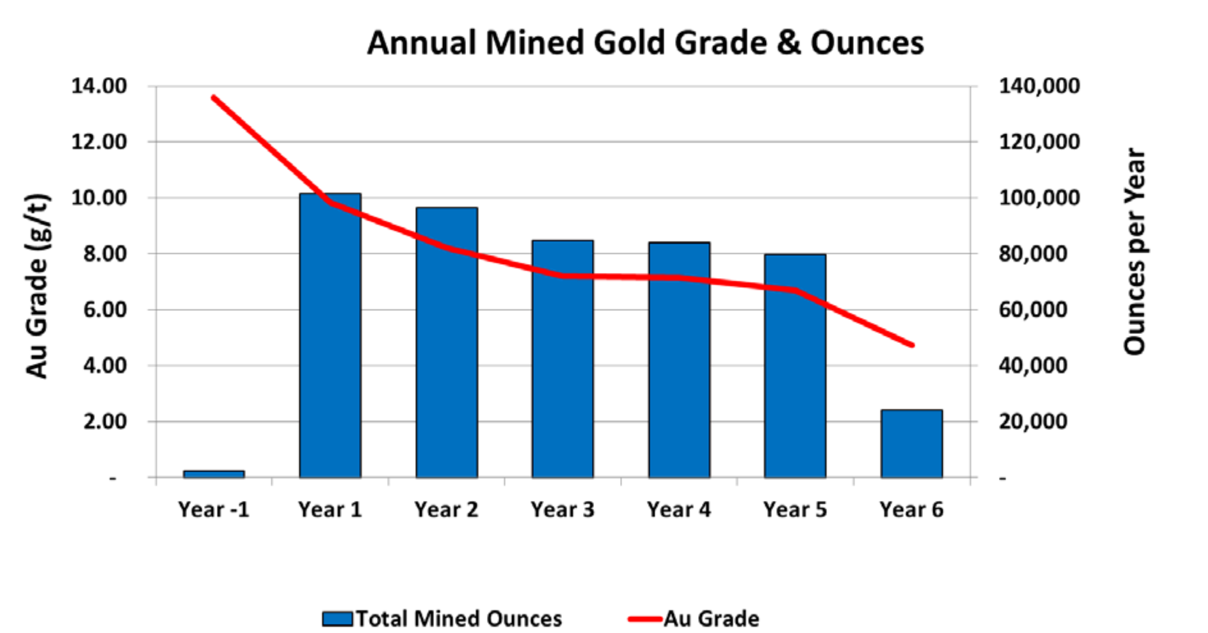
Mining recovery and dilution factors were applied to each mining shape based on the mining method used. Average external dilution for the production stopes was calculated to be 12%. Annual ore production by mining method is shown in Figure 1-2 and mined gold grade and contained ounces are shown in Figure 1-3.

Figure 1-2: Annual Ore Production by Mining Method



Source JDS (2017)

Figure 1-3: Annual Mined Gold Ounces & Grade



Source JDS (2017)

The deposit will initially be accessed from the existing portal plus an ancillary new portal accessing the top level of the mine, which will be used for ventilation exhaust and a secondary escape way. A third lower access portal, to be used for haulage, will be added in year 1 of the mine life. Access ramps will be driven at a maximum grade of 15% with a 4.5 m by 4.5 m profile to accommodate 30-tonne haul trucks.

Level spacing for the longhole zones will vary up to a maximum of 25 m. Mineralized zone development will be driven using a 4.0 m x 4.0 m profile. Cut and fill zones will be accessed by attack ramps with a maximum gradient of +/- 17%. Cemented rock fill (CRF) using crushed waste rock will be utilized in a majority of the completed stopes, with rock fill (RF) used in secondary cut and fill stopes and longhole stopes at the end of the zones. Ore/waste passes are planned for the Marc and AV zones once the lower access drift is completed.

The ventilation network will consist of primary ventilation fans with mine air heaters located outside of the upper portal, pushing air down the ramp and across the levels exhausting out the second upper and the lower portals. Level ventilation will be controlled by a combination of regulators, ducting, and auxiliary fans.

Mine water and ground water will be collected at the various level sumps and allowed to drain down via gravity to the main pump stations placed at strategic locations in the mine. Generally, there will only be two main pump stations in operation at any time. Pump stations have been designed for a peak inflow capacity of 10,000 m^3/day. Average inflow volumes were estimated to be 3,450 m^3/day.

The FS mine plan focusses on accessing and mining higher value material early in the mine life. The plan commences with the mining of Marc zone, followed by AV, and then JW and 141 zones. The mine production rate will be targeting 1,045 t/day, over 350 assumed operating days per year.

The production plan for the Red Mountain Project is summarized in Table 1-6.

Table 1-6: Mine Production Schedule

Parameter	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Ore Tonnage	kt	5	322	366	366	366	368	159	**1,953**
Gold Grade	g/t	13.59	9.82	8.20	7.20	7.14	6.70	4.72	**7.53**
Silver Grade	g/t	46.26	34.44	27.14	19.63	15.45	17.54	13.31	**21.86**
Gold Ounces	koz	2	102	96	85	84	80	24	**473**
Silver Ounces	koz	8	356	319	231	182	208	68	**1,373**
Lateral Development	m	1,100	4,800	5,000	5,000	5,100	5,100	2,700	**28,900**
Vertical Development	m	50	200	200	100	100	-	50	**700**
Cemented Rock Fill	m^3	-	90,000	90,000	84,000	68,000	62,000	33,000	**427,000**
Waste Fill	m^3	-	21,000	37,000	41,000	54,000	58,000	21,000	**232,000**

Source JDS (2017)

1.9 Recovery Methods

The results of the metallurgical test work, together with financial evaluation data, were used to develop metallurgical design criteria and the selected flowsheet for the process facility.

The test work has shown that Red Mountain mineralization can be treated using conventional mineral processing techniques for the recovery of gold and silver. A trade-off study was conducted to compare processing the mill feed material using either whole ore leach or a flotation/regrind/leach circuit. The CIL circuit was selected based on the results of the trade-off study and metallurgical test results.

The plant will consist of the following unit operations:

- 3-stage crushing;
- Primary and secondary grinding;
- Pre-leach thickening and CIL;
- Cyanide destruction;
- Carbon processing and gold refining; and
- Tailings disposal at the TMF.

The grinding circuit product size is targeted at approximately P_{80} of 25 µm, with CIL to recover gold and silver. The crushing circuit will operate at an availability of 75% while the milling and CIL circuits will operate 24 h/d, 365 d/y at an availability of 92%.

1.10 Infrastructure

The Project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 15 km year-round access road from Highway 37A to the processing plant site;
- Approximately 11 km year-round haul road from the processing plant site to the upper and lower mining portals near the top of Red Mountain;
- Electrical connection to BC Hydro power grid and a transmission line at 138 kV adjacent to the access road;
- Distribution powerline at 25 kV from processing plant site to the upper mine portal;
- Process plant located at Bromley Humps;
- TMF and impoundment located at Bromley Humps;
- Water management ponds to manage contact water runoff from around the Project site;
- Diversion ditches to divert non-contact water to the maximum practical extent;
- Temporary development of waste rock storage areas prior to being re-handled into the underground workings as backfill;
- Administration office, mine dry, maintenance shop and warehouse facilities;
- Mine operations office and emergency facilities at the mine portals;
- Tailings effluent water treatment plant;

- Process and fire water storage and distribution; and
- Temporary construction camp located in Stewart.

1.11 Environment & Permitting

The Project has been designed to minimize short- and long-term environmental impacts, and to maximize lasting benefits to local communities, employees, and shareholders. The goal of the company 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. IDM fully respects the traditional knowledge and culture of the Aboriginal peoples who have historically used or travelled through the Red Mountain Project area and IDM will continue to engage in a meaningful and respectful way with Nisg̲a'a Nation leadership and citizens.

The Project area watershed is relatively undisturbed by human activities with the exception of an access road that was constructed in the late 1990s but is currently decommissioned. A key objective of the company is to protect and retain the integrity of the current watershed and local ecosystem during the construction and operation phases of the Project. Upon closure and reclamation of the Project, the goal will be to return the relatively small-disturbed areas to the condition of pre-mine existence.

Pursuant to Section 3(1) of the Reviewable Projects Regulation, the proposed production capacity for the Project exceeds the criteria of 75,000 tonnes per annum (t/a) of mineral material for a new mineral mine and is required to undergo a provincial and federal environmental assessment under the British Columbia Environmental Assessment Act (BCEAA) and the Canadian Environmental Assessment Act (CEAA). IDM initiated this assessment process in October 2015 with the filing of a Project Description Report.

Since that time, a number of steps in the process have been undertaken successfully and IDM has filed a Project Application Report in July 2017 that fulfills the requirements of the federal and provincial environmental assessment processes. Approval for the Project under BCEAA and CEAA is expected in the second quarter of 2018. Provincial permitting for the Project is being pursued in a synchronous manner with the environmental assessment process.

Extensive work has been undertaken over the past 25 years on the geochemistry and weathering characteristics of the rock, both ore and waste, which is found at Red Mountain. Tailings and waste rock have been characterized as having potential for metal leaching/acid rock drainage (ML/ARD) over extended periods of time. Tailings process water is expected to contain residual metals and ammonia from destruction of cyanide solutions. The Project incorporates appropriate design features and mitigation measures consistent with best practices for waste and water management to address these issues including:

- Fully lined TMF with seepage collection and pump-back systems;
- Water treatment plant to treat effluent from the tailings pond during mine operations;
- Water collection ponds to control suspended sediment concentrations in seepage and runoff associated with the waste rock stockpiles and groundwater discharged from the mine;
- Backfilling of all underground development rock into the underground mine as part of the mining process; and

- Appropriate handling and management practices for all water and waste generated at the site.

At closure, the TMF supernatant pond will be drained and a geosynthetic liner installed over the surface of the exposed tailings beach. A graded earth-fill/rock-fill cover will be constructed on top of the liner and revegetated to facilitate runoff from the surface of the reclaimed TMF towards a permanent closure spillway and to minimize infiltration. The three underground portals will be hydrostatically sealed with engineered bulkheads to allow the mine to flood. Infrastructure will be removed and disturbed sites re-graded to natural slopes. The access roads will be deactivated in accordance with future permit conditions. A full closure and reclamation plan will be developed as part of the environmental assessment and provincial permitting process.

1.12 Operating & Capital Cost Estimates

1.12.1 Capital Cost Estimate

The capital cost (CAPEX) estimate includes all costs required to develop, sustain, and close the operation that has a planned six-year operating life. The construction schedule is based on an approximate 15-month build period. The accuracy of this CAPEX estimate is in accordance with the level of detail for an AACE Class 3 estimate. All currency in this report is in Canadian dollars (C$ or $) unless otherwise stated.

The summary CAPEX estimate is shown in Table 1-7. The initial or pre-production CAPEX is $135.7 M, with sustaining CAPEX totalling $66.8 M. Costs are expressed in Canadian dollars with no escalation (Q2-2017 dollars).

Table 1-7: Capital Cost Summary

Area	Pre-Production (M$)	Sustaining (M$)	Total (M$)
Mining	11.3	38.3	49.6
Site Development	9.0	2.2	11.2
Mineral Processing	37.7	0.4	38.0
Tailings Management	7.2	10.9	18.1
Infrastructure	23.7	-	23.7
Off-site Infrastructure	2.8	-	2.8
Project Indirects	9.3	-	9.3
EPCM	13.0	-	13.0
Owner's Costs	9.1	-	9.1
Subtotal Pre-Contingency	**123.0**	**51.7**	**174.7**
Contingency	12.7	5.2	17.9
Subtotal (incl. Contingency)	**135.7**	**56.9**	**192.6**
Closure Costs	-	12.4	12.4
Closure Contingency	-	1.2	1.2
Salvage Value	-	(3.8)	(3.8)
Total Capital Costs	**135.7**	**66.8**	**202.4**

Source: JDS (2017)

Preparation of the capital cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency, and uses defined and proven project execution strategies. The estimates were developed using first principles, applying directly-related project experience, and the use of general industry factors. Almost all of the estimates used in this Project were obtained from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The initial capital estimates include all pre-production mining activities in year -1 and are based on leased mining equipment. The capital estimate includes the down-payment on the leased equipment only, and lease payments are carried as mining operating costs.

The CAPEX estimate includes the costs required to develop, sustain, and close the operation for the planned six-year mine life, which includes a 15-month construction period. The sustaining capital estimate is based on required capital expenditure during operations for tailings storage, limited site development work, and mining infrastructure installations as defined by the mine plan. The closure and reclamation estimate is based on a preliminary estimation of a closure plan commencing in year 6 and continuing to year 11.

1.12.2 Operating Cost Estimate

The operating cost estimate (OPEX) for the Red Mountain Project is based on a combination of experience, reference projects, first principle calculations, budgetary quotes, and factors as appropriate for an FS.

The total life-of-mine operating costs are summarized in Table 1-8. Table 1-9 outlines the major assumptions used to build up the operating costs.

The operating cost estimate is based on leasing major mining equipment, and a year-round mining and processing operation.

Table 1-8: Operating Cost Estimate

Operating Cost	Unit Cost ($/t processed)	LOM Cost (M$)
Mining(1)	72.30	141.2
Processing	45.96	89.8
Site Services	10.40	20.3
General & Administrative (G&A)	11.36	22.2
Total	**140.02**	**273.5**

Source: JDS (2017)

Note: [1] Mining operating cost is $72.50/t mined during the production period.

Table 1-9: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.061
Average Power Consumption	MW	6.8
Overall Power Consumption (all facilities)	kWh/t processed	162
Diesel Cost (delivered)	$/litre	0.898
LOM Average Manpower (including contractors, excluding corporate)	employees	224

Source: JDS (2017)

1.13 Economic Analysis

1.13.1 Main Assumptions

An economic model was developed to estimate annual cash flows and sensitivities of the Project. All costs and economic results are reported in Canadian dollars (C$ or $) unless stated otherwise.

Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations.

The reader is cautioned that the metal prices and exchange rate used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that

they will be realized if the Project is taken into production. The gold price is based on many complex factors and there is no reliable method of predicting long term gold price.

Other economic factors considered include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario);
- Closure cost of $9.9 M (net of $3.8 M salvage value and $1.2 M in contingency);
- Nominal 2017 dollars;
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Results are presented on 100% ownership and do not include management fees or financing costs; and
- Exclusion of all pre-development and sunk costs (i.e., exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, permitting, financing, etc.).

The results of the economic analysis are shown in Table 1-10, with the key economic metrics highlighted in Table 1-11.

Table 1-10: Summary of Economic Assumptions and Results

Parameter	Unit	Value
Au Price	US$/oz	1,250
Ag Price	US$/oz	17.00
FX Rate	US$:C$	0.76
Mine Life	Years	5.4
Mill Feed	Mt	1.95
Throughput Rate	t/d	1,000
Average Au Head Grade	g/t	7.53
Average Ag Head Grade	g/t	21.86
Au Payable	koz	425
	koz/a	78
Ag Payable	koz	1,173
	koz/a	215
NSR (after Royalties)	M$	683.9
Operating Costs	M$	273.5
	$/t mined	140.02
All In Sustaining Cost (AISC)	US$/payable oz	658
AISC (Net of By-product)	US$/payable oz	611
Pre-production Capital	M$	123.0
Pre-production Contingency	M$	12.7
Total Pre-production Capital	**M$**	**135.7**

Parameter	Unit	Value
Sustaining & Closure Capital (Net of Salvage)	M$	60.4
Sustaining & Closure Contingency	M$	6.4
Total Sustaining & Closure Capital	**M$**	**66.8**
Total Capital Costs Incl. Contingency	**M$**	**202.4**
Working Capital	M$	4.0
Taxes	M$	63.2

Source: JDS (2017)

*All in sustaining costs (AISC) is calculated as (operating costs + refining costs + royalties + sustaining capital + closure capital) / payable gold ounces.

**All in sustaining costs (net of by-product) is calculated as (operating costs + refining costs + royalties + sustaining capital + closure capital – value of payable silver) / payable gold ounces.

The Project is economically viable with an after-tax internal rate of return (IRR) of 32% and a net present value using a 5% discount rate ($NPV_{5\%}$) of $104 M using the metal prices and exchange rates outlined in Table 1-10.

Table 1-11: Economic Results

Parameter	Unit	Pre-Tax Results	After-Tax Results
$NPV_{0\%}$	M$	208	145
$NPV_{5\%}$	M$	155	104
IRR	%	40	32
Payback Period	Production years	1.7	1.9

Source: JDS (2017)

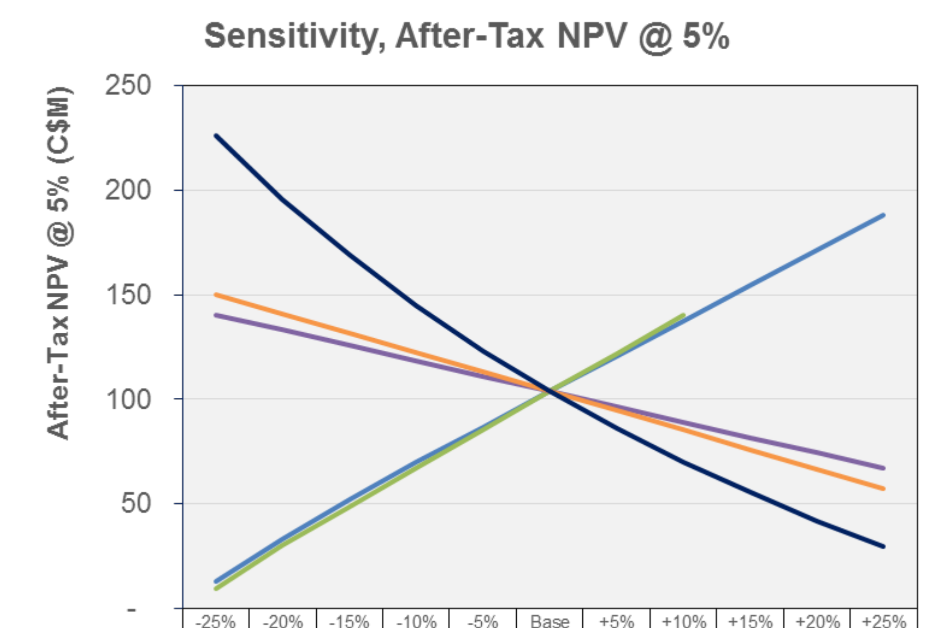
1.13.2 Sensitivities

Sensitivity analyses were performed using metal prices, exchange rate (FX), recovery, operating cost estimate (OPEX), and CAPEX as variables. The value of each variable was in changed in +/- 5% increments independently of each other while all other variables remained constant. As with most metal mining projects, the Project is most sensitive to recovery, metal prices (gold and silver), and FX. The FX sensitivity evaluation includes revenues only.

The Project is slightly more sensitive to CAPEX than OPEX.

The results of the sensitivity analyses are shown in Figure 1-4.

Figure 1-4: After-Tax NPV$_{5\%}$ Sensitivity Graph



	-25%	-20%	-15%	-10%	-5%	Base	+5%	+10%	+15%	+20%	+25%
Metal Price	13	33	52	70	87	104	121	138	155	171	188
Recovery	10	30	49	67	85	104	122	140			
OPEX	140	133	126	118	111	104	96	89	82	74	67
CAPEX	150	141	132	122	113	104	95	85	76	67	57
F/X	226	196	169	145	123	104	86	70	55	42	29

Source: JDS (2017)

1.14 Project Development

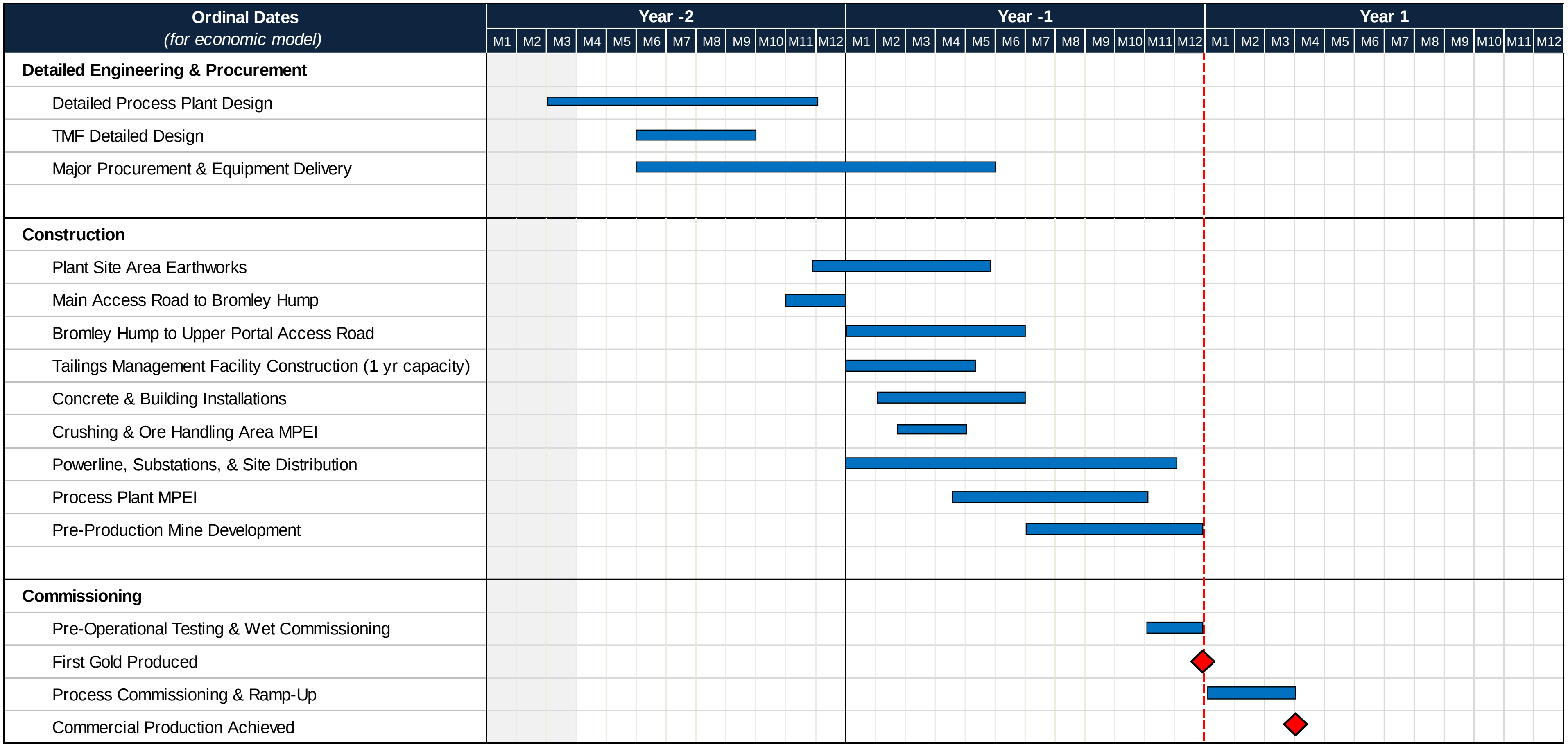
The overall construction and commissioning period for the Project is estimated to be approximately 15 months from the start of the site access road construction to first gold pour and will be preceded by nine months of engineering and procurement. Engineering, preliminary procurement, preliminary road work and some site prep will take place in year -2 of the Project. The remainder of the facilities will be built during year -1.

During year -1, underground mine development will commence approximately six months prior to the final commissioning of the process plant, once the haul road is completed to the upper portal. Initially, mining will commence from the upper portal to develop access to the year 1 production stopes in the Marc zone and will utilize as much of the existing underground development as possible.

Permanent mine surface infrastructure will be installed during years -1 and 1, while underground development is ongoing. This includes surface buildings, primary ventilation, water management and a cemented rock fill batch plant.

Process plant and TMF construction will begin early in year -1, as soon as the 15 km road from the plant site to Highway 37A is re-established to allow collection of runoff for reclaim to the plant site at mill startup. The preliminary development schedule is shown in Figure 1-5.

Figure 1-5: Project Development Schedule (Post-Permitting & Financing)



Source: JDS (2017)

1.15 Conclusions

Based on the assumptions used in the FS, the Red Mountain Gold Project shows 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 Project from undergoing further economic studies, permitting, financing and ultimately development.

JDS recommends that the Project be advanced to the permitting and detailed engineering stage.

1.15.1 Risks

It is the conclusion of the QPs that the FS summarized in this technical report contains adequate detail and information to support the positive economic result. To date, the QPs are not aware of any fatal flaws for the Project.

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 pro-active management.

1.15.2 Opportunities

The main opportunities associated with the Project are currently:

- 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 sulphidation system;
- The sediment-porphyry contact that controls mineralization in the Marc/AV/JW Zones can be traced in the SF Zone for a further 800 metres along strike to the north through sparse drilling with isolated gold intercepts. Further drilling could potentially expand resources both up and down-dip from the AV and JW Zones, and along strike from the 141 Zone and Marc Zone (to the south);
- Optimization of mine plans and production schedules is another opportunity that might add more value to the Project;
- The increased use of used processing equipment, which is presently available from several sources, would reduce the Project's equipment cost and overall Project CAPEX and potentially reduce the engineering, procurement and construction schedules;
- In the spring of 2016, IDM signed a memorandum of understanding (MOU) with Bridge Power Corp., an Independent Power Producer with run-of-river hydroelectric generation rights to Bitter Creek. The companies are committed to sharing environmental baseline data, and potentially sharing capital costs for construction of the access road and power line. This

would potentially result in substantial cost reductions to the capital and operating costs at Red Mountain;

- With road access in mineral-rich northwestern British Columbia, toll-treatment of nearby deposits could add value to the Project; and
- The recent drilling program has discovered additional mineralization that is not part of the current mineral resource. If this mineralization can be upgraded to an Indicated Mineral Resource and incorporated into the mine plan then the mine life could be extended.

1.15.3 Recommendations

Further work is recommended to gather the necessary technical information and complete the requirements for the BC Mines Act Permit (MAPA) and Environmental Management Act (EMA) application. The field investigations and engineering work to complete the MAPA and EMA is estimated to be approximately $6.6 M.

2 Introduction

2.1 Basis of Technical Report

IDM Mining commissioned JDS to complete an FS for the Red Mountain Gold Project located in northwestern British Columbia near the town of Stewart. The purpose of this study is to update and define the Project to an FS level and to declare a mineral reserve estimate. JDS conducted a preliminary economic assessment on the Project in 2016 (JDS 2016). The main changes from the JDS 2016 study are:

- Updated mineral resource estimate – conducted by ACS and based on additional drilling and reinterpretation which changed the overall mineral resources estimates as per Table 2-1.

Table 2-1: 2017 Mineral Resource Estimate with Comparison to the 2016 Estimate

Resource by Classification	Tonnes	In-Situ Gold Grade (g/t)	In-Situ Silver Grade (g/t)	Contained Gold (oz)	Contained Silver (oz)
2016 Historical Mineral Resource Estimate					
Measured and Indicated	1,641,600	8.36	26	441,500	1,379,800
Inferred	548,100	6.1	9	107,500	153,700
2017 Mineral Resource Estimate					
Measured and Indicated	2,074,700	8.75	24.8	583,700	1,655,700
Inferred	324,700	6.21	10.1	64,800	105,500
2017 to 2016 Mineral Resource Comparison					
Measured and Indicated	433,100	0.39	-1.2	142,200	275,900
Inferred	-223,400	0.11	1.1	-42,700	-48,200

Source: ACS (2016 & 2017)

- Revised process flow sheet – change from flotation and concentrate leach to whole ore leaching to improve gold recoveries across mineralization types;
- Addition of a water treatment plant;
- Addition of a construction camp in Stewart, BC;
- Revised mine production schedule – increased LOM tonnes, mine life, and recovered metal based on the improved 2017 mineral resource estimate;
- Revised mine operating schedule – change from a seasonal (8-month) mining operation to continuous year-round mining;
- Updated capital and operating costs based on improved detail, information, designs, and quotes;
- Updated metal prices and exchange rate to:
    - US$1,250/oz Au;
    - US$17.00/oz Ag; and
    - 0.76 US$:C$ exchange rate.

- Design and engineering considerations to minimize the environmental footprint and improve operational, sustainability of the operation and mine closure.

2.2 Scope of Work

The following companies contributed to this technical report and provided QP sign-off for their respective sections:

JDS:

- Overall Feasibility Study Lead;
- Introduction;
- Metallurgical Test Work Analyses;
- Mine Engineering;
- Processing Methods;
- Infrastructure;
- Cost Estimation;
- Project Execution Plan;
- Economic Analysis; and
- Conclusions, Risks, and Opportunities.

ACS:

- Mineral Resource Estimate.

Andrew Hamilton, P. Geo, Independent Consultant:

- Project Description;
- History;
- Deposit Type;
- Geology;
- Drilling;
- Exploration;
- Sample Preparation, Analyses, and Security; and
- Data Verification.

Knight Piésold Ltd. (KP):

- Site Geotechnical;
- Tailings Management;
- Water Management; and

- Closure.

SRK Consulting (Canada) Inc. (SRK):

- Mine Geotechnical;
- Hydrogeology;
- Water Treatment; and
- ARD/ML Considerations.

2.3 Qualified Person Responsibilities & Site Inspections

The Qualified Persons (QPs) preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, 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 IDM. The QPs are not insiders, associates, or affiliates of IDM. 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 IDM and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

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. The QPs are responsible for specific sections as listed in Table 2-2.

Table 2-2: Qualified Person Responsibilities

QP	Company	QP Responsibility/Role	Report Section(s)
Gord Doerksen, P.Eng.	JDS	Overall Responsibility, Costs and Economics	1, 2, 3, 18 (except 18.4, 18.7, 18.8.1, 18.8.4), 19, 20, 21, 22, 23, 24, 25, 26, 27, 28
Michael Makarenko, P.Eng.	JDS	Mining & Mineral Reserves Estimate	15, 16 (except 16.2)
Kelly McLeod, P.Eng.	JDS	Metallurgy & Processing	13, 17
Gilles Arseneau, P.Geo	ACS	Mineral Resource Estimate	14
Andrew Hamilton, P.Geo	Independent	Geology	4, 5, 6, 7, 8, 9, 10, 11, 12
Ken Embree, P.Eng.	KP	Site Geotech, Water and Tailings Management	18.4, 18.7, 18.8.4
Kelly Sexsmith, P.Geo.	SRK	ARD/ML Considerations	18.8.1
Bruce Murphy, P.Eng	SRK	Mine Geotechnical	16.2

Source: JDS (2017)

2.4 Site Visits and Inspections

QP site visits were conducted as per Table 2-3.

Table 2-3: QP Site Visits

Qualified Person	Date	Accompanied by	Description of Inspection
Gord Doerksen, P.Eng.	May 28, 2014 Aug 27, 2016	Rob McLeod	Complete underground and surface inspection of project and access road fly-over with helicopter.
Michael Makarenko, P.Eng.	March 25, 2016	NA	Inspection of existing underground workings and core storage in Stewart.
Kelly McLeod, P.Eng.	NA	NA	
Gilles Arseneau, P.Geo.	March 25, 2016	NA	Verified property access, logistics and surface geology. Examined underground workings and collected seven check samples 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.
Andrew Hamilton, P.Geo.	July 20-24, 2016	NA	Review of core handling, sampling, shipping of samples. Ensure QA/QC protocols are in place and being followed.
Ken Embree, P.Eng.	July 12-13, 2016	Rob McLeod	Inspection of location and surface conditions, geotechnical conditions of proposed tailings facility, and process plant areas.
Kelly Sexsmith, P.Geo.	Aug, 2000 Aug, 2003	NA	Underground decline, portal area cribs and waste rock piles, former camp site, historical water quality monitoring stations
Bruce Murphy, P.Eng.	September 27, 2016	NA	Review of rock mass conditions within the existing underground excavations and the review of selected drill core of the various mineralized zones.

Source: JDS (2017)

2.5 Units, Currency & Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Troy ounces (oz) are used throughout the report as they are an industry-standard unit. The currency used in this report is Canadian dollars (CAD, C$ or $) unless otherwise noted.

This 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 QPs do not consider them to be material.

2.6 Terms of Reference

The function of this report is to update and define the Project to an FS level and to declare a mineral reserve estimate for the Red Mountain Project. It is a compilation derived from the historical work performed by previous operators from 1986 to present, and first principles design and estimate work by JDS and other consultants listed in Table 2-2.

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 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 and Banks engaged in engineering studies as well as PEA in 2013 and conducting further TMF studies. JDS completed Preliminary Economic Assessment (PEA) studies for IDM in 2014 and 2016.

Engineering and geological information from historical documents was used in this report after determination by JDS that the work was performed by competent persons or engineering firms. Data derived from engineering companies, consultants, and authors are listed in the reference section of this report.

3 Reliance on Other Experts

The QP's opinions contained herein are based on information provided by IDM and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice includes:

- Allnorth Consultants Limited – Process plant and infrastructure design;
- Onsite Engineering Ltd. – Road design and cost estimation;
- Clean Energy Consulting Inc.– Transmission line design and cost estimation;
- Integrated Sustainability Consultants Ltd. – Water treatment plant cost estimation;
- Base Metallurgical Laboratories Ltd. – Metallurgical testing;
- MineFill Services Inc. – Cemented rock backfill testing and backfill plant design.
- Soucie Construction Ltd., Avalanche Risk Management and Mountain Safety Division – Avalanche control program recommendations and operating costs; and
- Wentworth Taylor, CPA - Taxation guidance.

4 Property Description & Location

4.1 Property Description & Location

Red Mountain is situated in northwestern British Columbia, approximately 18 km east-northeast of Stewart (Figure 1-1). The Project is located at 55° 57’ N latitude and 129° 42’ W longitude between the Cambria Ice Field and the Bromley Glacier at elevations ranging between 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. A road has been pioneered from Highway 37A up the Bitter Creek valley to the base of Red Mountain. A plan was developed by NAMC to extend this road to the Red Mountain portal site.

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 the Rio Blanco Creek. Both of these 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 border.

Stewart is situated at the head of the Portland Canal, a 120-km long fjord. Stewart is commonly referred to as Canada’s most northerly ice-free port. It is 880 km northwest 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 District of Stewart borders on the State of Alaska and extends some services to the community of Hyder, Alaska.

4.2 Mineral Title

The 47 contiguous claims that comprise an area of 17,125.2 ha (Table 4-1 and Figure 4-1) are wholly owned 100% by IDM, subject to underlying agreements and royalties.

All claims are in good standing until May 9, 2023 according to documents provided by IDM and information from the British Columbia Mineral Title Online web site:

https://www.mtonline.gov.bc.ca/mtov/home.do

Table 4-1: Red Mountain Claim Map

Tenure Number	Tenure Type		Hectares (ha)	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

Table 4-1: Red Mountain Claim Map (continued)

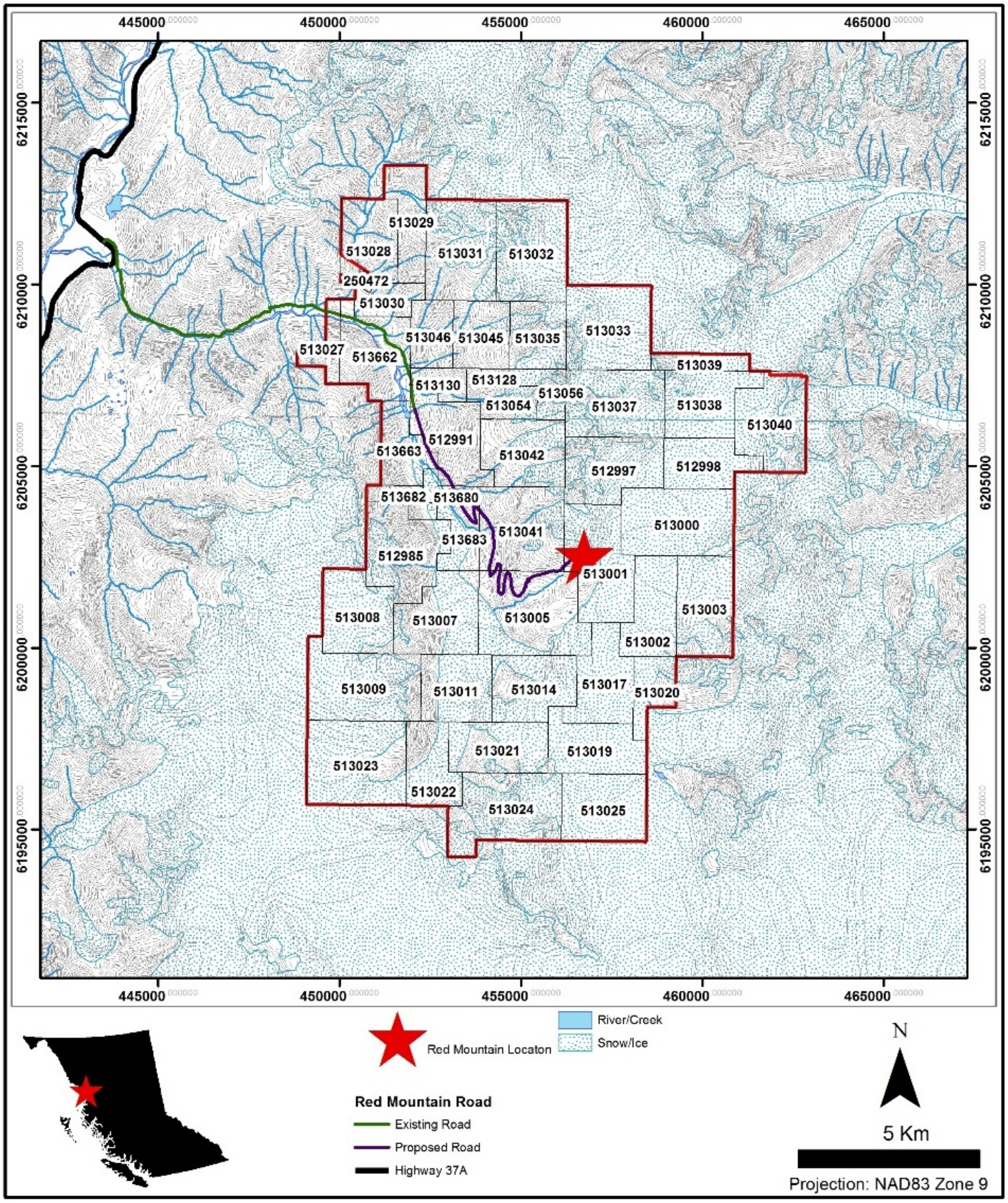
Tenure Number	Tenure Type		Hectares (ha)	Ownership (%)
513002	Mineral	CLAIM	362.3	100
513024	Mineral	CLAIM	580.5	100
513045	Mineral	CLAIM	289.3	100
513130	Mineral	CLAIM	108.5	100
513007	Mineral	CLAIM	452.8	100
513017	Mineral	CLAIM	380.5	100
512985	Mineral	CLAIM	488.8	100
513005	Mineral	CLAIM	670.2	100
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
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

Table 4-1: Red Mountain Claim Map (continued)

Tenure Number	Tenure Type		Hectares (ha)	Ownership (%)
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 (2017)

Figure 4-1: Red Mountain Claim Map



Source: IDM (2017)

The Red Mountain Gold Project is wholly owned by IDM, subject to certain underlying royalties and gold streaming arrangements described in Section 4.3.

4.3 Royalties, Agreements & Encumbrances

4.3.1 Royalties

The Red Mountain Gold Project is 100% owned by IDM and is subject to the payment of production royalties and, on the key Wotan Resources Corp. (Wotan) claim group, the payment of an annual minimum royalty of $50,000.

Production from the Wotan claims, which contain the Red Mountain gold deposit, is subject to two separate royalties aggregating 3.5% of NSR, comprising a 1.0% NSR payable to Franco Nevada (Franco) and a 2.5% NSR payable to Wotan.

Initially Barrick Gold Corporation (Barrick) was granted its 1.0% NSR royalty in 1995 on all of the then existing claims when it sold the property to Royal Oak. On November 1, 2013, Barrick transferred all of their right, title and interest in the 1.0% NSR to Franco. 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's successor, Lac). The agreements each provide for NSR royalties and one of them, the Wotan agreement, has an area of influence. 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 report are subject to two royalties: 1.0% NSR payable to Franco and a 2.5% NSR payable to Wotan.

4.3.2 Underlying Agreements

On April 15, 2014, IDM entered into an Option Agreement (Option Agreement) for the Red Mountain Gold Project with Seabridge. IDM has satisfied the option terms and on May 25, 2017 completed the acquisition of the mineral claims and certain other related assets comprising the Red Mountain Gold Project. IDM owns 100% of the property claims subject to two royalties. Franco holds a 1% NSR and a 2.5% royalty is payable to Wotan. A $50,000 annual advance royalty is payable to Wotan annually.

In accordance with the Option Agreement, IDM is obligated upon the commencement of commercial production, to make an additional one-time $1.5 million cash payment to Seabridge and Seabridge retains the right to acquire 10% of the annual gold production from the Property at a cost of US$1,000 per ounce up to a maximum of 500,000 ounces produced (50,000 ounces 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 principal agreements governing the Red Mountain Gold Project are listed below, along with a summary of the more salient provisions and identified by claim number in Table 4-2

The mineral resource defined in his report is subject to the Franco & Wotan Agreements only.

1. Franco Agreement: Separated Royalty Agreement dated May 25, 2017 between Franco and IDM. Pursuant to an Asset Purchase Agreement dated August 17, 1995, Royal Oak purchased its interest in Red Mountain from 1091064 (a wholly owned Barrick subsidiary which was voluntarily dissolved on April 19, 2016 and all of its assets were transferred and

assigned to Barrick) and granted 1091064 an uncapped 1.0% NSR royalty on production. Barrick is entitled to receive an additional $10.00 cash production payment per ounce on all ounces of gold produced from the property in excess of 1,850,000 ounces (Production Payments). On November 1, 2013, 1091064, Barrick and Franco-Nevada entered into a royalty deed and assignment pursuant to which Barrick transferred to Franco their right to the 1.0% NSR royalty and the Production Payments ("Assigned Interests"). On May 25, 2017, Barrick, Franco and IDM entered into a separation, assumption and amending agreement that separated the respective rights, obligations and liabilities of Franco and Barrick in the asset purchase agreement and pursuant to which, inter alia, IDM and Franco agreed to replace the provisions of the asset purchase agreement constituting the Assigned Interests with the Separated Royalty Agreement.

2. Wotan Agreement: 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 claim 513005, (which contain the known Red Mountain gold deposits) and from any other properties within a 2-km area of influence extending from the boundaries of the claim. By October 31st 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.

3. Krohman Sinitsin Agreement: IDM is obligated to pay Darcy Krohman and Greg Sinitsin a 1.0% NSR royalty on production from claims 513128 and 513190. IDM may buy out the royalty at any time for $500,000.

4. 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.

Table 4-2: Underlying Agreements by Claim Number

Claim #	Hectares	Barrick Agreement	Wotan Agreement	Sinitsin Krohman Agreement	Harkley Fegan Scott Agreement
512985	488.797	1			
512991	416.154	1			
512997	452.432	1			
512998	307.647	1			
513000	579.305	1			
513001	525.127	1			
513002	362.257	1			
513003	434.699	1			
513005	670.206	1	2		
513007	452.776	1			
513008	416.515	1			

Table 4-2: Underlying Agreements by Claim Number (continued)

Claim #	Hectares	Barrick Agreement	Wotan Agreement	Sinitsin Krohman Agreement	Harkley Fegan Scott Agreement
513009	597.805	1			
513011	362.383	1			
513014	398.677	1			
513017	380.539	1			
513019	380.734	1			
513020	199.338	1			
513021	380.738	1			
513022	308.159	1			
513023	634.389	1			
513024	580.530	1			
513025	435.383	1			
513027	126.577	1			
513028	361.393	1			
513029	289.073	1			
513030	162.691	1			
513031	542.145	1			
513032	542.161	1			
513033	542.426	1			
513035	289.308				
513037	506.513	1			
513038	397.977	1			
513039	126.596	1			
513040	470.395	1			
513041	543.126	1			
513042	416.200	1			4
513045	289.307				
513046	216.972				
513054	180.890	1			4
513056	144.704				
513128	36.173	1		3	
513130	108.522	1		3	
513662	434.001	1			
513663	253.327	1			
513680	181.046	1			
513682	108.596	1			
513683	90.495	1			
Total Hectares	**17125.20**				

Source: IDM (2017)

4.4 Environmental Liabilities & Permitting

4.4.1 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;
- the closure of the decline portal; and
- 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 from Goldslide Creek and Bitter Creek as part of the baseline program, on a monthly basis. No hydrocarbons have been noted in lab analyses.

4.4.2 Required Permits & Status

Pursuant to section 3(1) of the Reviewable Projects Regulation pursuant to the CEAA (2012), the proposed production capacity for the Project exceeds the criteria of 75,000 t/a of mineral ore for a new mineral mine and will require review pursuant to the BCEAA and the issuance of an Environmental Assessment Certificate (EAC). The BC Environmental Assessment Office (EAO) issued a Section 10 Order to IDM on November 2, 2015 confirming the Project will require an EAC. The EAO further issued a Section 11 Order on February 10, 2016 which outlined the requirements for the environmental assessment of the Project under the BC EAA. The submission date of the Application for the EAC was July, 2017. The Project will also require a review and decision pursuant to the CEAA 2012. The BCEAA and CEAA processes are coordinated and only one Application is required addressing both the CEAA Guidelines and the EAO Application Information Requirements.

IDM is pursuing synchronous permitting for provincial permits relating to the development and operation of the Project. Using this approach, provincial permit application review timelines will be coordinated and agreed upon through the Major Mine Permitting Office (MMPO) in consult with the issuing agencies. No decisions on commercial production related to provincial permits is possible until completion of the decision pursuant to the BCEAA.

It is anticipated that the Project will require approvals under the *Mines Act* (1996b), *Environmental Management Act* (2003), and *Land Act* (1996a). Additional details regarding provincial and federal permit requirements are discussed in Section 20.

IDM will continue to engage the appropriate provincial agencies and their representatives in the mine review committee to confirm permitting requirements related to the Project.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility & Transportation to the Property

Access to the property is currently by helicopter. Road access up the Bitter Creek valley from Highway 37A was partially developed for 13 km by Lac Minerals Ltd. 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 Climate

Climatic conditions at Red Mountain are dictated primarily by its altitude (1,742 masl 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 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.

5.3 Topography, Elevation & Vegetation

A view showing the topography of the Red Mountain area is provided in Figure 5-1.

Figure 5-1: View of Red Mountain & Camp Looking South (1,400 to 2,000 masl)

Source: IDM (2016)

From June 1993 to June 1994 and from 2014 to 2017, weather data were collected for the site. Several stations were monitored but the station most relevant to this study is the one at approximately 1,800 m on the Southeast side of Red Mountain (Table 5-1). For those periods, based on conditions in Stewart, it was noted that December and January were warmer than usual while February was colder than usual.

Table 5-1: Temperature Data on Red Mountain at Approximately 1,800 m Elevation

Month	Average (°C)	Max (°C)	Min (°C)
Jan	-3.3	8.1	-13.1
Feb	-9.8	7.3	-24.7
Mar	-3.4	6.8	-12.9
Apr	-0.7	5.7	-8.1
May	1.5	13.0	-4.8
Jun	3.1	7.0	0.0
Jul	5.9	20.5	-4.3
Aug	9.6	20.5	1.1
Sep	3.9	14.4	-3.1
Oct	3.2	13.7	-4.3
Nov	-4.2	2.1	-17.1
Dec	-4.1	1.6	-9.6
Average	**0.1**		

Source: LAC (1994)

5.3.1 Relative Humidity

The relative humidity is generally high year-round due to the proximity to the Pacific Coast. The relative humidity through 1993 and 1994 ranged from 67.5% to 89.4% with an average of 78.4% based upon the one-hour average relative humidity values.

5.3.2 Wind

Winds on Red Mountain are channeled by topography and windy conditions are frequent. Hourly average wind speeds regularly exceed 10 m/s and instantaneous wind speeds in excess of 30 m/s have been observed.

5.3.3 Precipitation

Precipitation data were collected for part of 1994 (April to August) at the west side of Red Mountain about 1600 masl; this data along with data collected at the base of Red Mountain in the Bitter Creek Valley were compared to the 1974 to 1992 Stewart Airport records. While there were insufficient data from the 1600 masl station for an accurate correlation with the Stewart Airport data, precipitation at the Stewart Airport was considered by LAC's consultants, to be representative of precipitation at the Red Mountain site.

The hypothesis that the precipitation at the Project site on Red Mountain (+1,600 masl) is equivalent to that of Stewart Airport (7 masl) may seem surprising given the large increase in precipitation generally associated with increasing elevation in the Coast Mountains. The similarity is explained by the fact that the Red Mountain site is separated from the Portland Canal by a topographic divide with elevations exceeding 2,000 m. Therefore, air masses reaching Red Mountain from the ocean have already lost moisture due to orthographic lifting from sea level.

The Stewart Airport precipitation data for the period 1974 to 1992 is shown in Table 5-2. As described above, the precipitation at the Red Mountain site is assumed to be the same as that at the Stewart Airport.

Table 5-2: Stewart Airport Precipitation

Month	Stewart Airport Precipitation (mm)
January	229.7
February	151.9
March	109.6
April	84.4
May	76.0
June	66.0
July	66.3
August	97.4
September	201.3
October	301.9
November	242.2
December	250.7
Annual Total	**1,877.4**

Source: LAC (1994)

At the Stewart Airport, an average of 35% of the precipitation falls as snow.

LAC operated two snow survey stations in the Project area during the winter of 1993-94 each comprising 10 sampling points. A sampling tube was used to collect a snow core sample at each sampling point on a monthly basis. Snow pack density and water equivalent were calculated on the basis of snow depth and core weight, as an average from the ten sampling points. One of the snow survey stations was located across Goldslide Creek from the exploration camp. This station is most relevant to the Project as currently planned.

Snow survey data were compared to the data collected by BC Ministry of Environment, Lands and Parks (MELP) from other stations in the area. Snow pack development at this site was very similar to snow pack development at the Bear Pass site until April when water equivalent peaked at Bear Pass. At Red Mountain, the peak was reached in early May. Snow densities are generally high in coastal British Columbia, reaching 50% by late winter.

Comparing snow pack data for the area, it appears that the Red Mountain site receives considerably less precipitation than other nearby sites. This corroborates the observation that the Cirque receives considerably less precipitation than suggested by its altitude due to its relatively sheltered location. This underscores the importance of aspect and direct exposure to the Portland Canal in determining local precipitation levels in the Project area.

The 1994 snow course data for the Red Mountain camp is shown in Table 5-3.

Table 5-3: 1994 Red Mountain Snow Course Data

Date (1994)	Snow Depth (cm)	Water Equiv. (mm)	Density (%)
Jan 1	-	-	-
Feb 1	167.7	584	35
Mar 1	158.7	653	41
Apr 1	187.9	840	44
May 1	201.7	975	49
Jun 1	142.7	740	52

Source: LAC (1994)

5.3.4 Seismic Activity

The National Building Code of Canada seismic source model (Horner 1994) places Stewart in Zone 2 for peak ground acceleration and Zone 4 for peak ground velocity, on a Risk Zone scale of 1 (low risk) to 6 (high risk). A site-specific seismic hazard assessment was carried out using the Cornell method incorporated in the McGuire program "RISKLL," and ground motion attenuation relationships. Annual probabilities of exceeding a range of return periods are shown in Table 5-4 with the corresponding peak ground accelerations and velocities. This analysis indicates that the Red Mountain Gold Project area is in a region of moderate seismic risk. Seismic events occurring in the earthquake prone zone, which runs along the length of the Coast Mountains (Horner 1994), may cause ground motion at the Red Mountain Gold Project area.

Table 5-4: Probabilistic Seismic Ground Motion Analysis

Annual Probability of Exceeding	Return Period (years)	Peak Ground Acceleration (g)	Peak Ground Velocity (cm/sec)
0.05	20	0.021	4.0
0.01	100	0.046	10.0
0.005	200	0.061	13.2
0.0021	476	0.083	18.2
0.001	1,000	0.104	23.0
0.0005	2,000	0.126	28.0
0.0001	10,000	0.188	41.9

Source: (LAC 2014).

5.3.5 Local Resources

Stewart provides a number of community services including air services, road transportation to the interior of BC, marine transport via the Portland Canal, water supply, sewage and waste management facilities, health services, and policing and emergency services. There is also a range of business services, parks and recreation services, and services and facilities for visiting tourists.

5.3.6 Operating Conditions

Road access in the higher elevation areas can be hampered during the late winter and spring by heavy snowfall and avalanche conditions. Current planning envisions a year-round mining and milling operation.

5.3.7 Surface Rights

The Project currently resides on Crown land and no private property is within the operating plan area.

5.4 Infrastructure

The Project is located approximately 13 km from the BC Hydro transmission line that runs adjacent to Highway 37A.

At the Project site, 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. 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.

5.5 Demographics

5.5.1 Population

Prior to 1914, the population of Stewart was in the order of 10,000 people. By 2001, the population declined to approximately 660 people, and then to 496 in 2006 (Government of Canada, 2006). The population of the District of Stewart was 494 in 2011 (Government of Canada, 2011).

At the time of the 2006 census by the Government of Canada, 32.4% of the population held a high school certificate or equivalent and the majority of employment was in the trades and transportation sectors. The unemployment rate was 8.2%.

According to the District of Stewart's Investment-Ready Community Profile, the largest employers in Stewart are in the mining, petroleum resources, highway maintenance, accommodation, education, and health care industries.

Nisga'a Nation has a population of approximately 5,581 citizens (Aboriginal Affairs and Northern Development Canada, 2014). The majority (67%) live off the reserve. The on-reserve population predominantly live in four Nisga'a Villages: Gitlaxt'aamiks (formerly New Aiyansh), Gitwinksihlkw, Laxgalts'ap, and Gingolx.

5.5.2 Economic Activity

Major industries operating around the District of Stewart include tourism, mining exploration, mining operations, and logging. The Stewart World Port and Stewart Bulk Terminals operate out of the Port of Stewart, which is North America's most northern ice-free port and a hub for shipping to Alaskan and Asian markets. Roadways and railways connect Stewart to other transportation hubs in British Columbia and North America.

Businesses in Stewart generally rely on resource industry companies and tourism opportunities related to the many hiking trails and outdoor recreation activities in and around Stewart.

6 History

6.1 Prior Ownership, Ownership Changes & Exploration Results

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. staked claims in 1988 and optioned the property to Bond Gold Canada Inc. (Bond) in 1989. Pre-1988 exploration history is outlined below:

- **1899/1902:** Discovery and small-scale mining of placer gold in Bitter Creek.
- **1912-1919 & 1940:** Hartley Gulch Area, three adits developed, grades to 0.79 oz/t Au found.
- **1915:** Shipment to Trail of 15 tonnes 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 tonnes 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.
- **1977-78:** Zenore Resources Ltd.: Logging and re-sampling of the 1967 drill core (samples assayed for molybdenum only); geological mapping, petrographic studies, rock geochemistry (assayed for copper, molybdenum, and gold).
- **1978-80:** Falconbridge Nickel Mines Ltd: Reconnaissance program for porphyry copper-molybdenum targets in the Stewart area.
- **1987-88:** 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.
- **1988-89** Staking of Red Mountain by Wotan Resources Inc. and optioned to Bond Gold Canada Inc.

In 1989, gold mineralization in the Marc and Brad zones was discovered by drilling. LAC Minerals Ltd. Acquired Bond in 1991. Surface drilling on the Marc, AV, and JW zones continued in 1991, 1992, 1993, and 1994. Underground exploration of the Marc zone was conducted in 1993 and 1994. In 1995, LAC was acquired by Barrick, who subsequently optioned the property to Royal Oak in 1996. NAMC purchased the property from the receivership sale of Royal Oak in 2000. NAMC subsequently sold the property in 2002 to Seabridge, who optioned the property to Banks. 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 carried out by IDM are given in Section 9.7.

Table 6-1 is a recent chronological summary of exploration efforts on Red Mountain from 1988 to 2017:

Table 6-1: Red Mountain 1988-2017 Exploration Summary

Year	Summary
1988-89	Staking of Red Mountain by Wotan Resources Inc.
1989	Red Mountain and Wotan properties optioned to Bond. Discovery of gold-silver mineralization by drilling in the Marc zone (3,623 m); airborne EM and magnetic survey.
1990	Exploration of Marc zone and adjacent area (11,615 m of drilling) by Bond.
1991	LAC acquired 100% of Bond. 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 and 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 metres of surface drilling.
1995	Red Mountain Gold Project acquired by Barrick following Barrick's take-over of LAC. No exploration work completed by 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	NAMC purchased the property and project assets from Price Waterhouse Coopers, conducted 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 PEAs.
2012-2013	Banks options property, two surface drill holes completed, completes PEA study.
2014	IDM optioned property, drilled 12 core holes, completed soil, rock and channel sampling and prepared a PEA.
2016	IDM drilled 11 surface holes and 51 underground holes totalling 8,123.44 metres, and completed surface rock and channel sampling. IDM also updated the 2014 PEA and carried out extensive environmental baseline and assessment studies
2017	Continued metallurgical test work was conducted on the 2016 drilling samples and a new UG definition and expansion drilling campaign was started however, no results from the 2017 resource drilling were used in this report.

Source: ACS (2017)

6.2 Stewart Area History

Stewart's history has been largely dictated by the fortunes of the mining industry. The first prospecting in the area, for gold, took place in the late 1890's and the town site was named in 1905. In the early 1900s, an estimated 10,000 people lived in the area attracted by the prospects of gold. Significant mines such as Premier Gold, Big Missouri, and Granduc Copper were later established in the area.

In 1992, the Premier mine suspended operations, thus starting the most recent hiatus in mineral production in the Stewart district.

6.3 Historic Mineral Resource Estimates

Several resource estimates for the Red Mountain deposit were completed in the past 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 6-2 are only stated for historical completeness and should not be relied upon as they are superseded by the Mineral Resources presented in Section 14 of this report.

Table 6-2: Historical Resource Estimates

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
2008	Seabridge[2]	Inferred	191,020	10.25	15.22	62,900	93,500
2013	Banks[3]	M&I	1,612,000	8.4	28.3	432,000	1,440,000
2013	Banks[3]	Inferred	807,000	5.4	10.2	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

Source: JDS (2017) with modifications.

Notes: (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 PEA.

6.4 Historic Production

No historical production has taken place on the property.

7 Geological Setting and Mineralization

7.1 Introduction

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 Regional Geology

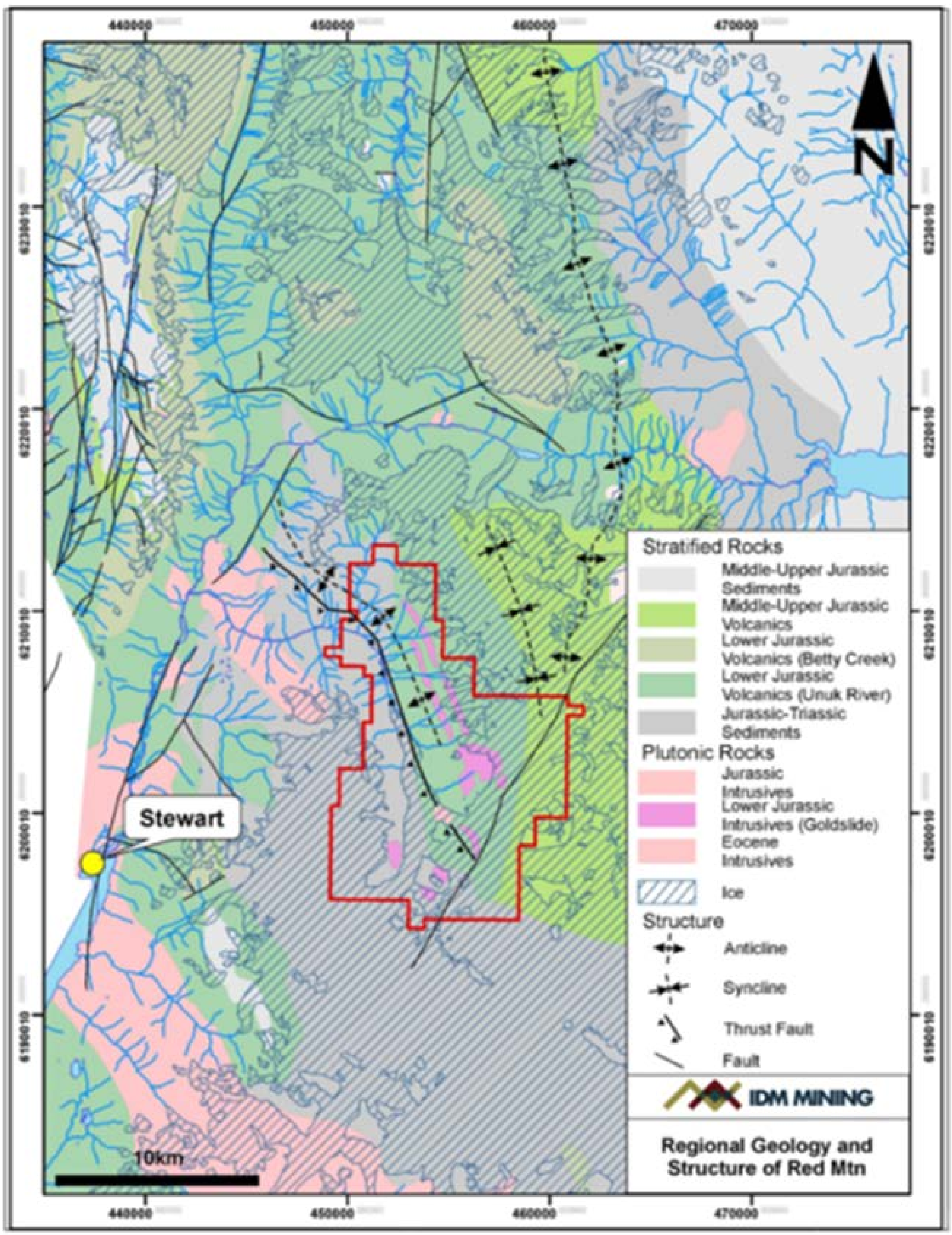
The regional geology of the Red Mountain area has been described by Greig et al., (1994), Alldrick (1993) and Rhys et al. (1995). The following description is drawn from these sources.

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt (Figure 7-1). There are three primary stratigraphic elements in Stikinia and all are present in the Stewart 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. Many primary textures are preserved in rocks from all of these groups, and mineralogy suggests that that the regional metamorphic grade is probably lower greenschist facies.

Intrusive rocks in the Red Mountain region range in age from Late Triassic to Eocene and form several suites. The Stikine plutonic suite is comprised of 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 gold deposits. 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.

Structurally, Red Mountain lies along the western edge of a complex, northwest-southeast trending, doubly-plunging structural culmination, which was formed during the Cretaceous. 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 (Greig, personal communication, 2001). 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).

Figure 7-1: Regional Geology



Source: IDM (2016)

7.3 Local Geology

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

200 Million Years (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 is 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 docks and formation of the Skeena fold belt starts. This event folded the rocks of the Stuhini, Hazelton and Bowser Lake groups.

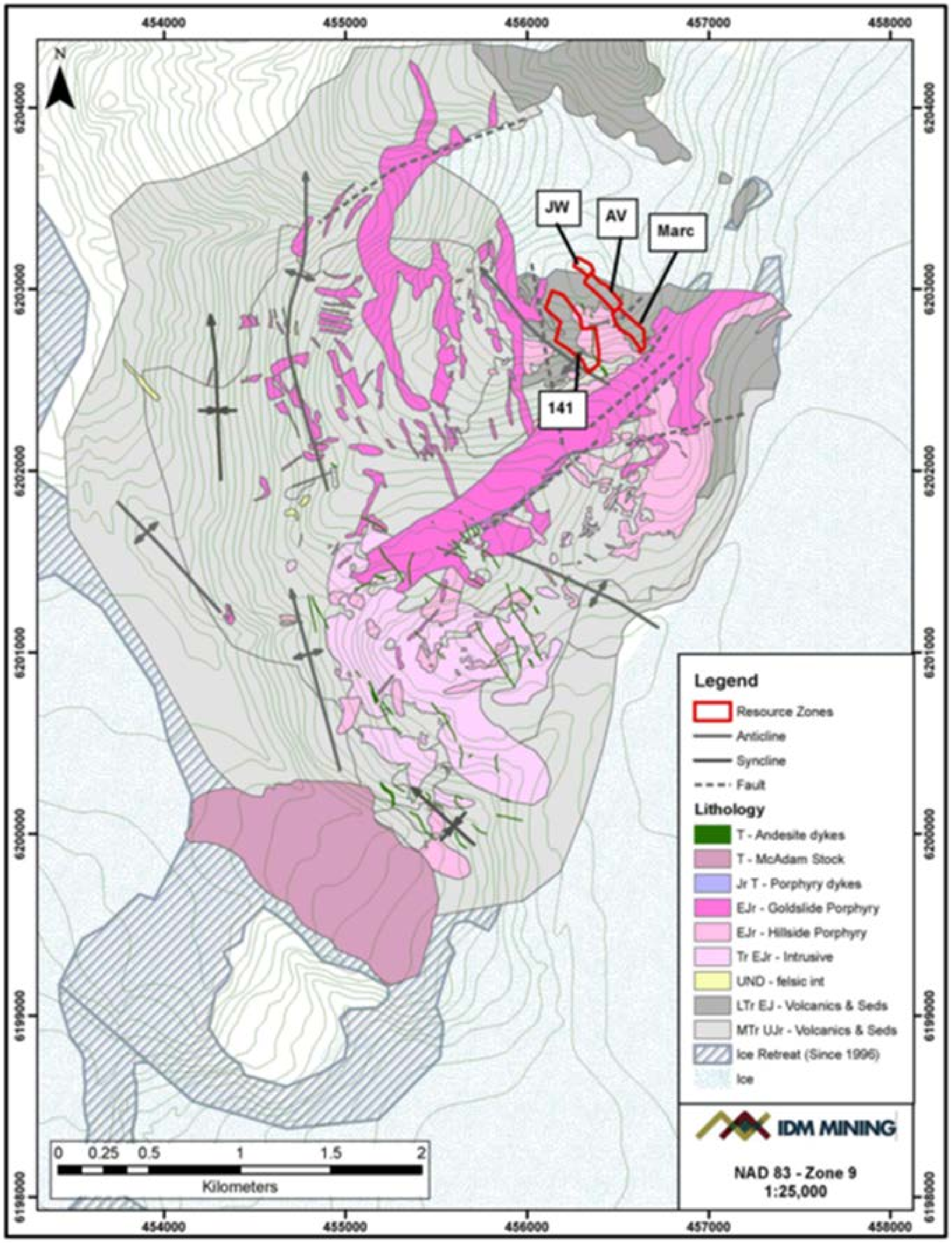
65 Ma (End of Cretaceous) – Deformation of Stikine terrain rocks is complete resulting in folded and doubly plunging structural culminations. The Red Mountain deposits have been rotated from a vertical orientation to a westerly dipping, northerly plunging orientation in the eastern limb of the Bitter Creek antiform. Alexandria 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 can be equated to formation of the Rick Fault and other property scale faults, offsetting the mineralized zones.

7.4 Property Geology

Property geology is shown on Figure 7-2. The oldest rocks, Middle to Upper Triassic mudstone, siltstone and chert of the Stuhini group outcrops 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 map area. Rocks of both groups are folded about axes, which plunge towards 345° and dip steeply to the southwest. An approximate contact between rocks of the two groups also follows this trend and occurs along the projected trace of the Bitter Creek antiform, a structure that 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 have been eroded away.

Figure 7-2: Red Mountain Property Geology



Source: IDM (2016)

Three phases of the Early Jurassic Goldslide intrusions are exposed in the map area. The Hillside porphyry, 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, biotite ± quartz Goldslide porphyry, 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. Finally, sills of the Biotite porphyry intrude Upper Triassic sediments on the west side of Red Mountain. It is distinguished from the Hillside porphyry by the presence of biotite phenocrysts and from the Goldslide porphyry by the small size of hornblende and plagioclase phenocrysts (Rhys et al., 1995). Contact breccias and strongly disrupted bedding are common along the contacts of these intrusions, particularly in association with the Hillside porphyry. In addition, the Hillside porphyry contains rafts of the sedimentary rocks ranging in size from one or two metres to several tens of metres.

Recent work indicates that the three phases of intrusive porphyry have all originated from the same source, and as such represent an evolution in the magma, seen as an enrichment in elements such as sodium and minerals like quartz, which are common markers in the Goldslide phase.

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 dated to 45 Ma (Rhys et al., 1995). Rather than being one large intrusion, the Lost Valley stock appears to be a series of nested structures, with sharp contacts between coarse and fine phases of quartz monzonite observed in several locations. Ductile shear structures do indicate that regular emplacement took place in quick succession and that the entire intrusion cooled as a whole sometime in the Eocene or Oligocene. 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.

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.

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 around 400 m apart. Sympathetic structures, such as riedel shears, normal and reverse faults have been observed propagating from these faults, with some evidence that late stage mineralization (unrelated to Red Mountain zones) tied these structural features.

Over all this brittle 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 have contain more gouge and have broader alteration envelopes than the former.

7.5 Significant Mineralized Zones

7.5.1 Mineralized Zones

The mineralized zones consist of crudely tabular, northwesterly trending and moderately to steeply southwesterly dipping gold bearing iron sulphide stockworks. Pyrite is the predominant sulphide, however locally pyrrhotite is important. The stockworks zones are developed primarily within the Hillside porphyry, and to a lesser extent, in rafts of sedimentary and volcaniclastic rocks. Although locally anomalous gold values are present within the Goldslide porphyry, significant auriferous sulphide stockwork zones have not been located in this rock unit, which generally lies less than 100 m below mineralized zones.

The stockwork zones consist of pyrite microveins, coarse-grained pyrite veins, irregular coarse-grained pyrite masses and breccia matrix pyrite hosted in a pale, strongly sericite altered Hillside porphyry. Vein widths vary from 0.1 cm to approximately 80 cm but widths of 1 to 3 cm are most common. The veins are variably spaced and 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 sulphosalts (Barnett, 1991). These mineral grains, which are typically 0.5 to 15 microns in size, 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 zone of disseminated pyrite and pyrrhotite alteration. Each of these sulphides, 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 it does occur within the pyrite stockwork, but generally only in peripheral areas where bleaching and pyrite vein density is weak.

The stockwork zones are also partially surrounded by a halo of light red coloured sphalerite. It comprises 0.5 to 4.0% of the rock and generally is more abundant in the footwall portions of the zones. The relationship between this sphalerite and the gold bearing pyrite stockworks is unclear. Locally the sphalerite halo contains low-grade gold values (0.5 to 2.0 g/t gold); however, these areas also contain sparse pyrite or pyrrhotite veinlets that could easily explain the gold values. The lack of a consistent relationship between the stockwork zones, gold grades and the distribution of sphalerite suggests that it is not necessarily related to the gold bearing system. A cross cutting relationship between pyrite, pyrrhotite and sphalerite mineralization was not observed during core re-logging in 2000.

8 Deposit Types

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 (2000c) 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. A third scenario has been presented by Barclay (2000) in which fracture formation was due to extension caused by cooling in a high-level intrusion and sulphide-gold deposition was from a locally derived, volatile-rich exsolving fluid. In this case, both mineral deposition and extension were ongoing and evolving.

A synthesis of these models, in particular using elements of the models proposed by Lang and Barclay, appears to best fit with geological and mineralogical observations. A series of schematic diagrams illustrating this is shown in Figure 8-1 and a brief description of each frame is as follows:

1. Intrusion of the Hillside porphyry into Stuhini and Hazelton Group strata. Large rafts of the host rocks are picked up by the intrusion;
2. The Hillside porphyry cools and contracts. The contraction causes the initial formation of a zone of extensional fractures. Pyrite deposited into these fractures starts from volatile fluids that are exsolving from the Hillside porphyry as it cools;
3. Ongoing cooling and extension with fracturing and brecciation of coarse-grained pyrite veins. Additional coarse-grained pyrite is deposited into open space. The gold telluride petzite is deposited as small inclusions in pyrite grains;
4. Intrusion of the Goldslide porphyry. The intrusion drives a pulse of hydrothermal fluids containing native gold, gold tellurides and sulphosalts into fractures in the coarse-grained pyrite veins where they are deposited; and
5. Final infilling of remaining fractures in the coarse-grained pyrite veins with gold minerals, fibrous quartz, calcite, feldspar and sericite.

A series of detailed diagrams illustrating vein formation and gold deposition are shown in Figure 8-2 (after Lang, 2000c).

The model proposes a plausible origin for the structures that host sulphide and gold mineralization, and puts forth a paragenetic sequence for mineral deposition that fits well with macroscopic and petrographic observations. The model also fits well with the random nature of a stockwork system and the variation in gold grades that are encountered over short distances in the diamond drill core.

Figure 8-1: Schematic Diagrams Showing Stages of Formation of Pyrite Veins & Timing of Au

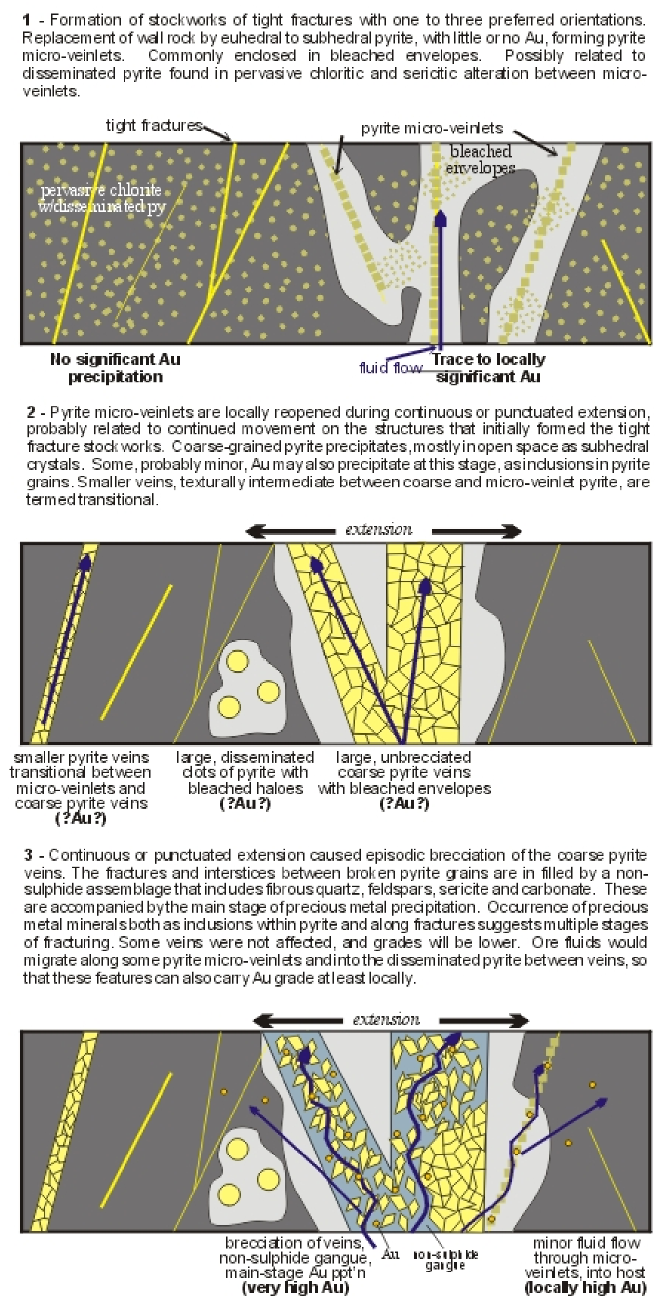
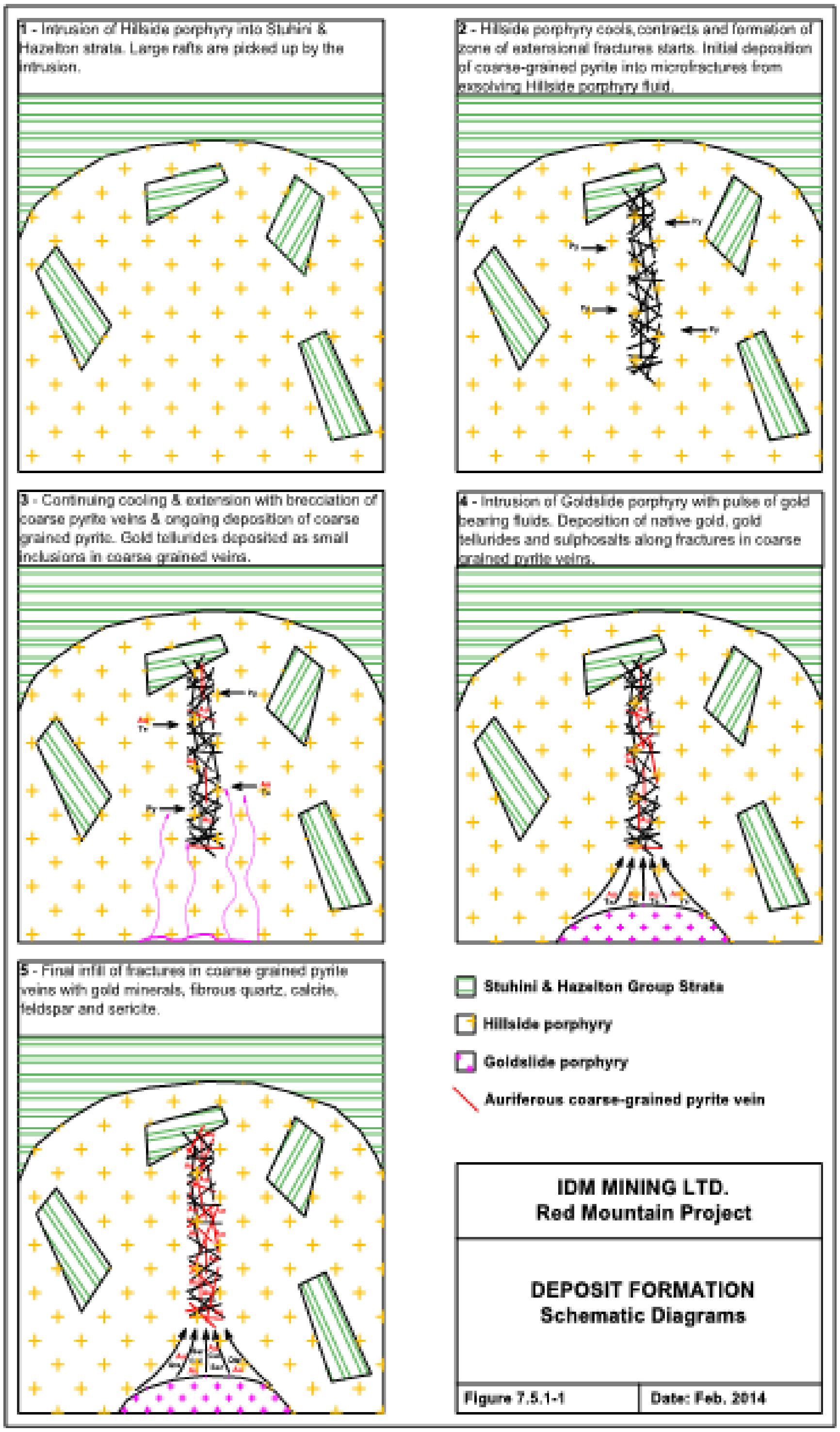


Figure 7.5.1-2: Schematic diagrams showing stages of formation of pyrite veins, and the timing of Au precipitation (from Lang, 2000C).

Source: IDM (2014)

Figure 8-2: Deposit Formation Models



Source: NAMC (2014)

9 Exploration

9.1 Introduction

The Red Mountain property has been explored by IDM, predecessor companies (e.g., LAC), or by contractors (e.g., geophysical surveys).

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; the third hole was abandoned prior to reaching the Mark Zone.

9.2 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 is comprised of 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 Geological Mapping

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

9.4 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, 5,800 rock samples and 890 whole rock samples.

9.5 Geophysics

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; and
- Downhole IP, magnetics, and UTEM.

9.6 Petrology, Mineralogy, & Research Studies

A significant number of research studies have been completed 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; and
- Age dating studies.

9.7 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 2223.0 m (McLeod, 2014). Additionally, historic core was re-logged and 68 infill core 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 on 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. Preliminary drilling in 2014 assessed two of these, the Oxlux and Wyy Lo’oop showings in the Cambria zone.

The 2016 exploration program focused primarily on underground drilling, which consisted of 51 holes, totalling 6385.44 m. The drilling program was designed to upgrade the mineral resource classification and to expand the known resources as well as to collect samples for metallurgical, geotechnical, and hydrological evaluation. Surface rock sampling, consisting of 509 samples, was focused on mineralized exposures in Lost Valley, which were later tested by a preliminary drilling program consisting of five holes. Five additional surface drill holes were also completed to test extensions of the 141 Zone and one hole tested the extension of the Bard Zone. Finally, additional samples collected historic core in the Marc and 141 Zones.

9.8 Exploration Potential

Exploration potential for the property is excellent. 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, particularly up and down-dip from the AV and JW zones, and to the north of the JW zone in a fault-offset extension called the SF zone.

9.9 Comment on Section 9

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 assessment.

10 Drilling

10.1 Introduction

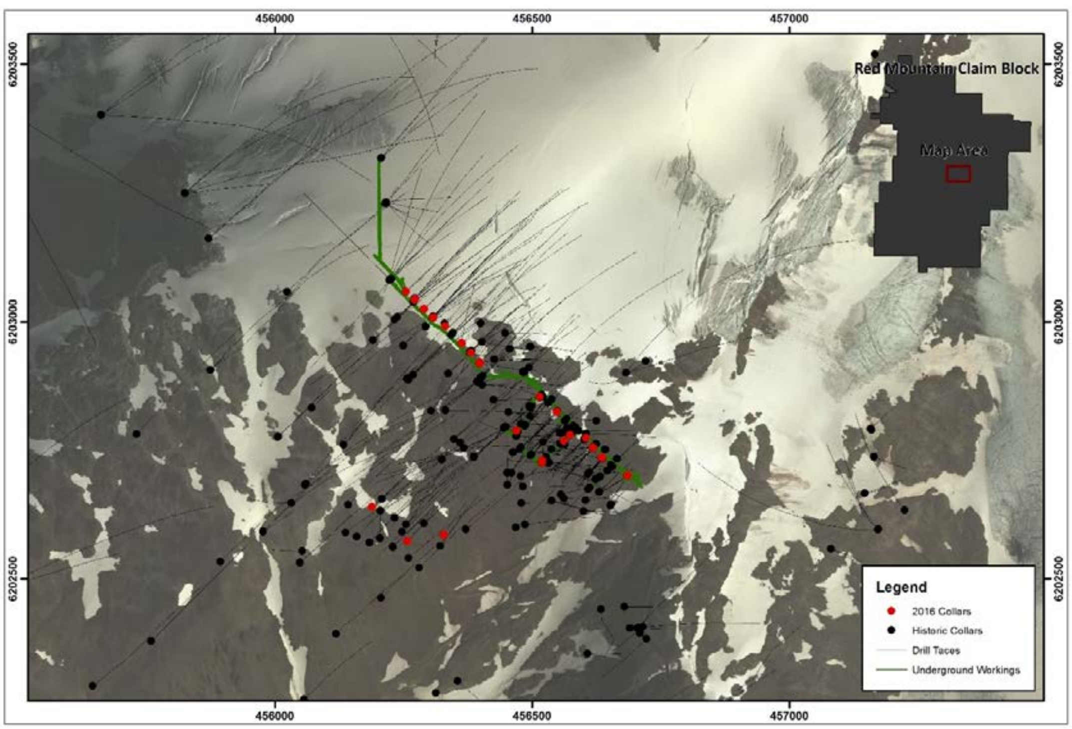
A total of 543 surface and underground diamond drill holes (141,104 m) have tested a variety of targets on the Red Mountain property. Of these, 406 holes totalling 100,298 m were drilled by Bond and LAC between 1989 and 1994, and 60 holes totalling 29,671 m were drilled by Royal Oak in 1996. No drilling was carried out by NAMC. During 2012, Banks completed 3 drill holes for 681 m in the Marc zone.

The majority of the historical drilling has 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 locations of a majority of drill holes on the property are shown on Figure 10-1, which is centred on the resource areas and main prospects.

In 2014 IDM Mining completed 12 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 a further 62 holes totalling 8,123 metres, including 51 underground 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.

Figure 10-1: Red Mountian Drill Plan Resource Areas & Main Prospects



Source: IDM (2016)

10.2 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 by J.T. Thomas Diamond Drilling Ltd. of Smithers, BC, from 1992 to 1994. 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.

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

The 2014 and 2016 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 have tested the Marc, AV, and JW zones. All holes were drilled parallel to the mine grid section lines. 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 remaining of the surface holes were drilled at off section orientations. Inclinations for the holes ranged from -45° to -90°.

10.3 Underground Drilling Contractors

J.T. Thomas Diamond Drilling Ltd. of Smithers, BC, carried out underground drilling programs in 1993 and 1994. As with the surface drilling, they used equipment suitable for producing 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.

MoreCore Drilling of Stewart, BC, using equipment suitable for production of HQ and NQ2 diameter core, carried out the 2016 IDM underground drilling program.

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°.

10.4 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.

10.5 Core Logging

10.5.1 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, rock quality designation (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; and
- 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 the 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.

10.5.2 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; the alteration codes were not used, only written descriptions. There is no written evidence of the sample intervals and sample numbers in their drill hole log files, only computer print outs with intervals, sample numbers, and results. None of the Royal Oak holes are within resource areas.

10.5.3 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 to 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.

10.5.4 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.

10.5.5 IDM Logging

IDM logged and sampled their core at the camp on Goldslide creek. Logging was carried out by directly entering data onto computers using a customized Access drill hole database, which includes all standard tables. Samples were laid out by a geologist, respecting geological boundaries.

10.6 Recovery

All operators have measured core recovery. A selection of 1993 and 1994 surface and underground drill holes with a total of 47,429 intervals of recovery data averaged 93.46%, indicating that the rocks intersected by drilling are generally solid.

10.7 Drill Collar and Downhole Surveys

10.7.1 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 set up 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 resurveyed 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 set up 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 set up orientations.

All three Banks Island drill holes were completed from a single pad. How the pad was located and surveyed is unknown.

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 all but two of the IDM 2016 underground drill holes were surveyed using a total station. For 26 of the holes, 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. A further 18 drill holes had collar orientations from gyro surveys. A few holes with no surveys of either type had ideal collar orientations entered in the database.

10.7.2 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 downhole 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 12 holes. The IDM 2016 drilling program used a combination of a Reflex multi-shot surveys and Reflex Gyro surveys. Details of the downhole surveys and collar surveys for all programs are given in Table 10-1.

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 10-1: 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
1996	Royal Oak	S	Y	Y for ~25% of holes	Sperry	Variable spacing, 50 to 100 m or more
1996	Royal Oak	UG	Y	N	Sperry	Variable spacing, 50 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 coordinates 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

Source: IDM (2016)

10.8 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 an 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 downhole 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 15 of the 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; and
- 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.

10.9 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.

10.10 Drill Spacing

Drill spacing on the Red Mountain Gold Project 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, 132, AV, and JW lower 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 132, AV, and JW Lower zones have 50 m sectional spacing and 50 to 100 m spacing on section.

10.11 Drill Intercepts

Table 10-2 shows a selection of intersections through the main resource zones to illustrate typical grades and widths the deposit.

Table 10-2: Typical Drill Intersections

Zone	Section	Hole ID	From	To	Length (m)	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	172.7	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

Source: IDM (2016)

10.12 Comments on Section 10

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 Red Mountain Gold Project 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 10-2, appropriately reflect the nature of the gold mineralization, and include areas of higher-grade intervals in low-grade drill intercepts; and
- 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 & Security

11.1 Sampling Methods

11.1.1 Soil Sampling

The methods used by Bond and LAC for collecting soil samples is unknown. 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.

11.1.2 Rock & Channel Sampling

The methods used by Bond and LAC for collecting rock samples is not known, however, the 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 in length and 5 cm in width and depth. The samples were chipped out using a chisel after being cut with the rock saw (McLeod, 2014).

11.1.3 Drill Sampling

11.1.3.1 Bond and LAC 1989 -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 to for sample preparation and analysis and the other half was kept for future reference at the core storage facility in Stewart, British Columbia.

11.1.3.2 LAC 1993 - 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.

11.1.3.3 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 was placed in the sample bag.

11.1.3.4 Banks Island 2013

Banks Island sampled over 0.25 to 1.5 m intervals that honoured 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.

11.1.3.5 IDM 2014-2016

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 in length and never crossed lithological boundaries. Sampling took place at the camp in Goldslide Creek. The core was cut 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 were the same in 2016 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, 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.

11.1.4 Whole Rock Samples

For drill holes from most Bond and LAC drilling programs, samples were collected 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 samples pulp was submitted for whole rock analysis.

11.1.5 1993-1994 LAC Underground Chip Samples

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.

11.1.6 1993-1994 LAC Bulk Samples

A muck sample was collected from every underground round, either from the main decline or from the crosscuts 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 Analytical Laboratories

Several primary laboratories have been used for Red Mountain samples over the history of the Project as shown in Table 11-1. For a majority of drill hole samples, Eco-Tech Labs was the primary laboratory.

Table 11-1: Laboratory Summary Table

Operator	Laboratory	Time Period	Sample Type Analyzed
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	Surface & UG drill samples, rock samples
IDM	ActLabs, Kamloops, BC	2016	Check assay on drill pulps

Source: ACS (2017)

The ISO accreditations of all labs from 2000 and prior is unknown. AGAT Labs, Acme (Bureau Veritas), ALS Global, and ActLabs are all ISO 9001:2008 accredited laboratories. All laboratories are also ISO/IEC 17025:2005 accredited for some specific tests including fire assays with AA and gravimetric finishes.

11.3 Sample Preparation & Analysis

11.3.1 Sample Preparation

Sample preparation for drill samples includes drying as required, crushing, and selection of a sub-split which is then pulverized to produce a pulp sample sufficient for analytical purposes. Table 11-2 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 11-2: Sample Preparation Procedures

Laboratory	Procedure
Min-En	Dry, two-stage crushing to -1/8”, 500 g split pulverised to 95% passing -120 mesh.
Bondar-Clegg	Dry, crush and pulverize to -150 mesh (onn 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, 1000 gram split pulverized to 85% passing -200 mesh

Source: ACS (2017)

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.

11.3.2 Sample Analysis

The analytical methods used on drill core and check assays from Red Mountain are summarized in Table 11-3.

Table 11-3: Analytical Methods

Laboratory	Procedure
Min-En	Fire assay for gold a 30 g sample with an AA finish. Results over 0.5 oz/T Au (~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 0.2 oz/T Au (~7 g/t) re-assayed with a gravimetric finish.
Acme	Fire Assay for gold on a 30 g sample, Mutli 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. Ag 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. Ag assayed using an aqua regia digestion and an AA finish. Also multi element ICP on 1993 over flow samples. Whole rock analyses by XRF.
AGAT	Fire assay for gold on a 30g 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 30g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. Ag by Acid digestions with AA finish, repeated if >100 g/t Ag, 48 element 4 acid, ICP-MS package.

Source: ACS (2017)

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 fire assay and ICP work was done at the Kamloops facility. For the 1996 Royal Oak samples, all analytical work was carried out in Kamloops.

11.4 Quality Assurance & Quality Control

The QA/QC for the Red Mountain drilling programs has previously been presented by Anderson (2000) and reported in Craig (2001). All historic QA/QC data was recompiled and assessed in early 2016.

11.4.1 Bond & LAC QA/QC 1989-1992

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 (1121 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.

11.4.2 LAC QA/QC 1993-1994

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 11-4. Note that in 2000, +/-2 standard deviations were used as failure limits for all standards. Current industry standards are to use +/-2 standard deviations as a warning limit and +/-3 standard deviations as failure limits, and this has been followed here.

Table 11-4: Red Mountain Canmet Standards

Standard Name	Value Au (g/t)	+3SD (g/t)	-3SD (g/t)
MA-1b	17.00	16.55	17.45
MA-2b	2.39	2.47	2.31
MA-3	7.49	7.78	7.2

Source: ACS (2017)

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, four site specific standards were created by CDN Resource Laboratories of Delta, BC, for this program 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 +/-3SD failure limits, based on the round-robin results and analysis, are presented in Table 11-5.

Table 11-5: Red Mountain LAC Site Specific Standards

Standard Name	Value Au (g/t)	+3SD (g/t)	-3SD (g/t)	%RSD
LAC #1	1.90	2.35	1.45	8.06
LAC #2	3.19	3.70	2.68	5.35
LAC # 3	6.35	7.34	5.36	5.19
LAC #4	14.15	16.07	12.23	4.54

Source: ACS (2017)

The results of the standard insertions from the 1993 and 1994 LAC drilling programs are summarized in Table 11-6.

Table 11-6: Summary of Standard Insertions

Standard	Number of Analyses	Mean of Analyses (g/t)	Expected Value (g/t)	Percentage difference (%)	No. High Fails	No. Low Fails
MA-1b	22	17.1	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

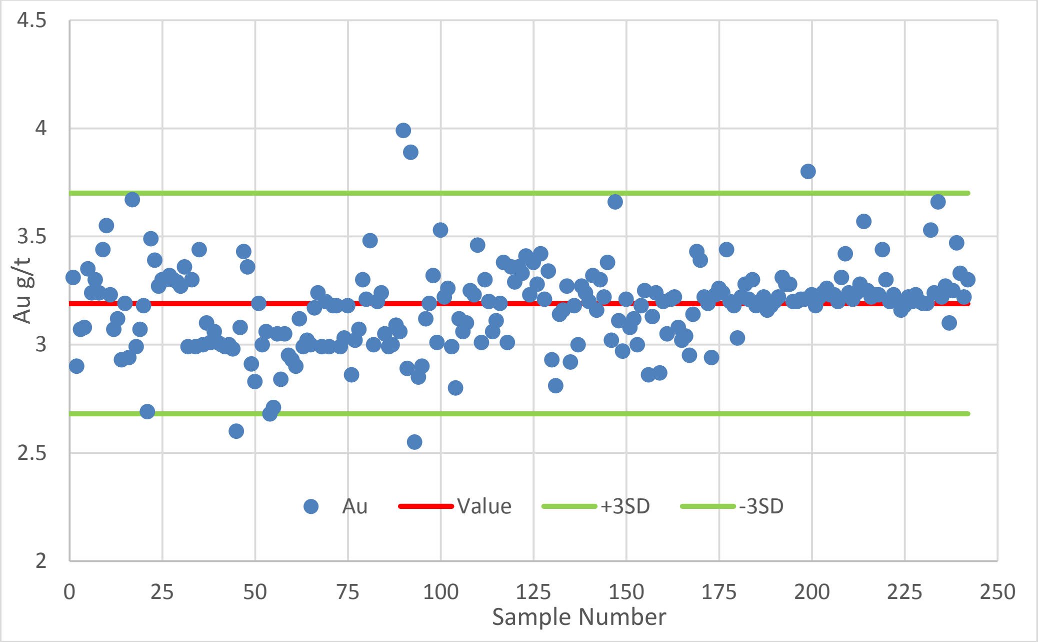
Source: ACS (2017)

In general, the Canmet standards did not perform well relative to their +/-3 standard deviation 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 standard deviation failure limits. Examples of timeline plots are shown in Figure 11-1 (LAC 2) and Figure 11-2 (LAC 3). Note that a tightening of results relative to the expected values is evident in both plots at approximately samples 185 and 210 respectively, corresponding to the moving of all analytical work from the Stewart Eco-Tech facility to the Kamloops facility.

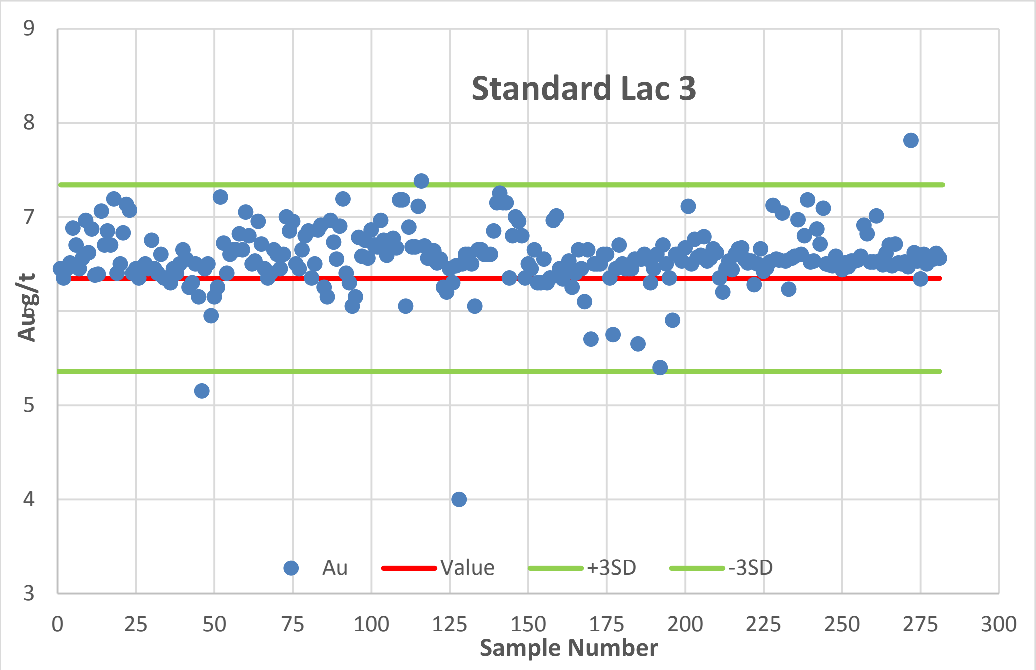
During the LAC programs, analytical results for standards were tracked and if results were out of acceptable limits, the lab was asked to re-assay all samples that were analyzed in the same batch as the standard.

Figure 11-1: Timeline Plot for Standard LAC 2



Source: ACS (2017)

Figure 11-2: Timeline Plot for Standard LAC 3



Source: ACS (2017)

11.4.2.1 Check Assays

A rigorous check assay program was implemented by LAC in 1993 with a protocol whereby 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 check samples 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. Nine hundred and twenty-five check assays in the historic Access database were not included in the compilation as the material, pulp or reject, could not be determined. Results are summarized in Table 11-7.

Table 11-7: Summary of 1993-1994 Check Assays

Material	Number	Eco Tech Au (g/t)	Chemex Au (g/t)	% Diff	Eco Tech Ag (g/t)	Chemex Ag (g/t)	% Diff
Pulp	371	1.71	1.83	+7.0	7.31	7.37	+0.8%
Reject	2689	3.02	2.84	-6.0	13.44	13.04	-3.0%

Source: ACS (2017)

The pulp check assays 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. There is a consistent low bias by Chemex at all grade levels compared to Eco-Tech with the reject checks. 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.

11.4.2.2 Duplicates

Anderson (2000) reported a LAC 1993-94 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 sulphide veins in opposing halves of core.

11.4.2.3 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 11-8 summarizes the weighted assay averages for the higher-grade intervals in the four drill holes from 13 to 29 m.

Table 11-8: 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

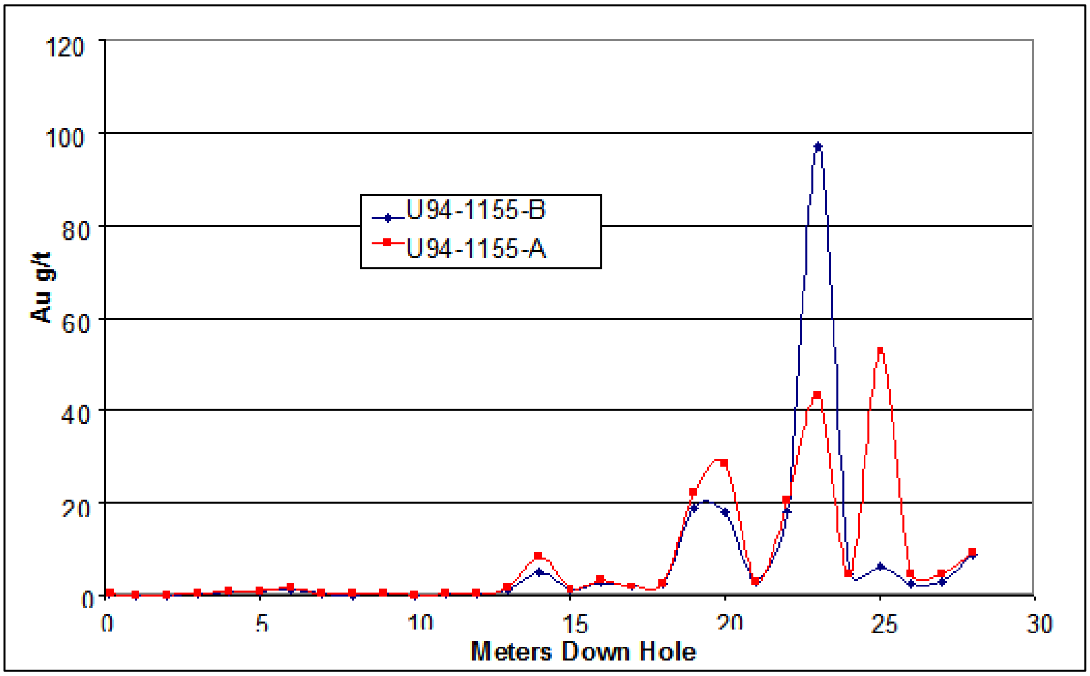
Source: ACS (2017)

Figure 11-3 to Figure 11-5 display the downhole assay comparisons for each half of the core for holes U94-1155 to U94-1157. Figure 11-6 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 comparisons 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.

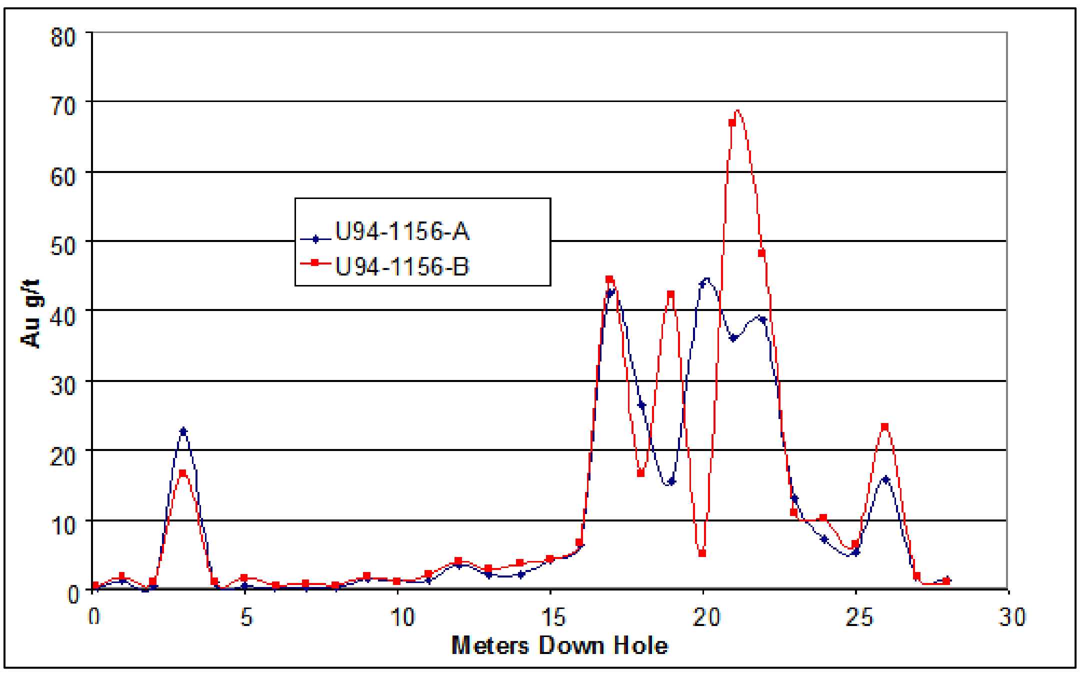
This variance is evident in the four individual drill holes (Figure 11-3 to Figure 11-6). When these plots considered in conjunction with the mean results for the 369 duplicates presented above, which suggest extremely good global precision, it is evident that variability on an individual sample basis can vary considerably, particularly at higher grades, as can be seen in Figure 11-7.

Figure 11-3: U94-1155 Gold Assay on Both Halves of Core



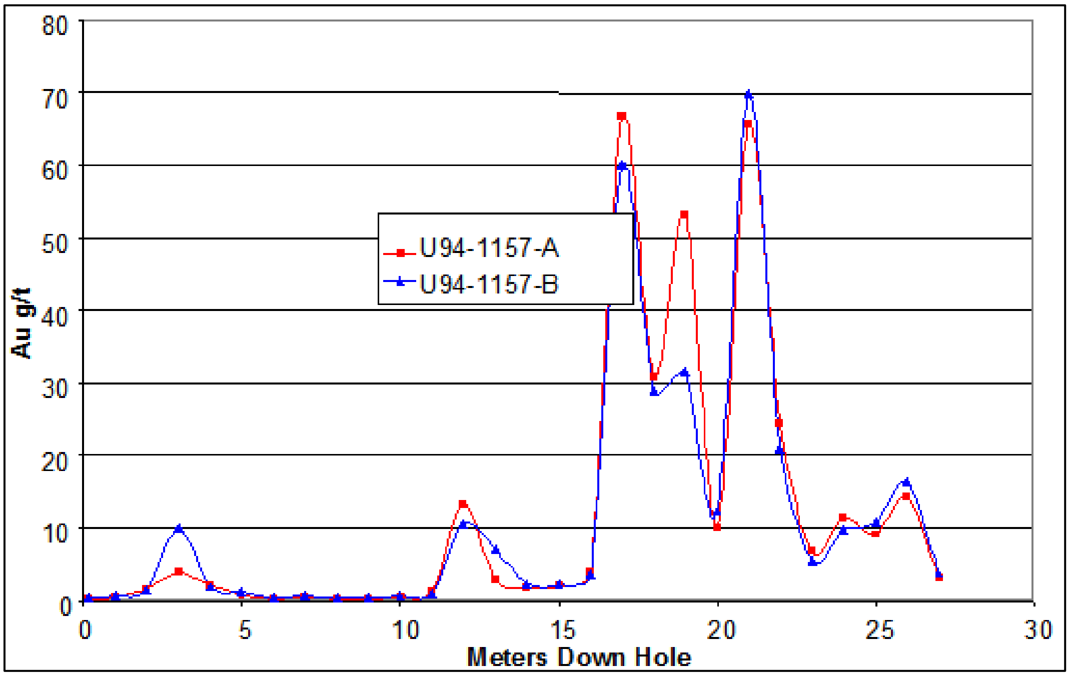
Source: NAMC (2001)

Figure 11-4: U94-1156 Gold Assays on Both Halves of Core



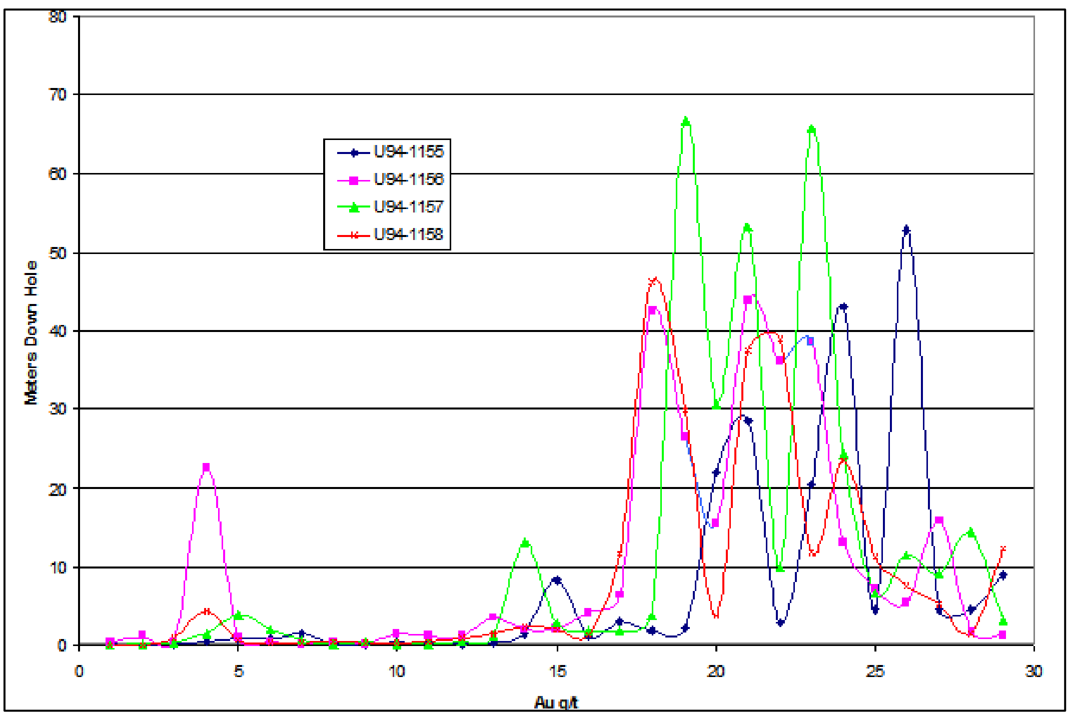
Source: NAMC (2001)

Figure 11-5: U94-1157 Gold Assays on Both Halves of Core



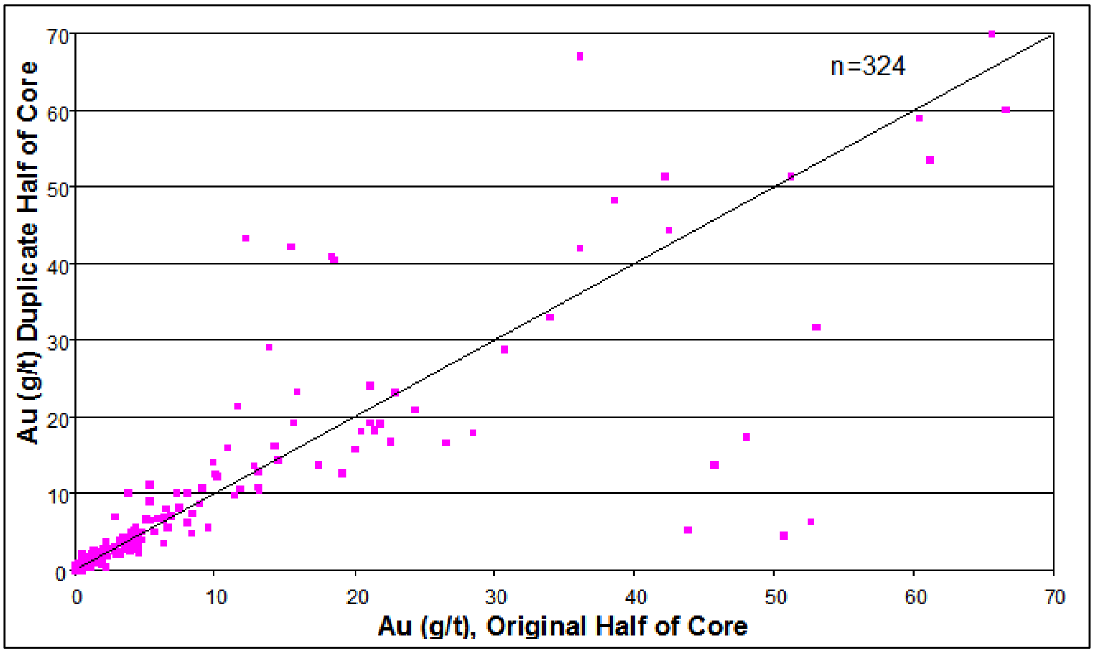
Source: NAMC (2001)

Figure 11-6: Gold Assay Comparison for DDH U94-1155, -1156, -1157 and -1158



Source: NAMC (2001)

Figure 11-7: Comparison of Original Gold Assays vs. Duplicate Halves of Core



Source: NAMC (2001)

11.4.2.4 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 on a regular basis during the 1993 and 1994 programs by a LAC geologist.

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 is suspect. In order to resolve this, Eco Tech built a separate sample preparation facility in July 1993, which was inspected by a sampling consultant from Vancouver who considered the updated facilities adequate.

In 1994 a second consultant, Jack Stanley (Stanley, 1994a, 1994b, & 1994c) was contracted to visit the Eco-Tech lab and audit sample preparation, assaying procedures and the internal lab's QA/QC. He made two visits and on each occasion noted some issues that were subsequently addressed.

11.4.2.5 Extra Sample Splits

In 1994 at least one in 40 samples had two assay splits from the coarse (-10 mesh) sample taken and one 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.

11.4.3 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.

11.4.4 NAMC QA/QC 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.

11.4.5 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 11-9. The coarse field blank used came from a local quarry along the highway near the mouth of Bear Creek. The rock was from a barren Bitter Creek pluton of quartz monzonite composition.

Table 11-9: Banks Island Standard Reference Material

Standard	Grade Au (g/t)	+3SD (g/t)	-3SD (g/t)	Grade Ag (g/t)	+3SD (g/t)	-3SD (g/t)
PM929	5.1	5.81	4.39	65.0	72.5	57.5
PM451	1.77	1.95	1.59	NA	NA	NA
BL118	<0.005	NA	NA	<0.3	NA	NA

Source: ACS (2017)

A total of six standard insertions were made, with all returning values within the +/-3SD 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 indicates a good assay precision.

11.4.6 IDM QA/QC 2014

IDM inserted one standard every 20 samples and one blank every 20 samples into its 2014 drill sample shipments. No duplicate was inserted and no check assays were done.

The standards used were from CDN Labs in Vancouver with values and limits as shown in Table 11-10. 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 11-10: 2014 CDN Labs Standards

Standard	Au Grade (g/t)	+3SD (g/t)	-3SD (g/t)	Ag Grade (g/t)	+3SD (g/t)	-3SD (g/t)
GS13A	13.2	14.28	12.12	NA	NA	NA
GS3M	3.10	3.45	2.85	95.4	103.8	87.0

Source: ACS (2017)

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 ppb so failure limit is 15 ppb).

11.4.7 IDM QA/QC 2016

IDM used a stronger QA/QC program in 2016 consisting of a QC material once every 10 samples rotating between standards, blanks and field duplicates.

Four new standards, two from CDN Labs and two OREAS standards were used. Expected values and limits are shown in Table 11-11. Timeline plots show good assay accuracy for both gold and silver.

Table 11-11: 2016 Red Mountain Standards

Standard	Au Grade (g/t)	+3SD (g/t)	-3SD (g/t)	Ag Grade (g/t)	+3SD (g/t)	-3SD (g/t)
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
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

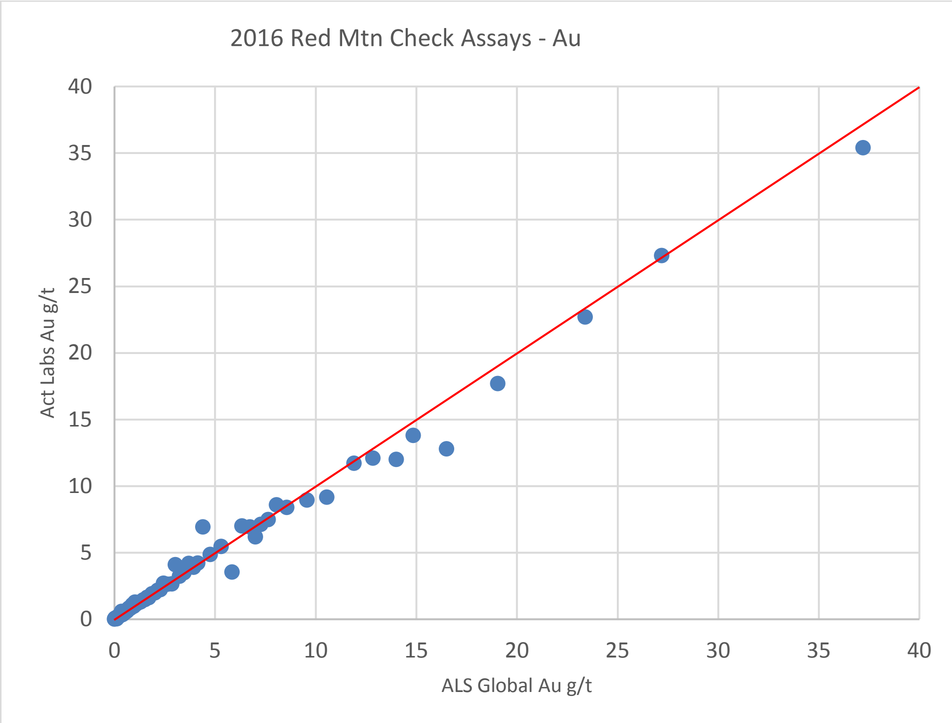
Source: ACS (2017)

The same field blank was used as in 2014. For gold, there were a few mild failures (20 to 30 ppb) and one failure at 0.11 g/t. The latter was found to follow a sample with 63.8 g/t Au and is attributed to mild contamination from the previous sample. One lesser failure of 30 ppb was found to follow a sample grading 18.85 g/t. A single silver failure of 2.3 g/t Ag could not be explained. The number and tenor of 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 11-8.

Figure 11-8: 2016 Check Assay Comparison



Source: ACS (2017)

11.5 Comments on QA/QC

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 dates over which the bulk of the work 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 dataset is quite small.

Historic 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, respectively, 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.

11.6 Databases

Information in 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 Microsoft (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.

11.7 Security and Storage

11.7.1 Security

For all Red Mountain drilling programs, samples were under the control of drill contractors and Project staff until they left the immediate Project 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 located 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, BC. The same procedure was used in 2016 except that sample shipments were delivered to the ALS Global sample preparation facility in Terrace, BC.

11.7.2 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, BC. The authors are unaware of the current location of the Banks Island core or if it still exists.

Core from the 2014 and 2016 IDM drilling programs is stacked on pallets at the Goldslide Creek camp. 2016 sample rejects and pulps are currently stored at the ALS Global facility in Terrace but storage in Stewart is being arranged.

11.8 Comments on Section 11

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 analysed 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 Project site or logging facilities, and delivered to the lab either directly by Project staff or commercial trucking companies;
- The data that was collected was entered in databases and validated through visual checks prior to being imported into the master drill database(s); and
- Current sample storage procedures and storage areas are consistent with industry standards.

12 Data Verification

12.1 Geology, Drilling and Assaying

12.1.1 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 of 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 to ensure all data was entered, checked, and that data printouts were with completed logs in the files. Merging new data into the master drill databases was done by the system manager.

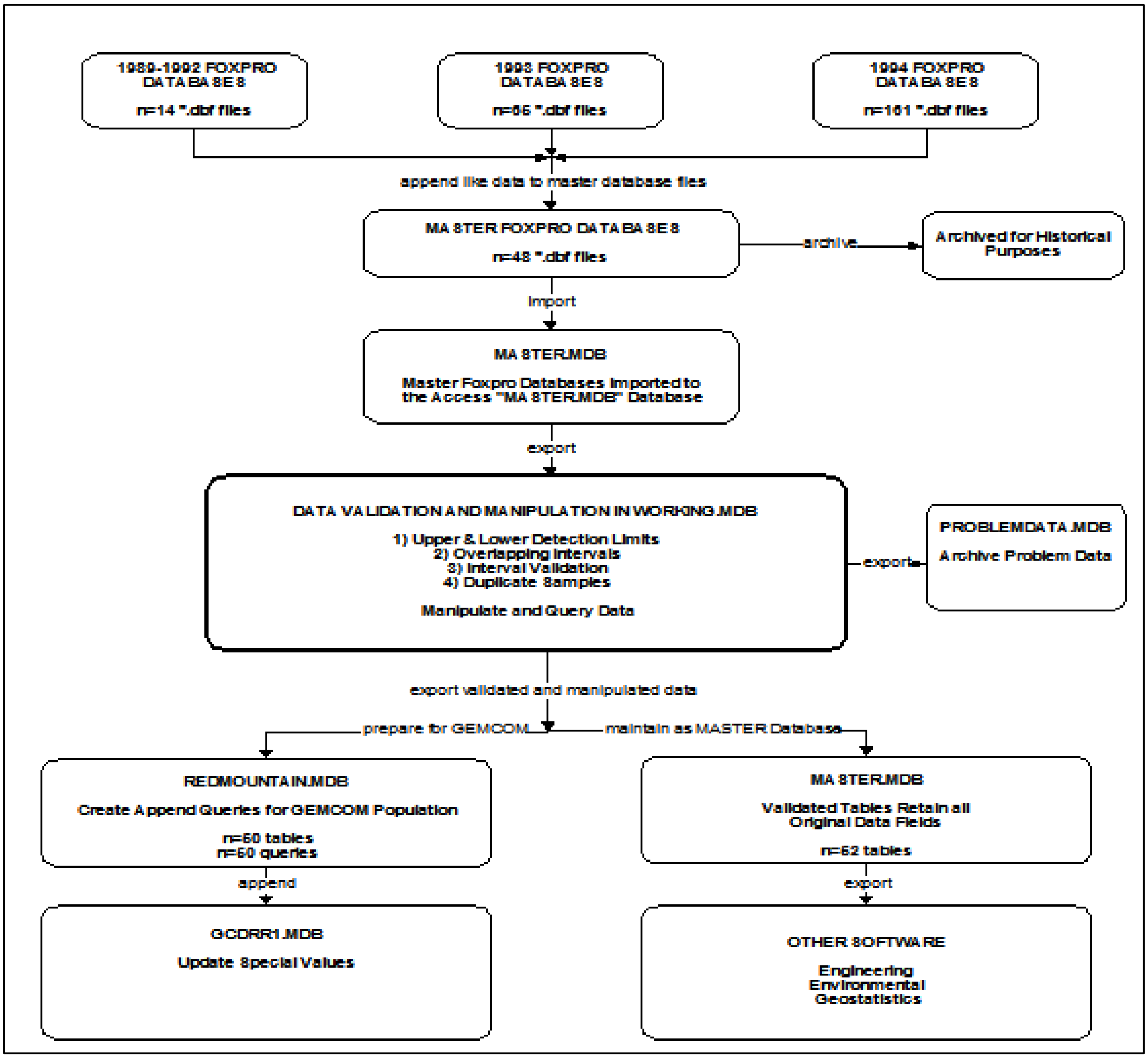
12.1.2 Electronic Data Verification

LAC collected and organized over one gigabyte of electronic information during their work on the Red Mountain property during 1993 and 1994. As the Project 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 the Project data, undertook to create and validate an MS Access database that held all of 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).

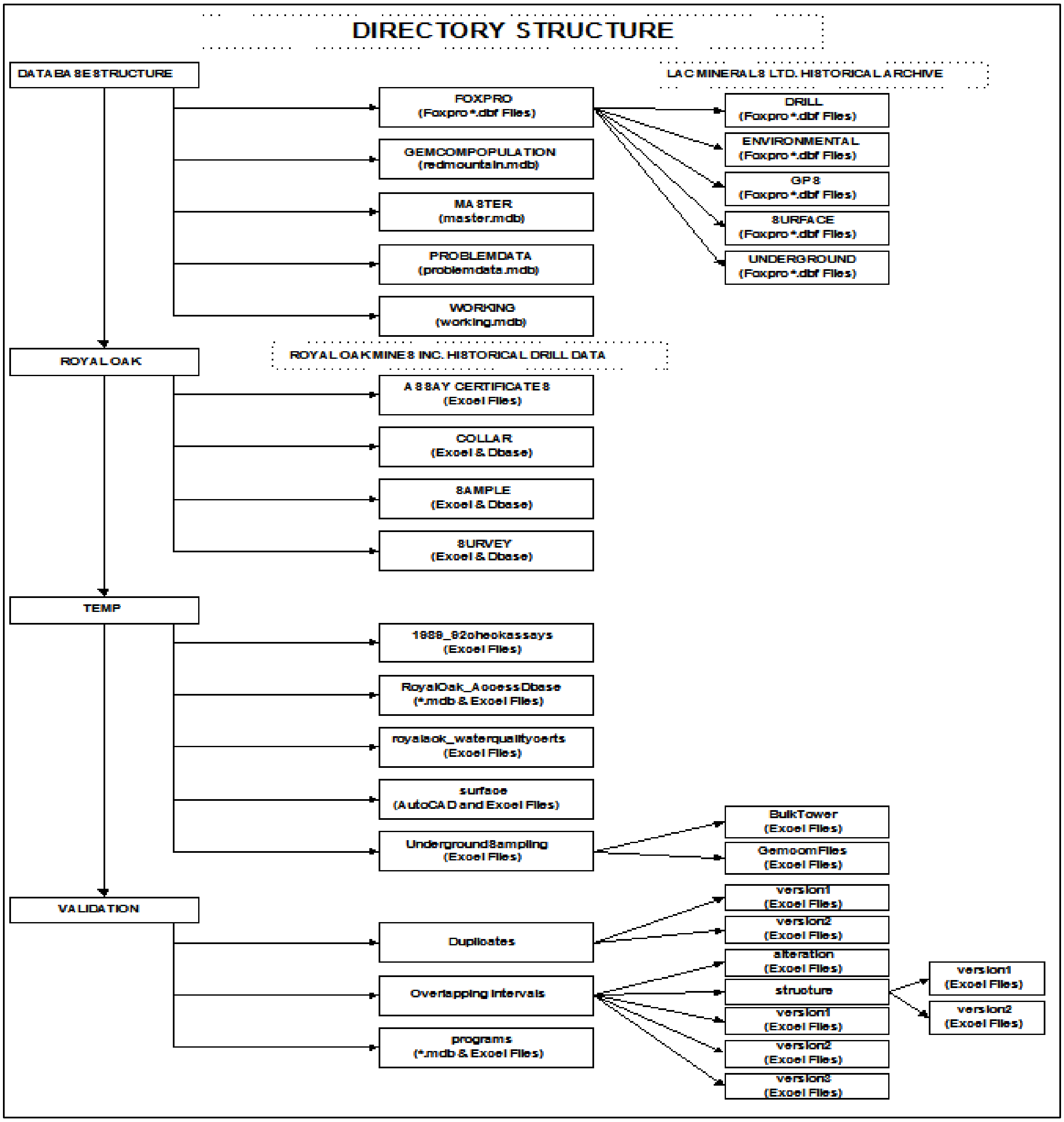
Flow sheets illustrating the database compilation procedures and resulting directory structure as shown in Figure 12-1 and Figure 12-2, respectively.

Figure 12-1: Data Validation Flowchart



Source: NAMC (2001)

Figure 12-2: Directory Structure



Source: NAMC (2001)

12.1.3 NAMC Metallurgical Composites

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 5-15 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 pulverized to >90% -150 mesh. The pulps were then sent to IPL Laboratories in Vancouver, BC for FA/AAS for Au and FA/Grav in Ag 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 an Au and Ag assay verification and as a large-scale quality control device.

Table 12-1: DDH Composite Assays vs. Metallurgical Composites

Metallurgical Composite	DDH Comp Average Grade Au (g/t)	Met Comp Average Grade Au (g/t)	DDH Comp Average Grade Ag (g/t)	Met Comp Average Grade 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)

12.1.4 2016 Data Verification

For the resource update, some of the key tables in the GEMs database were audited for holes affecting the resource solids.

12.1.4.1 Collar Table

Drill collar locations were audited through examination in three dimensional 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.

12.1.4.2 Survey Table

The downhole survey table form the GEMs database was checked by examining the changes from one survey to the next in all holes for both azimuth and dip. 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.

12.1.4.3 Assay Table

Most original assay certificates are available in the Red Mountain files. A check was made between the gold and silver values in the GEMs data base and values on the 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 assays checked for each zone is given in Table 12-2.

Table 12-2: Assay Validation Summary

Zone	No. Holes Checked	No. of Assays in Solid	No. of Assays Checked	% Checked
Marc	10	1978	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
Totals	**33**	**3057**	**447**	**14.6**

Source: ACS (2017)

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 assays were validated by comparing the data base values to certificates obtained directly from ALS Global. In total, assays from 20 certificates representing 825 of 6,022 assays, or 13.6% were evaluated. No discrepancies were found.

12.1.4.4 Site Visit & Check Samples

ACS carried out a site visit to the Red Mountain Gold Project on March 25, 2016 for one day. During the site visit, ACS verified the property access, logistics, and surface geology. 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 re-sampling program carried out by 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 only collected to verify that the Red Mountain Gold Project did host gold and silver mineralization in the range of grades that have been reported for the Project in the past.

Table 12-3: Results of 2016 Re-Sampling 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 cross cut	4.95	7.43	26	16
I95067	1200 cross cut	1.3	0.1	1.7	<5
I95068	1295 cross cut	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

Source: ACS (2017)

12.1.5 Comments on Section 12

The resource 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, the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation.

12.2 Mining

Mining design data was verified through review of studies and reports. Any studies and reports referred to were thoroughly reviewed and summarized in this report and align with the FS mine design and mine plan. All mining data was verified and is adequate for this FS Technical Report as required by NI 43-101 guidelines.

12.3 Metallurgy

The first steps taken to verify the metallurgical data were to review previous test work, reports and confirm the location of new drill core used in the most recent test program completed by Basemet Laboratories. The drill holes were located in each of the three zones (Marc, AV, and JW) to provide spatially representative samples. Segments of the drill core were used to create variability and global composites for each zone. AV and JW zones were increased in size based on the new mine plan and resource. Variability samples to include the new areas were added to the test program as well as new global samples created to confirm the flowsheet. The global samples were blended from the drill core based on advice from the geologist to target the LOM average grade. It is the QP’s opinion that there is sufficient data and test work to estimate the metallurgical recoveries and define the flowsheet for this FS Technical Report, as defined in the NI 43-101 guidelines.

13 Mineral Processing & Metallurgical Testing

13.1 Introduction

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 test work conducted between 1991 and 1997 focused on cyanide leaching as the primary process for extracting gold and silver from the deposit.

In the spring of 2000, a metallurgical test program was conducted at Process Research Associates (PRA) under the direction of Dr. Morris Beattie, P.Eng. This test work focused on producing a saleable gold and silver rich flotation concentrate.

In 2015, test work was completed by Gekko Systems (Gekko). Gravity, flotation and comminution test work 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 the following reports and were summarized in the 2016 PEA.

- Lakefield Research, 1991. Project No. L.R. 4048: The Recovery of Gold from Red Mountain Project Samples;
- Brenda Process Technology, 1994. Final Report on Metallurgical Testing of Red Mountain Gold - Silver Ores;
- A.R. MacPherson Consultants Ltd., 1994. Project No. 8300: Proposed Grinding System for Red Mountain Project;
- International Metallurgical and Environmental Inc., 1997. A Metallurgical Investigation into the Reduction of the Acid Generation Potential of Red Mountain Tailings;
- Beattie Consulting Ltd., 2001. Red Mountain Project Flotation Study (Summarizes the test work completed by Process Research Associates); and
- Gekko Systems, 2015. Project No. T1306: Red Mountain Python Amenability Test work.

This section of the FS will focus exclusively on the 2016-2017 test work program completed by Base Met Laboratories in Kamloops, BC (BL0084, BL0184). The recovery method and process design criteria outlined in Section 17 were based primarily on the results from this program.

The 2016-2017 test work program was completed on variability and composite samples for Marc, AV, JW, and 141 zones. Initially, the test work focused on the PEA flowsheet, which included rougher flotation followed by concentrate leach. 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, WOL became the focus of the program. Optimization continued primarily on the Marc zone master composite and was confirmed with the AV, JW, and 141 samples. The final flowsheet included a primary grind to 80% passing (P_{80}) 25 microns (µm) followed by CIL recovery of gold and silver.

The metallurgical test procedures and results for the 2016-2017 test program are documented in the following reports:

- Base Met Labs, 2017. Project No. BL0084: Metallurgical Testing Red Mountain Project (Base Met, 2017a); and
- Base Met Labs, 2017. Project No. BL0184: Additional Metallurgical Testing Red Mountain Project (Base Met, 2017b).

The QP confirms that test samples are generally representative of the various deposits and styles of mineralization and the mineral deposit as a whole and there is no indication of any processing factors or deleterious elements that could have a significant effect on potential economic extraction.

13.2 Base Met Labs 2016 to 2017 Metallurgical Testing

The test work program was split into three phases. The first two phases, designated BL0084, focused on the following objectives:

- Create variability and master composites for the Marc, JW, and AV deposits;
- Define the metallurgical responses of both flotation/concentrate regrind/leach and whole ore leach;
- Select the recovery method for optimization;
- Identify the parameters affecting process response for the chosen recovery method;
- Assess the metallurgical variability of the deposit by testing discrete subsamples of the various geographical zones;
- Generate advanced process engineering data for equipment selection; and
- Generate tailings samples for environmental testing.

Upon completion of BL0084, additional work was carried out in BL0184. This third phase included the following objectives:

- Create new master composites for the JW and AV deposits based on lithology;
- Create additional variability samples for the JW and AV deposits that represent the new areas in the FS mine plan;
- Create variability samples for the 141 deposit;
- Test the effect of pre-oxidation, cyanide concentration, and carbon concentration on reducing operating costs;
- Assess the metallurgical variability of the 141 deposit using the optimized flowsheet; and
- Generate additional process engineering data for equipment selection.

13.2.1 Process Selection: Flotation/Regrind/Leach vs. Whole Ore Leach

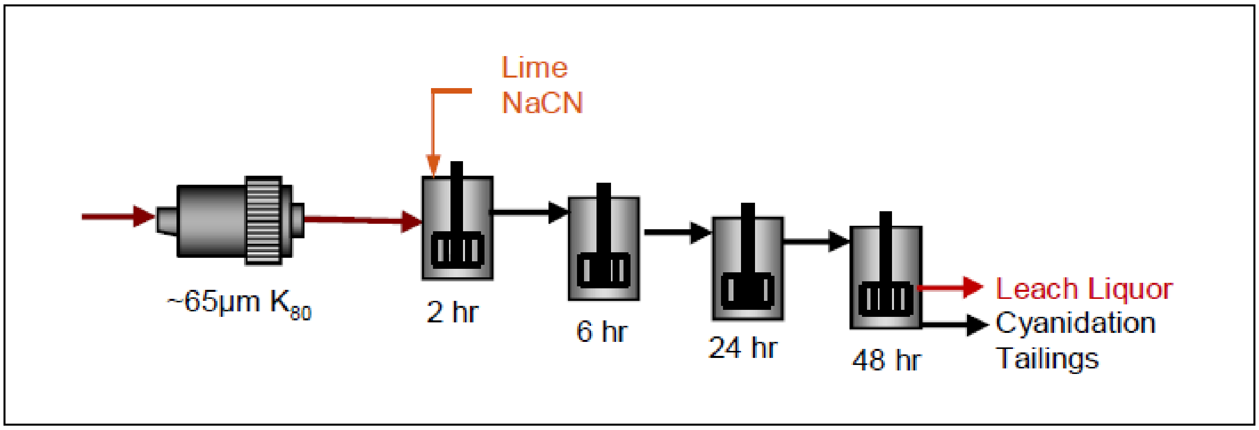
Historical testing had identified two potential processing options for recovering precious metals from Red Mountain material:

- Direct cyanide leaching, often referred to as whole ore leaching (WOL); and

- Flotation of gold-bearing sulphide material to produce a rougher concentrate followed by regrinding and cyanide leach (FRL).

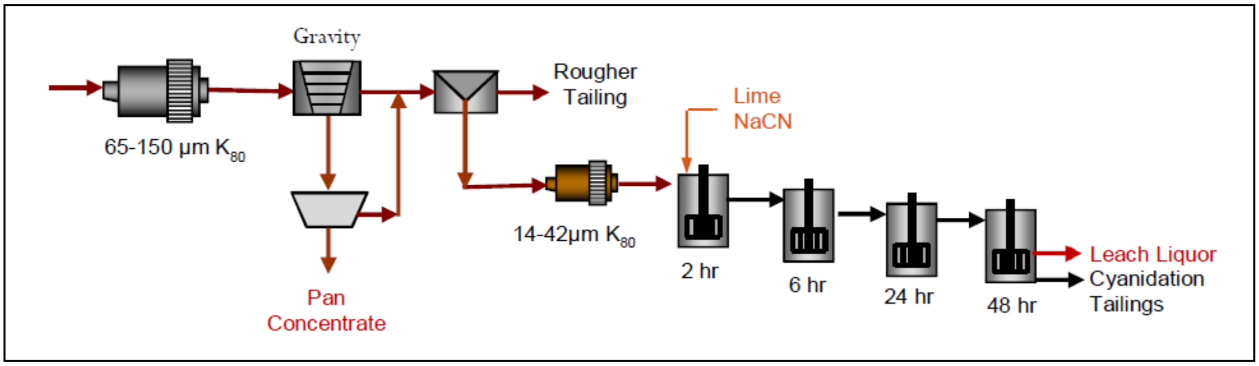
The corresponding flowsheets for each option are shown in Figure 13-1 and Figure 13-2.

Figure 13-1: Whole Ore Leach Flowsheet



Source: Base Met (2017a)

Figure 13-2: Flotation/Regrind/Leach Flowsheet



Source: Base Met (2017a)

Using these two recovery procedures, variability and master composite samples from Marc, AV, and JW were tested to evaluate metallurgical response. The results formed the basis for recommending WOL.

13.2.1.1 Variability Composite Sample Selection

Samples for the test program were drilled in summer 2016 and were received at Base Met Labs immediately after they were logged and split on site. The final samples arrived at the end of November 2016. Once all of the drill core assays were completed, test composites were constructed.

A total of 36 variability samples were constructed from three zones: 18 from Marc zone, designated as MV, and nine from both AV and JW zones. The composites were selected to test a range of feed grades and geological lithologies in each zone, including various sulphur levels. Composites were constructed from contiguous half core intervals from the same drill hole to maintain spatial representation. In some instances, the gold content was quite variable between adjacent samples.

The chemical contents of key elements for the variability samples are displayed in Table 13-1.

Gold and silver values were quite variable throughout the samples, ranging from 0.2 to 31 g/t gold, and about 1 to 130 g/t silver. This wide range allowed analysis of metallurgical performance at high and low ends of anticipated mine feed grades. Sulphur and iron values in the samples were also quite variable, measuring between 3.4 to 18.8% sulphur, and 5.5 to 15.1% iron. Total organic carbon (TOC) was measured at minor amounts in the samples ranging between 0.01 and 0.19%. MV9 and MV10 were found to have the highest TOC and could lead to preg-robbing in the cyanide leach circuit.

Table 13-1: Head Assay Data for BL0084 Variability Composites

Composite ID	Zone	Chemical Content				
		Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
MV1	Marc Zone	5.79	42	7.5	6.66	0.03
MV2	Marc Zone	9.06	46	9.0	9.85	0.04
MV3	Marc Zone	3.52	7.4	6.7	4.96	0.03
MV4	Marc Zone	3.96	41	6.1	5.50	0.02
MV5	Marc Zone	12.3	68	11.2	13.2	0.02
MV6	Marc Zone	5.10	24	9.8	10.4	0.03
MV7	Marc Zone	3.73	5.9	8.0	5.39	0.03
MV8	Marc Zone	5.81	7.4	7.0	4.68	0.01
MV9	Marc Zone	8.53	42	7.5	5.38	0.11
MV10	Marc Zone	7.14	18	7.7	7.09	0.19
MV11	Marc Zone	0.87	0.9	7.4	5.37	0.05
MV12	Marc Zone	22.4	12	11.2	11.7	0.04
MV13	Marc Zone	5.33	50	8.5	9.83	0.02
MV14	Marc Zone	3.94	2.4	9.3	5.98	0.01
MV15	Marc Zone	14.7	16	10.8	12.2	0.02
MV16	Marc Zone	4.84	2.7	7.7	4.81	0.02
MV17	Marc Zone	16.2	71	13.0	17.4	0.02
MV18	Marc Zone	2.35	38	8.4	8.94	0.01
JW1	JW Zone	0.86	0.7	6.4	3.80	0.02
JW2	JW Zone	5.70	26	10.4	13.0	0.02

Table 13-1: Head Assay Data for BL0084 Variability Composites (continued)

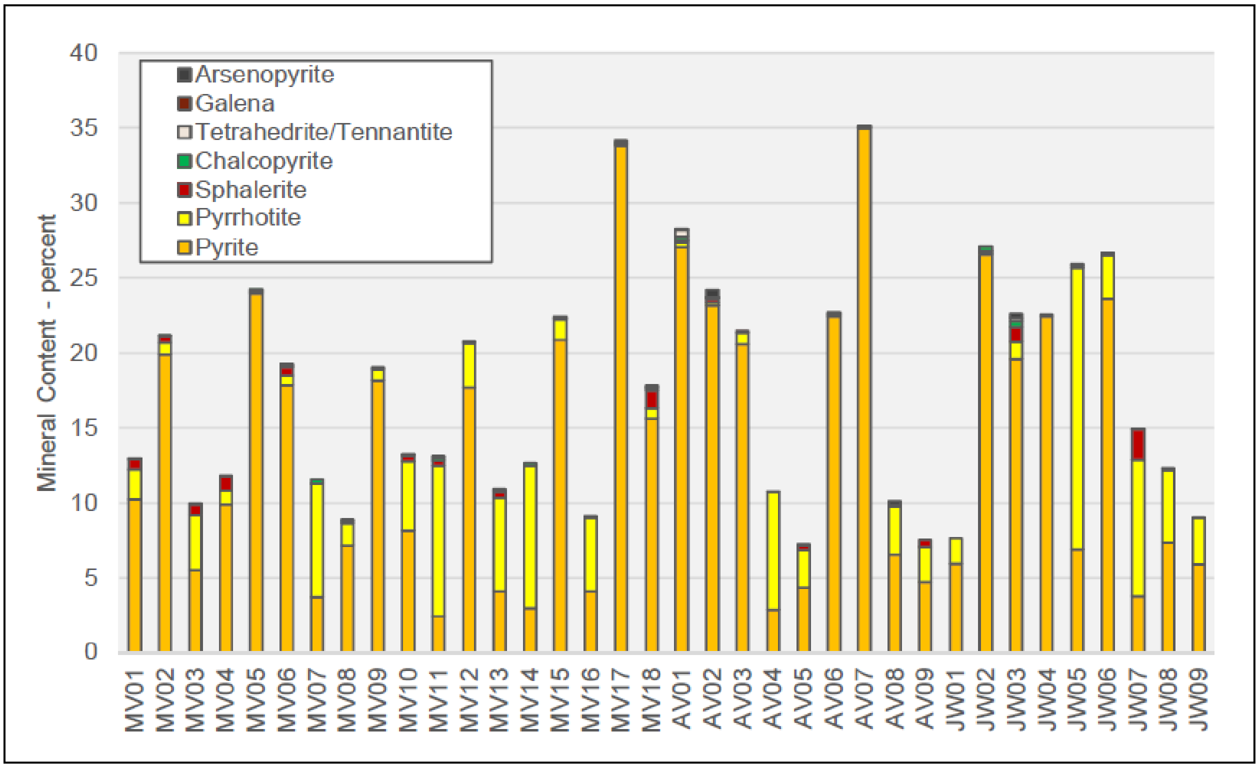
Composite ID	Zone	Chemical Content				
		Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
JW3	JW Zone	6.77	130	9.2	10.2	0.02
JW4	JW Zone	7.31	11	8.9	10.6	0.02
JW5	JW Zone	5.35	3.8	15.1	11.4	0.04
JW6	JW Zone	2.82	15	12.2	14.4	0.02
JW7	JW Zone	2.75	1.1	9.8	6.33	0.02
JW8	JW Zone	6.10	6.2	9.25	6.00	0.01
JW9	JW Zone	1.83	1.5	7.2	4.37	0.01
AV1	AV Zone	6.96	22	11.9	14.1	0.03
AV2	AV Zone	3.90	11	9.9	11.7	0.02
AV3	AV Zone	4.76	15	9.7	10.9	0.05
AV4	AV Zone	6.41	1.4	7.4	4.65	0.01
AV5	AV Zone	0.38	1.7	7.6	4.13	0.02
AV6	AV Zone	5.15	28	10.0	11.5	0.02
AV7	AV Zone	31.0	49	14.1	18.8	0.02
AV8	AV Zone	29.5	42	11.2	11.9	0.01
AV9	AV Zone	0.21	3.0	5.8	3.42	0.01

Source: Base Met (2017a)

13.2.1.2 Mineralogical Characterization of Variability Composites

The mineral composition of each variability sample was determined by QEMSCAN - Bulk Mineral Analysis (BMA) determinations. The main non-sulphide minerals included muscovite, quartz, feldspars, and chlorite. The sulphide minerals, which are of particular interest, are shown in detail in Figure 13-3. Pyrite and pyrrhotite represented the majority of the sulphide minerals in the samples, at levels up to about 35% in some samples. Pyrrhotite is a highly reactive mineral and susceptible to oxidation, which could negatively affect flotation performance.

Figure 13-3: Sulphide Mineral Content for BL0084 Variability Composites

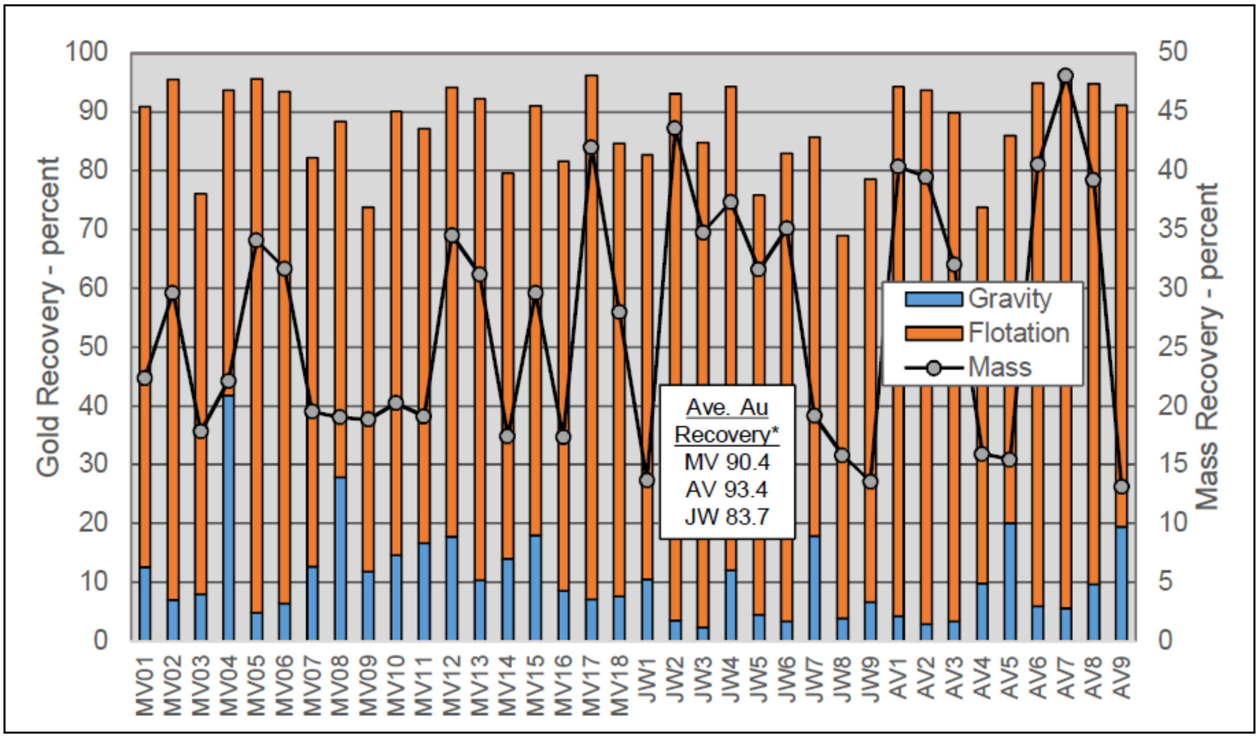


Source: Base Met (2017a)

13.2.1.3 Gravity and Rougher Flotation Testing on Variability Composites

Gravity concentration followed by bulk sulphide rougher flotation tests were conducted on all variability composites at a primary P_{80} grind size of 150 µm. The flotation tests were conducted at natural pH and used Potassium Amyl Xanthate (PAX) as the sulphide mineral and gold collector. A graphical summary of the gravity - flotation tests is displayed in Figure 13-4.

Figure 13-4: Gravity and Rougher Flotation Results for BL0084 Variability Composites



Source: Base Met (2017a)

Gravity performance was quite poor for many of the samples, averaging 11% gold recovery. Only two samples achieved over 20% gold recovery; the maximum and minimum were about 41% and 2%, respectively. Based on this data, there appears to be very little coarse gold in the deposit, therefore, the addition of a gravity concentrate circuit was not included in the process design.

Overall, gravity followed by rougher flotation recovered between 69-96% of the gold, 41-98% of the silver, and 69-98% of the sulphur. Due to the high sulphide content in the samples, the mass recoveries to the rougher concentrates were quite high, up to 48% and averaging 27%. This reduces the benefit of a flotation circuit prior to regrinding and subsequent leaching, as a significant portion of the feed mass still requires fine regrinding.

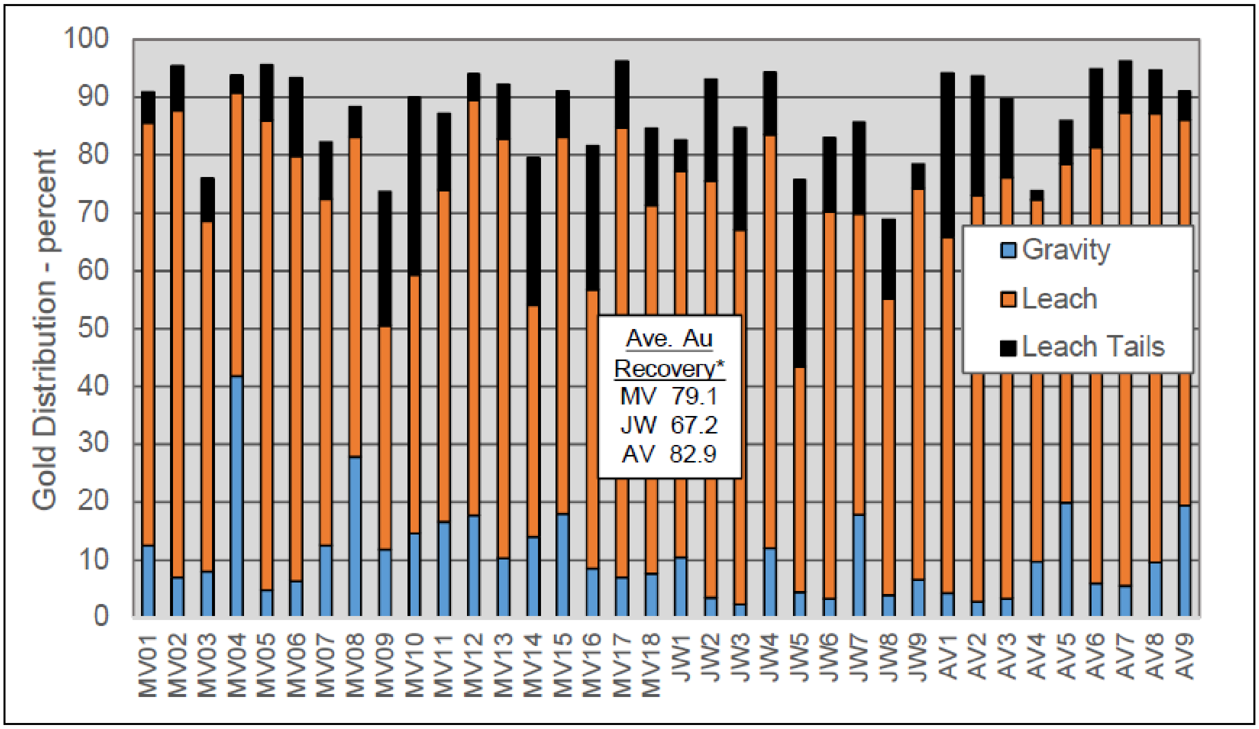
13.2.1.4 Rougher Concentrate Cyanidation Tests on Variability Composites

The rougher concentrates were subsequently reground and leached with cyanide to determine the resulting gold and silver extraction. The average regrind P_{80} of the concentrate was 27 µm. Tests were conducted at a pH of 11.0, and a sodium cyanide (NaCN) concentration of 1,000 ppm. Pre-oxidation was not utilized, but leaches were sparged with oxygen at sampling intervals.

A summary of gold and silver distributions, including in the cyanide leach tailings, are shown in Figure 13-5 and Figure 13-6. The cumulative sums of the blue and orange bars are considered the final recovery position of the combined gravity, flotation, and flotation concentrate leach process. The average final recovery for each zone is shown in the inset table. The black portion of the bar indicates the metal lost to the leach tailings.

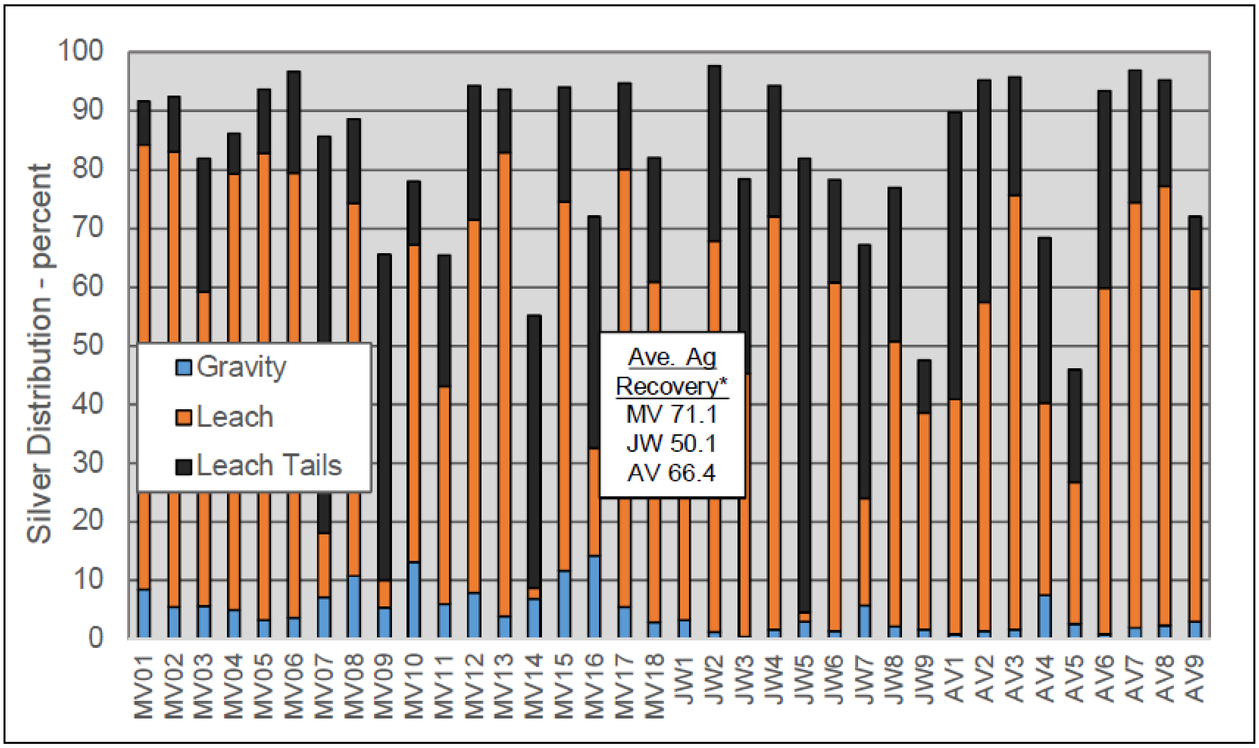
As shown, many of the samples had significant gold and silver losses to the leach tailings. Overall recovery, which includes gravity concentrate and leach extraction, ranged from 43 to 91% gold and 5 to 84% silver.

Figure 13-5: Overall Gravity/Flotation/Leach Au Recoveries for BL0084 Variability Composites



Source: Base Met (2017a)

Figure 13-6: Overall Gravity/Flotation/Leach Ag Recoveries for BL0084 Variability Composites

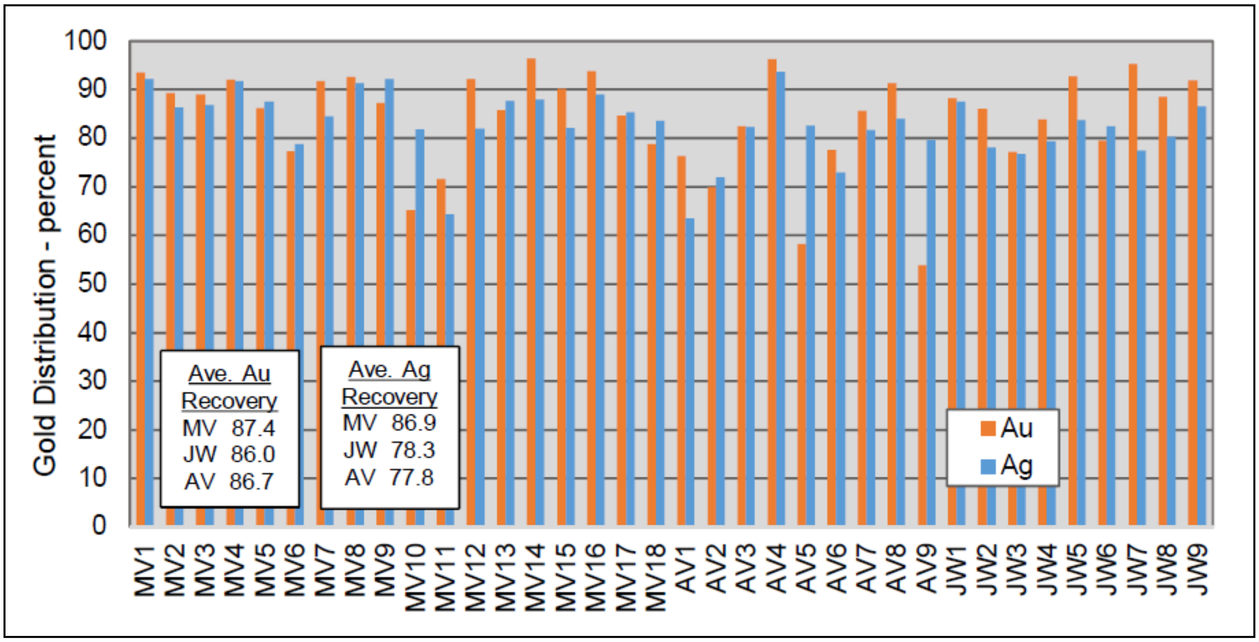


Source: Base Met (2017a)

13.2.1.5 Whole Ore Leach Tests on Variability Composites

Whole ore carbon in leach tests were conducted on all variability samples at primary grind sizes targeting a P_{80} of 40 µm. Tests were conducted at a pH of 11.0, and a NaCN concentration of 1,000 ppm. Lead nitrate was added at 250 g/t, and carbon was added at 50 g/L. Oxygen was sparged at 2, 6, 24, and 48 hour time intervals. The gold and silver recoveries are shown in Figure 13-7. The inset tables display weighted average recoveries by feed grade.

Figure 13-7: Whole Ore Leach Results for BL0084 Variability Composites



Source: Base Met (2017a)

13.2.1.6 Master Composite Sample Selection

Master Composites for the three zones were constructed using the variability composite ratios depicted in Table 13-2.

The master composites were constructed to target the average LOM head grades for each zone. Some variability composites were not included in the master composites since they were outside the projected mine plan.

Table 13-2: Composition of BL0084 Master Composites

Marc Master Composite		AV Master Composite #1		JW Master Composite #1	
Composite ID	Contribution (%)	Composite ID	Contribution (%)	Composite ID	Contribution (%)
MV1	7	AV1	20	JW1	11
MV2	11	AV2	22	JW2	26
MV3	0	AV3	8	JW3	0
MV4	10	AV4	10	JW4	34
MV5	5	AV5	13	JW5	9
MV6	6	AV6	12	JW6	20
MV7	0	AV7	6	JW7	0
MV8	7	AV8	0	JW8	0
MV9	3	AV9	9	HW9	0
MV10	8				
MV11	8				

Table 13-2: Composition of BL0084 Master Composites (continued)

Marc Master Composite		AV Master Composite #1		JW Master Composite #1	
Composite ID	Contribution (%)	Composite ID	Contribution (%)	Composite ID	Contribution (%)
MV12	5				
MV13	10				
MV14	0				
MV15	2				
MV16	2				
MV17	6				
MV18	10				
Total	**100**	**Total**	**100**	**Total**	**100**

Source: Base Met (2017a)

Head assays for the Master Composites are shown in Table 13-3. The JW zone had limited material for composite construction and the overall composite was relatively low in weight. Therefore, subsequent optimization testing in this program focused on the Marc and AV master composites. Final conditions derived from the Marc and AZ zone were then applied to the JW master composite.

Table 13-3: Head Assay Data for BL0084 Master Composites

Composite ID	Chemical Content										
	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 Composite - Head 1	9.1	33	8.98	8.5	0.05	355	162	2,290	471	75	52
Marc Master Composite - Head 2	11.0	32	8.77	8.7	-	341	181	2,530	502	76	53
Average	**10.1**	**33**	**8.88**	**8.6**	**0.05**	**348**	**173**	**2,410**	**487**	**76**	**53**
AV Master Composite #1 - Head 1	6.45	16	10.9	9.2	0.01	657	58	1,130	481	313	35
AV Master Composite #1 - Head 2	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**
JW Master Composite #1 - Head 1	5.63	13	12.3	10.2	0.02	467	36	606	307	94	29
JW Master Composite #1 - Head 1	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**

Source: Base Met (2017a)

13.2.1.7 Gold Mineralogical Assessment on Master Composites

All three master composites were submitted for gold trace mineral searches to investigate gold occurrences in the samples. Due to the limited number of gold occurrences found, the results should be taken as indicative only. A summary of the mineral associations is shown in Table 13-4.

Although telluride gold minerals were observed, the majority of the gold weighted by mass was observed as native gold. The gold particles were generally very small, all less than 10 µm in diameter and often finer. Much of the gold was un-liberated, locked with gangue minerals, most of which being pyrite.

Table 13-4: Master Composite Gold Deportment Percents by Mineral Species

	Native Gold Au	Electrum Au/Ag	Petzite Ag_3AuTe_2	Sylvanite $(Au,Ag)_2Te_4$	Hessite $(Ag,Au)_2Te$	Aurostibite $Au(TeSb)_2$	Calaverite $AuTe_2$
Marc Master	94.8	0.0	2.7	1.2	1.2	0.0	0.0
AV Master 1	81.8	0.2	2.7	6.1	1.9	7.4	0.0
JW Master 1	91.6	0.0	3.2	0.8	1.8	0.0	2.6

Source: Base Met (2017a)

13.2.1.8 Master Composite Metallurgical Testing

Master composite samples were used to assess metallurgical response for various test conditions. Both the direct cyanide leaching process and the gravity/flotation/leach option were evaluated. Due to poor flotation performance of several composites in the variability testing campaign, most of the focus was placed on direct cyanide leaching.

A series of leach tests were conducted on the whole feed of the Marc and AV Master Composites. A summary of test conditions and results is summarized in Table 13-5.

Table 13-5: Master Composite Whole Ore Leach Test Results

Test	Grind K_{80} (µm)	Gravity	pH	$Pb(NO_3)_2$ (g/t)	NaCN (ppm)	Carbon (g/L)	O_2 Sparged	Pre-Ox 24hr	Extraction Au (%)	Extraction Ag (%)
Marc Master Composite										
43	68	No	11.0	0	1,000	0	Yes	None	**80**	**76**
44	68	Yes	11.0	0	1,000	0	Yes	None	**82**	**84**
45	50	No	11.0	0	1,000	0	Yes	None	**85**	**82**
46	37	No	11.0	0	1,000	0	Yes	None	**86**	**86**
47	68	No	11.0	0	500	0	Yes	None	**81**	**79**
48	68	No	11.0	0	2,000	0	Yes	None	**81**	**77**
49	68	No	11.0	0	3,000	0	Yes	None	**81**	**81**
50	68	No	10.0	0	1,000	0	Yes	None	**81**	**78**
51	68	No	11.5	0	1,000	0	Yes	None	**81**	**79**
52	68	No	12.0	0	1,000	0	Yes	None	**79**	**79**
53	68	No	11.0	250	1,000	0	Yes	None	**81**	**74**
54	68	No	11.0	500	1,000	0	Yes	None	**80**	**72**
55	68	No	11.0	0	1,000	0	No	Air	**82**	**75**
56	68	No	11.0	0	1,000	0	Yes	O2	**82**	**76**
77	37	No	11.0	250	1,000	50	Yes	None	**89**	**90**
78	68	No	11.0	250	1,000	50	Yes	None	**86**	**86**
117	37	No	11.0	250	1,000	50	Yes	None	**90**	**89**
118	21	No	11.0	250	1,000	50	Yes	None	**92**	**92**
119	17	No	11.0	250	1,000	50	Yes	None	**93**	**93**

Table 13-5: Master Composite Whole Ore Leach Test Results (continued)

Test	Grind K_{80} (µm)	Gravity	pH	$Pb(NO_3)_2$ (g/t)	NaCN (ppm)	Carbon (g/L)	O_2 Sparged	Pre-Ox 24hr	Extraction Au (%)	Extraction Ag (%)
AV Master Composite #1										
57	70	No	11.0	0	1,000	0	Yes	None	**79**	**70**
58	70	Yes	11.0	0	1,000	0	Yes	None	**76**	**74**
59	50	No	11.0	0	1,000	0	Yes	None	**81**	**67**
60	41	No	11.0	0	1,000	0	Yes	None	**83**	**71**
61	70	No	11.0	0	500	0	Yes	None	**75**	**54**
62	70	No	11.0	0	2,000	0	Yes	None	**78**	**59**
63	70	No	11.0	0	3,000	0	Yes	None	**78**	**66**
64	70	No	10.0	0	1,000	0	Yes	None	**78**	**65**
65	70	No	11.5	0	1,000	0	Yes	None	**80**	**69**
66	70	No	12.0	0	1,000	0	Yes	None	**76**	**61**
67	70	No	11.0	250	1,000	0	Yes	None	**80**	**65**
68	70	No	11.0	500	1,000	0	Yes	None	**79**	**61**
69	70	No	11.0	0	1,000	0	No	Air	**77**	**57**
70	70	No	11.0	0	1,000	0	Yes	O2	**80**	**62**
79	41	No	11.0	250	1,000	50	Yes	None	**82**	**64**
80	70	No	11.0	250	1,000	50	Yes	None	**77**	**65**
120	41	No	11.0	250	1,000	50	Yes	None	**83**	**78**
121	23	No	11.0	250	1,000	50	Yes	None	**87**	**84**
122	16	No	11.0	250	1,000	50	Yes	None	**89**	**85**

Source: Base Met (2017a)

Primary grind size and CIL were dominant influences on gold and silver extraction. Preliminary tests were conducted at P_{80} grind sizes of 70 µm and 40 µm with two different carbon configurations. The first was carbon in pulp (CIP) conducted without carbon or lead nitrate, while the second was performed as CIL with lead nitrate. The second test series also investigated even finer grinding, down to a P_{80} of 16 µm. The highest gold and silver recoveries were achieved at the finest primary grind size using the CIL configuration.

Gravity concentration prior to leaching was tested on both master composites. Gold recovery was negligible at a P_{80} of 40 µm and gravity concentration was not explored further in the test program.

Flotation tests were conducted on the Marc and AV Master Composites to determine metallurgical response using a bulk sulphide flotation flowsheet. Initial testing on variability samples indicated gold losses in the flotation stage outweighed additional processing costs of WOL. Finer primary grinding prior to flotation, as well as the addition of copper sulphate were tested on the master composites to assess if an increase in gold recovery was achievable with either of these methods. The results are summarized in Table 13-6.

Gold recovery to the rougher concentrate averaged about 92% for both Marc Master Composite and AV Master Composite #1. Finer primary grinding and the addition of copper sulphate marginally increased the gold recovery for Marc Master, but was unchanged for AV Master #1. Subsequent leaching of reground rougher concentrate would incur further gold losses. Testing was not carried out and overall gold and silver extraction would likely be lower than recoveries obtained through WOL.

Table 13-6: Master Composite Flotation Test Results

Test	Composite	Primary Grind K_{80} (µm)	Reagents		Flotation Recovery			
			PAX (g/t)	$CuSO_4$ (g/t)	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 #1	150	35	0	30	92	91	93
75	AV Master #1	70	50	0	27	92	90	93
76	AV Master #1	150	35	150	30	91	92	94

Source: Base Met (2017a)

13.2.1.9 Analysis

Upon completion of the variability and master composite testing, a trade-off study was carried out to select the recovery method for the FS. The following technical aspects were considered in concluding that WOL was the preferred option over FRL.

- **Throughput** - WOL will process the full feed to the plant to recover gold and silver. To optimize the recovery, additional power (approximately 20-35%) will be required in the grinding circuit to produce a fine P_{80} grind size of 25 to 40 µm. In order to process the higher slurry volumes resulting from the increased tonnage, the size of thickeners, tanks, pumps, and CIL circuit will be correspondingly larger than for FRL.

  FRL will process a portion (30%) of the plant feed tonnage. The lower tonnage and slurry volumes will decrease the equipment size required to leach and recover the precious metals. The product size from the primary grinding circuit will be coarser at a P_{80} of 150 µm, requiring less power and decreased mill sizes. Only the concentrate will be processed through the regrind and leach circuits, resulting in lower equipment and operating costs.

- **Recovery** - The optimized flotation recovery was achieved with a primary P_{80} grind size of 150 µm, 8-minute laboratory float time, PAX/$CuSO_4$/MIBC reagent addition, and a 30% mass pull. The average gold and silver recovery to the rougher concentrate was 92.6% and 87.3%, respectively. Using WOL results, subsequent leach recoveries of 90.4% Au and 94.7% Ag were predicted at a P_{80} regrind size of 20 µm. Overall recoveries of 83.7% Au and 82.6% Ag were used for the analysis.

Based on Tests 118 and 121 in in Table 13-5.

  Table 13-5, WOL recovery was projected to be 90.8% Au and 88.6% Ag. This was achieved with a primary P_{80} grind size of 25 µm, 250 g/t $Pb(NO_3)_2$ addition and a CIL configuration.

  With less unit operations, the WOL option would have decreased gold and silver loses, increasing precious metal recovery.

- **Oxygen and Cyanide Consumptio**n - WOL will process the entire feed, and the ratio of sulphides and gold to both cyanide and oxygen would be significantly lower. For this reason, low dissolved oxygen levels and high cyanide consumptions will generally not be an issue compared to leaching sulphide-rich flotation concentrate.

  The relatively high levels and considerable variability of sulphur will have an impact on leaching flotation concentrate. Samples with high levels of sulphide minerals can result in high cyanide and oxygen consumption, as well as additional metals in solution, which can increase cyanide destruction costs.

- **Operability** - WOL will be an easier circuit to operate. The process uses less equipment and will result in lower maintenance requirements and fewer plant operators.

  FRL will be a more complex circuit to operate. The addition of flotation and regrind circuits will increase maintenance costs and require more plant operators.

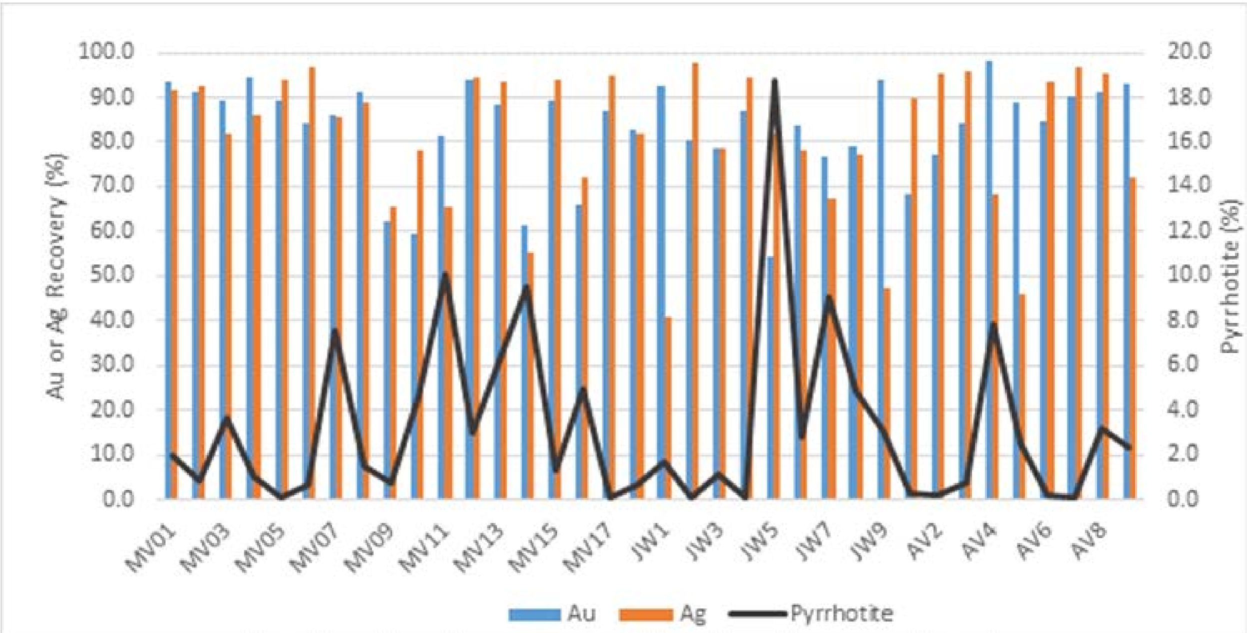
- **Flotation Variability** - There were several composites with very low flotation performance for gold. Further analysis of the variability data indicated that only TOC levels and gold feed grade had a small influence on results, but much of the variance in the recovery data remained. The overall performance of the FRL process was highly variable which was influenced by several factors which included: low initial flotation recovery, increased oxygen demand during leaching, variability in pyrrhotite levels, and variability in sulphur grade.

  Pyrrhotite levels varied significantly in the deposit, for some samples representing most of the sulphide mineral mass. Pyrrhotite is very reactive and oxidizes rapidly, degrading flotation performance. Stockpiling ores containing pyrrhotite would also likely have a negative impact on flotation. The effect of pyrrhotite on flotation performance is shown in Figure 13-8.

  The variable sulphur grade will impact the flotation circuit design as the mass recovered in the concentrate will vary proportionally with the sulphur level. The downstream sulphide concentrate leach circuit will need to be sized for the maximum levels of concentrate mass, or strategic mine planning and stockpiling will need to be implemented to reduce variability and ensure an average design sulphur grade to the mill.

- **Capital and Operating Costs -** With less processing steps, WOL will have a lower capital cost than FRL; however operating costs will be higher due to the increased energy costs associated with grinding the entire feed to a P_{80} of 25 µm. Based on an economic analysis, these increased operating costs are offset by increased gold and silver recovery, making WOL the more economically viable option.

Figure 13-8: Pyrrhotite Content vs. Flotation Recovery



Source: Base Met (2017)

13.2.2 Comminution Testing

SAG Mill Comminution (SMC) tests and Bond ball mill work index tests were completed on each variability composite. A summary of results is shown in Table 13-7. The SMC test results were on the hard end of the spectrum, mostly measuring between 80th and 95th hardness percentiles in the JK database.

Based on Bond work index testing, the majority of the samples would be considered hard or very hard for ball mill grinding, with average work indices of 19, 19, and 22 kWh/t, respectively for Marc, AV, and JW. It should be noted that the tests were conducted using a closing screen of 106 µm.

Table 13-7: Comminution Results for BL0084 Variability Composites

Composite ID	Bond Ball Mill Work Index (kWh/t)	SMC Test Results							
		DWI (kWh/m)	Mia (kWh/t)	Mib (kWh/t)	Mic (kWh/t)	A	b	t_a	SAG Circuit Specific Energy (kWh/t)
MV1	18.9	9.43	23.9	18.9	9.8	68	0.45	0.27	11.78
MV2	17.5	8.16	20.7	15.9	8.2	62.3	0.58	0.32	11.04
MV3	20.5	10.45	26.4	21.3	11	76.2	0.36	0.25	12.3
MV4	19.7	9.26	24	18.9	9.8	66	0.46	0.28	11.64
MV5	16.8	6.38	17.1	12.6	6.5	64.6	0.72	0.41	9.71
MV6	19.6	9.53	22.2	17.6	9.1	73.3	0.45	0.27	11.96
MV7	21.7	11.24	28	23	11.9	100	0.25	0.23	12.92
MV8	20.7	9.73	25.1	20	10.3	70	0.41	0.27	11.94
MV9	20.0	10.8	26.7	21.7	11.2	84.3	0.32	0.24	12.58
MV10	19.6	9.13	23.1	18.2	9.4	67.7	0.47	0.28	11.61
MV11	19.7	11.18	27.4	22.4	11.6	72.4	0.36	0.23	12.82
MV12	16.9	9.82	23.5	18.7	9.7	63.2	0.49	0.26	12.15
MV13	17.4	7.11	18.3	13.7	7.1	59.9	0.71	0.36	10.26
MV14	21.8	8.56	23	17.9	9.3	73.2	0.44	0.3	11.12
MV15	17.5	8.73	22.8	17.8	9.2	74.8	0.44	0.3	11.23
MV16	21.2	8.92	25.4	19.9	10.3	75.5	0.39	0.29	11.21
MV17	15.3	7.74	20.1	15.3	7.9	71.7	0.53	0.33	10.69
MV18	19.0	9.81	24.2	19.3	10	73.6	0.41	0.26	12.06
JW1	27.3	11.1	27.7	22.7	11.7	90.2	0.28	0.23	12.85
JW2	18.2	9.34	21.8	17.2	8.9	74.1	0.45	0.28	11.9
JW3	20.5	10.87	24.6	20	10.4	77.9	0.37	0.24	12.89
JW4	17.4	9.41	22.4	17.7	9.2	68.9	0.47	0.28	11.96
JW5	19.6	8.87	21.5	16.8	8.7	76.1	0.45	0.29	11.57
JW6	21.4	8.62	19.8	15.4	8	78.9	0.48	0.3	11.25
JW7	21.6	9.89	24.9	19.9	10.3	75.5	0.39	0.26	11.98
JW8	25.1	9.34	23.9	18.9	9.8	73.7	0.42	0.28	11.65
JW9	24.9	10.76	26.9	21.9	11.3	81.6	0.32	0.24	12.66
AV1	22.5	8.34	20.1	15.5	8	63.5	0.59	0.31	11.16
AV2	17.2	10.14	24.3	19.5	10.1	68	0.44	0.26	12.29
AV3	18.0	10.42	24.3	19.6	10.1	86.5	0.34	0.25	12.59
AV4	21.8	11.7	28.3	23.4	12.1	85.8	0.29	0.22	13.18
AV5	23.7	10.69	27.2	22.1	11.4	83.1	0.32	0.24	12.39
AV6	16.0	9.38	22.8	18	9.3	66.7	0.48	0.28	11.88
AV7	16.1	5.65	13.1	9.4	4.8	55.2	1.12	0.46	8.78
AV8	16.9	9.96	23.2	18.5	9.6	77.5	0.4	0.26	12.33
AV9	21.7	9.37	24.5	19.4	10	76.1	0.39	0.28	11.68

Source: Base Met (2017a)

Two sets of master composites were generated for Marc, AV, and JW testing. One set of master composites was constructed exclusively for comminution testing to determine the crushing and abrasion indices for each zone. The results are summarized in Table 13-8.

The crusher work index ranged from 9.5 - 12.1 kWh/t. Based on these values, the JW sample would be considered soft, while the Marc and AV samples would be considered average. The abrasion index for these composites ranged from 0.24 - 0.30, classifying these samples as abrasive.

Table 13-8: Crushing Work Index and Abrasion Index by Zone

Zone	Bond Crushing Work Index (kWh/t)	Bond Abrasion Index (g)
Marc	11.7	0.24
AV	12.1	0.30
JW	9.5	0.29

Source: Base Met (2017a)

13.2.3 Cyanide Destruction Testing

Cyanide destruction testing was conducted on the three master composites. Optimization testing was conducted on leach tailings from Marc Master at a P_{80} of 44 µm. These conditions were applied to AV and JW leach tailings. Tests were also then conducted on the three master composites at a P_{80} of 25 µm. A summary of conditions and results are shown in Table 13-9.

Table 13-9: Master Composite Cyanide Destruction Test Results

Test	Particle Size K_{80} (µm)	Retention Time (mins)	Reagents Used		Number of Displace ments	Solution Concentration (ppm)			
			SO_2 (g/g CN_{WAD})	Cu (mg/L)		CN_{WAD}	Cu	Fe	Calc' CN_{TOT}
D1 - Marc	44	57	6	0	3.2	135.0	49.1	104.6	427.2
D2 - Marc	44	54	6	600	3.4	1.8	1.71	0.0	1.9
D3 - Marc	44	54	5	810	2.2	3.0	6.67	0.0	3.0
D4 - Marc	44	55	5	300	2.4	1.2	2.61	0.0	1.2
D5 - Marc	44	84	6	300	3.2	1.0	1.40	0.0	1.0
D6 - Marc	44	86	5	300	3.2	1.7	2.97	14.8	43.1
D7 - Marc	44	78	4	300	3.5	0.9	2.19	0.5	2.4
D8 - AV	41	69	4	300	4.6	1.4	3.09	0.0	1.4
D9 - JW	40	86	4	300	3.7	1.6	4.69	0.0	1.6
D10 - Marc	21	86	4	300	2.9	48.2	90.4	30.1	132.2
D11 - AV	23	86	4	300	2.6	61.3	98.6	6.2	78.6
D12 - JW	25	86	4	311	1.6	55.3	97.8	38.7	163.5
D13 - JW	25	86	4	515	1.6	78.7	129.8	8.6	102.8

Source: Base Met (2017a)

Initial tests on Marc Master leach tailings, at a P_{80} of 44 µm, indicated 300 mg/L copper, SO_2 to CN_{WAD} ratios of four and six and about 80 minutes' retention time were required to reduce CN_{WAD} to about 1 ppm. When these conditions were applied to JW and AV leach tailings at 40 µm, the final CN_{WAD} concentration was about 1.5 ppm. When conditions were applied to leach tailings at 25 µm, the CN_{WAD} concentrations in the tailings were quite high, at 48 to 79 ppm. Additional optimization testing was carried out in BL0184 to reduce these levels.

13.2.4 Solid-Liquid Separation Testing

Three slurry samples of Marc Master, AV Master #1, and JW Master #1, at a P_{80} of 25 µm, were shipped to TAKRAF Canada Inc in Burnaby, BC for solid-liquid separation testing. The results are documented in the following report:

- Tenova Delkor, 2017. Test No: D1724–Red Mountain TW_TCAN.TH.FP Test Report.

The objective of the test work was to determine the pre-leach thickener operating parameters and to determine whether the tailings material is suitable for filtration. The scope of the test program included flocculant selection, settling tests, optimum dilution tests, flocculant dosage tests, compaction tests, rheology, and rise rate or thickener loading selection. It also included the selection of filter press operating parameters and equipment design.

Thickening results indicated that a flocculant dosage of 20 - 25 g/t AF304HH or its equivalent, produced the best overflow clarity, while a rise rate of 2.1 - 2.3 $m^3/m^2/h$ and a solids loading of 0.19 to 0.23 $t/hr/m^2$ should be used for thickener design. Tenova Delkor recommended an 18 m Pre-Leach Thickener with 3 m tank wall and a floor slope of 9 degrees to achieve a target underflow density of 55% solids. To maintain a stable thickener operation, they recommended a feed dilution of 8% solids.

The possibility of producing a dry stackable tailings product was also investigated by Tenova Delkor. They concluded that a Fluid Actuated Screw Technology (F.A.S.T.) Filter press model F.A.S.T. FP 1500/99/40/10/R/A (1500 mm plate, 99 chambers, 40 mm chamber depth, 10 bar feeding pressure, Recessed Plate, opening all at once) could achieve a cake moisture content of 16.5% - 18.5% at a total estimated cycle time of 16 minutes.

13.2.5 Optimized Test Work – BL0184

Upon completing BL0084, Base Met Labs carried out a follow-up program to optimize test conditions in the CIL circuit and reduce operating costs. Cyanide concentration, carbon concentration, and a pre-oxidation stage were investigated on Marc Master Composite to evaluate their effect on precious metal recovery. The optimal conditions were then applied to freshly created AV, JW, and 141 composites.

Comminution, cyanide destruction, and CIL oxygen consumption test work was also completed to generate additional design parameters for the Feasibility Study.

13.2.5.1 Sample Selection

Marc zone test work was carried out using master composite samples generated during the BL0084 test program. Additional AV and JW composites were created for this test program to enhance the variability datasets within each deposit. Four variability composites were also created for the 141 zone, a deposit projected to encompass approximately 4% of the total tonnage mined. The chemical composition of each sample is shown in Table 13-10.

Table 13-10: Head Assay Data for BL0184 Variability Composites

Composite ID	Zone	Chemical Content				
		Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
JW10	JW Zone	11.65	20	8.53	10.1	0.02
JW11	JW Zone	19.95	128	9.08	9.62	0.02
JW12	JW Zone	7.37	12	7.36	7.36	0.02
JW13	JW Zone	9.84	3	21.05	18.5	0.01
JW14	JW Zone	7.51	26	11.40	16.1	0.02
JW15	JW Zone	9.88	16	9.42	7.18	0.02
JW16	JW Zone	8.22	31	7.32	7.25	0.02
JW17	JW Zone	8.17	27	9.82	11.8	0.01
JW18	JW Zone	3.51	12	9.33	8.82	0.02
AV10	AV Zone	7.2	22.2	10.7	13.3	0.04
AV11	AV Zone	5.6	27.8	11.2	15.2	0.03
AV12	AV Zone	19.2	16.9	11.4	13.6	0.02
AV13	AV Zone	7.8	15.6	10.0	12.4	0.02
AV14	AV Zone	10.8	13.7	12.2	15.2	0.02
AV15	AV Zone	8.0	11.9	9.9	11.1	0.02
Z141-1	141 Zone	6.9	1.1	4.7	2.6	0.02
Z141-2	141 Zone	3.7	0.5	8.8	5.0	0.01
Z141-3	141 Zone	5.1	4.0	7.6	7.3	0.01
Z141-4	141 Zone	6.3	16.6	10.7	10.7	<0.01

Source: Base Met (2017b)

AV and JW master composites were also generated from the variability composites. The composition of each composite was developed with direction from the Project geologist in order to ensure that the anticipated lithologies within each deposit were well represented in the masters. Based on the FS mine plan, a few variability samples from BL0084 were also included to improve spatial representation. The composition of each master composite is summarized in Table 13-11 and the resulting head grades are show in Table 13-12.

Table 13-11: Composition of BL0184 Master Composites

JW Master Composite #2		AV Master Composite #2	
Composite ID	Contribution (%)	Composite ID	Contribution (%)
JW2	6.0	AV1	0.2
JW3	4.6	AV2	0.3
JW4	3.0	AV3	9.3
JW5	9.6	AV4	8.8
JW6	15.0	AV6	7.8
JW10	12.9	AV7	2.0
JW11	4.0	AV10	20.0
JW12	10.0	AV11	11.0
JW13	5.8	AV12	3.0
JW14	1.2	AV13	12.0
JW15	15.6	AV14	4.5
JW16	2.6	AV15	21.1
JW17	3.1		
JW18	6.6		
Total	**100**	**Total**	**100**

Source: Base Met (2017b)

Table 13-12: Head Assay Data for BL0184 Master Composites

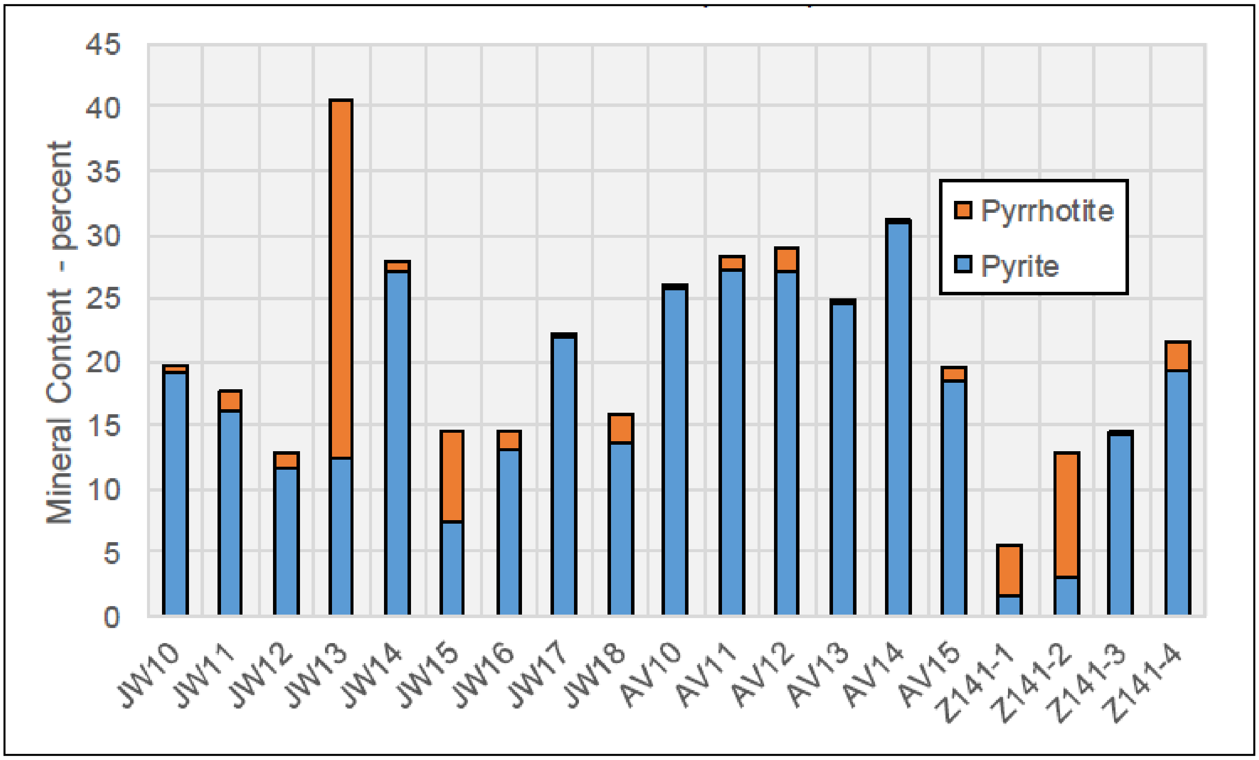
Composite ID	Chemical Content									
	Au (g/t)	Ag (g/t)	S (%)	Fe (%)	TOC (%)	Cu (g/t)	Pb (g/t)	Zn (g/t)	As (g/t)	Sb (g/t)	Te (g/t)
Marc Master Composite - Head 3	7.63	33	8.6	8.37	0.07	303	181	2,510	452	83	46
Marc Master Composite - Head 4	8.34	40	8.9	8.60	0.05	327	206	2,780	469	101	51
Average	**8.0**	**36**	**8.7**	**8.5**	**0.06**	**315**	**194**	**2,645**	**461**	**92**	**49**
AV Master Composite #2 - Head 1	8.82	17	11.8	9.58	0.02	728	71	527	630	223	43
AV Master Composite #2 - Head 2	8.51	17	11.5	9.53	0.02	729	77	474	633	213	46
Average	**8.57**	**17**	**11.7**	**9.6**	**0.02**	**729**	**74**	**501**	**632**	**218**	**44.5**
JW Master Composite #2 - Head 1	7.60	24	11.6	11.2	0.02	404	44	1,070	276	89	42
JW Master Composite #2 - Head 1	7.40	25	11.8	11.0	0.01	401	49	1,080	270	95	42
Average	**7.5**	**25**	**11.7**	**11.1**	**0.02**	**403**	**47**	**1,075**	**273**	**92**	**42**

Source: Base Met (2017b)

13.2.5.2 Mineralogical Characterization

The mineralogy of each variability sample was determined by QEMSCAN using the BMA protocol. The analysis was focused to determine sulphide mineral species present. Figure 13-9 shows the iron sulphide content in each sample. Iron sulphides represent the majority of the sulphur in the sample, averaging about 99%. Most samples consist primarily of pyrite; however, there are a few samples that have high levels of pyrrhotite, namely JW13, JW15, Z141-1, and Z141-2.

Figure 13-9: Iron Sulphide Content in BL0184 Variability Composites



Source: Base Met (2017b)

13.2.5.3 Comminution

Bond ball mill work index testing at a closing screen of 106 µm was carried out on AV and JW variability composites where sufficient sample was available. The results are summarized in Table 13-13. The new AV samples were found to be moderately hard while the JW samples were found to be hard to very hard.

Table 13-13: Bond Ball Mill Work Index Results for BL0184 Variability Composites

Composite ID	Bond Ball Mill Work Index (kWh/t)
AV11	14.8
AV12	16.7
AV13	16.7
JW11	18.9
JW12	22.3
JW14	19.1
JW15	22.5
JW16	22.1
JW17	17.7
JW18	19.2

Source: Base Met (2017b)

Preliminary fine grinding tests, based on the Levin procedure, were completed on each of the master composites. The results indicated that a ball mill would require 54.2 kWh/t (Marc), 50.7 kWh/t (AV) and 68.5 kWh/t (JW) to grind material from an F_{80} of 75 µm to a P_{80} of 25 µm. Since vertical stirred mills utilize different breakage mechanisms than ball mills, these results are not indicative of what energy will be required for the Red Mountain secondary grinding circuit. Based on discussions with vendors, the design currently includes a 1,475 kW vertical stirred mill. Confirmation testing by the chosen vendors is recommended in the next phase of the project.

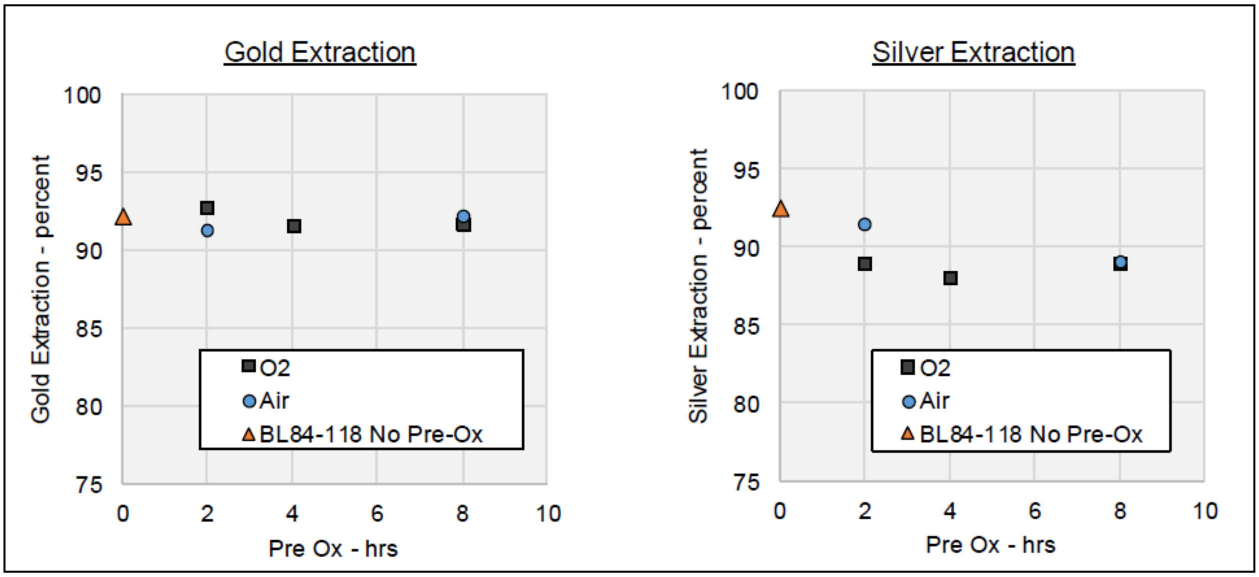
13.2.5.4 Pre-Oxidation Test Work

The original test work had indicated that pre-oxidation of the sample prior to leaching did not ultimately result in an increase in extraction of gold or silver, given that oxygen was used as the sparge gas during leach. Further examination of the data showed that with pre-oxidation, cyanide consumptions were reduced and dissolved iron in the effluent solutions were dramatically reduced. Iron in solution required high levels of copper sulphate during the cyanide destruction process. Due to high processing costs, this parameter was re-evaluated.

Using Marc Master Composite, pre-oxidation times of eight, four, and two hours were examined. Oxygen was used for the sparging gas. Air was also examined as the oxidation gas for eight and two hours. A graphical presentation of the related tests is displayed in Figure 13-10 and Figure 13-11, along with a test from the previous program with no pre-oxidation.

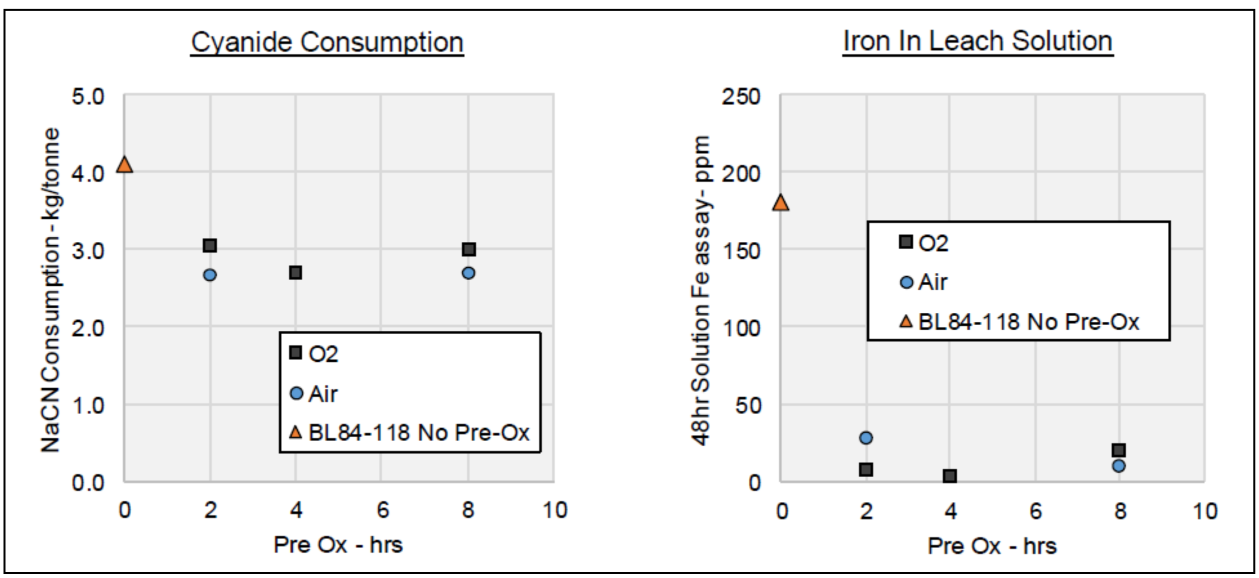
Based on the reduction of cyanide consumption and reduction of iron in the effluent, the use of a two-hour pre-oxidation step was included in the process design. Oxygen was selected for use in the pre-oxidation stage because it was shown to be beneficial to gold extraction in the kinetic leaching stages. Oxygen will also provide more aggressive oxidation for samples that have higher levels of sulphide mineral, particularly pyrrhotite.

Figure 13-10: The Effect of Pre-Oxidation on Gold and Silver Extraction



Source: Base Met (2017b)

Figure 13-11: The Effect of Pre-Oxidation on Cyanide Consumption and Iron Content



Source: Base Met (2017b)

13.2.5.5 CIL Oxygen Consumption

Oxygen conditioning tests were completed on Marc, AV, and JW master composites to estimate the quantity of oxygen needed for the process. The test was conducted in an agitated open vat, with the gas added below the agitator with an air stone sparger. Gas was initially added to achieve a volume hold-up of about 10% in the pre-oxidation stage. Once the target dissolved oxygen (DO) value of 15 ppm was reached, the oxygen flow was reduced to maintain the 15 ppm target during CIL. The results were combined with the projected plant mass balance to generate the oxygen consumptions estimates shown in Table 13-14.

Table 13-14: Projected Oxygen Consumption for Pre-Oxidation and CIL

Composite	Test	Pyrrhotite Content	Grind P_{80} (µm)	Plant Feed (tpd)	Slurry Volume (m^3/day)	Slurry Flow (m^3/hr)	Total O_2 (m^3/hr)	Total O_2 (m^3/t)	Total O_2 Consumption (kg/t)
AV Master #2	51	1.2	25	1,000	1,336	60.5	48.8	1.08	1.5
JW Master #2	52	5.5	25	1,000	1,336	60.5	71.7	1.58	2.3
Marc Master	123	2.6	44	1,000	1,336	60.5	80.4	1.78	2.5

Source: Base Met (2017b); JDS (2017)

13.2.5.6 Leach Variability Test Work

A leach test was performed on each of the variability composites based on the results from Marc master composite testing. The test conditions included: two-hour pre-oxidation, 500 ppm NaCN in solution, pH 11, primary grind P_{80} of 25 µm, 30 g/L carbon and 250 g/t $Pb(NO_3)_2$. The results are presented in Table 13-15.

Table 13-15: Leach Results for BL0184 Variability Composites

Test No.	Composite ID	Head Grade		Recovery		Reagent Consumption	
		Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	NaCN (kg/t)	Lime (kg/t)
11	JW10	11.65	20	91.5	85.3	1.7	1.7
12	JW11	19.95	128	94.2	95.6	1.9	1.5
13	JW12	7.37	12	95.4	92.7	1.5	1.8
14	JW13	9.84	3	93.7	74.5	2.9	3.1
15	JW14	7.51	26	79.5	84.7	3.0	2.0
16	JW15	9.88	16	94.8	92.9	1.7	1.7
17	JW16	8.22	31	93.5	91.6	1.6	1.7
18	JW17	8.17	27	91.4	92.3	1.7	1.7
19	JW18	3.51	12	89.9	90.0	1.8	2.5
20	AV10	7.2	22.2	85.8	86.7	1.9	2.0
21	AV11	5.6	27.8	75.8	74.1	3.1	1.3
22	AV12	19.2	16.9	93.7	87.8	2.5	1.7
23	AV13	7.8	15.6	82.2	82.6	1.5	2.3
24	AV14	10.8	13.7	83.8	81.8	2.0	1.7
25	AV15	8.0	11.9	83.1	45.9	2.4	2.0
27	Z141-1	6.9	1.1	87.1	92.8	1.5	1.1
28	Z141-2	3.7	0.5	98.0	93.8	1.9	1.7
29	Z141-3	5.1	4.0	94.9	84.3	1.7	1.8
30	Z141-4	6.3	16.6	83.9	84.1	2.4	1.9

Source: Base Met (2017b)

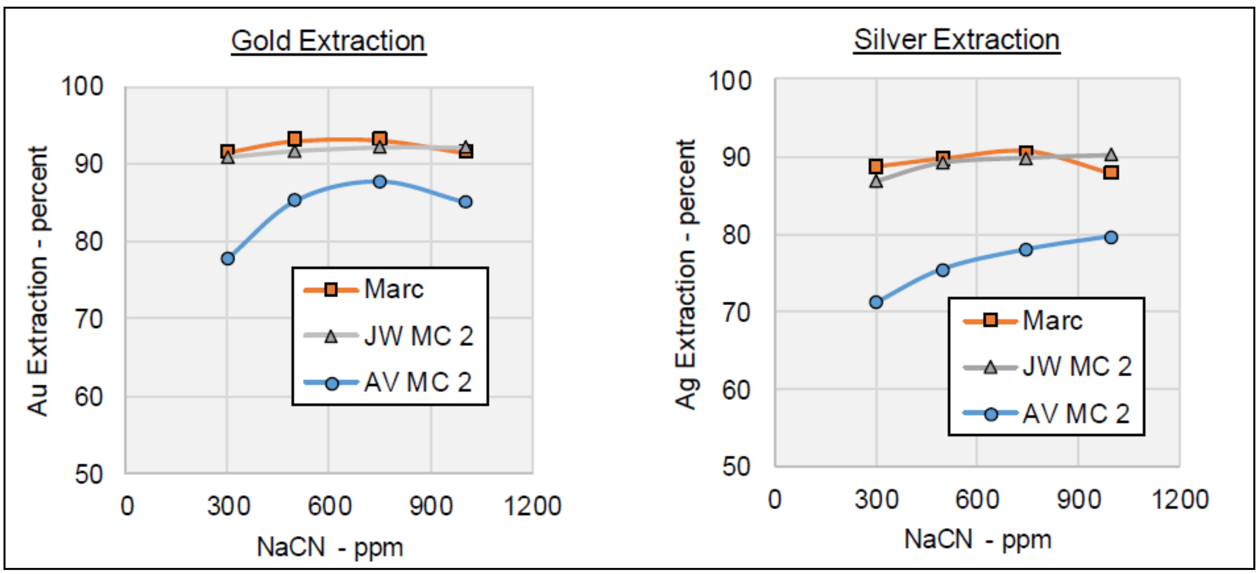
13.2.5.7 Leach Optimization Test Work

Additional leach test work was carried out on Marc, AV, and JW master composites in an attempt to maintain gold and silver recoveries while reducing operating costs in the CIL and cyanide destruction circuits. Cyanide dosage and carbon loading were varied to establish a balance between recovery and cost.

Cyanide dosages were varied from 300 ppm to 1,000 ppm in solution to determine the effect on gold and silver extraction. For each test, free cyanide levels were measured at the noted time intervals and cyanide was added as required to maintain these levels. All other variables were maintained constant.

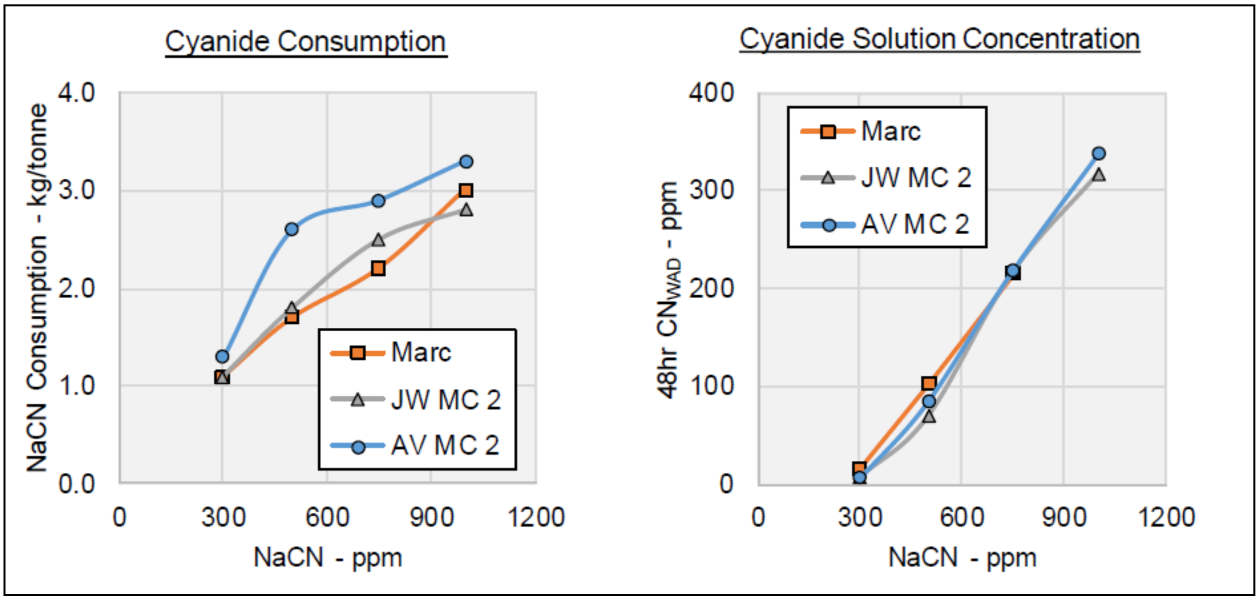
The effect of cyanide dosage on gold and silver extraction rate is presented in Figure 13-12. The resulting cyanide consumption and effluent levels are summarized in Figure 13-13. The trade-off between gold and silver extraction and cost of the process will be dominated by extraction of gold at current metal and reagent pricing. Current data would suggest operating at 500 ppm cyanide for the Marc zone and 750 ppm for the other zones.

Figure 13-12: The Effect of Cyanide Concentration on Gold and Silver Extraction



Source: Base Met (2017b)

Figure 13-13: The Effect of Cyanide Concentration on Cyanide Consumption and Effluent Level

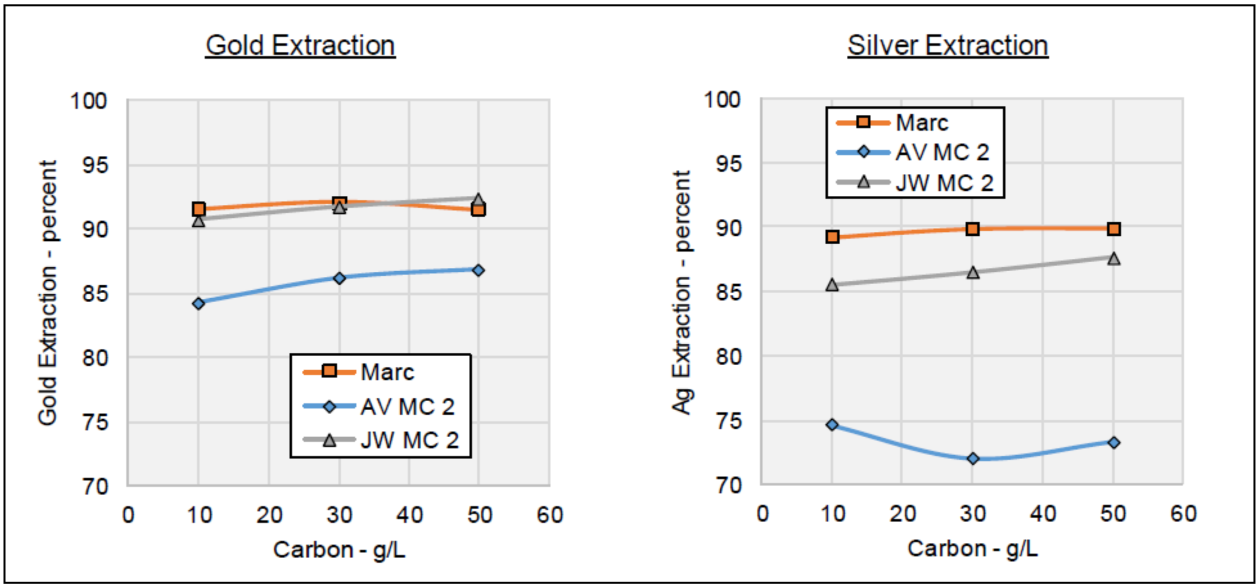


Source: Base Met (2017b)

A series of tests were performed on each composite to determine the impact of carbon loading. Three levels were tested on each composite; 10, 30, and 50 g/L. The effect on gold and silver extraction is shown in Figure 13-14. The resulting cyanide consumption and effluent levels are summarized in Figure 13-15. The carbon loading rate had a measurable effect on gold and silver extraction rates, with higher loading values resulting in the best extraction rates.

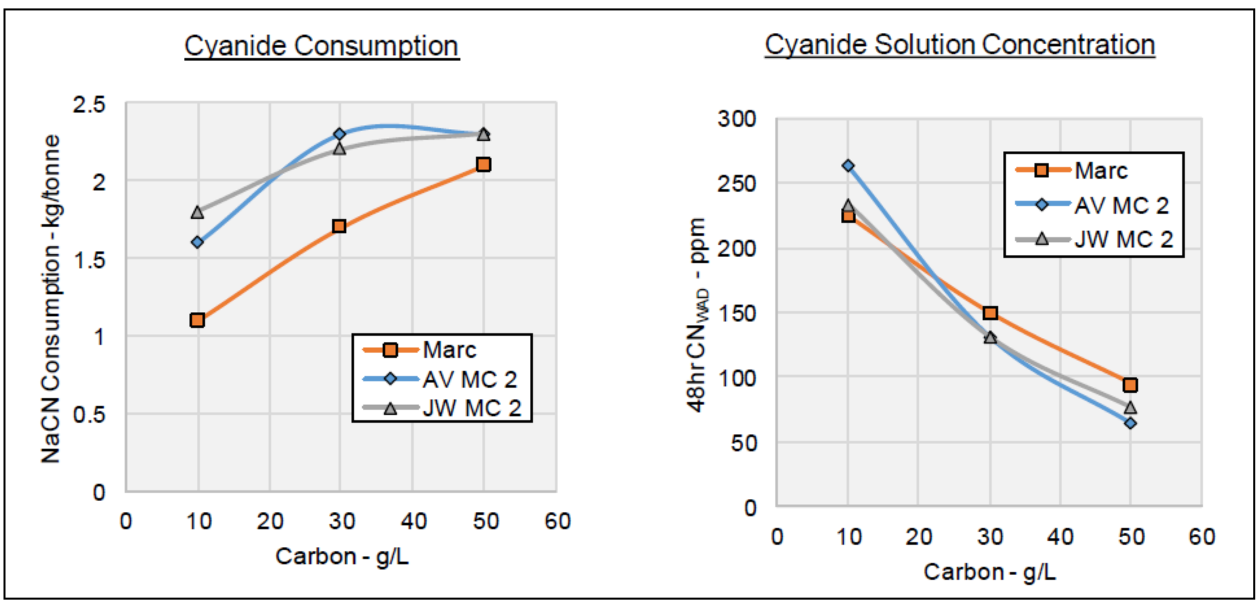
Carbon loading also affected the cyanide consumption and the final effluent CN_{WAD} values. Cyanide consumption was reduced as the carbon loading was reduced. Conversely, the final CN_{WAD} values were inversely proportional to carbon loading. The minor increase in metallurgical performance and the reduction of CN_{WAD} in the effluent would drive selection of the carbon loading to the higher levels of 30 or 50 g/L.

Figure 13-14: The Effect of Carbon Loading on Gold and Silver Extraction



Source: Base Met (2017b)

Figure 13-15: The Effect of Carbon Loading on Cyanide Consumption and Effluent Level



Source: Base Met (2017b)

13.2.5.8 Cyanide Destruction Testing

The refined test conditions from Section 13.2.5.7 were then performed on 9 kg of Marc Master composite, with the tailing going directly to cyanide destruction. The optimized conditions included:

two-hour pre-oxidation, 500 ppm NaCN in solution, pH 11, primary grind P_{80} of 25 µm, 50 g/l carbon and 250 g/t $Pb(NO_3)_2$.

Cyanide destruction testing ran for seven days using the SO_2/air process. The feed had a measured CN_{WAD} level of 80 ppm and copper and iron levels of 41 and 16 ppm, respectively. The new conditions represented a large drop in CN_{WAD} and iron. Previous leach conditions resulted in CN_{WAD} levels over 269 ppm and iron levels of 200 ppm. The results are summarized in Table 13-16.

Reagent requirements of 80 ppm copper and 10:1 SO_2 to CN_{WAD} were found to achieve the lowest CN_{WAD} and total cyanide (CN_{TOT}) levels. Optimization work is recommended to reduce these costs further.

Table 13-16: Cyanide Destruction Test Results on Marc Master Composite

Test	pH	Retention Time (mins)	Reagents Used		Number of Displacements	Solution Concentration (ppm)			
			SO_2 (g/g CN_{WAD})	Cu (mg/L)		CN_{WAD}	Cu	Fe	Calc' CN_{TOT}
D1	9.60	89	4.0	0	3.0	59.5	39.5	16.2	104.8
D2	9.00	89	5.0	0	1.0	60.4	38.8	16.0	105.1
D3	8.75	88	5.0	75	1.0	13.4	17.3	0.1	13.7
D4	8.62	120	6.5	0	1.0	8.8	4.6	3.9	19.8
D5	8.68	120	6.4	65	1.0	8.2	5.3	0.1	8.3
D6	8.54	120	6.4	130	1.0	9.9	14.6	0.1	10.0
D7	8.88	120	5.0	10	1.0	8.0	-	-	-
D8	8.61	150	9.6	10	0.8	4.4	-	-	-
D9	9.37	88	5.0	10	3.1	7.5	7.5	12.3	41.8
D10	9.05	87	5.0	15	1.6	19.6	1.3	7.7	41.1
D11	9.26	180	5.0	40	1.5	12.1	3.2	5.0	26.0
D12	8.86	176	5.0	80	0.5	10.8	2.9	2.5	17.8
D13	8.42	176	7.5	80	0.5	10.7	0.9	1.4	14.7
D14	8.25	170	7.5	120	0.3	12.7	1.7	0.2	13.3
D15	8.23	171	10.0	80	3.2	2.3	1.1	0.2	2.7

Source: Base Met (2017b)

13.3 Process Selection

The results from the 2016-2017 Base Met Labs test program were used to develop the process design criteria for the Red Mountain project. Whole ore leach was selected as the preferred recovery method (see Section 13.2.1.9) and a P_{80} grind size of 25 µm was chosen to maximize precious metal recovery.

The levels of TOC observed in the majority of variability composites would not typically interfere with the leaching process; however, there were a few samples with higher levels of TOC: MV9 and MV10. If the carbon has high activity levels it may interfere with cyanide leaching and adsorption of gold and silver. This phenomenon is referred to as “preg-robbing” and the organic carbon sequesters gold and silver in the leach residue, reducing recovery.

Based on test work results, some variability samples contained highly active organic carbon, which drastically affected gold leach recovery, including a test with decreasing gold recovery over time. This indicates that organic carbon may be preg-robbing, reducing the amount of leached gold adsorbing onto activated carbon. The CIL process was selected to mitigate the risk of this preg-robbing material. Subsequent test work (see in Table 13-5) found improved recoveries with the addition of carbon during the leach.

13.4 Relevant Results for Process Design

The following process design criteria were used to size the process plant.

13.4.1 Comminution Design Criteria

Comminution test work was completed on 43 variability samples, representing the Marc, AV, and JW deposits. Ore hardness was highly variable with Bond ball mill work indices ranging from 14.8 to 27.3 kWh/t. The grinding circuit was designed to accommodate this variability, with mill sizing capable of handling the 75th percentile value of 21.7 kWh/t. A summary of the key comminution design criteria is presented in Table 13-17.

Table 13-17: Key Comminution Design Criteria

Description	Units	Value	Source
Bond Crushing Work Index	kWh/t	11.1	Average of three Bond crushing work index tests completed on master composites from Marc, AV, and JW (BL0084)
Bond Ball Mill Work Index –- Average	kWh/t	19.6	Average of 43 Bond ball mill work index tests completed on variability composites from Marc, AV, and JW (BL0084, BL0184)
Bond Ball Mill Work Index - 75th Percentile	kWh/t	21.7	75^{th} percentile of 43 Bond ball mill work index tests completed on variability composites from Marc, AV, and JW (BL0084, BL0184)
Bond Abrasion Index	g	0.274	Average of three Bond abrasion index tests completed on master composites from Marc, AV, and JW (BL0084)

Source: JDS (2017)

13.4.2 Leach Design Criteria

Extensive cyanide leach test work was carried out on variability and master composites from the Marc, AV, and JW deposits. As a result of the optimization testing in BL0184, the leach process will include a pre-oxidation stage prior to CIL, which will reduce cyanide consumption, as well as cyanide destruction costs. The optimized conditions included two-hour pre-oxidation, 500 - 750 ppm NaCN concentration, pH 11, a primary grind P_{80} of 25 µm, 30 - 50 g/L carbon and 250 g/t $Pb(NO_3)_2$. A summary of the key process design criteria for the leach area is presented in Table 13-18.

Table 13-18: Key Leach Circuit Design Criteria

Description	Units	Value	Source
Pre-Leach Thickener Loading	$t/hr/m^2$	0.21	Based on an average of 0.19 - 0.23 $t/hr/m^2$ (Tenova Delkor, 2017)
Pre-Oxidation	hr	2	Based on pre-oxidation testing with Marc master composite (BL0184 Tests 1-10)
Leach Feed F_{80}	µm	25	Based on grind size vs. recovery analysis on Marc and AV master composites (BL0084 Tests 78,80,117-122)
Leach Retention Time	hrs	48	Based on variability composite leach kinetic curves (BL0084)
NaCN Concentration	ppm	500-750	Marc performed best at 500 ppm, while AV and JW required increased NaCN concentrations of 750 ppm (BL0184 Tests 4,9,10,31-39,)
Operating pH	-	11.0	Based on master composite testing on Marc and AV (BL0084 Tests 43,50-52,57,64-66)
Lead Nitrate Addition	g/t	250	Based on master composite testing on Marc and AV (BL0084 Tests 43,53,54,57,67,68)
Carbon Concentration	g/L	30-50	Based on master composite testing on Marc, AV, and JW (BL0184 Tests 40-48)
Leach Circuit Recovery			
Au - Marc Zone	%	92.8	Optimized Marc master composite results at 500 ppm NaCN (BL0184 Test 10)
Ag - Marc Zone	%	90.1	Optimized Marc master composite results at 500 ppm NaCN (BL0184 Test 10)
Au - AV Zone	%	88.1	Optimized AV master composite #2 results at 750 ppm NaCN (BL0184 Test 33)
Ag - AV Zone	%	78.3	Optimized AV master composite #2 results at 750 ppm NaCN (BL0184 Test 33)
Au - JW Zone	%	92.1	Optimized JW master composite #2 results at 750 ppm NaCN (BL0184 Test 37)
Ag - JW Zone	%	90.3	Optimized JW master composite #2 results at 750 ppm NaCN (BL0184 Test 37)
Au - 141 Zone	%	89.9	Weighted average of 4 variability samples based on head grade (BL0184 Tests 27-30)
Ag - 141 Zone	%	84.9	Weighted average of 4 variability samples based on head grade (BL0184 Tests 27-30)
Lime Consumption	kg/t	1.5-1.9	Based on master composite testing: 1.5 kg/t for JW and 1.9 kg/t for Marc and AV (BL0184 Tests 9,33,37)
Cyanide Consumption	kg/t	1.2-1.8	Based on optimized cyanide concentrations of 500 ppm for Marc and 750 ppm for AV and JW (BL0184 Tests 4,9,10,31-39)

Source: JDS (2017)

13.4.3 Cyanide Destruction Design Criteria

The cyanide destruction test work conditions were optimized in BL0184 using Marc master composite sample at the optimized leach test conditions. A summary of the key process design criteria for cyanide destruction is presented in Table 13-19.

Table 13-19: Key Cyanide Destruction Circuit Design Criteria

Description	Units	Value	Source
Retention Time	min	90-180	Based on cyanide destruction test work using Marc, AV, and JW (BL0084, BL0184)
Operating pH	-	8.00	Based on Marc master composite cyanide destruction test work (BL0184 D15)
Air Flow Requirement	Nm^3/hr	540	Based on industry benchmark of 5 $Nm^3/hr/m^3$
SO_2 Consumption	g SO_2 / g CN_{WAD}	10	Based on Marc master composite cyanide destruction test work (BL0184 D15)
Copper Consumption	g/t	300	Based on Marc master composite cyanide destruction test work (BL0184 D15)

Source: JDS (2017)

13.5 Preliminary Recovery Estimate

The recoveries used for each zone were estimated based on optimized leach results from Marc Master Composite, AV Master Composite #2, and JW Master Composite #2. Cyanide concentration and carbon loading were used to find a balance between operating costs and metallurgical recovery. Marc was found to perform best at a NaCN concentration of 500 ppm, while AV and JW required a higher NaCN concentration of 750 ppm. Table 13-20 presents the preliminary recovery estimates used for economic projections. The 141 zone recoveries were calculated using a weighted average of test work results, while the overall recovery is a weighted average of recovery by zone and projected zone tonnage.

Table 13-20: Preliminary Recovery Projections

Recovery	Au (%)	Ag (%)
Marc Zone	92.8	90.1
AV Zone	88.1	78.3
JW Zone	92.1	90.3
141 Zone	89.9	84.9
Overall Recovery (Weighted average based on the projected mine plan tonnages)	**90.9**	**86.3**

Source: JDS (2017)

14 Mineral Resource Estimate

14.1 Introduction

The mineral resource model prepared by ACS utilized a total of 538 drill holes, 74 of which were drilled by IDM, 12 in 2014 and 62 in 2016. The resource estimation work was completed by Dr. Gilles Arseneau, P. Geo. (APEGBC) an appropriate independent "qualified person" within the meaning of NI 43-101. The effective date of the Mineral Resource statement is January 23, 2017.

This section describes the resource estimation methodology and summarizes the key assumptions considered by 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 Red Mountain Gold Project 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.

The database used to estimate the Red Mountain mineral resources was audited by ACS. ACS 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.

14.2 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; and
- Preparation of mineral resource statement.

14.3 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) re-logged 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 an additional 62 holes in 2016 to better define 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 Red Mountain Gold Project in 2014. Table 14-1 summarizes the drill holes used for each mineralized zone estimated.

Table 14-1: Drill Hole Used in Resource Estimate Update

Zone	Number of Holes	Metres
Marc	156	21,531
AV	45	14,060
JW	47	14,711
141	25	7,281
132	6	3035

Source: ACS (2017)

There are a total of 58,057 records in the assay database, of these 3,232 represent samples taken from the mineralized horizons. Table 14-2 summarizes the basic statistical data for all the assays in the database. Table 14-3 summarizes the gold assays contained within the mineralized zones and Table 14-4 summarizes the silver assays.

Table 14-2: Basic Statistical Information for all Assays in Database

Zone	All	All
Assays	Au (g/t)	Ag (g/t)
Valid cases	58057	58057
Mean	0.96	3.45
Variance	63	794
Std. Deviation	7.9	28.2
Variation Coefficient	8.3	8.2
Minimum	0	0
Maximum	1320	2152
1st percentile	0.007	0
5th percentile	0.01	0
10th percentile	0.02	0
25th percentile	0.03	0.1
Median	0.11	0.4
75th percentile	0.4	1.2
90th percentile	1.3	3.3
95th percentile	3.1	10.7
99th percentile	15.72	56.3

Source: ACS (2017)

Table 14-3: Basic Statistical Information of Gold Assays within the Mineralized Zones

Zone	Marc	AV	JW	141	132
Assays	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)
Valid cases	1963	571	253	350	95
Mean	12	11.15	8.28	3.73	2.50
Variance	573	3474.71	134.27	94.43	9.33
Std. Deviation	23.9	58.95	11.59	9.72	3.05
Variation Coefficient	2.0	5.29	1.40	2.60	1.22
Minimum	0	0.04	0.09	0.09	0.19
Maximum	502	1321	80.60	169.30	15.88
1st percentile	0.08	0.09	0.11	0.18	0.00
5th percentile	0.50	0.65	0.57	0.40	0.24
10th percentile	0.98	1.16	1.15	0.66	0.34
25th percentile	2.58	2.63	2.51	1.16	0.83
Median	5.32	4.38	4.59	1.97	1.53
75th percentile	12.40	7.92	8.99	3.30	2.51
90th percentile	26.59	16.34	16.01	7.64	6.28
95th percentile	43.90	25.68	33.13	12.17	9.48
99th percentile	104.44	118.42	60.72	22.58	12.30

Source: ACS (2017)

Table 14-4: Basic Statistical Information of Silver Assays within the Mineralized Zones

Zone	Marc	AV	JW	141	132
Assays	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)
Valid cases	1963	571	253	350	95
Mean	47.99	20.72	35.69	7.64	3.76
Variance	14530.05	1469.00	11224.25	361.30	162.61
Std. Deviation	120.54	38.33	105.94	19.01	12.75
Variation Coefficient	2.51	1.85	2.97	2.49	3.40
Minimum	0.00	0.00	0.00	0.05	0.05
Maximum	2152.00	504.20	889.00	203.30	73.20
1st percentile	0.00	0.05	0.00	0.10	0.00
5th percentile	0.90	0.05	0.05	0.15	0.05
10th percentile	2.37	0.80	0.05	0.40	0.08
25th percentile	7.30	4.30	1.10	0.90	0.10
Median	20.10	9.60	8.18	2.20	0.30
75th percentile	42.10	21.70	24.05	5.85	0.70
90th percentile	92.78	50.76	58.62	14.76	3.64
95th percentile	169.20	71.26	169.49	32.02	38.20
99th percentile	551.48	190.69	720.97	112.10	48.30

Source: ACS (2017)

14.4 Design of 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 sulphide 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:

1. Basic lithology, including major structural features, with appropriate textural modifiers.
2. The limits of pyrite, and more rarely pyrrhotite, stockwork. These limits are often, but not always coincident with a 1 g/t gold assay outline. Inside this outline, sulphide 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, sulphide occurs as disseminations and sparse micro-veinlets with an average pyrite content of 1.5%.
3. The shift from a pyrite-dominated stockwork to a pyrrhotite-dominated alteration halo is sharp and often corresponds to a 1 g/t gold outline, except in rare cases where pyrrhotite abundance, style and gold content mimics the pyrite stockwork.
4. 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.
5. 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.

14.5 Solid Modelling

New three-dimensional 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 three-dimensional 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 coincides 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 for the Marc, AV, and JW zones were derived from vertical sections in Gemcom software. The vertical section outlines were digitized as closed polylines that were snapped to the actual 3D 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.

Wireframes for the 141, 132, Marc Footwall, AV, and JW Lower zones were designed from sectional interpretation on 25 m sections. Each wireframe was assigned a unique rock code as outlined in Table 14-5.

Table 14-5: Rock Codes Assigned to Wireframes

Zone	Rock Code
Marc	101
AV	201
JW	301
141	401
132	132
Marc Footwall	102
Marc Hanging wall	103
Marc NK	104
AV Lower	202
JW Lower	302

Source: ACS (2017)

14.6 Bulk Density

The bulk density of the Red Mountain gold deposits has been tested by two sampling programs. During 1993 and 1994, LAC had 4,225 specific gravity (SG) determinations made on drill core that was submitted to the Eco-Tech lab in Stewart. In 2000, NAMC collected 58 samples that were subjected to bulk density analysis. Of the 4,283 samples, 1,290 are from sample intervals within the solids used for resource calculation. Average specific gravity values for different subsets of the entire data set are given in Table 14-6.

Table 14-6: 1993-1994 Bulk Density Sample Results

Zone	# Samples	Range of Values	Avg. SG	Pyrite %
All samples	4283	1.44 - 4.12	2.86	N/A
Within mineralized zones	1290	1.85 - 4.04	2.95	6.11
Marc zone	1058	2.03 - 4.04	2.95	6.43
AV zone	194	1.85 - 3.85	2.99	5.83
JW zone	38	2.67 - 3.12	2.90	1.91
Mineralized zones > 5.0 g/t Au	667	2.48 - 4.04	3.01	8.66
Mineralized zones < 5.0 g/t Au	623	1.85 - 3.58	2.89	3.43

Source (NAMC 2001)

In addition to the above data, Banks Island collected 170 density measurements from their drilling in 2013 and IDM collected 120 bulk density samples from the 2016 drill program for a total of 4,573 density readings for the Red Mountain Project. Results of the IDM density program are summarized in Table 14-7.

Table 14-7: 2016 Bulk Density Sample Results

Zone	# Samples	Range of Values	Avg. SG
All samples	120	2.41 – 4.39	2.90
Within mineralized zones	46	2.74 – 4.39	3.06
Marc zone	0	ND	ND
AV zone	26	2.76 – 4.23	3.03
JW zone	20	2.74 – 4.39	3.10
Waste zone	74	2.41 – 3.04	2.81

Source : ACS (2017)

14.7 Composite Statistics

14.7.1 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 14-8 summarizes the basic statistical data for uncapped gold composites used in the resource estimates and Table 14-9 shows the statistics of the silver composited data.

Table 14-8: Descriptive Statistics of 1.5 m Gold Composites

Zone	Au (g/t) MARC	Au (g/t) AV	Au (g/t) JW	Au (g/t) 132	Au (g/t) 141
Valid cases	1443	404	178	83	273
Mean	11.6	10.8	8.0	2.3	3.6
Variance	364	2241	89.6	4.8	34.0
Std. Deviation	19.1	47.4	9.5	2.2	5.8
Variation Coefficient	1.6	4.4	1.2	1.0	1.6
Minimum	0	0.05	0.3	0.0	0.1
Maximum	282	909	61.0	11.1	60.7
1st percentile	0.1	0.1	0.6	0.0	0.2
5th percentile	0.7	0.9	1.0	0.3	0.6
10th percentile	1.4	1.7	1.2	0.6	0.9
25th percentile	3.0	3.1	3.0	1.2	1.3
Median	6.2	4.8	4.7	1.7	2.1
75th percentile	1.7	7.9	9.3	2.2	3.2
90th percentile	25.0	15.6	14.5	5.5	8.4
95th percentile	39.4	23.4	27.4	7.8	11.5
99th percentile	101.9	110.3	58.1	10.1	31.0

Source: ACS (2017)

Table 14-9: Descriptive Statitics of 1.5 m Silver Composites

Zone	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)
	MARC	AV	JW	132	141
Valid cases	1443	404	178	83	273
Mean	46.5	20.5	31.4	2.9	7.6
Variance	9856	1026.8	7077.0	114.5	310.0
Std. Deviation	99.3	32.0	84.1	10.7	17.6
Variation Coefficient	2.1	1.6	2.7	3.6	2.3
Minimum	0	0.0	0.0	0.0	0.1
Maximum	1102	337.0	578.4	72.3	177.3
1st percentile	0.1	0.1	0.0	0.0	0.1
5th percentile	1.3	0.1	0.0	0.1	0.3
10th percentile	2.8	1.3	0.1	0.1	0.4
25th percentile	8.7	4.9	1.1	0.1	1.0
Median	21.5	11.4	9.8	0.3	2.4
75th percentile	44.2	23.3	22.0	0.9	7.0
90th percentile	89.5	46.2	51.3	2.7	16.4
95th percentile	157.9	70.7	134.7	22.0	26.2
99th percentile	531.8	178.1	521.0	63.5	108.9

Source: ACS (2017)

14.7.2 Top Cut Applied to Composites

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 composites was warranted. ACS decided to cap gold composites to 55 g/t Au. This 55 g/t Au top cut was used in the interpolation runs for all of the mineralized zones.

Silver values were top cut to 220 g/t, which is slightly lower than the 97.5% of all combined Ag composite values. A 220 g/t Ag top cut was used in the interpolation runs for all of the mineralized zones.

14.8 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 structure (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 Marc zone. The Marc zone is the most densely drilled and provides the greatest opportunity for determining the short-range character of the correlogram. Correlogram derived form the Mac and AV zones were used to interpolate grades into the Marc, AV and JW zone.

Correlograms for the Marc Footwall, 132, 141 AV Lower, and JW Lower zones were somewhat inconclusive, and, for this reason, ACS decided to use the correlograms derived from the main Marc, AV, and JW data to estimate these zones. ACS recommends that the data for these zones be re-examined after additional drilling is carried out to determine if more robust Variography can be achieved with additional information. Table 14-10 summarizes the correlogram parameters used to estimate gold and silver in the block model.

Table 14-10: Correlogram Parameters Used for Grade Estimation

Metal	Model Type	Nugget (C_0)	C_1 & C_2	Rotation			Range		
				(Z)	(Y)	(Z)	Rot X	Rot Y	Rot Z
Au	Exponential	0.1	0.589	-37	8	19	8	8	19
			0.313	-37	8	19	9	170	82
Ag	Exponential	0.1	0.90	21	79	8	21	79	8
			NA				NA	NA	NA

Source: ACS (2017)

14.9 Block Model

A 3D block model was created using Geovia GEMs Version 7.3 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 a robust estimate of the resource.

The parameters for the block model are listed in Table 14-11. 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 above. All waste blocks were assigned a default rock code of 99.

Table 14-11: Model Parameters for the Red Mountain Block Model

Coordinates			Origin Coordinates	Block Size (m)	Number of Blocks
Axis Direction	Axis				
Easting	X	Column	4500	4	200
Northing	Y	Row	1000	4	200
Elevation	Z	Level	2000	4	135

Source: ACS (2016)

Gold grades were interpolated within the individual zones using ordinary kriging and multiple passes as outlined in Table 14-12. Grades were only interpolated into blocks if the blocks had not been interpolated by a previous pass.

Table 14-12: Interpolation Parameters Used for Grade Interpolation

Zone	Pass	Rotation (°)			Search Ellipse Size (m)			No of composites		Max no per hole
		Z	Y	Z	X	Y	Z	Min	Max	
Marc	1	0	-75	0	30	30	10	5	15	3
Marc	2	0	-75	0	60	60	15	2	15	1
Marc	3	0	-75	0	20	20	15	2	15	none
AV	1	0	-60	0	30	30	10	5	15	3
AV	2	0	-60	0	60	60	15	2	15	1
AV	3	0	-60	0	20	20	10	2	15	none
JW	1	0	-45	0	30	30	10	5	15	3
JW	2	0	-45	0	60	60	10	2	15	1
JW	3	0	-45	0	20	20	15	2	15	none
141	1	0	-45	0	30	30	10	5	15	3
141	2	0	-45	0	60	60	20	2	15	1
141	3	0	-45	0	20	20	15	2	15	none
Marc FW	1	0	-22	0	30	30	10	5	15	3
Marc FW	2	0	-22	0	60	60	15	2	15	1
Marc FW	3	0	-22	0	20	20	15	2	15	none
Marc Outlier	1	0		0	30	30	10	5	15	3
Marc Outlier	2	0		0	60	60	15	2	15	1
Marc Outlier	3	0		0	20	20	15	2	15	none
Marc NK	1	0	0	0	30	30	10	5	15	3
Marc NK	2	0	0	0	60	60	15	2	15	1
Marc NK	3	0	0	0	20	20	15	2	15	none
AV Lower	1	0	-25	0	30	30	10	5	15	3
AV Lower	2	0	-25	0	60	60	20	2	15	1
AV Lower	3	0	-25	0	20	20	15	2	15	none
JW Lower	1	0	-22	0	30	30	10	5	15	3
JW Lower	2	0	-22	0	60	60	35	2	15	1
JW Lower	3	0	-22	0	20	20	15	2	15	none
132	1	0	-22	0	30	30	10	5	15	3
132	2	0	-22	0	60	60	20	2	15	1
132	3	0	-22	0	20	20	15	2	15	none

Source: ACS (2016)

Bulk density and iron grades were 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 14-13.

Table 14-13: Block Model Default Densities by Rock Codes

Rock Code	Average Density (t/m^3)
99	2.82
101	2.96
102	3.00
103	2.90
104	No data
201	2.99
202	No data
301	2.90
302	2.96
401	2.89

Source: ACS (2016)

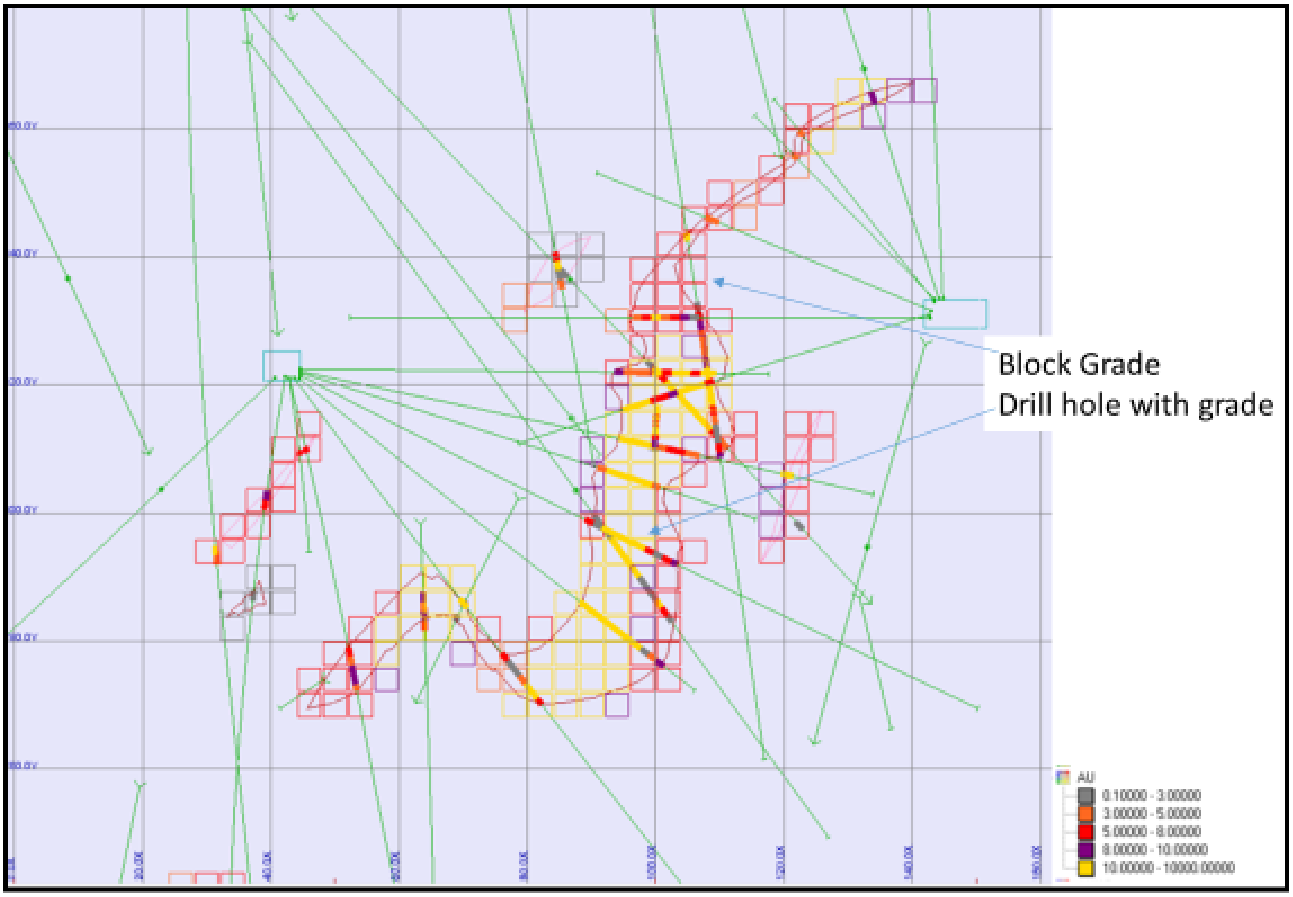
14.10 Model Validation

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.

14.10.1 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 14-1 and Figure 14-2).

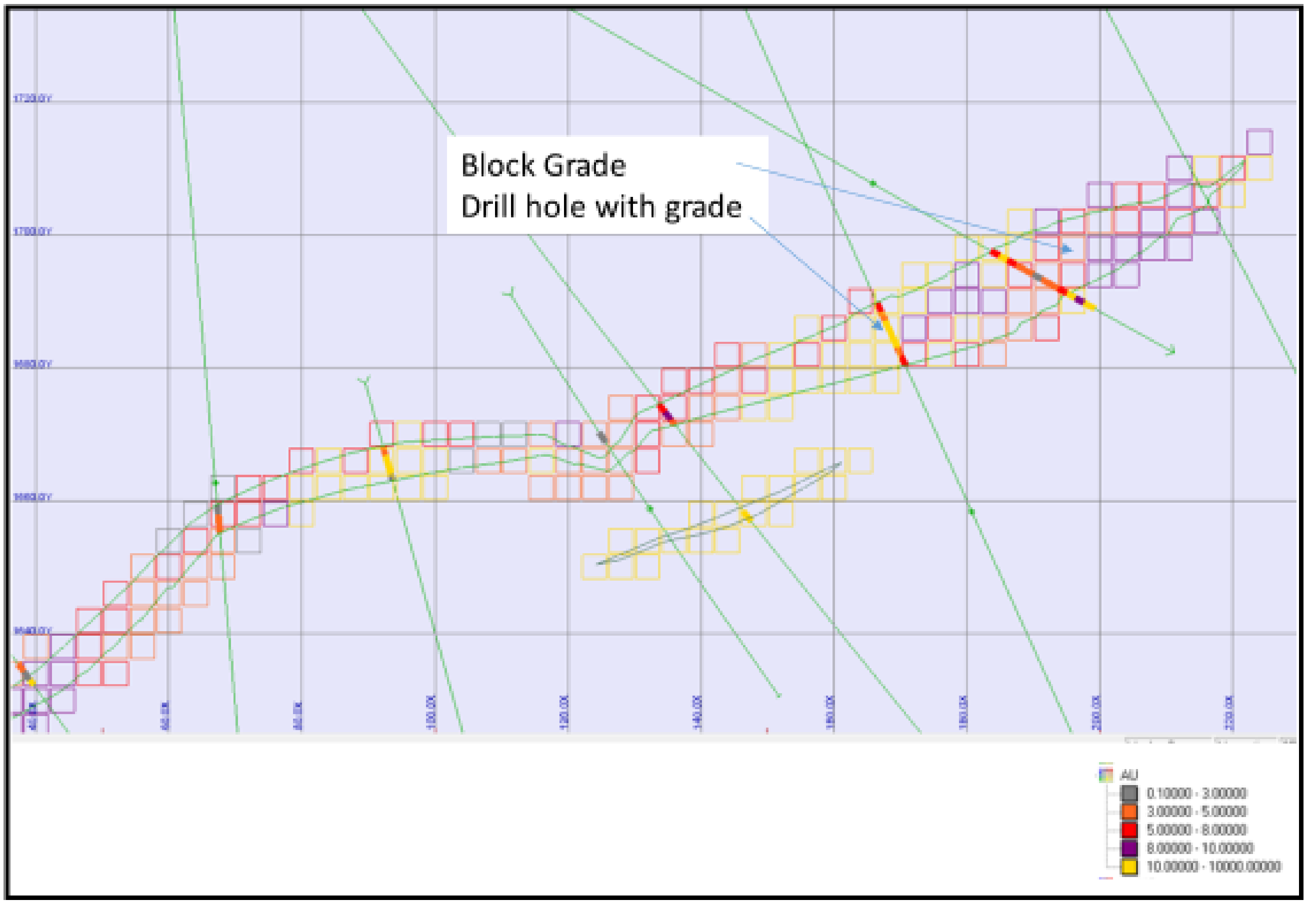
Figure 14-1: Section 1250N Showing Block Drill Hole Composites and Estimated Gold Grades



Source: ACS (2016)

Note: Grid lines are 20 m apart and blocks are 4 m by 4 m

Figure 14-2: Section 1525N Showing Drill hole Composite and Estimated Gold Grades



Source: ACS (2016)

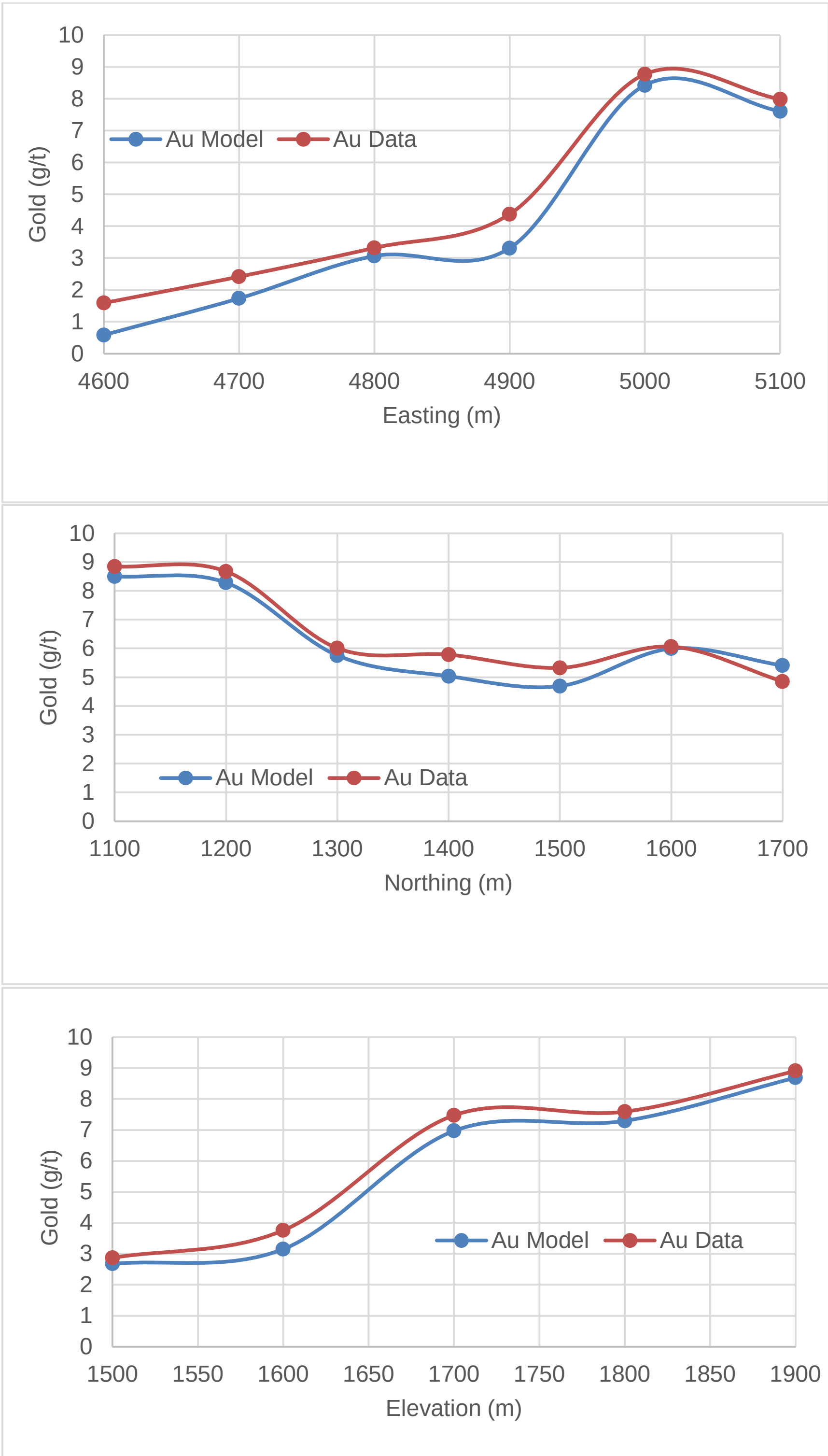
Note: Grid lines are 50 m apart and blocks are 4 m by 4 m

14.10.2 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 14-3 shows the swath plot for gold. On average, the estimated data agree well with the composited data with the estimated values being slightly more smoothed than the composite data.

Figure 14-3: Swath Plot for Gold Values



14.11 Resource Classification

Block model quantities and grade estimates for the Red Mountain Gold Project 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 "qualified person" 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 to 60 m.

ACS considers that blocks in the Marc, AV, and JW zones estimated during pass one and from at least four drill holes could be assigned to the Measured category. Blocks interpolated during pass one with at least three drill holes in all zones could be assigned to the Indicated category. Blocks that had not been interpolated during pass one were assigned to the Inferred category within the meaning of the CIM Definition Standards.

14.12 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 longhole 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 gold representing an approximate mining cost of C$160 and a minimum mining width of 2 m. The reader is cautioned that there are no mineral reserves at the Red Mountain Gold Project.

ACS is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political issues that may adversely affect the Mineral Resources presented in this Report.

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 Red Mountain Gold Project are summarized in Table 14-14.

Table 14-14: Red Mountain Mineral Resource Statement at a 3 g/t Gold Cut-off Effective January 23, 2017

Zone	Tonnage (tonnes)	In-situ Gold Grade (g/t)	In-situ Silver Grade (g/t)	Contained Gold (troy ounces)	Contained Silver (troy ounces)
Marc Zone					
Measured	682,000	10.62	38.3	232,800	840,500
Indicated	32,300	9.69	32.6	10,100	33,800
Inferred	4,500	10.43	43,4	1,500	6,200
AV Zone					
Measured	519,400	7.73	20.0	129,100	334,500
Indicated	236,300	9.07	19.2	60,700	146,300
Inferred	43,300	8.13	15.4	20,400	21,400
JW Zone					
Measured	44,600	10.11	13.2	14,500	18,900
Indicated	314,200	8.54	18.0	86,300	181,600
Inferred	111,700	6.78	7.4	24,400	26,500
141 Zone					
Indicated	188,600	4.91	11.1	29,700	67,300
Inferred	15,100	4.67	4.7	2,300	2,300
Marc Footwall					
Indicated	18,100	6.15	12.1	3,600	7,000
Inferred	12,600	5.12	6.4	2,100	2,600
Marc Outlier Zone					
Indicated	4,200	3.43	16.8	500	2,300
Inferred	7,300	6.54	27.4	1,500	6,400
Marc NK Zone					
Indicated	10,700	5.58	7.6	1,900	2,600
Inferred	7,300	5.98	9.0	1,400	2,100
JW Lower Zone					
Indicated	24,300	8.15	26.6	6,400	20,800
Inferred	2,000	13.94	9.3	900	600
AV Lower Zone					
Inferred	42,500	5.55	6.6	7,600	8,300
132 Zone					
Inferred	78,700	4.73	11.5	12,000	29,100
Total Measured & Indicated	**2,074,700**	**8.75**	**24.8**	**583,700**	**1,655,700**
Total Inferred	**324,700**	**6.21**	**10.1**	**64,800**	**105,500**

Source: ACS (2017)

Mineral Resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve 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, socio-economic and other factors.

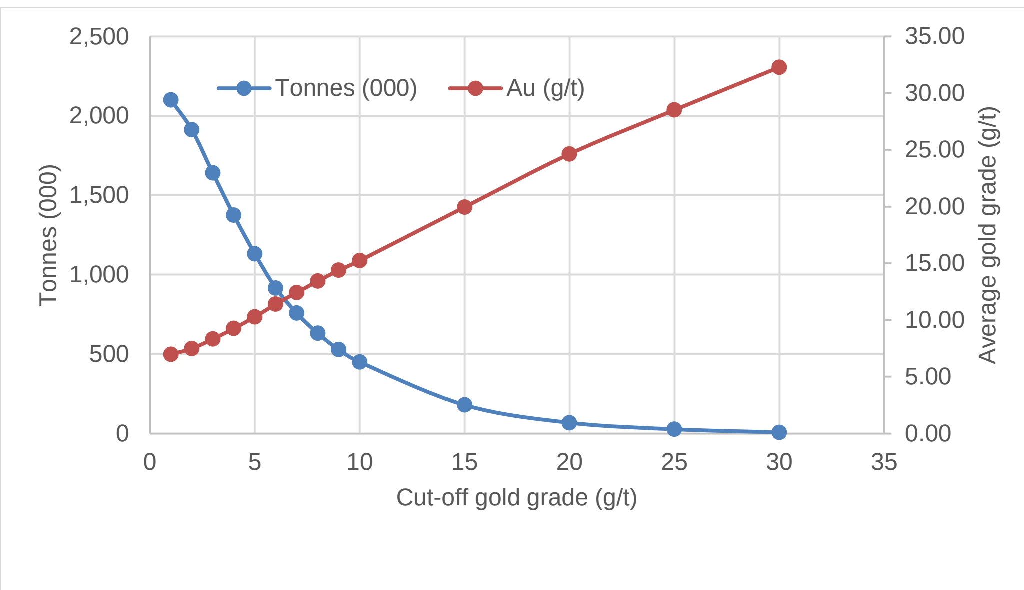
Mineral Reserves can only be estimated based on the results of an economic evaluation as part of a PFS or FS. As such, no Mineral Reserves have been estimated by ACS. There is no certainty that all or any part of the Mineral Resources will be converted into a Mineral Reserve.

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.

14.13 Grade Sensitivity Analysis

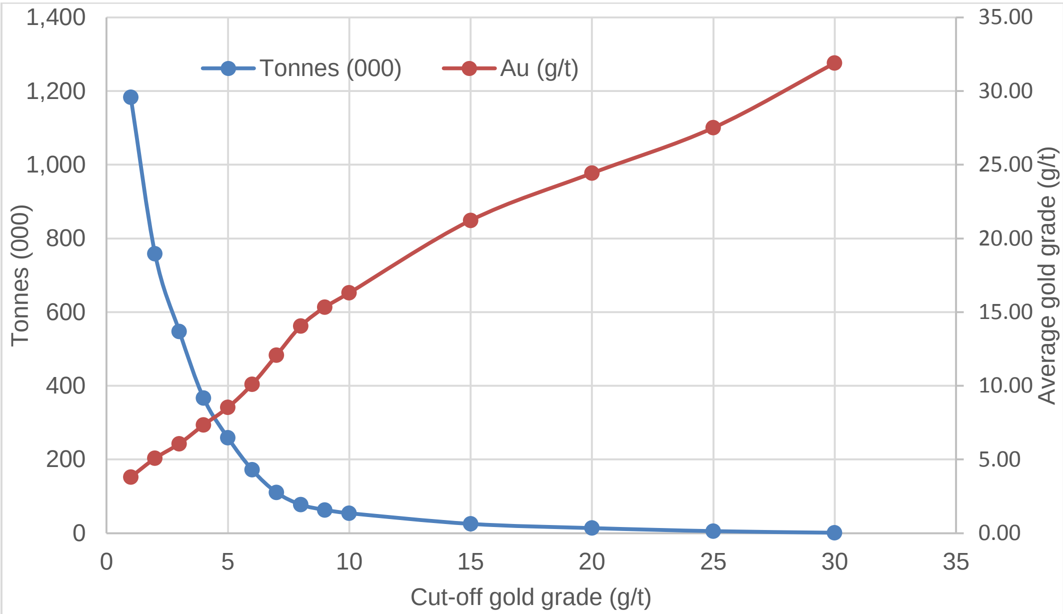
The Mineral Resources at the Red Mountain are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model quantities and grade estimates of the Measured and Indicated Resource are presented in Figure 14-4 and the Inferred Resources are presented in Figure 14-5. 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.

Figure 14-4: Grade Tonnage Curve for Measured and Indicated Mineral Resource at Red Mountain



Source: ACS (2017)

Figure 14-5: Grade Tonnage Curve for Inferred Mineral Resource at Red Mountain



Source: ACS (2017)

14.14 Previous Mineral Resource Estimates

Mineral Resources have been estimated for the Red Mountain Gold Project in the past. IDM reported Mineral Resources in a Technical Report dated May 6, 2016. The Mineral Resources were included in a PEA report published by IDM in July of 2016. The 2016 Mineral Resources are summarized in Table 14-15.

Table 14-15: Previous Mineral Resource Statement for Red Mountain

Class	Tonnes	Gold Grade (g/t)	Silver Grade (g/t)	Contained Gold (oz)	Contained Silver (oz)
Measured and Indicated	1,641,900	8.36	26	441,500	1,379,800
Inferred	548,100	6.1	9	107,500	153,700

Source: ACS (2017)

The previous mineral resources are presented here only as a means of comparing the previous estimate with the current estimate present in Table 14-15 above. As can be seen, the tonnage of the mineral resource has increased in all categories as a result of the 2016 drilling and the grade of the Mineral Resource has remained relatively consistent.

15 Mineral Reserve Estimates

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource demonstrated by at least a PFS. This FS includes adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral Reserves are those parts of Mineral Resources, which, after the application of all mining factors, result in an estimated tonnage and grade that is the basis of an economically viable project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the economic mineralized rock and delivered to the treatment plant or equivalent facility. The term “Mineral Reserve” need not necessarily signify that extraction facilities are in place or operative, or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Mineral Reserves are subdivided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

The reserve classifications used in this report conform to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of NI 43-101 Resource and Reserve definitions, and the Companion Policy 43-101CP. These are listed below.

A “Proven Mineral Reserve” is the economically mineable part of a Measured Mineral Resource demonstrated, at least, by a 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 QP has the highest degree of confidence in the estimate with the consequent expectation 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, at least, by a PFS. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.1 Resource Model Sub-Blocking

JDS used the resource block model, discussed in Section 14 of this report, for mine planning purposes. The block model was sub blocked down to 0.5 m x 0.5 m x 0.5 m to gain resolution of mineralized material blocks near the waste/mineralized material contact and to better estimate planned mine dilution.

Table 15-1 summarizes the change in block model resource at a 3.0 g/t Au cut-off before and after the sub blocking exercise.

Table 15-1: Mineral Resource Before and After Sub-Blocking

		JDS Sub Blocked Model			ACS Regularized Model		
Zone	**Class**	**Au (g/t)**	**Ag (g/t)**	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Tonnes**
Marc	Meas	10.62	38.33	682,000	10.62	38.3	682,000
Marc	Ind	9.69	32.59	32,300	9.69	32.6	32,300
Marc	Inf	10.42	43.09	4,500	10.43	43.4	4,500
AV	Meas	7.73	20.03	519,400	7.73	20.0	519,400
AV	Ind	9.06	19.25	236,100	9.07	19.3	236,300
AV	Inf	8.12	15.39	43,400	8.13	15.4	43,300
JW	Meas	10.11	13.21	44,600	10.11	13.2	44,600
JW	Ind	8.54	17.98	314,200	8.54	18.0	314,500
JW	Inf	6.78	7.39	111,700	6.79	7.5	111,900
141	Ind	4.91	11.10	188,600	4.90	11.10	188,600
141	Inf	4.67	4.69	15,100	4.67	4.69	15,200
Marc FW	Ind	6.15	12.05	18,100	6.15	12.06	18,100
Marc FW	Inf	5.12	6.38	12,600	5.11	6.36	12,600
Marc NK	Ind	5.58	7.57	10,700	5.57	7.57	10,700
Marc NK	Inf	5.98	9.05	7,300	5.98	9.04	7,300
JW Lower	Ind	8.15	26.58	24,300	8.15	26.60	24,300
JW Lower	Inf	13.94	9.26	2,000	13.89	9.26	2,000
AV Lower	Inf	5.55	6.05	42,500	5.56	6.08	42,800
132	Inf	4.73	11.51	78,700	4.73	11.51	78,600
Grand Total		**8.41**	**22.83**	**2,388,100**	**8.41**	**22.83**	**2,389,000**
Total Difference (%)		**0.03**	**0.04**	**0.04**			

Source: JDS (2017)

Note: Comparison shows tonnes and grade for the regularized and sub blocked model using a 3.0 gram per tonne gold cut-off grade.

15.2 Cut-Off Grade Criteria

Mining Reserve values were calculated from block model tonnes and grades, and defined by a gold cut-off grade (COG). The COG is based on expected operating cost, gold recovery, mining dilution, and commodity price assumptions. Mineable longhole stopes and cut and fill drifts were defined based on COG values greater than 3.55 g/t Au and 4.10 g/t Au respectively. The parameters used for the calculation were based on the data shown in Table 15-2.

Table 15-2: Cut-Off Grade Criteria

Item	Unit	Value
Gold Price	US$/oz	1,200
Payable Metal	% Au	99.0
Refining / Transport	$/oz	6.00
Royalty	$/oz	53.10
Total Operating Cost Longhole	$/t milled	128
Total Operating Cost Cut and Fill	$/t milled	148
Average Process Recovery	%	89.3

Note: Assumptions stated in this table were used to establish mining cut-off grade only.

Source: JDS (2017)

15.3 Dilution

Two types of dilution were applied to the stope and development designs:

- External dilution – additional material that is mined outside of the mineralized vein; this material is considered to carry a grade which is included in the reserves; and
- Backfill dilution – ROM waste, and/or CRF expected to fall into the stope being mined from adjacent stopes and/or inadvertently scraped off the stope floors during mucking. This material is considered zero grade dilution.

Additional sources of dilution include Inferred Resource dilution. Any Inferred Resource class material within the mining reserve stope and development shapes has been treated as waste and has been assigned zero metal grades. Inferred dilution comprises approximately 2,923 t or 0.15% of the reserve respectively.

The total external, backfill, and inferred dilution is approximately 12% of the total mining reserve.

15.3.1 External Dilution

15.3.1.1 Longhole External Dilution

External dilution for longhole stopes will come from the hangingwall and footwall contacts. Dilution was estimated based on each stope’s individual dimensions. External dilution estimates have been defined by geotechnical rock mass domains, stope dimensions, and mining method. The dilution estimates are based on the Equivalent Linear Overbreak Slough (ELOS) method.

Geotechnical rock mass domains have been defined by SRK for the Marc, AV, and JW zones and are based on core photo review and geotechnical logging. The rock mass domains are:

- Good (Green) high rock strength, low jointing RMR_{90} of 60-65;
- Moderate (Yellow) high rock strength, increased jointing RMR_{90} of 55-60;
- Moderate-Poor (Pink) same as Moderate but in faulted zones; and
- Poor (Red) poor rock strength increased jointing RMR_{90} of 25-55.

The geotechnical rock mass domains are discussed in more detail in Section 16.2.

External dilution estimates by longhole stope type are shown in Tables 15-3 to 15-5. The FS mine design does not include longhole stopes in the JW zone, so they have been excluded from the tables. The 141 zone has limited geotechnical data collected so it was assumed to have same dilution dimensions as the 'AV Yellow' geotechnical domain.

Table 15-3: Transverse (Primary) Longhole Stope Dilution – 25 m Sublevel Spacing

Zone	Geotechnical Domain	Stope Height (m)	Stope Length (m)	Stope Width (m)	Hangingwall Dilution (m)	Footwall Dilution (m)
Marc	Green	29	20	15	0.35	0.20
Marc	Yellow	29	20	15	0.45	0.35
AV	Green	29	20	15	0.45	0.35
AV	Yellow	29	20	15	0.80	0.45
141	AV Yellow	29	20	15	0.80	0.45

Source: JDS (2017)

Table 15-4: Transverse (Secondary) Longhole Stope Dilution – 25 m Sublevel Spacing

Zone	Geotechnical Domain	Stope Height (m)	Stope Length (m)	Stope Width (m)	Hangingwall Dilution (m)	Footwall Dilution (m)
Marc	Green	29	20	10	0.45	0.30
Marc	Yellow	29	20	10	0.60	0.35
AV	Green	29	20	10	0.55	0.35
AV	Yellow	29	20	10	0.90	0.50
141	AV Yellow	29	20	10	0.90	0.50

Source: JDS (2017)

Table 15-5: Longitudinal Longhole Stope Dilution – 25 m Sublevel Spacing

Zone	Geotechnical Domain	Stope Height (m)	Stope Length (m)	Stope Width (m)	Hangingwall Dilution (m)	Footwall Dilution (m)
Marc	Green	29	25	10	0.50	0.35
Marc	Yellow	29	20	10	0.70	0.45
AV	Green	29	25	10	0.50	0.35
AV	Yellow	29	20	10	0.70	0.45
141	AV Yellow	29	20	10	0.70	0.45

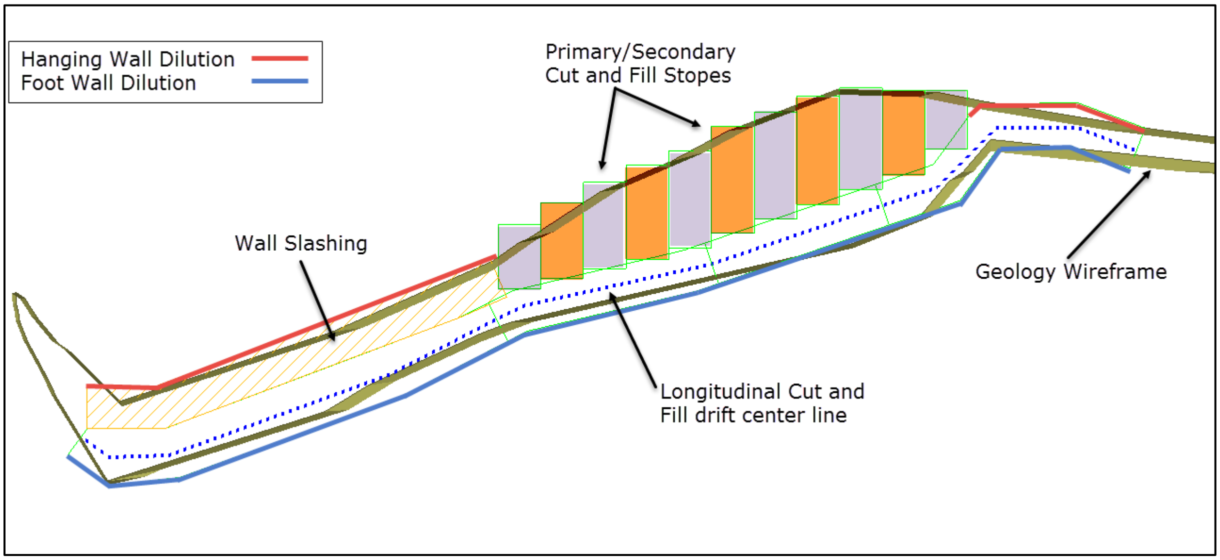
Source: JDS (2017)

Average grades for longhole external dilution were taken from the block model and included in the average grades for the stope shape. The average grade of the external dilution is 3.39 g/t Au.

15.3.1.2 Cut and Fill External Dilution

External dilution for cut and fill comes from the side walls of the stope that run along the hangingwall and/or footwall contact. In the wider zones where a primary/secondary extraction sequence will be used, no external dilution was added. Figure 15-1 shows an example of where external dilution was added. External dilution for cut and fill stopes was estimated to be 0.25 m along each wall.

Figure 15-1: Cut and Fill Dilution Example – AV Zone 1676 Level



Source: JDS (2017)

The average dilution grade for cut and fill stopes was calculated by creating a section of each cut and fill level and expanding the section by the dilution width of 0.25 m. This was used to calculate the average grade for each level. The average grade for the level was applied to the external dilution tonnes for each stope on that particular level.

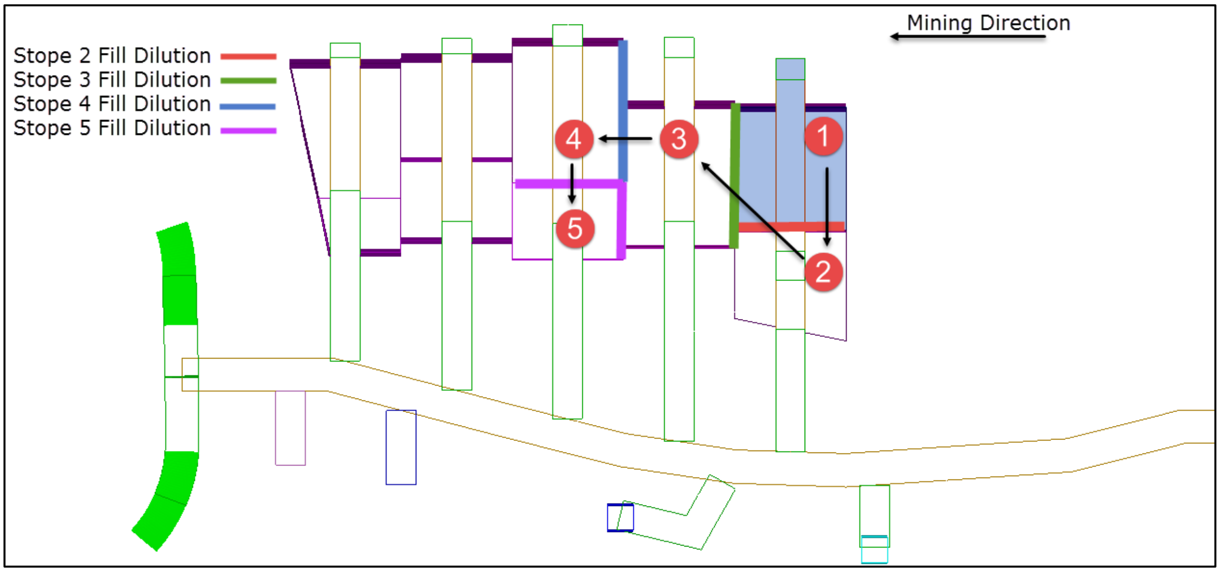
15.3.1.3 Other External Dilution

There are two near/at surface stopes that are being mined with uppers drilled from a lower drift. These stopes have an assumed external dilution of 30% at zero grade due to proximity to the surface.

15.3.2 Backfill Dilution

Backfill dilution will come from adjacent mined and filled stopes. Longhole stopes are being extracted in a way so that pillars are not created. As a result, longhole stopes will generally only have one wall next to fill. In zones that are wider than 20 m, two longhole stopes will be required transversely. The second stope in sequence will have an additional wall on the hangingwall side exposed to fill. Figure 15-2 shows examples of the longhole stope extraction sequence and the resulting wall fill dilution.

Figure 15-2: Backfill Dilution Example – AV Zone Partial 1745 Level Plan



Source: JDS (2017)

For cut and fill stopes, wall fill dilution will only occur in secondary fill stopes. Mucking or floor dilution of 0.3 m has been applied to all stopes. Table 15-6 summarizes the backfill dilution applied to the different mining methods.

Table 15-6: Backfill Dilution

Mining Method	Floor Fill Dilution (m)	Wall Fill Dilution (m)
Longhole	0.3	0.5
Longhole Upper	0.0	0.0
Cut and Fill (Primary/Single Drift)	0.3	0.0
Cut and Fill (Secondary)	0.3	1.0
Cross-cut Development	0.0	0.0

Source: JDS (2017)

15.4 Mining Recovery

Mining or extraction recovery is a function of mineralized material left behind due to operational constraints typical in the mining process. The longhole mining method is largely dependent on the accuracy of longhole drilling and explosive detonation to properly fracture the ore. Where holes deviate from the ore limits, some material will remain hung up and may never report to the stope floor for recovery. Lesser factors considered to affect recoveries in longhole mining include ragged mucking floors, limited visibility for remote mucking, and operator error.

Mining recoveries by mining type and rock mass domain have been applied based on industry norms as well as JDS operational experience in stopes and drifts of similar size and dip, and are summarized in Table 15-7.

Table 15-7: Recovery by Mining Method

Mining Method	Mining Recovery (%)
Longhole	95
Longhole Uppers	80*
Cut and Fill (Primary/Single Drift)	95
Cut and Fill (Secondary)	95
Crosscut Development	95

Source: JDS (2017)

Note: * Average recovery for near surface stopes.

15.5 Mineral Reserve Estimates

The stope and development designs with external and backfill dilution, and ore recovery factors applied determined the Mineral Reserve estimate shown in Table 15-8.

Table 15-8: Red Mountain Mineral Reserve Estimate

Category	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Au (koz)	Contained Ag (koz)
Proven	1,308	7.82	25.09	329	1,055
Probable	645	6.93	15.32	144	318
Total	**1,953**	**7.53**	**21.86**	**473**	**1,373**

Source: JDS (2017)

Notes: The Qualified Person for the Mineral Reserve estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.

Mineral Reserves were estimated using a $1,200/oz gold price and gold cut-off grade of 3.55 g/t for longhole mining and 4.10 g/t for development and cut and fill mining.

Other costs and factors used for gold cut-off grade determination were mining, process, and other costs of $128/t for longhole mining and $148/t for cut and fill mining, transport, and treatment charges of $6.00 /oz Au. A royalty of $53.10 /oz Au and a gold metallurgical recovery of 89.3% were assumed.

Silver was not used in the estimation of cut-off grades but is recovered and contributes to the revenue stream.

Tonnages are rounded to the nearest 1,000 t, gold grades and silver grades are rounded to two decimal places. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces. Rounding as required by reporting guidelines may result in summation differences.

The Mineral Reserves identified in Table 15-8 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified. This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production.

Table 15-9 and Table 15-10 list the reserves by zone and mining method, respectively.

Table 15-9: Reserves by Zone

Deposit	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Au (koz)	Contained Ag (koz)
Marc	744	8.59	31.34	205	749
AV	776	6.95	17.10	173	427
JW Main	336	7.26	14.40	78	155
141	83	4.77	10.84	13	29
JW Outliers	15	5.85	25.44	3	12
Total	**1,953**	**7.53**	**21.86**	**473**	**1,373**

Source: JDS (2017)

Table 15-10: Reserves by Mining Method

Mining Method	Tonnes (kt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Au (koz)	Contained Ag (koz)	Dilution (%)
Longhole	1,232	7.54	24.01	299	951	11
Cut and Fill	648	7.43	17.29	155	360	14
Development	58	8.71	28.67	16	54	0
Longhole Uppers	14	5.87	15.43	3	7	30
Total	**1,953**	**7.53**	**21.86**	**473**	**1,373**	**12**

Source: JDS (2017)

16 Mining Methods

16.1 Introduction

The mine design and planning for Red Mountain is based on the resource model completed by ACS in 2017 as detailed in Section 14 of this report. The following underground mining methods were selected:

- Transverse longhole stoping;
- Longitudinal longhole stoping;
- Overhand cut and fill; and
- Longhole uppers.

Mining method selection was driven primarily by vein geometry, geotechnical rock quality, and vein continuity. Unless geotechnical and geometry characteristics required cut & fill (C&F) mining, longhole mining was the preferred mining method due to higher productivities and lower mining costs.

16.2 Geotechnical Analysis & Recommendations

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.

16.2.1 Data Sources

Geotechnical drilling and underground mapping from the 2016 field program formed the basis of the geotechnical interpretation. Historical geotechnical data from preliminary resource investigations in the early 1990's were also considered during the geotechnical investigations and evaluations.

16.2.2 SRK 2016 Geotechnical Field Program

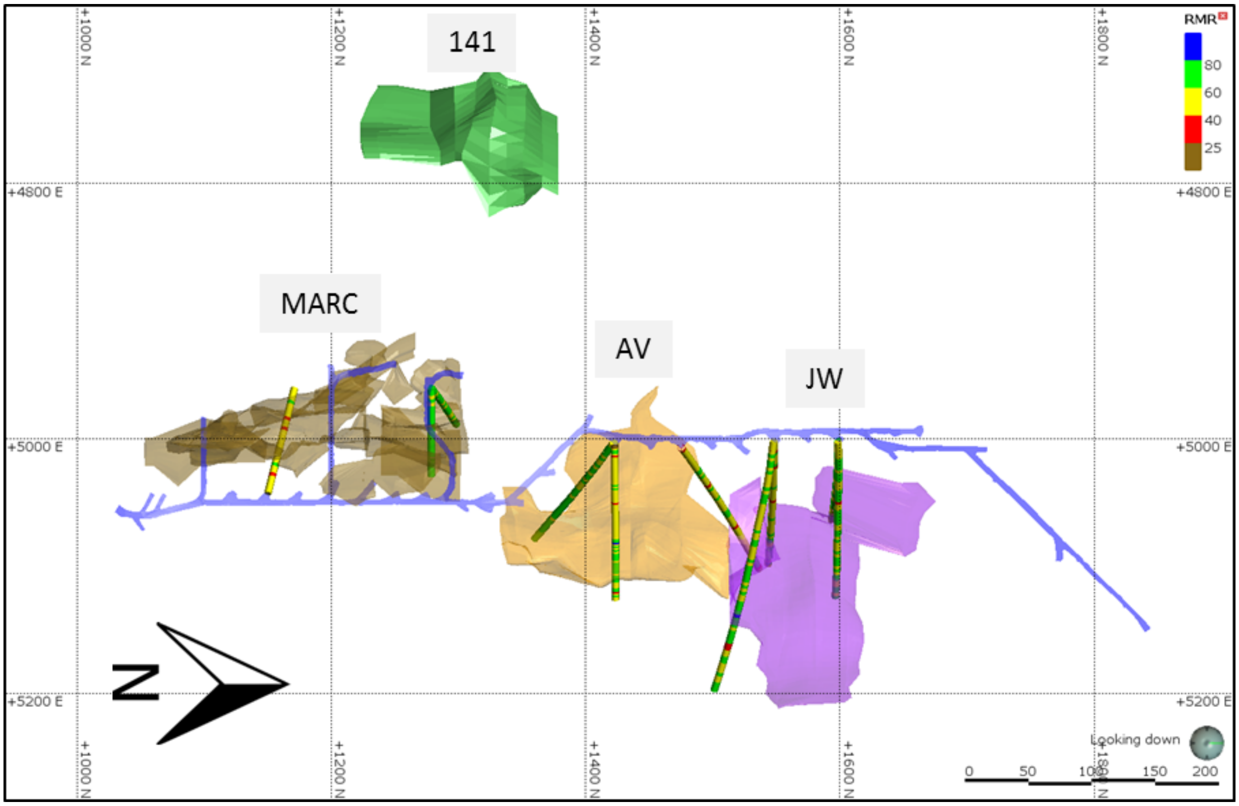
The 2016 geotechnical field program was campaigned with the resource drilling program. The program included detailed geotechnical core logging, underground geotechnical mapping, and laboratory testing. SRK staff were onsite for the duration of the field program, and were responsible to oversee the data collection, training, and quality assurance. Detailed geotechnical core logging was implemented for selected resource exploration drill holes, and was carried out by IDM under the supervision of SRK.

The 2016 feasibility geotechnical data acquisition/investigation program included the following:

Geotechnical Core Logging: Eleven drill holes had a combined resource and geotechnical purpose (Figure 16-1). Three boreholes were drilled into Marc, three into AV, and five were drilled into JW.

No geotechnical drilling intersected the 141 zone. 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 drillhole orientation to intersect these structures.

Figure 16-1: Boreholes Logged for Detailed Geotechnical Conditions in 2016



Source: SRK (2017)

Rock Strength Testing: Point load testing was undertaken approximately every 3 m during core logging and core samples from the ore zone and surrounding host rock (hangingwall 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.

16.2.3 Structural Geology

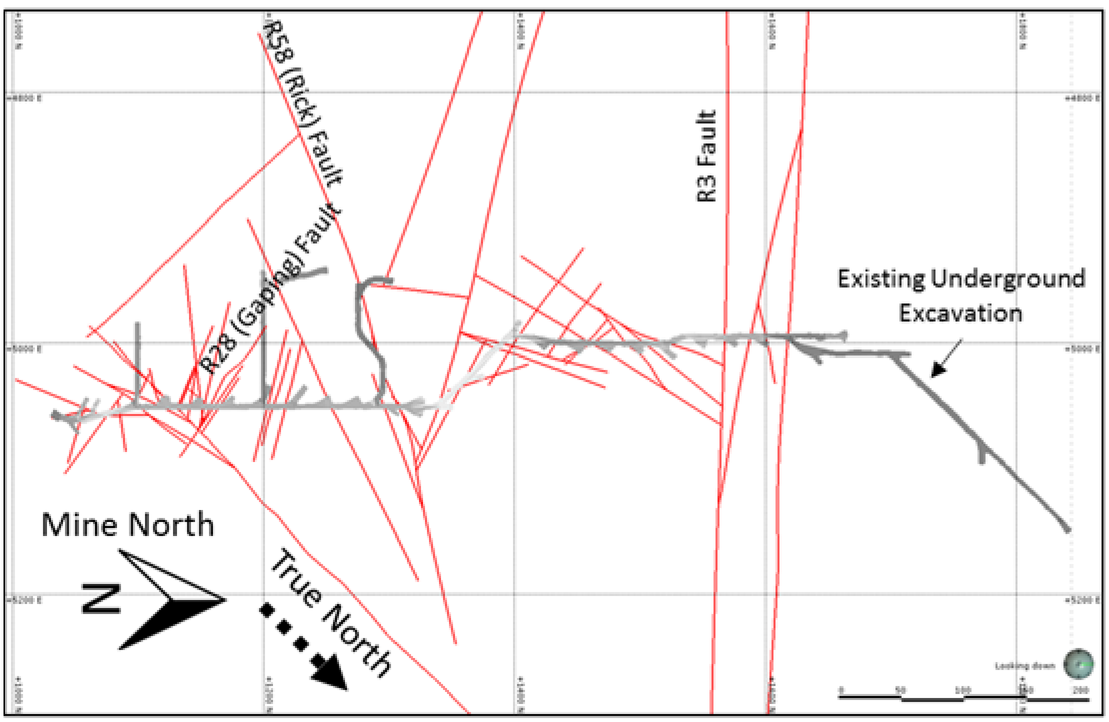
A three-dimensional structural model was developed for the deposit (Figure 16-2). Faults offsetting mineralization are labelled and mine grid north and true north are referenced in the figure. The model was built from detailed underground mapping, surface LIDAR review and lineament analysis, and borehole intercept data including historic and new geotechnical logging data. The data is referenced here to true north (as opposed to mine north) and strike (right hand rule). Three dominant trends were found for the deposit:

- **NE set**: This is the dominant fault set. Faults may dip to the SE or NW. This orientation includes faults that offset mineralization;
- **NNW set:** This corresponds to the orientation of minor Goldslide porphyry dyke offshoots and regional fold axes; and
- **NNE set:** Mostly steeply easterly dipping structures. The trend is well represented regionally.

Subordinate orientations included the following:

- **NE conjugate set:** These faults have the same trend as the main NE set but dip to the SE;
- **NE steep set:** This set strikes 033 and dips steeply to the SE;
- **EW set:** This set dips moderately to the south; and
- **NW set:** This set dips moderately to the NE. Regionally this trend is represented by the Bitter Creek trend.

Figure 16-2: Plan Section through Existing Infrastructure Showing the Traces of the 2017 Structural Model and the Existing Underground Excavations

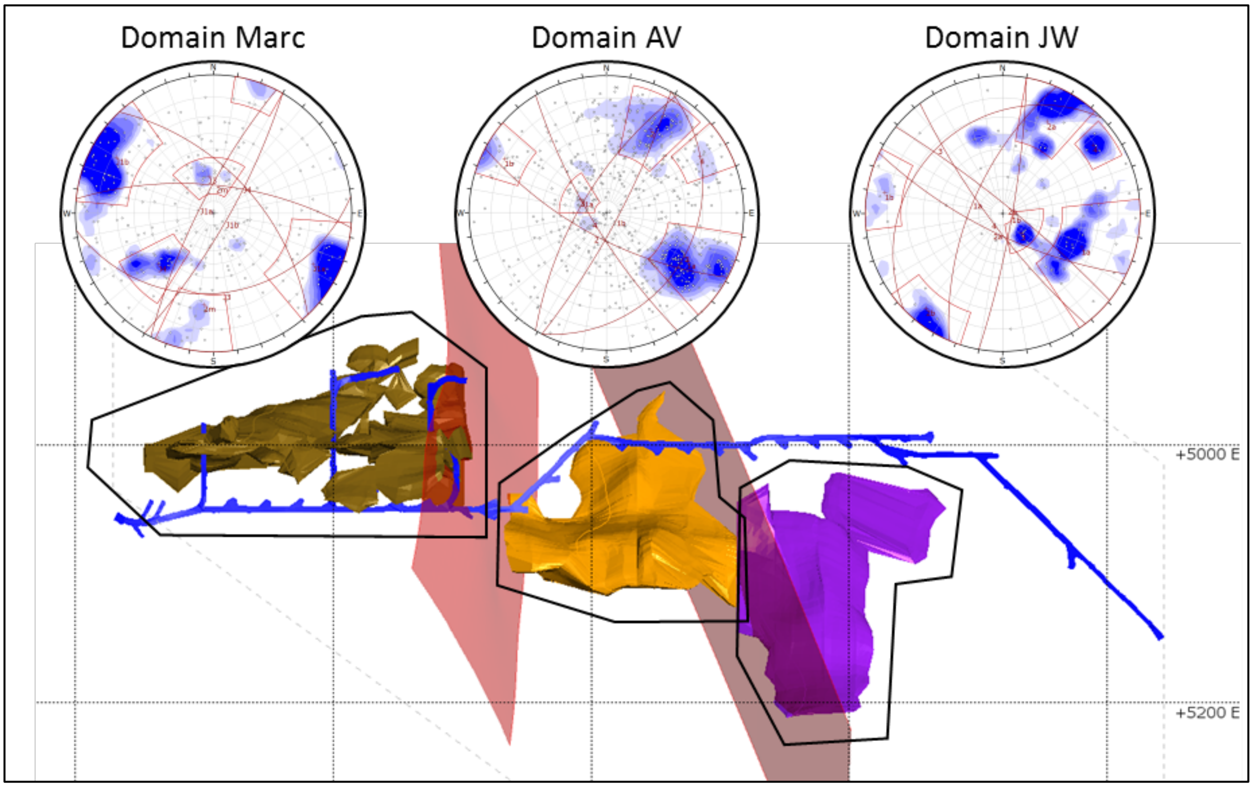


Source: SRK (2017)

16.2.3.1 Structural Domains

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 16-3).

Figure 16-3: Structural Domains for Red Mountain Orebody Areas



Source: SRK (2017)

16.2.4 Hydrogeology

16.2.4.1 Field Program

Between 1990 and 2016, a number of 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.

16.2.4.2 Impact to Mining

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 m^3/d in year 2 and then decreasing from this point onward to about 2,640 m^3/d (i.e., Base Case), while under more conservative assumptions (i.e., Upper Case), predictions were respectively 6,400 m^3/d and 4,400 m^3/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.

16.2.5 Geotechnical Design

16.2.5.1 Geotechnical Domains

Longhole open stoping has been selected as the primary mining method, with C&F and drift and fill (D&F) being applied in the narrower areas. Four geotechnical domains were derived based on the defined geotechnical conditions and expected performance of open stopes. The domaining approach considered stope stability/performance, unplanned dilution, and support requirements:

Green Domain: The most competent rock mass (RQD: > 95%, RMR: 60 - 65) with no prominent brittle structures likely to affect stope stability. This domain is suitable for open stoping with a primary-secondary sequence.

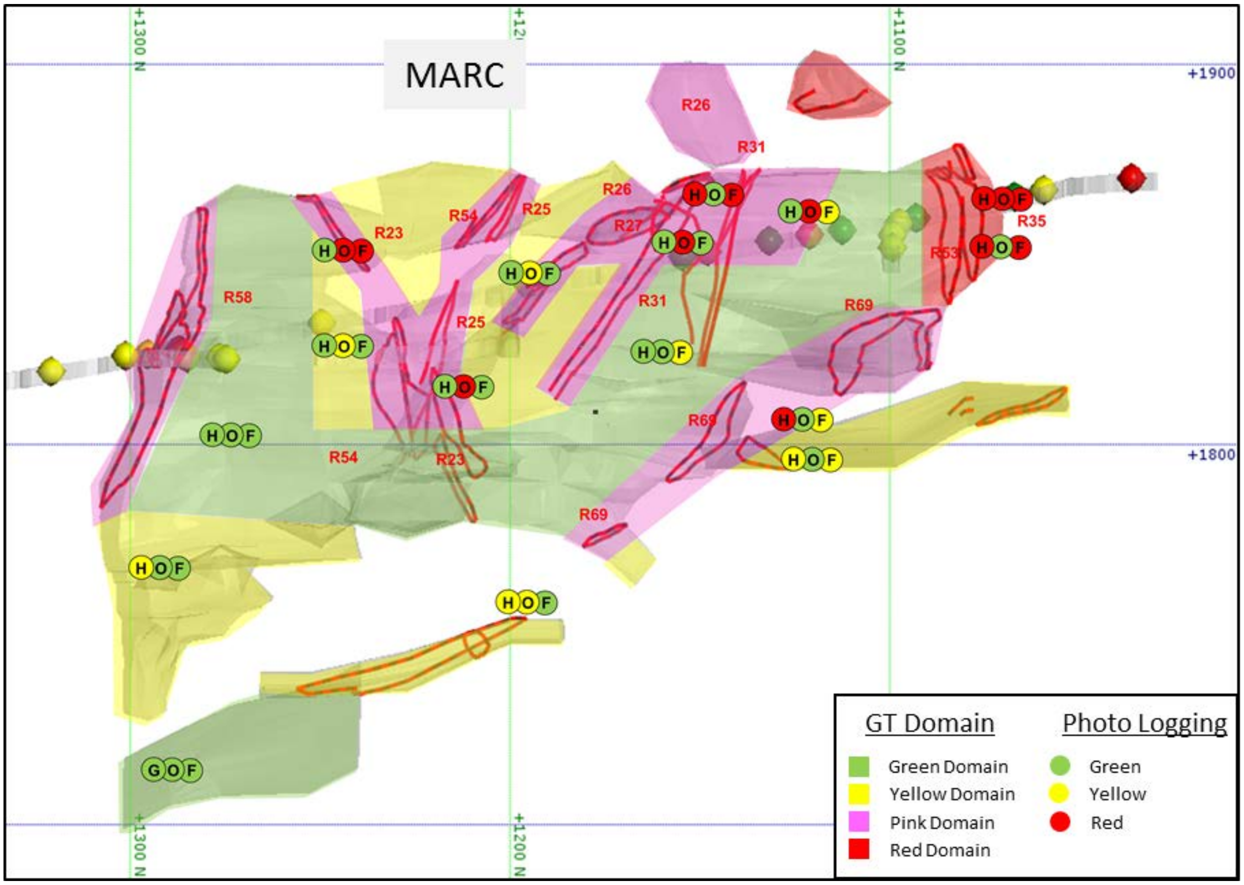
Yellow Domain: Comprised of fair rock mass conditions (RQD: 80 - 95%, RMR: 55 - 60) not affected by structures. Increased dilution is expected compared to the Green Domain. Open stoping with a primary-secondary sequence will be appropriate within this domain.

Pink Domain: This domain is similar to the Yellow in terms of the global rock mass conditions (RQD: 80 - 95%, RMR: 55 - 60), but faults are expected to impact stope stability and overall performance. Open stoping on a primary-primary sequence is achievable with the necessary support to address the more challenging ground conditions around faults.

Red Domain: The poorest ground conditions (RQD: 20 - 60%, RMR: 25 - 55) are found in the crown pillar near surface which is also associated with faults. Longhole stoping is not recommended within the Red Domain. C&F or D&F mining with the appropriate support are the recommended mining methods.

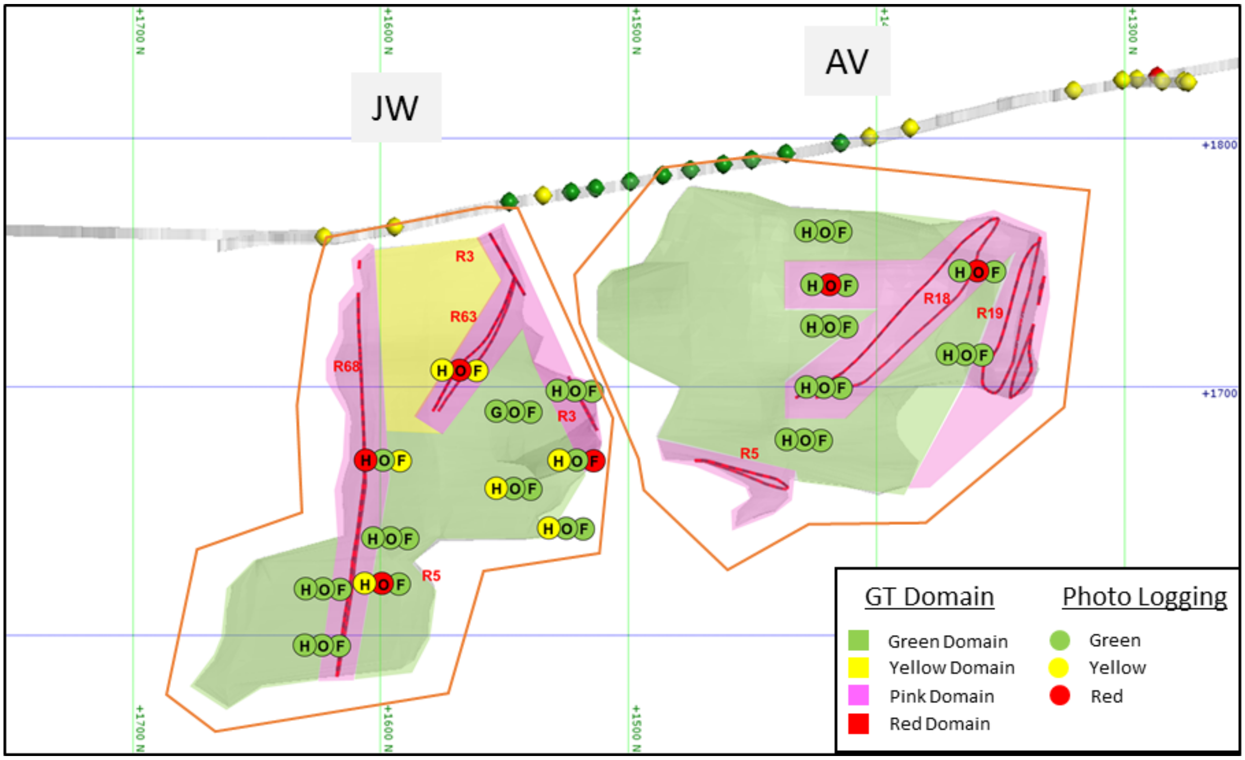
The geotechnical domains were assessed for each of the orebodies: Marc (Figure 16-4), AV, and JW (Figure 16-5). The geotechnical data, core photographs, structural model, and recommended mining methods were considered.

Figure 16-4: Geotechnical Domains within the Marc Zone



Source: SRK (2017)

Figure 16-5: Geotechnical Domains for the JW and AV Zones



Source: SRK (2017)

16.2.6 Excavation Design

The excavation stability assessment is based on well-established empirical and semi-empirical relationships, engineering judgement, and an understanding of the current excavation performance. This approach was used to estimate the expected conditions during mining, stabilize excavation sizes and to determine the support requirements. The expected excavation performance was assessed for the four defined geotechnical domains.

16.2.6.1 Man Entry Excavations

The development and production excavations requiring man entry have been assessed using the critical span curve after Ouchi et al. (2004). The design spans (4 – 10 m) fall within the stable to potentially stable region for the Green, Yellow, and Pink domains. This suggests that the excavations are expected to be stable with typical ground support. The Red Domain consists of weaker rock mass intersected by faults. The design spans (4 – 6 m) are expected to be potentially unstable to unstable, which will necessitate additional ground support and adjustments to the mining sequence to ensure excavation stability.

16.2.6.2 Production Excavations

Longhole stope design (no man-entry) was completed using the modified Mathews stability curve after Stewart and Forsyth (1995). The stope design was constrained by practical mining limitations and assessed based on stability and dilution. A range of sublevel spacing's and strike dimensions were considered while the orebody thickness dictated the third dimension. Transverse stoping was considered for the wider (>10 m) part of the orebody with narrow zones to be mined using a longitudinal stoping method. The orebodies vary in dip from sub-vertical to 30º dip. Longhole open stoping was considered in the wider and steeper areas (dip > 55º) with D&F and C&F elsewhere. The findings of the stope stability assessment concluded that:

Overhand open stoping was recommended for the Green, Yellow and Pink domains based on the 25 m sublevel spacing (floor to floor). Stope length, width and mining sequence should be based on the ground conditions in the various geotechnical domains:

- Longitudinal stopes length and width should be limited to 20 m and 10 m, respectively;
- Transverse stopes in the Green and Yellow domains could be mined on a primary-secondary sequence. Stope width (15 m primary and 10 m secondary) was based on stope stability and dilution. Stope length is based on the orebody width but individual stopes should not exceed 20 m; and
- The Pink domain contains faults that will impact stability and dilution. Transverse stopes (15 m wide) should be mined on a primary-primary (end slicing) sequence to manage stability. Increased ground support, including cable bolting, will be required to manage stability around the fault damage zones. Dilution is expected to be similar to the Yellow domain; however, the fault interactions with the stopes pose a risk of additional unplanned dilution (i.e., large wedges).

Open stoping mining methods are considered high risk in the Red Domain. These (Red) domains are more suited to a C&F / D&F mining method.

16.2.7 Dilution

The mineralized zone is not defined by discrete structures; dilution from both the hangingwall 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 – 0.90 m) and Longitudinal (0.85 m) stopes; and
- Yellow and Pink Domain: Transverse (0.80 – 1.40 m) and Longitudinal (1.15 m) stopes.

Kinematic wedge analysis using UNWEDGE™ identified potential unstable wedges forming in the hangingwall of the longhole stoping. These wedges are expected to increase dilution estimates by 0.19 m and 0.08 m for the Marc and AV/JW zones, respectively.

16.2.8 CRF

The selected mining methods, longhole open stoping and C&F / D&F require backfill to achieve the planned extraction and manage stability. CRF will be used in all longitudinal stopes and primary transverse stopes as well as the primaries in D&F areas. Geotechnical considerations do not require CRF in secondary stopes and C&F mining provided the fill will not be exposed by future mining.

CRF strength (UCS > 2 MPa) was 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. No underhand mining or sill mining is planned which eliminates the need to work underneath previously placed backfill.

16.2.9 Excavation Interaction & Sequence Evaluation

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. 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.

16.2.9.1 Sequence

The current mining design is based on an overhand longhole open stoping in the wider and steeper areas with overhand D&F / C&F mining being used in the narrower and flatter areas.

Longhole Open Stoping

The adopted primary-primary sequence broadly entails a sequence that commences at the bottom and middle of the stoping area expanding upwards and outwards. This method minimizes backfill exposure to one sidewall but requires CRF in all stopes. Numerical modelling suggests that zones of high induced stress and rock mass failure will be limited using the proposed extraction sequence. Stoping may affect development infrastructure by reducing confinement around the excavations close to the stoping horizon. The recommended ground support should be adequate to manage this risk.

D&F / C&F

Overhand D&F and C&F mining is planned to extract the narrower and flatter (< 55°) portions of the deposit. This includes the JW zone and smaller sections of the Marc and AV zones. The mining will begin at the bottom of the mining blocks and progress upwards. D&F mining in the Marc zone will occur in close proximity to previously mined and backfilled open stopes. Higher induced stresses are expected between D&F mining and previously mined open stopes. This will necessitate the usage of CRF in the stopes to mitigate potential interaction risks. Limited mining induced stress issues are anticipated in AV and JW zones.

16.2.10 Crown Pillar Assessment

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. D&F and longhole stoping will be used to mine in close proximity to surface. The crown pillar will not be stable long term and will require sufficient support to excavate. The planned D&F excavations have a minimum crown pillar of 12.5 m thickness, and longhole stoping will be conducted with a minimum pillar thickness of 20 m that results in a temporary stable crown pillar for a maximum 8 - 10 m span. The current mine plan honors these recommendations. Long-term crown pillar stability will require tight filling of the D&F and long-hole excavations with CRF.

There are two areas where the crown pillar will be mined. These areas include the second upper portal and another from a ramp nearby that will not breakthrough to surface until excavation. The pillars will be mined using uppers at the end of mine life.

16.2.11 Ground Support Design

The ground support requirements have been evaluated for the planned development and production excavations in each of the geotechnical domains using empirical design charts (i.e., Grimstad and Barton, 1993) and experience from similar operations. The empirical support requirements were 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. The general support recommendations for development, open stopes and C&F / D&F are summarized below:

16.2.11.1 Infrastructure Development

Green, Yellow, and Pink domains – Resin rebar (1.8 m), welded wire mesh and cable anchors around faults.

Red domain – Coated Swellex (2.1 m), welded wire mesh and shotcrete (50 mm).

16.2.11.2 C&F / D&F

Green, Yellow, and Pink domains – Resin rebar (1.8 to 2.1 m), welded wire mesh (high traffic areas) with cable anchors (4.0 m) around faults and in wider spans.

Red domain – Coated Swellex (2.1 m), welded wire mesh and shotcrete (50 mm).

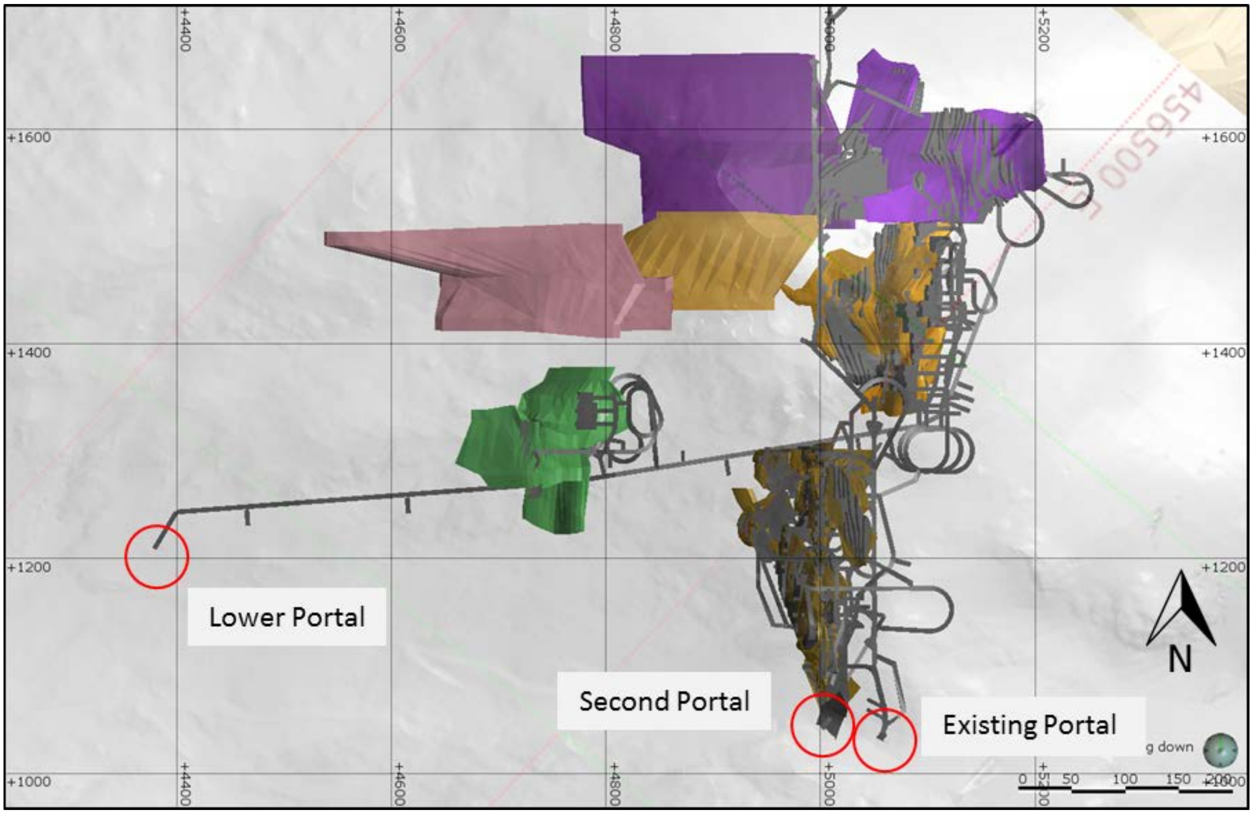
16.2.11.3 Transverse and Longitudinal Stopes

Resin rebar (1.8 to 2.1 m), welded wire mesh and cable anchors (4.0 to 6.0 m) in the wider spans (>6.0 m) and around faults.

16.2.12 Mine Access

Two additional portals are proposed: A lower portal closer to the processing facilities, and an upper portal adjacent to the existing portal. The proposed lower portal (Figure 16-6) is approximately 700 m to the south of the existing portal and removed from all available geotechnical data and no detailed structural interpretation exists in the area. There is a large regional fault (Slippery Jim) to the west of the proposed site. The second upper portal is planned near the existing decline for access to the crown pillar mining and to provide ventilation. Rock mass conditions are assumed to be similar to the upper existing portal, but will need further investigation. A geo-hazard rock fall and avalanche assessment of these locations is still to be undertaken.

Figure 16-6: Plan View of Existing & Planned Portals with Mine Design & Mineralized Solids



Source: SRK (2017)

16.2.12.1 Portal Design

Portal design is conceptual lacking detailed drilling or geotechnical data at either of the proposed locations. Preliminary design requires clearing talus from the slope, or excavation of talus to a stable slope angle and/or supported. The suitability of talus as a road bed should also be considered. The conceptual portal design is modelled after the existing portal.

The portals (4.5 m x 4.5 m, with an arched back) should be excavated with at least 5 m competent rock above the back and rockfall protection must be included in the design. Ground support around the portal should include:

- 2.4 m resin rebar at 1 m spacing with chain link;
- 6 m cable bolts at 2 m spacing (two rows above portal);
- Sidewalls in rock cut to 45º and cleared of talus;
- Face angle cut to 70º, with a minimum of 5 m of competent rock above portal entrance;
- Solid structure to be installed outside the portal (minimum 3 m) and for the first 3 m into the decline to mitigate rockfall risk from the slope above the portal. This can be a timber structure like existing portal, or segmented steel (Armtec); and

- Portal to be designed to accommodate frame for rockfall protection.

16.2.13 Geotechnical Conclusions & Recommendations

SRK undertook field investigations designed to characterize the rock geotechnical conditions, and provide appropriate analysis of the data for use in underground mine and infrastructure design, in support of the feasibility study.

Based on the geotechnical assessment, the orebody has been subdivided into four geotechnical domains. Three domains are considered suitable for longhole open stoping and the fourth requires D&F or C&F mining. Geotechnical design recommendations were derived based on rock mass conditions and in situ stress levels.

A 3D structural model was created for the deposit from underground mapping, core logging data, and core photo review. The brittle structures are expected to have some impact on the planned mining and they form the boundaries between the three ore bodies (Marc, AV, and JW).

Dilution (ELOS) from the longhole open stopes is expected to range between 0.55 and 1.4 m depending on the domain and rock mass conditions and stope dimensions.

Based on the current mine design, the crown pillar is expected to be temporarily stable. Long-term stability will require installation of the recommend support and tight filling of the excavations with CRF.

Groundwater inflows are predicted to be relatively low, reaching an annual average rate of about 3,810 m^3/d in year 2 and then decreasing from this point onward to about 2,640 m^3/d (i.e., Base Case), while under more conservative assumptions (i.e., Upper Case), predictions were respectively 6,400 m^3/d and 4,400 m^3/d. Seasonal water inflows into stoping areas could complicate mining and backfill operations in faulted areas.

The support design is based on a range of ground conditions (geotechnical domains) and specific spans which should be maintained. Additional support requirements are required in the existing accesses due to stress changes related to production stoping.

There is some level of uncertainty related to stoping and excavation performance in the JW and 141 zones as a result of limited geotechnical data for those zones. Additional joint orientation data for the JW and geotechnical data for the 141 zone will improve the level of confidence.

Geotechnical conditions at the lower portal site have been assumed to be similar to the existing portal. A geotechnical drilling program and geo-hazard assessment will need to be undertaken.

16.3 Mine Planning Criteria

The mine planning criteria for Red Mountain are listed below:

- The pre-production mine development period will be approximately six months, with the duration being split between surface preparations, portal construction at the second, upper portal location, and underground ramp and infrastructure development from the existing exploration decline. Ore will be extracted in the first quarter of year 1 and ramps up over the first and second quarter to the full 1,045 t/d average production rate. After year 1, the mine and mill will operate at a steady state of 365,750 t/a;
- It was estimated that due to weather conditions, access to the underground mine would be limited to 350 days per year with snow clearing equipment and avalanche control;
- Underground mining and maintenance will be carried out by owner;
- Contractors will be utilized for Alimak raise development;
- Conventional, trackless diesel-electric mining equipment will be utilized; and
- Mined voids will be filled with rock fill or CRF.

Other key mine planning criteria are summarized in Table 16-1

Table 16-1: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	350
Shifts per Day	Shifts	2
Hours per Shift	Hours	10
Work Roster	Days On/Off	4/4
Nominal Ore Mining Average Rate	t/d	1,045
Annual Ore Mining Average Rate	T	365,750
Ore Density	t/m^3	3.00
Waste Density	t/m^3	2.89
Swell Factor	%	35

Source: JDS (2017)

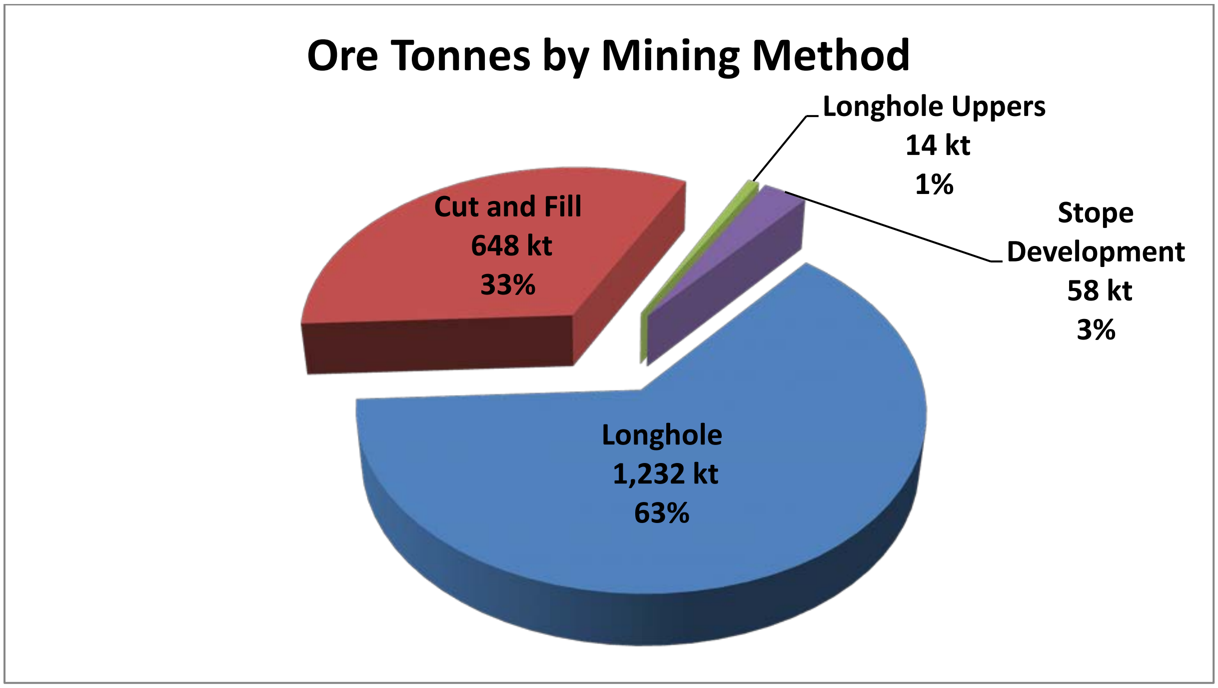
16.4 Mining Method Selection

The Red Mountain mine plan is made up of four major zones: Marc, AV, JW, and 141. All four zones share a general NE-SW strike, with dips varying between 30° and 80°. The zones range from 1 m to 40 m in width, 70 m to 200 m in strike length, and 60 m to 100 m in height.

JDS selected a combination of transverse and longitudinal longhole stoping, to be mined on a pillar less primary-primary basis with a majority of the stopes backfilled using CRF. Rock fill will be used as fill for the remaining stopes. The method will be used as the principal mining method at Red Mountain due to its high productivity, low cost, and successful history of application for deposits of this nature.

C&F mining will be used where conditions are not amenable to longhole stoping, this includes areas of shallow dips, weaker ground and where greater selectivity is required. Two stopes at or near surface will be mined with longhole uppers in a retreat sequence, this extraction method contributes less than 1% of the total reserve. Figure 16-7 illustrates the total reserve by mining method. Stope access development will contribute 3% of the planned ore produced.

Figure 16-7: Ore Tonnes by Mining Method

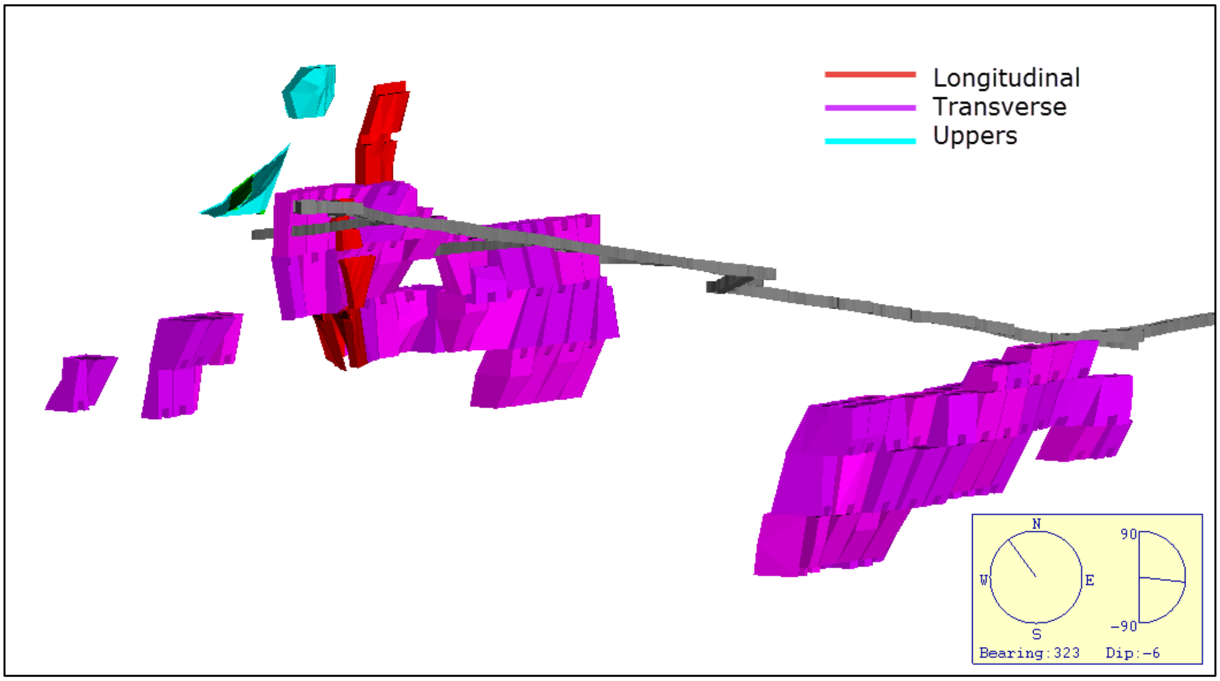


Source: JDS (2017)

16.4.1 Longhole Mining

In the planned longhole stopes at Red Mountain, a top and bottom drift vertically delineate the stope. Level spacing for the longhole sublevels varies from 15 m up to a maximum of 25 m and has been designed to maximize resource recovery and minimize dilution. The maximum overall stope height is 29 m including the 4 m height of stope development. Stopes have been designed at 15 m wide with a maximum length of 20 m. Where the orebody thickness exceeds 20 m, two stopes have been designed to enable mining of the full thickness of the orebody. Where the hangingwall stope would be mined and filled followed by mining of the footwall stope. Figure 16-8 shows the designed longhole stopes for the deposit.

Figure 16-8: Designed Longhole Stopes (Oblique View)



Source: JDS (2017)

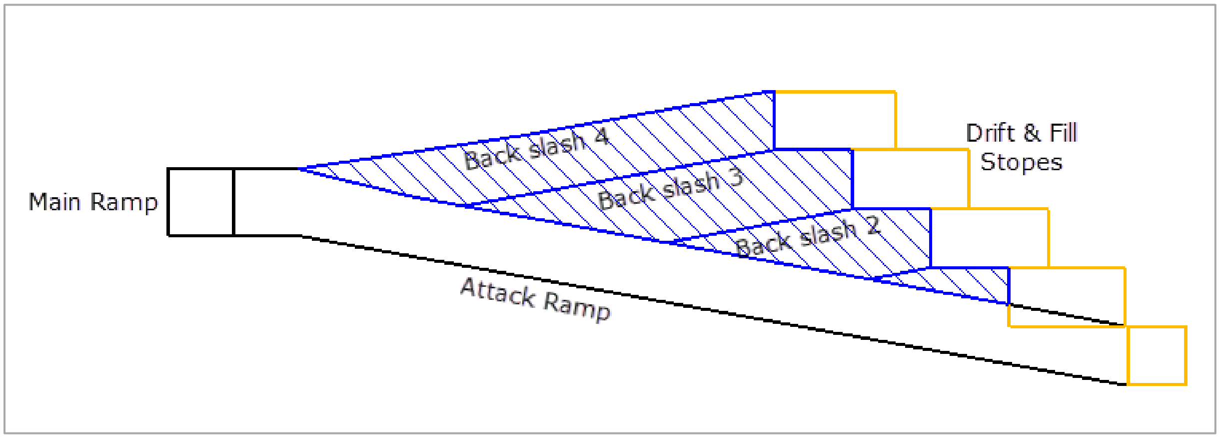
Once the stope is developed, the longhole mining cycle will begin with drilling of the stope. The drill holes will be loaded with explosives and the stope blasted, with broken material falling to the bottom drift for extraction. In longhole stopes, remote controlled load haul dump machines (LHDs) will be required to remove the blasted material from the stope once blasting commences.

Initially the slot raise will be pulled through from the lower extraction drift up to the drill drift, once the slot is blasted and mucked, the void provides a free face for the first longhole rings.

C&F mining will also be used at Red Mountain for areas of the deposit, which fall below an allowable dip for longhole stoping, where weaker rock conditions are expected to be encountered, or where more selective mining will be required. Portions of the Marc, AV, and 141 zone will be mined using the C&F method. The JW and JW Outlier zones will be mined using only C&F.

C&F lifts will be accessed by 4 m x 4 m attack ramps driven at a maximum of +/- 15%, these ramps generally will run perpendicular to strike of the zones. Figure 16-9 shows an example of how the C&F lifts will be accessed by the attack ramp.

Figure 16-9: C&F Stoping (Section View)



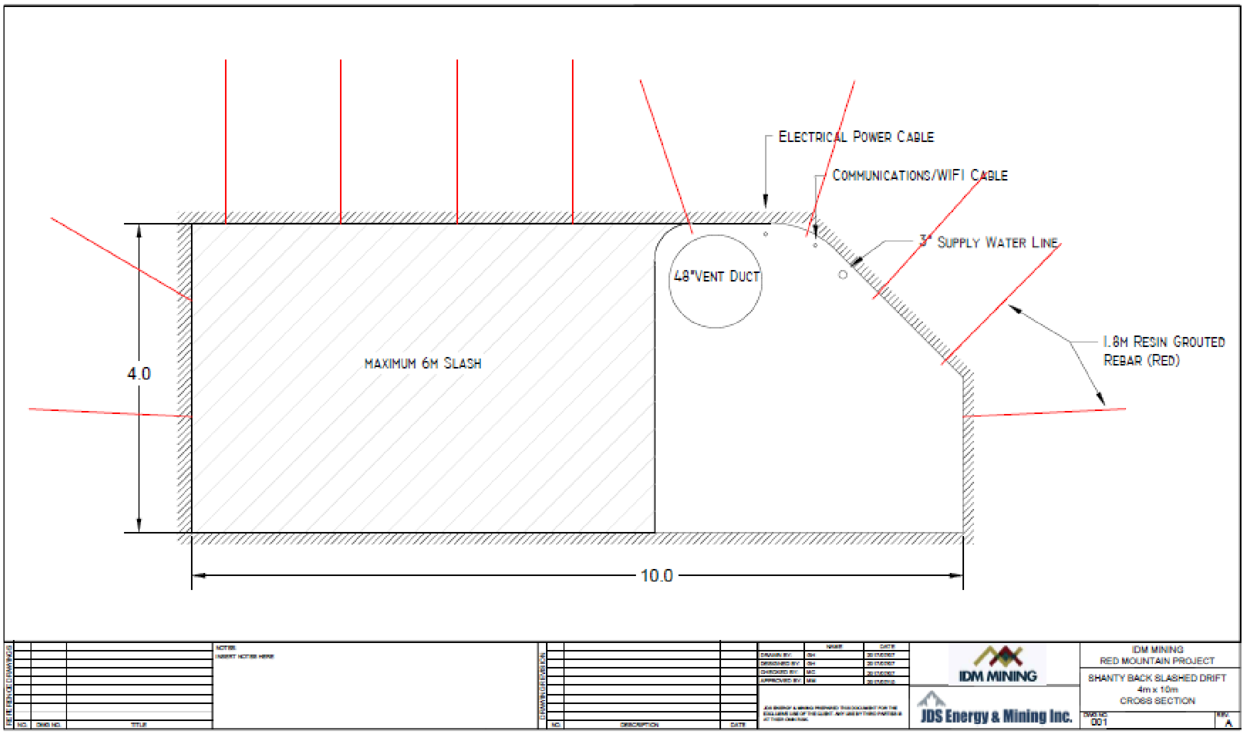
Source: JDS (2017)

Each C&F lift will be mined out overhand using one and/or a combination of the following development stopes:

1. 4 m x 4 m square development;
2. 4 m x 4 m shanty back development;
3. Wall slashing (to a maximum span of 10 m); and
4. 4 m x 4 m primary/secondary stopes mined from a 4 m x 4 m footwall drift (where orebody exceeds 10 m width).

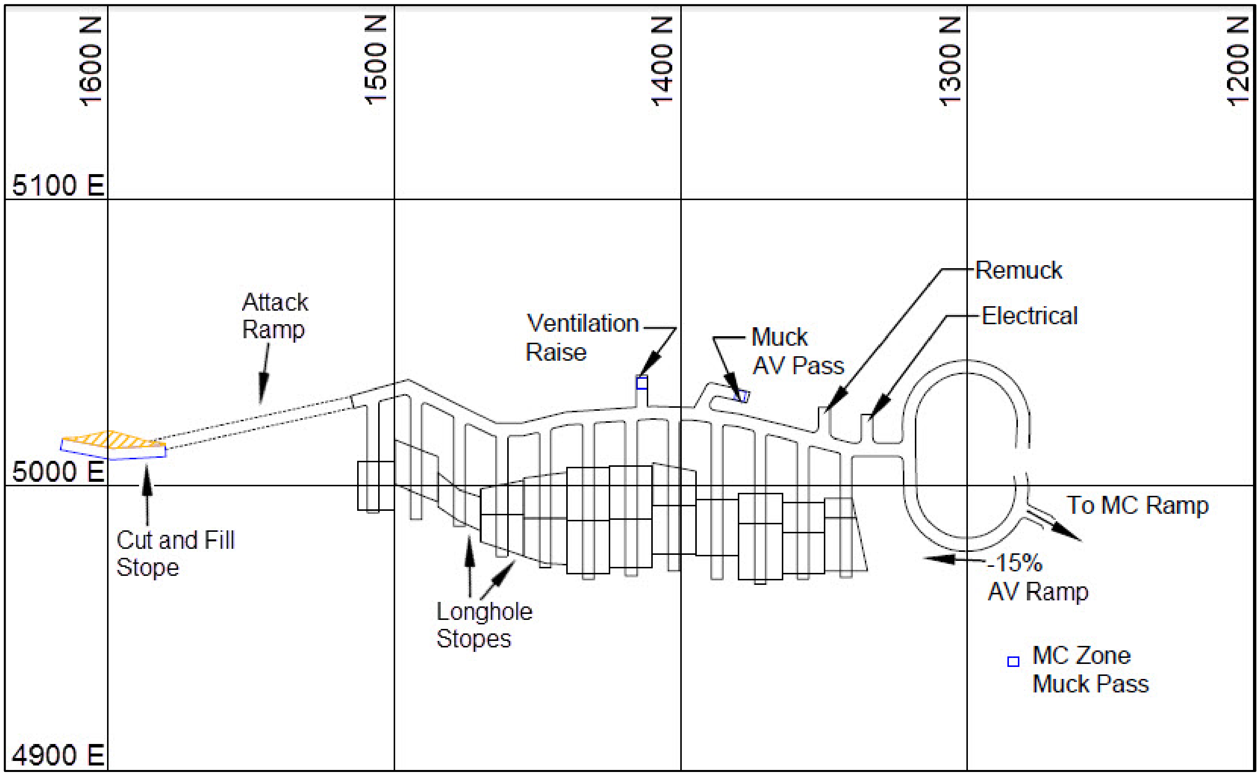
Figures 16-10 and 16-11 depict the example of the shanty back profile used in conjunction with wall slashing.

Figure 16-10: Shanty Back Profile & Wall Slash



Source: JDS (2017)

Figure 16-11: Level Plan



Source: JDS (2017)

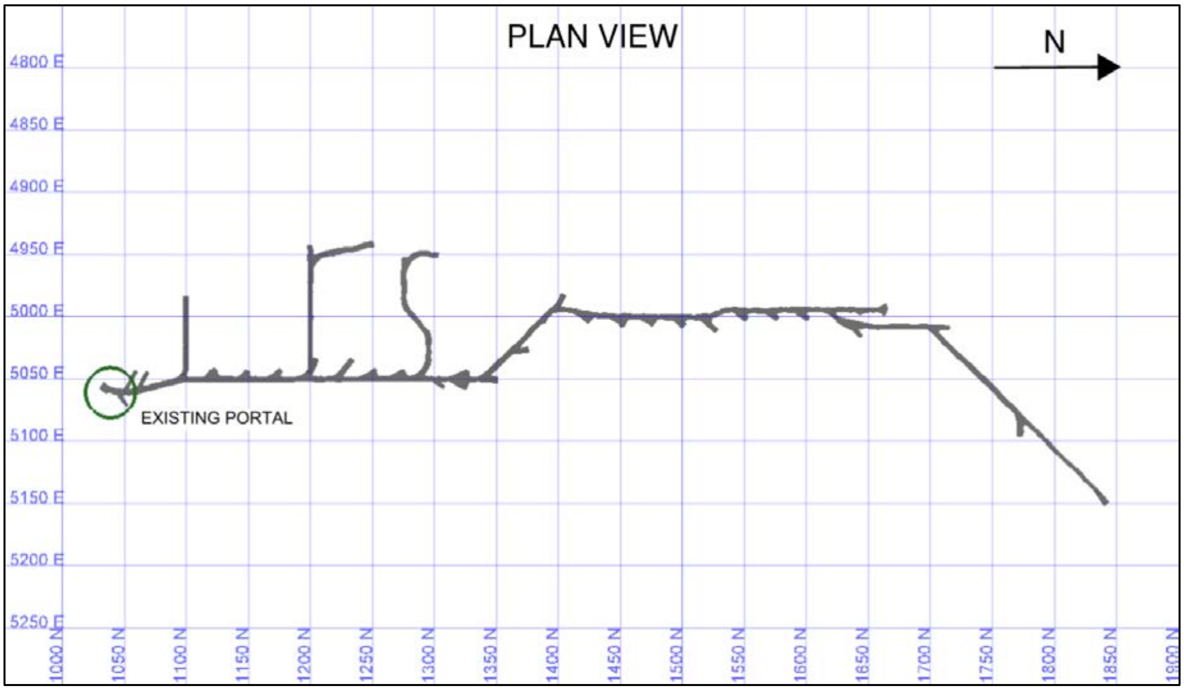
C&F stopes will be filled with rock fill, with the exception of the primary stopes which will require CRF to allow mining of the adjacent secondary stopes.

16.5 Mine Design

16.5.1 Mine Access

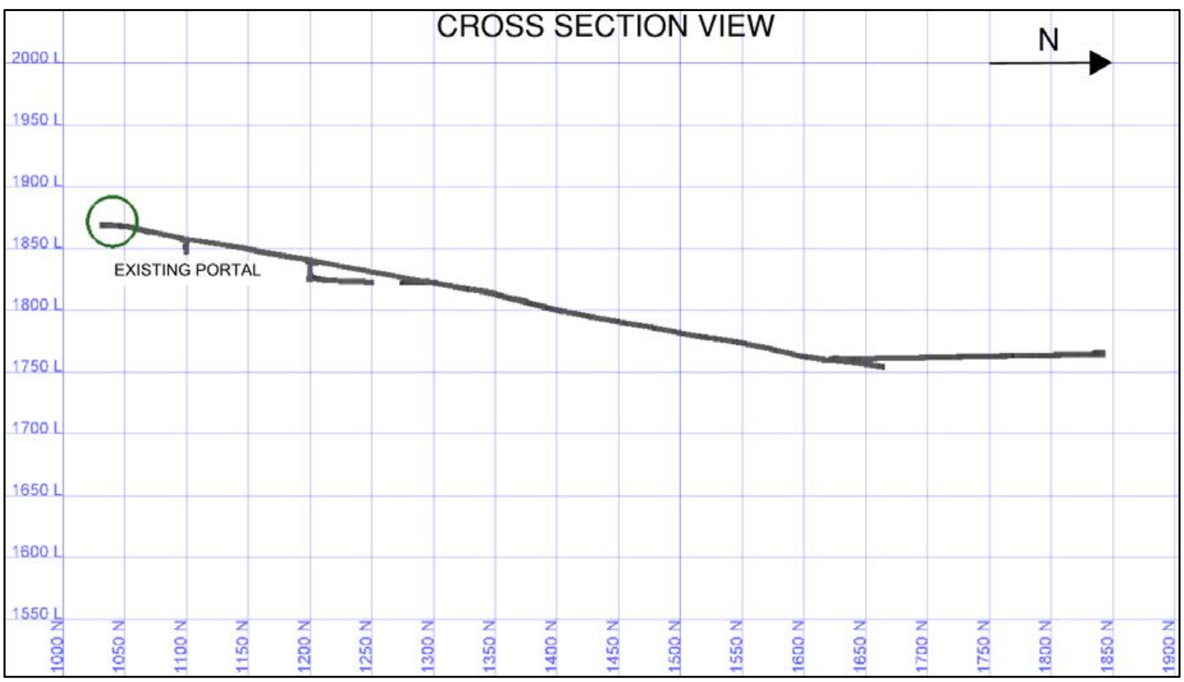
The Red Mountain deposit consists of a mineable resource extending 310 vertical metres. An exploration decline currently extends from a portal at 1,860 masl over the entire strike length of the known resource, well situated at the top of the mineralized zones. The existing decline is about 5 m wide by 4 m high and was driven at a grade of -17%. The existing workings as shown in Figures 16-12 and 16-13 have been incorporated and will be utilized for access and ventilation, and will be the main access in and out of the mine until the lower portal and access drift is completed in year 1.

Figure 16-12: Existing Underground Workings - Plan View



Source: JDS (2017)

Figure 16-13: Existing Underground Workings - Section View



Source: JDS (2017)

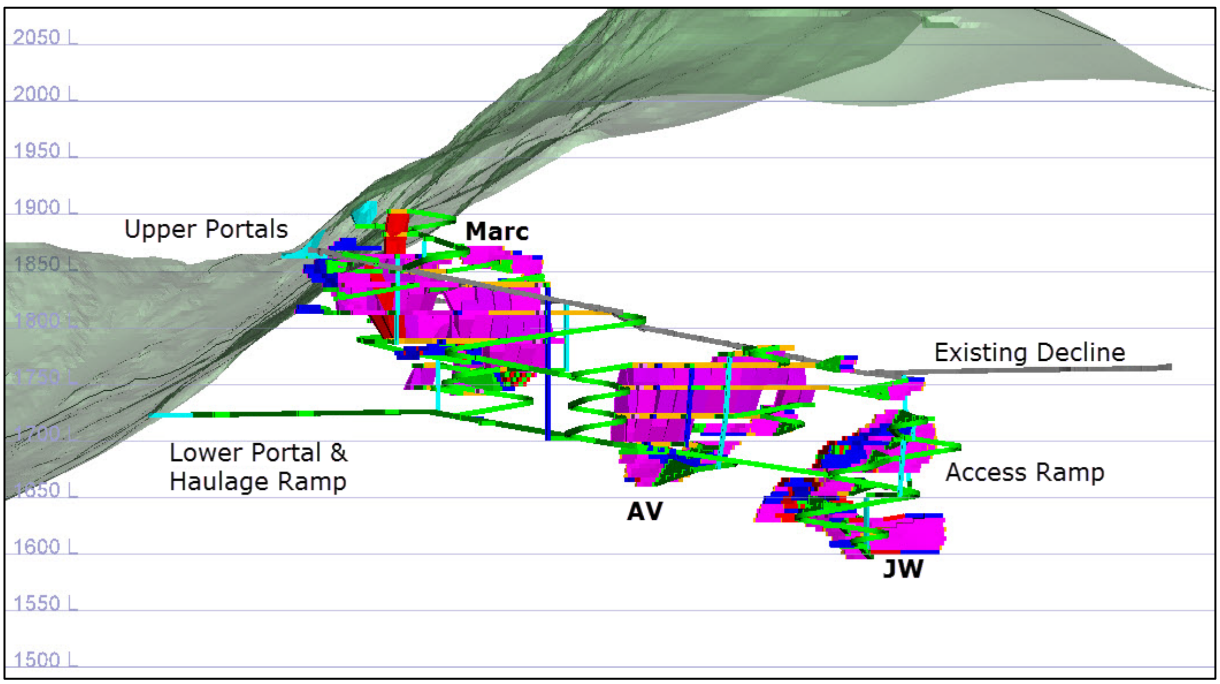
A second portal at the 1,860 m elevation will be mined at 4.5 m x 4.5 m. This portal will connect into the Marc zone level development and is required for ventilation early in the mine life.

The deposit is situated on a mountainside, and as such, there is an opportunity to establish another portal and drive an access towards the bottom of the AV zone. This would provide the mine with some gravity drainage to prevent mine flooding, a shorter haulage route to the process plant, and the potential to gravity feed broken muck via a muck pass in both the Marc and AV zones.

The lower portal is designed at 1,720 masl and is 300 m from the 'lower portal' laydown area. The incline will be driven at a grade of +1.0% until the connection with the 141 incline is reached, then declines down at -10% where it will connect with the AV zone development.

An internal ramp system would be developed for each of the four zones to provide mobile equipment access to the production levels. Access ramps would be driven at maximum grade of 15% at a 4.5 m by 4.5 m profile. Mineralized zone development would be on a 4.0 m by 4.0 m profile. Figure 16-14 below depicts the mine access points.

Figure 16-14: Mine Access Oblique View



Source: JDS (2017)

16.5.2 Mine Design Criteria

Development profiles and gradients shown in Table 16-2 were selected based on the equipment specifications, ventilation requirements, and stope design.

Table 16-2: Lateral Development Design Criteria

Development Heading	Width (m)	Height (m)	Maximum Gradient (%)
Mine Ramp	4.5	4.5	15
Remucks & Sumps	4.0	4.0	15
Footwall & Level Access	4.5	4.5	2
Vent Access	4.0	4.0	2
Orepass Access	4.0	4.0	2
Alimak Nest	4.5	8.0	2
Attack Ramp	4.0	4.0	15
Longhole Stope Access	4.0	4.0	2
Cut & Fill Drift	4.0	4.0	2
Cut & Fill Shanty Drift	4.0*	4.0*	2

Source: JDS (2017)

Note: Cut & Fill shanty drift designed with a 45 degree chamfer from the midpoint of the drift

Ore passes in the Marc and AV zones will be driven by Alimak at 3.5 m x 3.5 m. Ventilation raises longer than 30 m will also be mined by Alimak at 3.5 m x 3.5 m. Where ventilation raises are less than 30 m in length drop raises drilled using the longhole machines will be mined.

16.5.3 Stope & Mine Plan Optimization

Mine planning for the Red Mountain Project was completed by using Maptek© Vulcan 3D (Vulcan) and Minemax© iGantt software.

Vulcan's stope optimization software was utilized to determine where Longhole stoping was feasible in the deposit, select appropriate level spacing and level location, and generate initial stope shapes within the resource block model using the calculated cut-off grade.

The resulting stope optimizer shapes were reviewed in plan and section and adjusted as necessary to improve grade, tonnage, and/or mineability of each stope shape.

Longhole stopes were restricted to Green, Yellow, and Pink geotechnical zones as defined by SRK. For the remaining resource above COG, C&F stopes were designed.

The mine plan has been sized for a 1,045 t/d operation. Cycle times of the different mining methods were considered along with the tonnes per vertical metre and layout of the mine in determining the production rate. iGantt scheduling software was used to optimize the mine production schedule by maximizing the net present value (NPV), subject to constraints including maximum lateral development rates, maximum production rates, and extraction sequence.

Key scheduling constraints were as follows:

- 220 m per month per drill jumbo;
- 2 x two-boom jumbos; and
- Average 1,045 t/d mine production.

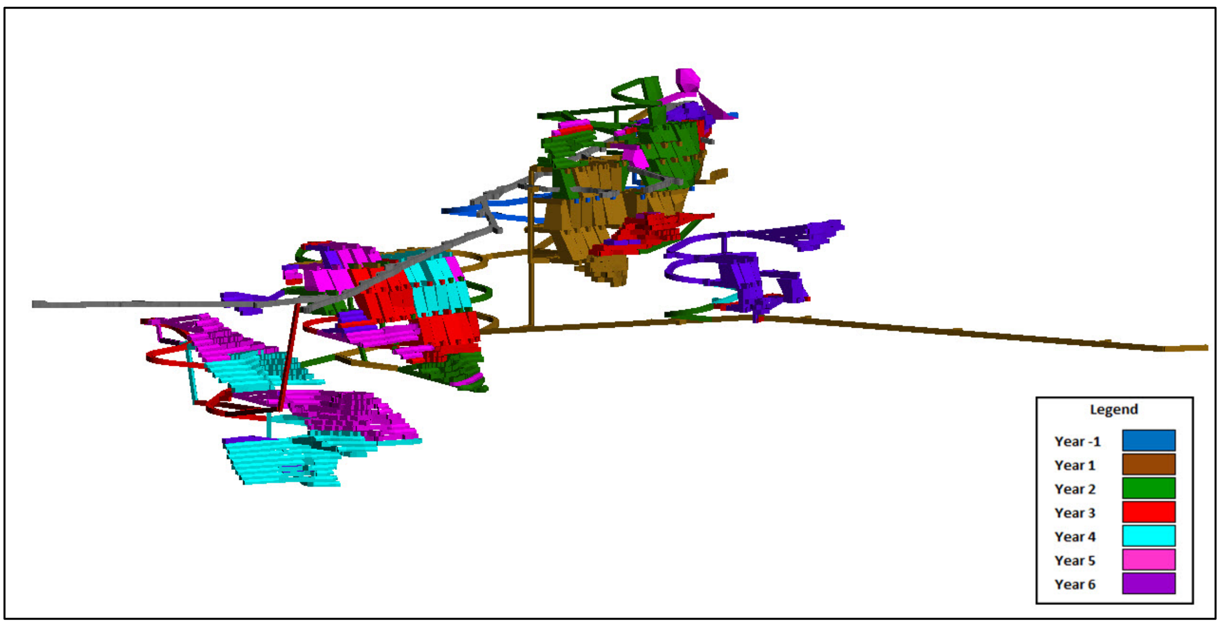
16.5.4 Production Sequencing

Production in longhole stoping zones will be mined with a bottom-up sequence in a pillar less primary to primary fashion, where once the initial stope in the sequence is mined, backfilled and cured the immediately adjacent stope would be next in sequence. 93% of longhole stopes will be backfilled with CRF with the remaining 7% filled with waste rock fill.

C&F zones will be mined in a bottom-up fashion from a main access drift. From the main ramp, a drift will access the production area with a +/-15% attack ramp. C&F stopes will be mined and filled progressively on the level. Once backfilling is complete on that level, the attack ramp will be slashed along the back and backfilled on the floor to allow access to the next level above, where the mining process will be repeated. Primary transverse C&F stopes will be filled with CRF the remainder of cut and fill stopes will be filled with rock fill. In the JW and AV zones a sill pillar exists between the 2 C&F mining fronts.

Figure 16-15 depicts the planned mining sequence on an annual basis for the mine plan.

Figure 16-15: Annual Mine Plan Sequencing



Source: JDS (2017)

16.6 Mine Dilution & Recoveries

See sections 15.3 and 15.4 for details on mining dilution and recovery parameters that were used for stope design.

16.7 Mine Services

16.7.1 Mine Ventilation

The ventilation network and fresh air supply quantities were designed to comply with Canadian ventilation standards. Required airflows were determined at multiple stages during the mine life, using equipment fleet numbers, utilizations, specific engine types and exhaust output, and number of working faces. For Red Mountain, the following was determined:

- Equipment being purchased and utilized underground will meet the new Tier 3 or Tier 4 U.S. Environmental Protection Agency (EPA) diesel emission standards;
- During peak production, 87 m^3/sec (185 kcfm) will be required to remove diesel emissions;
- Active working faces will require 20.7 m^3/sec (44 kcfm); and
- 10% leakage factor allowance.

Total designed ventilation capacity is 122 m^3/sec (260 kcfm). Initially in year -1, auxiliary fans and vent ducting will be used to advance underground development from the existing exploration adit and the second upper portal location.

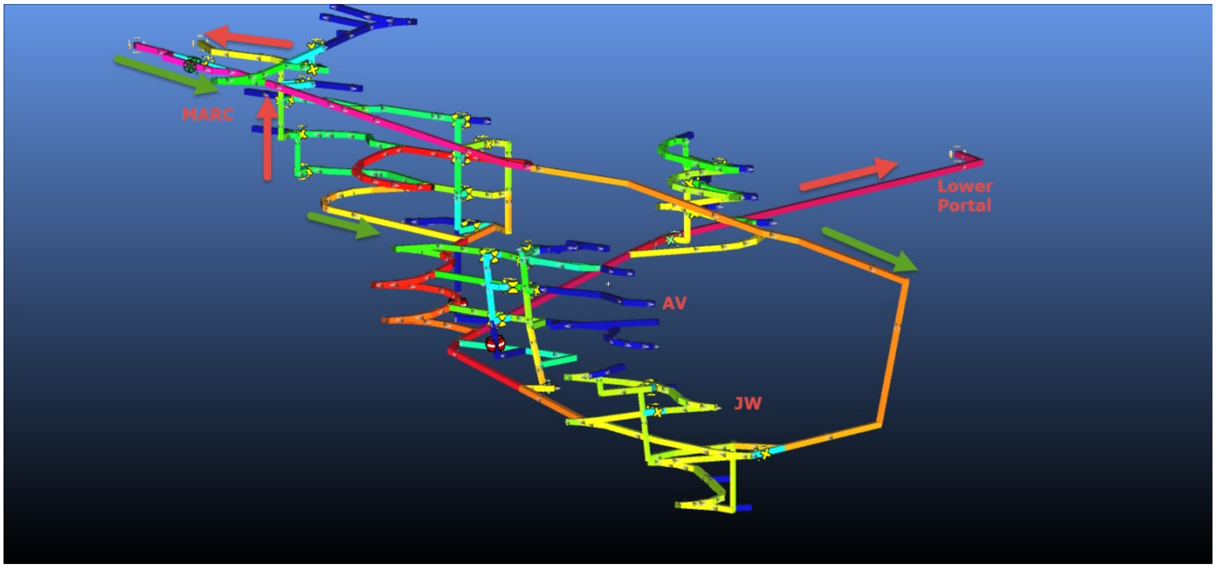
Once the connection is made via a vent drift from the Marc zone footwall, development to the second upper portal at the end of the pre-production phase the mine air heating, catenary, fans, and vent doors will be installed at the top of the exploration portal.

Designed as a ‘push’ system with ducting and fans running on the portal catenary overhead and through the automatic opening airlock doors, the fans will pull air through the direct propane fired mine air heaters located outside the portal and push into the mine.

Booster fans will be used on the ventilation raises for the 141 and AV zones. Fresh air would be distributed from the existing portal via the mine ramp, level development and eight ventilation raises. The second upper portal and lower portal will serve as exhausts.

Ventsim® ventilation software was used to estimate power requirements for the ventilation network (Figure 16-16). The main fans are two 1.9 m diameter 45kW fans, each fan delivering 61.4 m^3/sec (130 kcfm).

Figure 16-16: Ventilation Model in Ventsim Software



Source: JDS (2017)

16.7.2 Mine Air Heating

Mine air will be heated to + 1.5 °C by a direct-fired propane heater located outside the exploration portal in line with the ventilation fan. Heating calculations were based on average monthly temperatures collected at the mine site from October 2015 to September 2016. It was calculated that 3.6 M ft^3 of propane would be required per year for mine air heating.

16.7.3 Service Water Supply

Service water for drilling, dust control, washing, dewatering pump gland water, and fire suppression is sourced from a sump at the top of the underground workings and distributed in 76 mm diameter steel piping. Due to the high inflow of water in the underground workings, no external source of water is required to support peak production equipment demands. Water requirements are expected to peak at 31 m^3/hr. A 58 hp submersible pump will be installed at the 1859 m elevation sump and pump station with a 76 mm discharge line purposed for mine supply water.

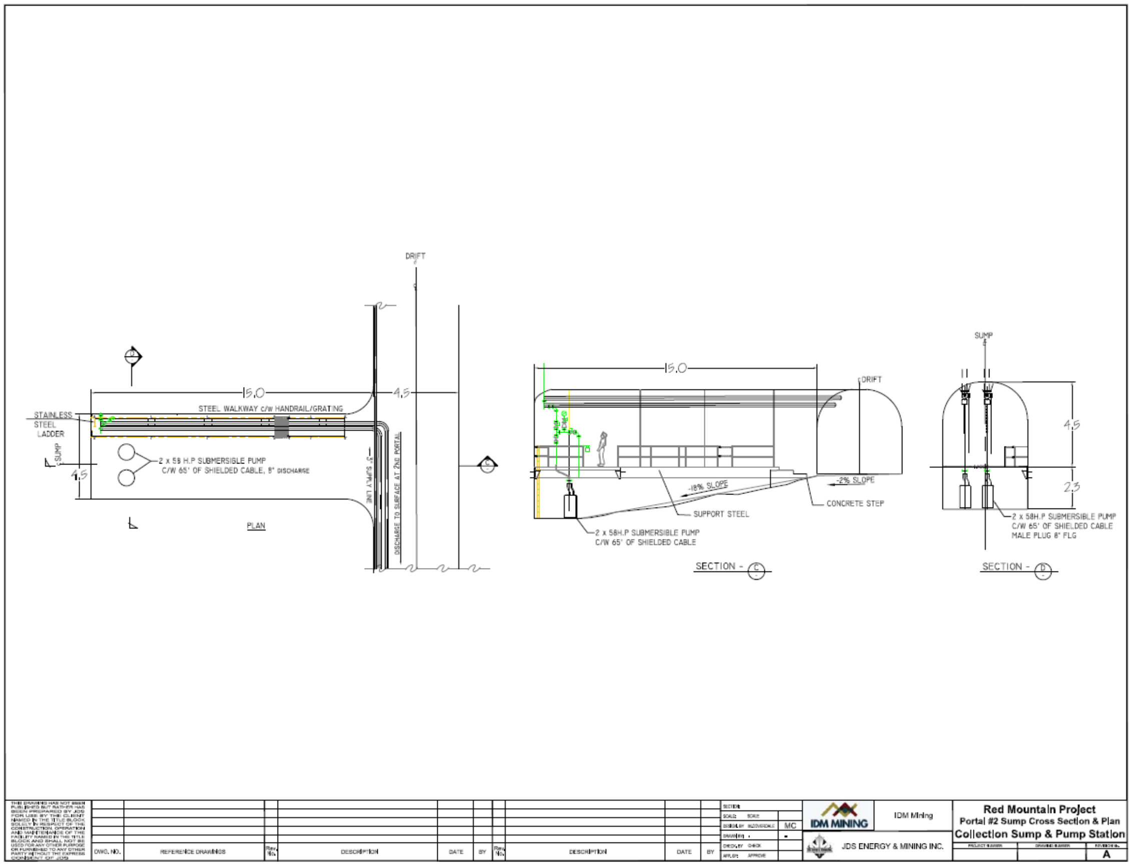
16.7.4 Dewatering

Groundwater inflows into the mine will vary throughout the year. Increased flow rates can be expected during the snowmelt in spring. Peak flowrates were modelled and estimated at 417 m^3/hr and a minimum flowrate of 42 m^3/hr 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 footwall drive, production/development faces, and main dewatering sumps. 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; Figure 16-17 depicts the general mine dewatering sump design.

Figure 16-17: Collection Sump & Pump Station Design



Source: JDS (2017)

A total of five pumping stations are to be installed during the Red Mountain mine life. The location of each pumping station is relevant to the deepest development for years' -1, 1, 2, and 3; development does not mine deeper in years 4 through 6.

Pump setup #1 is located at 1859 m elevation, situated near the existing portal. The pump station consists of two x 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 1,752 masl, situated at the deepest part of development driven in year -1. The pump station is equipped with two x 188 kW centrifugal pumps, each with their own 200 mm discharge lines. Pump setup #2 is designed to pump water to pump setup #1.

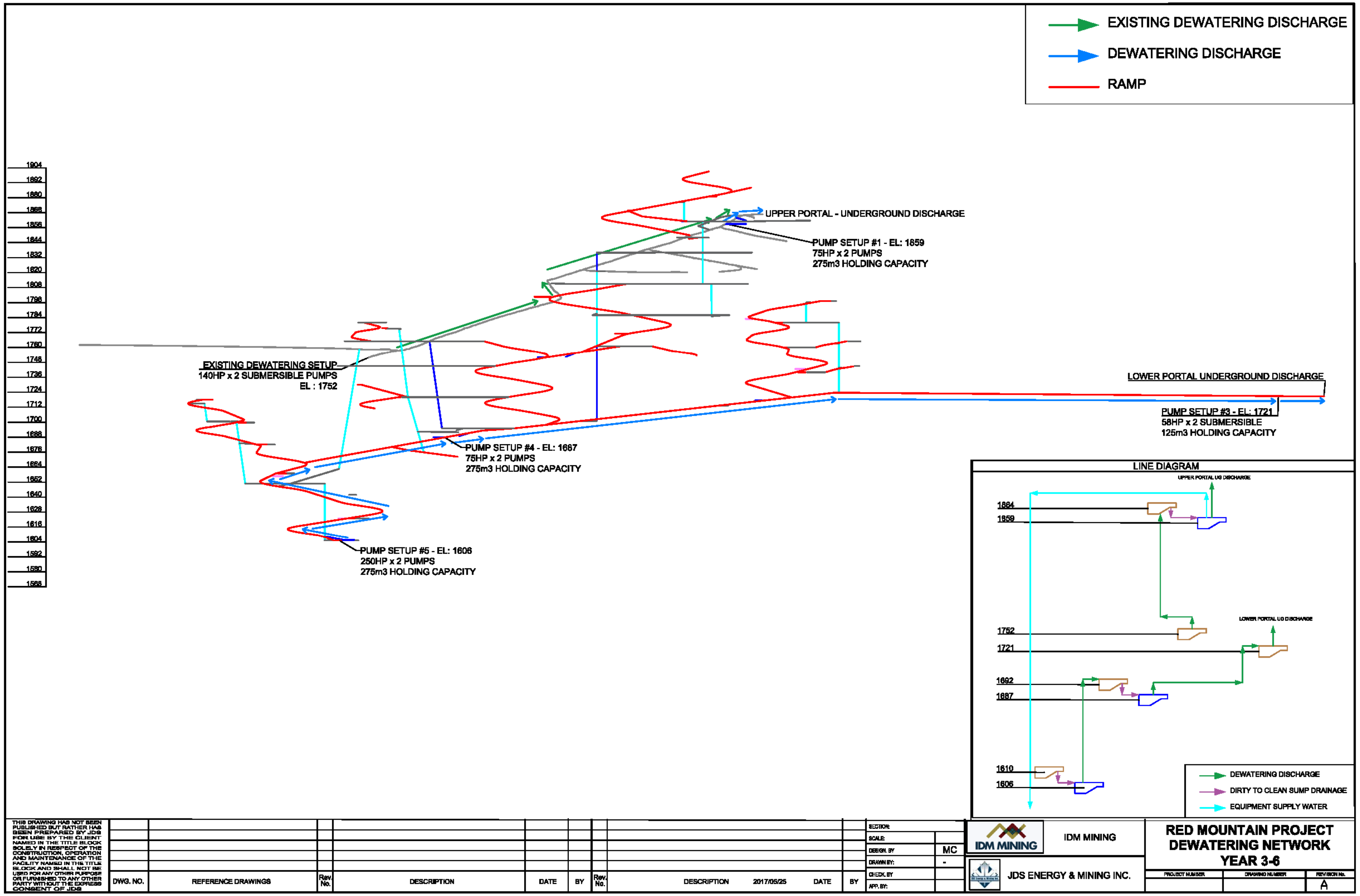
Pump setup #3 is installed during year 1 when the lower portal development reaches 1,721 masl. The sump is situated near the portal entrance and is equipped with two x 58 hp submersible pumps each with their own 200 mm discharge line.

Pump setup #4 is located at 1,687 masl near the deepest part of development driven in year 2. The pump station is setup with two x 56 kW centrifugal pumps each equipped with their own 200 mm discharge lines. Water is pumped from pump setup #4 to pump setup #3 at the second portal before reaching surface.

Pump setup #5 is located at 1606 m near the deepest part of development during the Red Mountain mine life. This pump station consists of two x 250 hp centrifugal pumps with two x 200 mm discharge lines. Water is pumped from pump setup #5 to pump setup #4. Figure 16-18 shows an example 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 16-18: Dewatering Single Line Diagram for Years 3 to 6



Source: JDS (2017)

16.7.5 Electrical Distribution

The Red Mountain underground mine will be fed by a 25 kV overhead powerline from the substation at the plant site. The mine power distribution single line diagram is shown in Figure 16-19.

The electrical distribution setup is similar at both the upper and lower portals. There will be a tap off at each portal electrical room (ER-2 and ER-3) to a 25kV to 4160V transformer. A modular electrical room housing 5 kV switchgear (with feeder protection and ground fault detection and protection relays) will be located outside each portal. The switchgear will include a generator-bus breaker and tie-breaker for the event when an outage occurs and the generator will provide backup power supply to the underground loads. The 4160 V switchgear will feed a 2500 kVA, 4160 V to 600 V transformer for general upper portal loads such as ventilation fans, dewatering pumps and the CRF plant. A second feeder will supply the 4160 V, 200A underground field splitter and third feeder will be installed as a spare.

Underground electrical power requirements are summarized in Table 16-3

Table 16-3: UG LOM Power Requirements

Year	Total Connected Load (kW)	Total Utilized Load (kW)
Year -1	1,790	460
Year 1	2,293	836
Year 2	2,371	774
Year 3	2,371	837
Year 4	2,371	829
Year 5	2,196	780
Year 6	2,196	778

Source: JDS (2017)

16.7.5.1 Surface Infrastructure

A modular electrical room housing 5 kV switchgear (with feeder protection and ground fault detection and protection relays) will be located outside each mine portal. The switchgear will include a generator-bus breaker and tie-breaker for the event when an outage occurs and the generator will provide backup power supply to the underground loads.

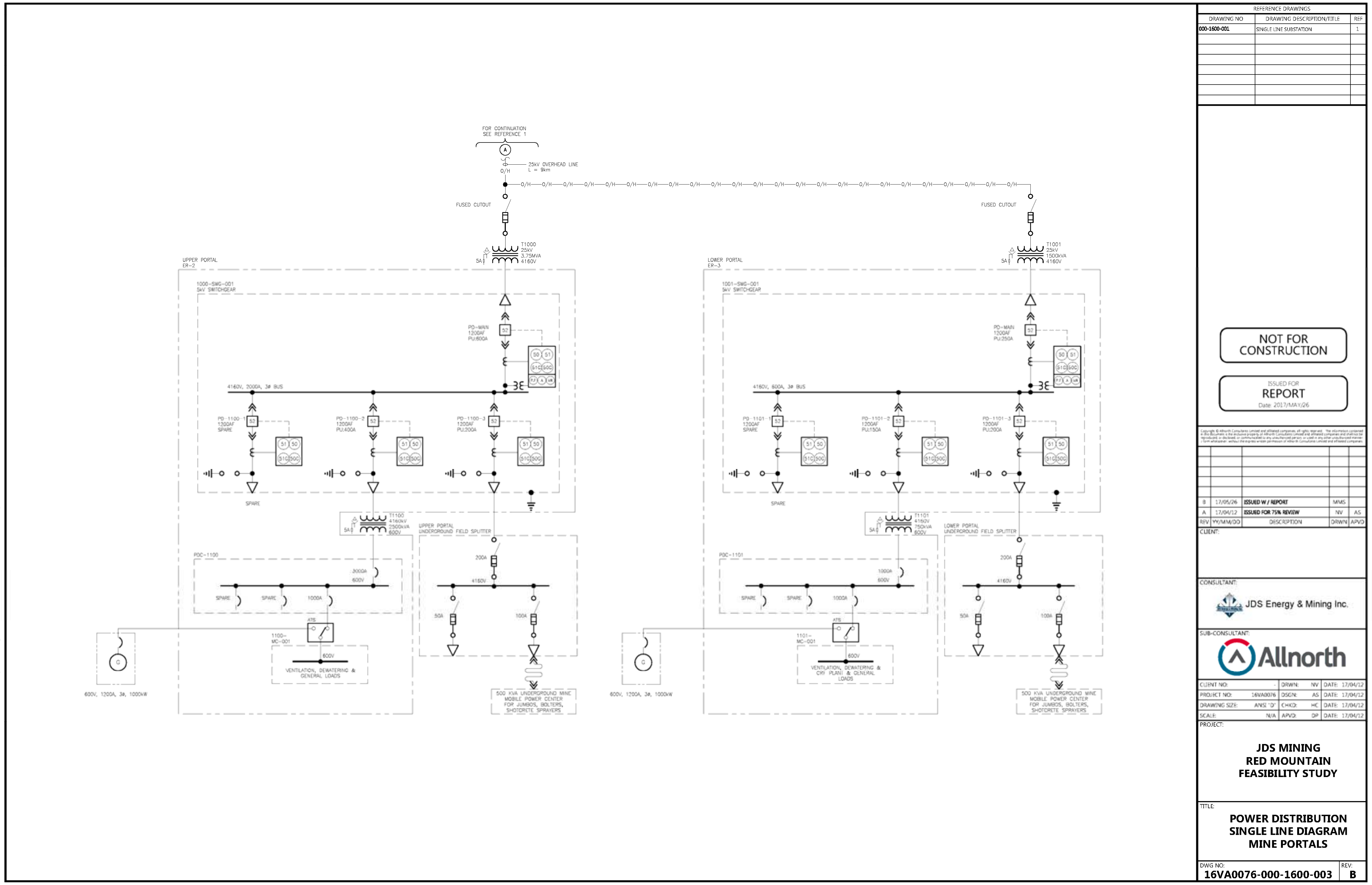
A 1 MW, 600 V standby rated generator and associated control equipment will also be located outside each mine portal. These two 1 MW generators will be used for temporary power during the entire pre-production development of the underground mine.

Temporary generator power is required for year -1 and will be used for backup power supply during operations. The backup generators will supply power to the underground area to ensure ventilation, emergency lighting, sump pumps, and refuge chambers remain operational in a power outage event. These generators will also keep other essential systems (controls, communications, etc.) operational.

16.7.5.2 Underground Infrastructure

Two, 500 KVA, 4160 V-600 V portable Mobile Power Centers (MPC) will be located underground to power mining equipment. The MPC's will be sized to handle the largest anticipated load and can be easily relocated, as required, by the mining sequence. All required power feeder cables will be routed through the portal declines and cross-cut development sections via permanently installed overhead electrical cables. The mining equipment feeder cables will have quick-disconnect type cable connecters installed to the MPCs. This allows for dynamic and rapid movement of MPCs from one area to another which minimizes labour hours.

Figure 16-19: UG Power Distribution Single Line Diagram



Source: Allnorth (2017)

16.7.6 Mine Communications

An underground network with wireless communications will be installed, including fleet and personnel tracking via cap lamps and WiFi tags. Mobile equipment operators, light vehicles, and supervisors will be equipped with hand-held radios for communication.

16.7.7 Compressed Air

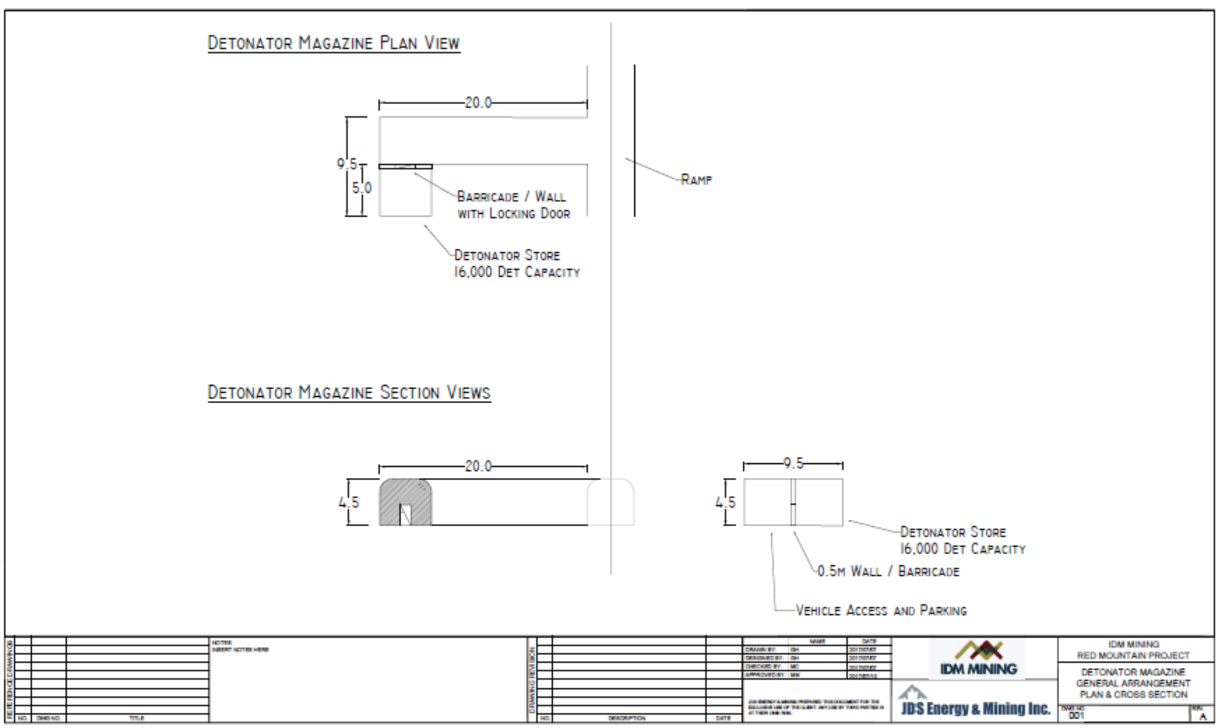
The larger mining equipment selected at Red Mountain have built-in air compressors and do not need an overall mine-wide compressed air system. However, compressed air will be required by certain mining activities and for small scale drilling activities with airlegs. Location specific demands for compressed air will be met using portable compressors which can be relocated to meet requirements.

16.7.8 Explosives and Detonator Storage

Explosives will be stored underground in permanent magazines, while detonation supplies (NONEL, electrical caps, detonating cords, etc.) will be stored in a separate underground magazine. Underground powder and cap magazines will be on the main ramp at elevation 124 m and 119 m, a distance of 35 m from each other. Day boxes will be used as temporary storage for daily explosive consumption. Additional storage of explosives on surface will be required during the first two years.

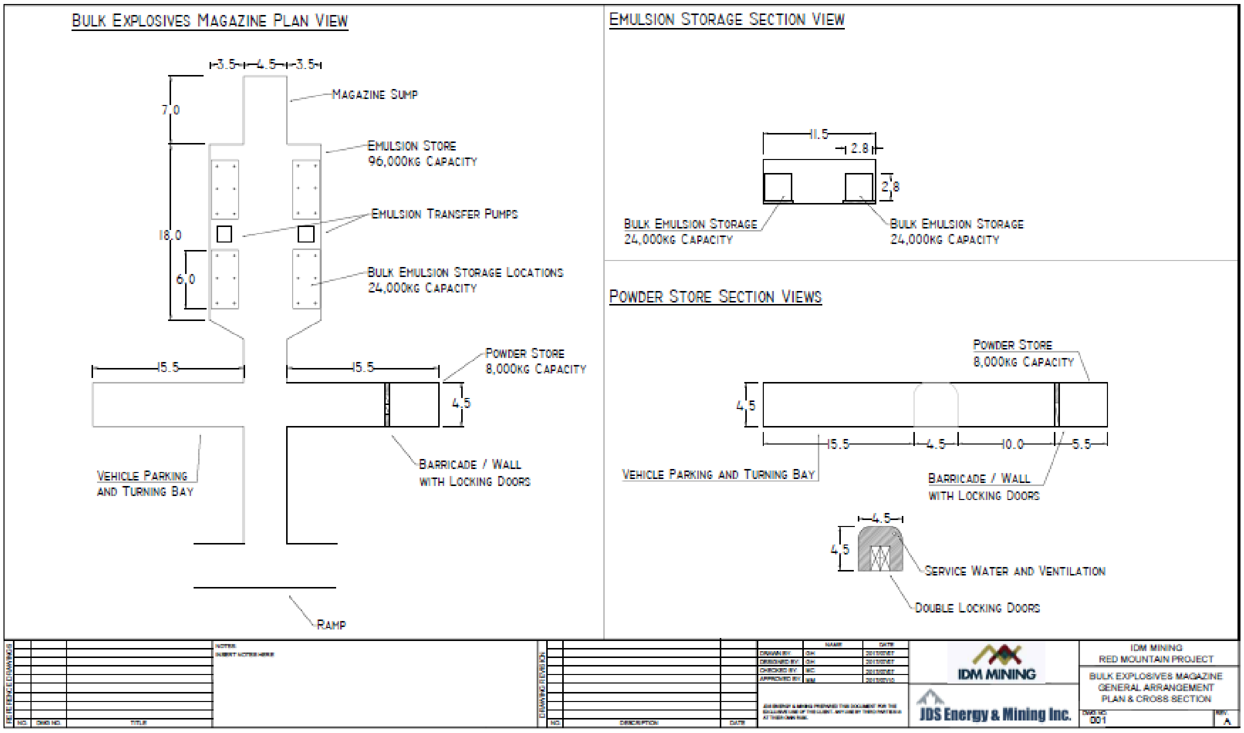
The design and layout of the detonator magazine and bulk explosives magazine are shown in Figures 16-20 and 16-21, respectively.

Figure 16-20: Detonator Magazine



Source: JDS (2017)

Figure 16-21: Bulk Explosives Magazine



Source: JDS (2017)

16.7.9 Fuel Storage and Distribution

Haul trucks and auxiliary mobile equipment would be re-fueled on surface at a fuel station setup at the portal laydown area. Drilling equipment and LHDs would be re-fueled underground with the fuel/lube truck.

16.7.10 Mobile Equipment Maintenance

Mobile equipment will be maintained in the surface shop located outside the upper portal. A mechanics truck will be used to perform minor repairs and service issues underground.

A maintenance supervisor will provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. The supervisor will also provide training for the maintenance workforce. A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability.

16.7.11 Mine Safety

Self-contained portable refuge stations will be provided in the main underground work areas. The refuge chambers are designed to be equipped with air scrubber, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers will be capable of being sealed to prevent the entry of gases. The portable refuge chambers will be moved to the new locations as the working areas advance.

Fire extinguishers will be provided and maintained in accordance with regulations and best practice. Locations include refuge stations, electrical substations, pump stations, fueling areas, explosives magazines, and other strategic areas. Every vehicle would carry at least one fire extinguisher, the correct size and type would be specific to the type of vehicle. It is recommended that underground heavy equipment is equipped with automatic fire suppression systems.

A fully trained and equipped mine rescue team is essential for the safe operation of any mine. The rescue team would be trained for surface and underground emergencies.

A stench gas system will be installed on the ventilation system and would be triggered to alert underground personnel in the event of an emergency.

16.8 Unit Operations

16.8.1 Development & Production Drilling

Drilling activities will be undertaken by the following equipment:

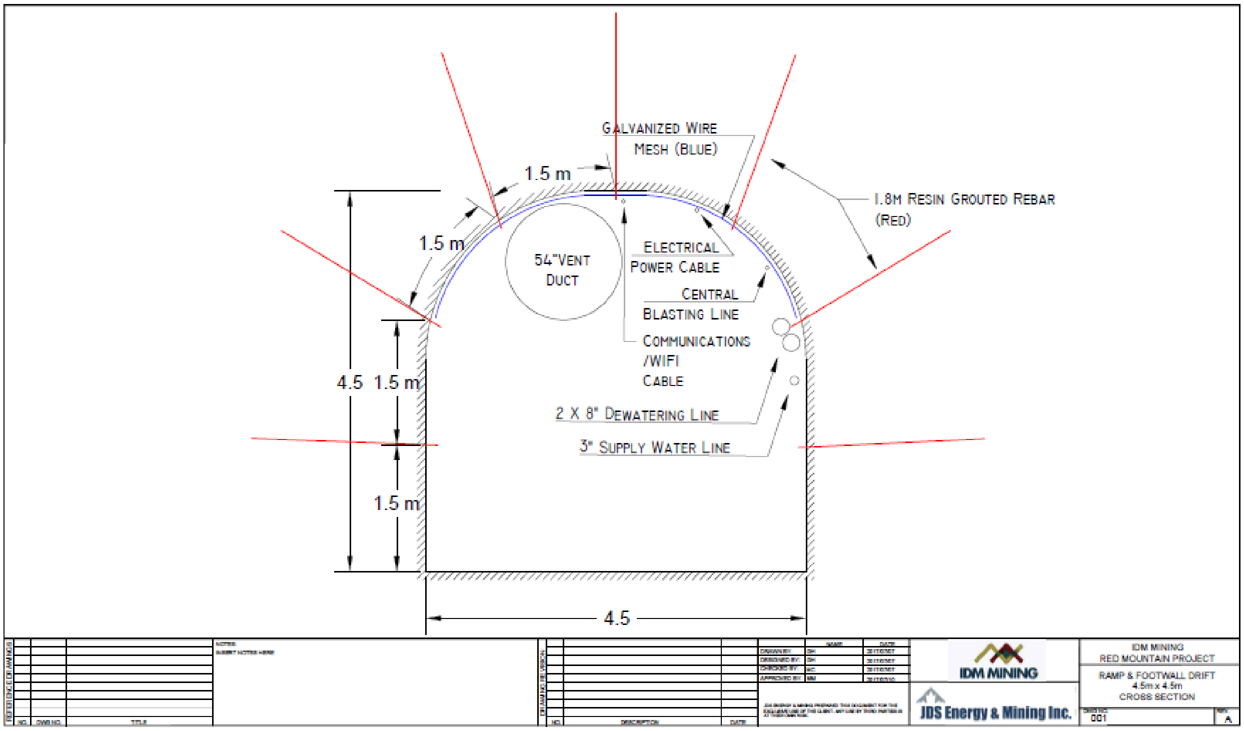
- Twin boom jumbo;
- Longhole drill; and
- Jacklegs/stopers.

Drilling productivities (metre/percussion hour) were built up from first principles by drilling machine type and heading dimensions. Jumbo drilling rates average 78.5 m/hr in a 4.5 m x 4.5 m heading, and longhole drill machines average 11.2 m/hr.

Capital ramps, cross-cuts, footwall drives and other large headings will be developed by two-boom electric jumbo drills. Jumbos will be equipped with 4.88 m (16”) drill steel and will advance an average 7.2 m/d per machine throughout the mine, which equates to approximately 1.5 rounds per day per machine. The smaller 4 m x 4 m headings will be drilled to 4.3 m per round.

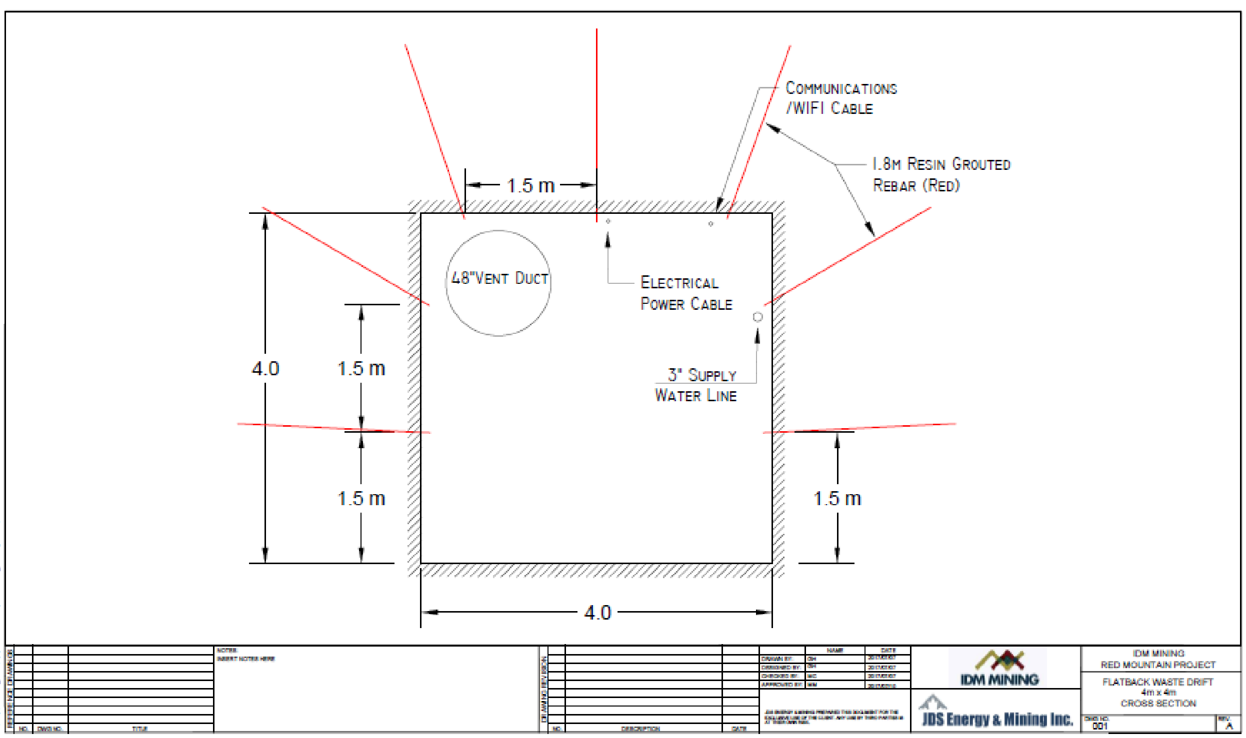
Typical jumbo drill patterns for Red Mountain development and C&F mining are depicted in Figures 16-22 to 16-24.

Figure 16-22: 4.5 m x 4.5 m Ramp Profile



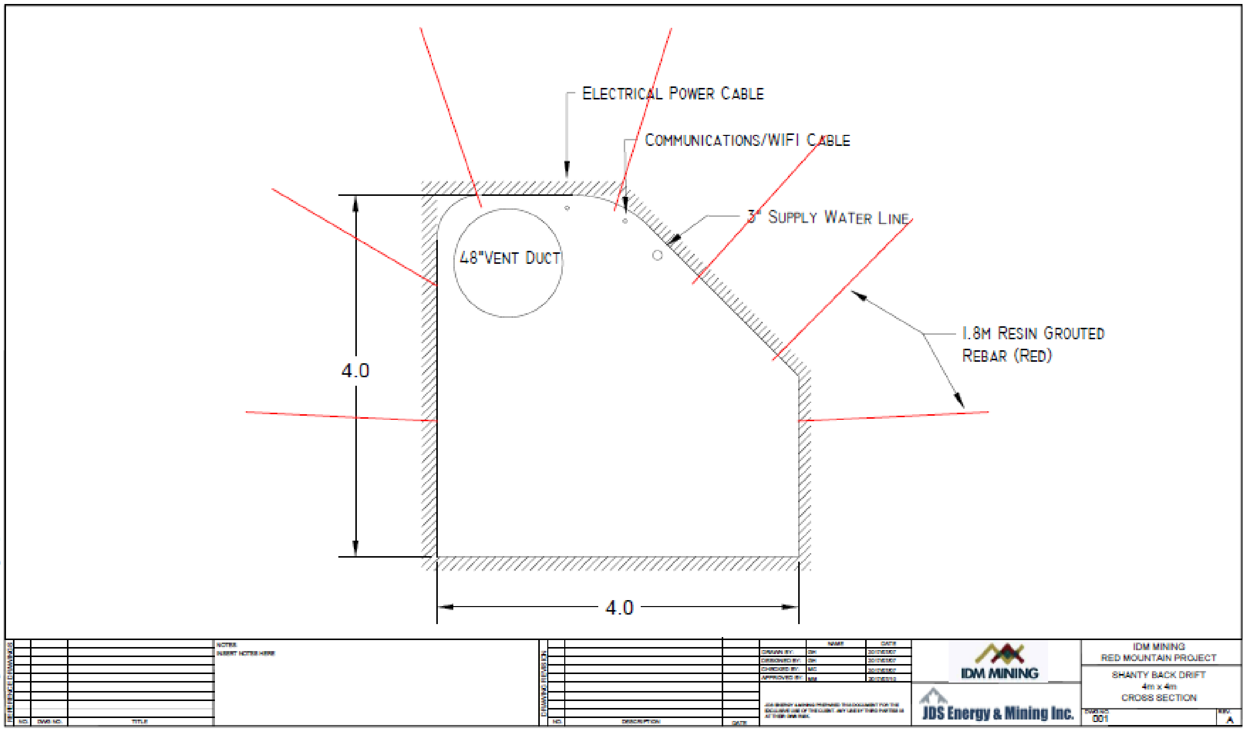
Source: JDS (2017)

Figure 16-23: 4 m x 4 m Waste Drift Profile



Source: JDS (2017)

Figure 16-24: 4 m x 4 m Shanty Profile



Source: JDS (2017)

Longhole drilling of mainly down holes with 89 mm diameter is planned at sublevel spacing of 15 m to 25 m. Some stoping would include drilling of upholes, being 20 m in length to ensure emulsion can be held in the hole.

16.8.2 Blasting

Bulk emulsion will be used for production blasting and development rounds. Boosters, primers, detonators, detonation cord, and other ancillary blasting supplies will also be utilized. Smooth blasting techniques may be used as required in headings, with the use of trim powder/bulk emulsion for loading the perimeter holes.

For longhole mining, slots would be developed by drop raise prior to blasting any of the production rings. Emulsion would be used for longhole blasting.

During the pre-production period, blasting in the development headings will be done at any time during the shift when the face is loaded and ready to blast. All personnel underground will be required to exit the workings and be accounted for during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of each shift.

16.8.3 Ground Support

Ground support will be installed in accordance with specifications based on geotechnical analysis provided by SRK for the various rock qualities expected (see Section 16.2). Electric-hydraulic bolters and shotcrete spraying machines will be used. Some ground support may be installed using jackleg drills. Shotcrete will be applied when required as a dry mix, which is mixed at the nozzle by specialized dry shotcrete machines.

Ground support will be installed post-mucking of the blasted drift. No additional development will be commenced in the heading prior to the installation of ground support. At no time will mine workers be under unsupported ground.

Different ground support criteria are recommended for various types of ground conditions, rated from good to poor, and largely associated with the fault zones. Discretion will be made by the development lead as to which ground support is required, with additional review and recommendations provided by the on-site geotechnical engineer.

Regular pull tests will be conducted on-site to ensure adequate installation of resin rebar, split set, and cables bolts are being done. Shotcrete, when required, will also be sampled by use of splatter boards and in-situ coring to be tested for strength and adequacy.

Jackleg and stoper drills will be available and used in areas the bolter cannot access or during times of maintenance.

16.8.4 Mucking

The LHD selected for all mucking activities at Red Mountain has a 6 yd^3 (10t) nominal capacity. For development, LHD's will typically muck a blasted round to a nearby re-muck bay in order to clear the working face prior to ground support installation. Rock temporarily stored in the re-muck is then either trammed to a rock pass or loaded into a haul truck.

Stope ore will be mucked with LHD and either trammed to the rock pass system, direct loaded into a haul truck, or temporarily stored in a re-muck. In longhole stopes once the drawpoint brow is open, mucking via remote will be utilized in order to keep personnel away from open stopes.

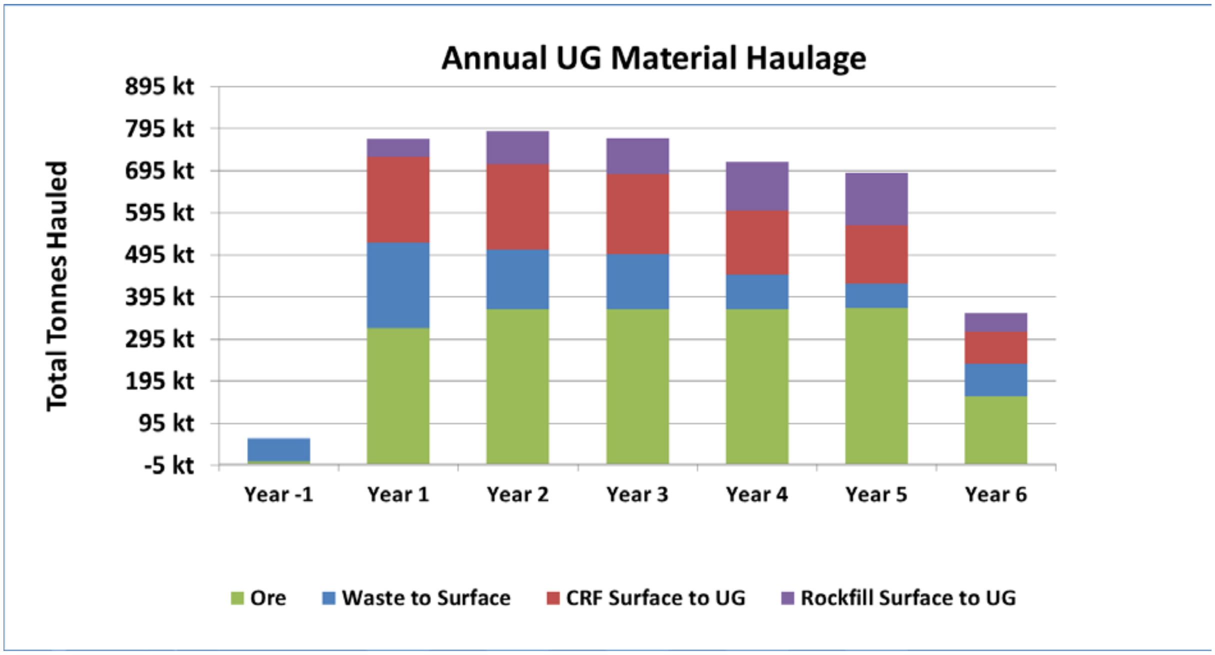
Mucking will be performed by LHD machines for all C&F and waste development drifts. Muck will be trammed to a nearby re-muck drift or directly into haul trucks.

16.8.5 Hauling

Muck will be hauled to surface by 30 t underground haul trucks. Trucks will be loaded at re-muck bays, and muck passes by LHD machines. In most cases, trucks will be restricted to loading at re-muck stations due to the increase back height requirements for LHDs to load over the side of the truck box. Trucks will haul muck to surface, which will then be dumped in ore and waste stockpiles, to be re-handled by surface equipment for transport to the mill site. Muck will be hauled to the upper portal site until the lower haulage ramp is mined at the end of year 1 at which point muck will be hauled and dumped at the lower portal stockpile area.

Figure 16-25 below gives a summary of the material haulage tonnes by activity and year.

Figure 16-25: Annual Underground Haulage Summary

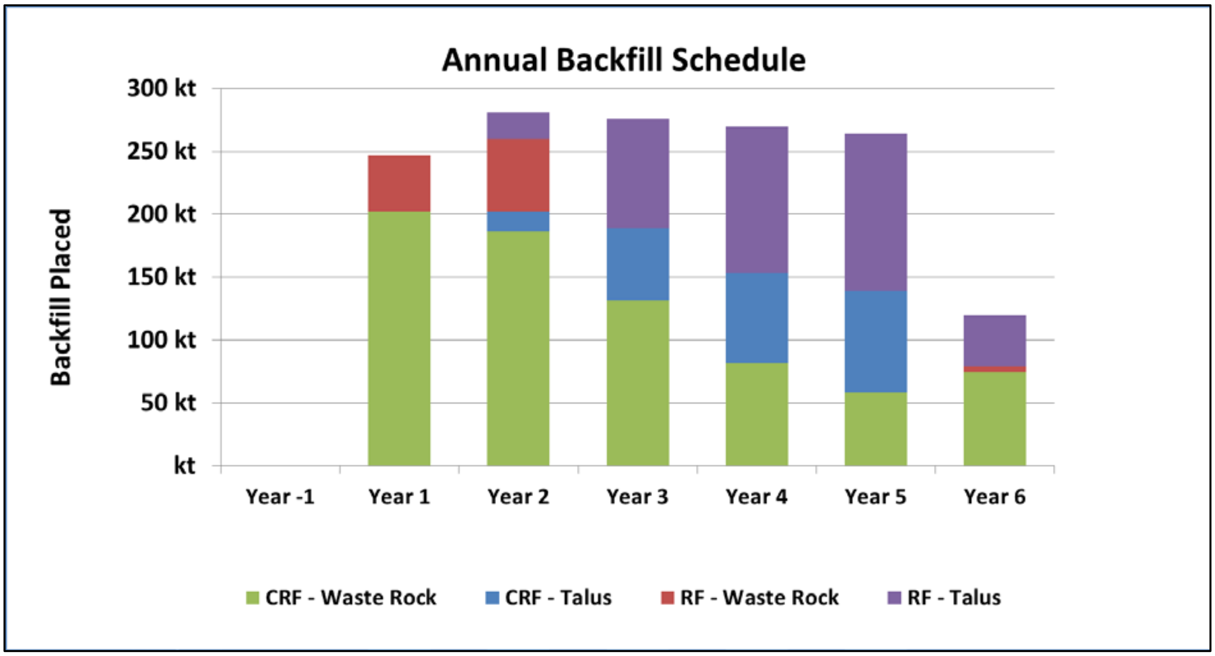


Source: JDS (2017)

16.8.6 Backfill

The principle method of backfilling at Red Mountain will be with CRF comprising of either crushed waste development rock or quarried talus from surface. RF will be used to fill ore voids everywhere where structural fill is not required. Annual backfill placement by fill type is shown on Figure 16-26.

Figure 16-26: Backfill Schedule & Material Source



Source: JDS (2017)

All of the development waste produced would be used as mine backfill. The existing historical waste stockpile at the upper portal is estimated at 90,000 t and would also be consumed as backfill. No development waste remains on surface at the end of the mine life. The remaining backfill deficit is planned to be sourced from the talus quarry which is located close to the lower portal and lower portal laydown/facilities area. Approximately 617 kt of talus material will be required.

RF will be dosed with quicklime (CaO) for geochemical purposes at a rate of 2 kg/t placed for mined waste, and 3 kg/t placed for talus material.

CRF consists of crushed rock mixed with cement slurry, a cement binder content of 5% was assumed for cost purposes. CRF would be produced at the CRF batch plant as outlined in Section 16.8.7. The CRF plant will be located at the upper portal until such a time that talus material is being mined in year 2 which is when the waste stockpiles would be depleted.

16.8.7 CRF Plant

MineFill Services Inc. (MFS) carried out a CRF system design for Red Mountain Mine. The design and plant design is summarized in MineFill's "*Feasibility, Cemented Aggregate Fill System Red Mountain Mine, British Columbia"* report dated June 27, 2017.

The backfill plant needs to be moved after the first year from near the upper portal to the lower portal. Therefore, the plant is designed to be modular for easy relocation.

To develop a CRF recipe, the laboratory testing was carried out at Lafarge lab in Seattle, Washington, using development waste rock and talus materials collected at the mine site. The backfill plant has a production rate that will produce sufficient CRF to fill the voids created underground with a catch-up allowance. The assumptions and data used in the design are summarized in Table 16-4.

Table 16-4: CRF Plant Process Design Criteria

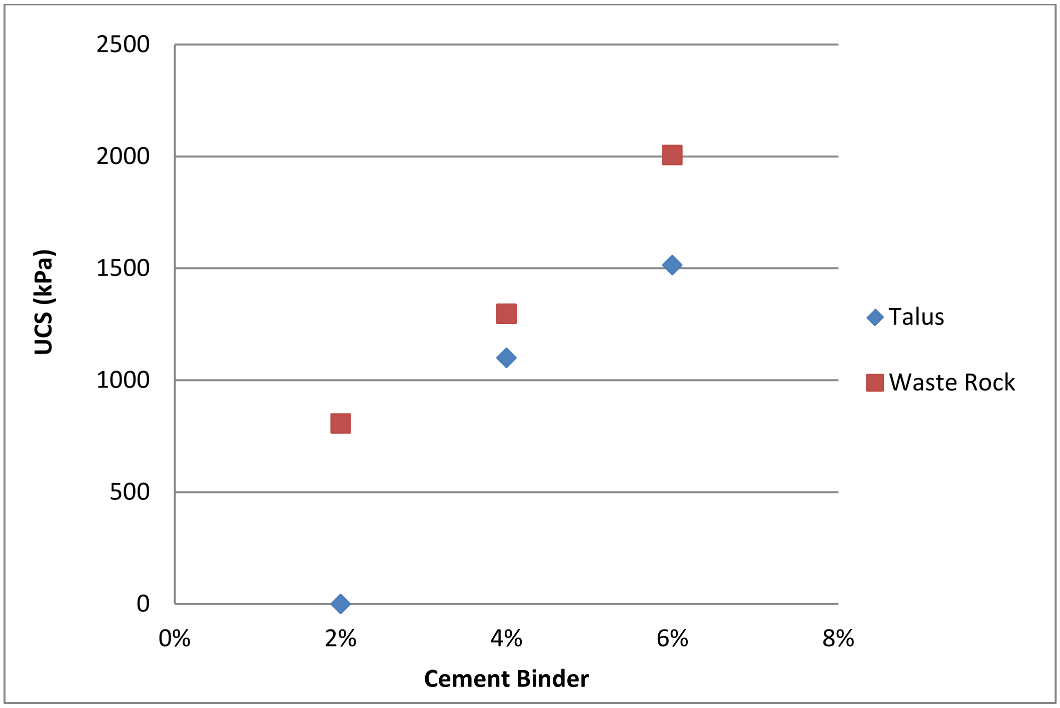
Parameter	Unit	Value
Nominal Mine Production	t/d	1,045
Maximum Mine Production	t/d	1,555
Operating Days	day/year	350
Mechanical Availability	%	80
System Availability	%	75
Overall Stopes filled with CRF	%	75
Ore Density	t/m³	3.00
Waste Rock Density	t/m³	2.82
Binder Density	t/m³	3.00
Binder Dosage	Wt %	5
Maximum Annual CRF Required (Year 1)	m³	62,000
Minimum Annual CRF Required (Year 5)	m³	3,000

Source: MFS (2017)

16.8.7.1 Test Results

The cement binder dose was varied at 2%, 4%, and 6%. At 4% binder, Talus, and waste rock sample developed more than twice the required strength after 28 days. Note that the talus sample with 2% binder did not bind properly becoming unsuitable for UCS testing. Considering the variability during the operation such as the variation in particle size distribution (PSD) of the aggregate and snow affecting the moisture content, 4% binder is the recommended binder dose for making cemented aggregate fill. Test results are shown in Figure 16-27.

Figure 16-27: Unconfined Compressive Strength vs. Cement Binder Content



Source: MFS (2017)

There are opportunities to fine-tune the CRF recipe such as blending talus and development waste rock. Collecting data on the 28-day UCS test with different fine to coarse ratio at 3%, 4%, and 5% allows a better control of the CRF quality. Also, technically 2% binder is sufficient for making CRF with development waste rock according to the data.

While these approaches may lower the binder cost in the CRF, blending the two types of rocks requires proper planning and potentially additional space for the stockpiles. Furthermore, using different binder contents for different rocks would require the cement dose setting to be manually adjusted. Despite that these are opportunities for reducing operating costs, there is a potential of producing inconsistent CRF due to human error. With further testing and proper training, there is an opportunity to reduce the binder consumption.

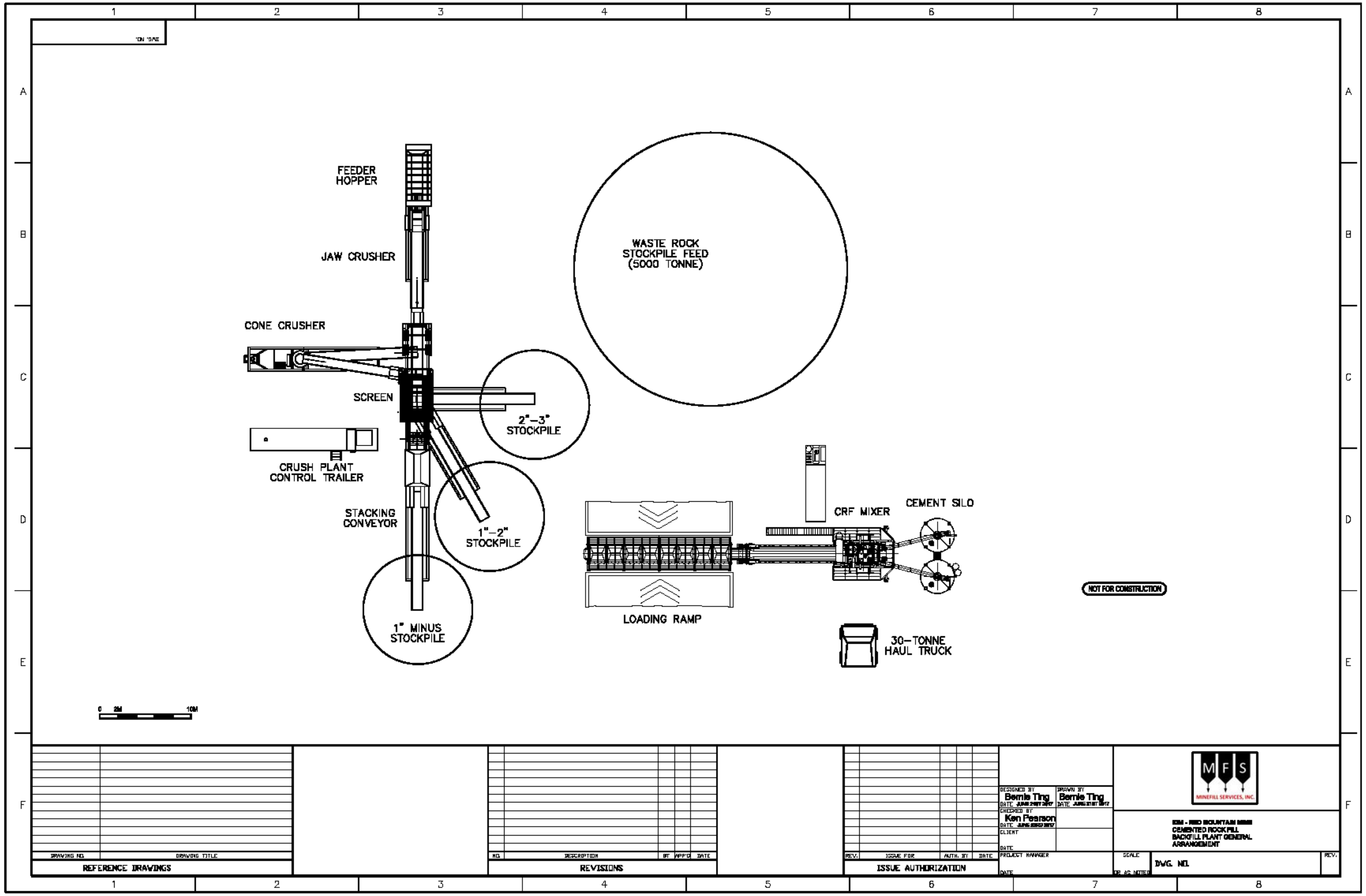
16.8.7.2 System Description

The CRF system at Red Mountain Mine can be divided into crushing and mixing. The crushing operation uses a jaw crusher, a cone crusher, and a screen deck. An FEL feeds the crusher with development waste rock and locally sourced talus. The rock is crushed and sized into three piles: a 25 mm minus pile, a 25 mm to 51 mm pile and a 51 mm to 76” pile.

Taking from the crushed pile, the aggregate is fed to a hopper of the batch plant by a front end loader (FEL). A belt conveyor will feed the aggregate into the mixer. In parallel, the cement binder is weighed and fed into the mixer along with water at 1:1 weight ratio. The twin-shaft batch mixer blends the material for up to three minutes from the start of the new batch. Then the CRF is discharged into the haul truck and a new batch begins subsequently. Three batches are made and loaded into a 30 t haul truck. The truck carries the CRF to the designated stope. The layout is shown in Figure 16-28.

At the end of each shift, the batch mixer is cleaned using the built-in high-pressure wash, and the station will be cleaned by an operator.

Figure 16-28: CRF Plant Layout



Source: MFS (2017)

16.8.8 Mine Equipment

All underground mine equipment required to meet the life of mine plan is summarized in Table 16-5.

Major equipment productivities are as follows:

- Jumbo drilling: 86 m/hour;
- Longhole drilling: 11 m/hour;
- Bolter: 8-10 bolts/hour;
- Mucking: 99 t/hour; and
- Trucking: 45 t/hour.

Table 16-5: Underground Mobile Equipment Fleet

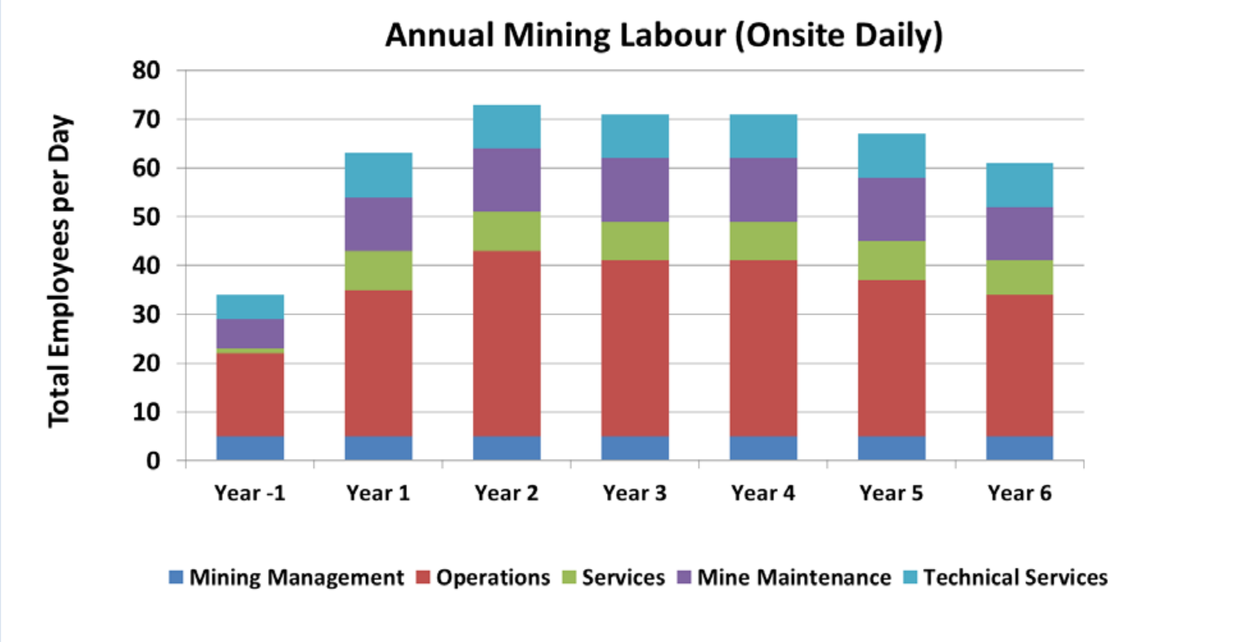
Description	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
30 t Haul Truck - Sandvik - TH430	1	5	5	4	4	4	4
LHD – 10 t / 6 yd - Sandvik LH 410	1	2	2	2	2	2	2
2 Boom Jumbo - Sandvik DD321-40	1	2	2	2	2	2	2
Bolter - Sandvik DS311	1	2	2	2	2	2	2
Explosives Truck - AC3 Emulsion Charger	1	2	2	2	2	2	2
Longhole Drill - Sandvik DL311	-	1	1	1	1	1	1
Scissor Lift - Walden SLX5000	1	1	1	1	1	1	1
Shotcrete Sprayer - Aliva 246	1	2	2	2	2	2	2
Personnel Carrier - Maclean PC3	1	2	2	2	2	2	2
Lube Service Truck - Maclean FL-3	-	1	1	1	1	1	1
Boom Truck - Maclean BT-3	-	1	1	1	1	1	1
Motor Grader - CAT12K	-	1	1	1	1	1	1
Utility Vehicle - Etrac 1300	-	1	1	1	1	1	1
Backhoe with Rockbreaker - JCB 3cx Compact	-	1	1	1	1	1	1
Telehandler - JCB 535-140	1	1	1	1	1	1	1
Mechanics Truck - Toyota MT	1	2	2	2	2	2	2
Toyota PC	2	4	4	4	4	4	4

Source: JDS (2017)

16.9 Mine Personnel

Life of Mine (LOM) personnel requirements are summarized in Figure 16-29.

Figure 16-29: Mine Labour Requirements



Source: JDS (2017)

Tables 16-6 to 16-10 show minimum and maximum numbers of personnel planned for the LOM for mine management, operations, mine services, mine maintenance, and technical services, respectively. These numbers don't include the milling plant personnel.

Table 16-6: Mine Management Personnel

Position	Average	Maximum	Pre-Production
Mine Superintendent	1	1	1
Maintenance Manager	1	1	1
Technical Services Manager	1	1	1
Mine Foreman	1	1	1
Mine Clerk	1	1	1
Total	**5**	**5**	**5**

Source: JDS (2017)

Table 16-7: Mine Operations (Production) Personnel

Position	Average	Maximum	Pre-Production
Mine Supervisor / Shift Boss	4	4	4
Safety / Trainer / Mine Rescue	2	2	2
Blaster	7	8	4
Development Services	6	6	4
Jumbo Operator	7	8	4
Production Drill Operator	3	4	-
LHD Operator	7	8	4
Truck Driver	15	20	4
Ground Support / Bolter / Shotcrete	7	8	4
Mine Helper	3	4	-
Utility Vehicle Operator / Nipper	4	4	4
Total	**65**	**76**	**34**

Source: JDS (2017)

Table 16-8: Mining Operations (Services) Personnel

Position	Average	Maximum	Pre-Production
CRF Plant Operators	3	4	-
Backfill Miner	3	4	-
Electrician	4	4	2
Utility Vehicle Operator/Nipper	3	4	-
Total	**13**	**16**	**2**

Source: JDS (2017)

Table 16-9: Mine Maintenance Personnel

Position	Average	Maximum	Pre-Production
Mine Maintenance Supervisor	1	1	1
Maintenance Planner	1	1	1
HD Mechanic	7	8	4
Mechanic Helper	3	4	-
Welder	2	2	-
Electric Hydraulic Mechanic	7	8	4
Total	**21**	**24**	**10**

Source: JDS (2017)

Table 16-10: Mine Technical Services Personnel

Position	Average	Maximum	Pre-Production
Senior Mine Engineer	1	1	1
Geotechnical Engineer / Backfill	1	1	-
Chief Geologist	1	1	1
Mine Ventilation/Project Engineer	1	1	-
Surveyor/Miner Technician	2	2	2
Production Geologist	1	1	-
Geotechnical Technician/Sampler	1	1	-
Senior Mine Technician	1	1	1
Total	**9**	**9**	**5**

Source: JDS (2017)

16.10 Mine Production Schedule

The following factors were considered in the estimation of the underground mine production rate:

- Mining inventory tonnage and grade;
- Geometry of the mineralized zones;
- Amount of required development;
- Stope productivities; and
- Sequence of mining and stope availability.

The underground mine production rate, at an average of 1,045 t/d, is considered appropriate due to the high degree of mechanization and potential high productivities of the selected stoping methods and available working faces and/or stopes. Based on the extent and thickness, the mineralized zones, and ability to have production from different sublevels, JDS considers the underground production rate to be achievable.

16.10.1 Mine Development

Mine development is divided into two periods: pre-production development (prior to commercial production) and ongoing development (during commercial production). The objective of preproduction development is to provide an access to higher grade areas of the Marc zone and prepare enough resources to support the mine production rate when access to other zones is being established.

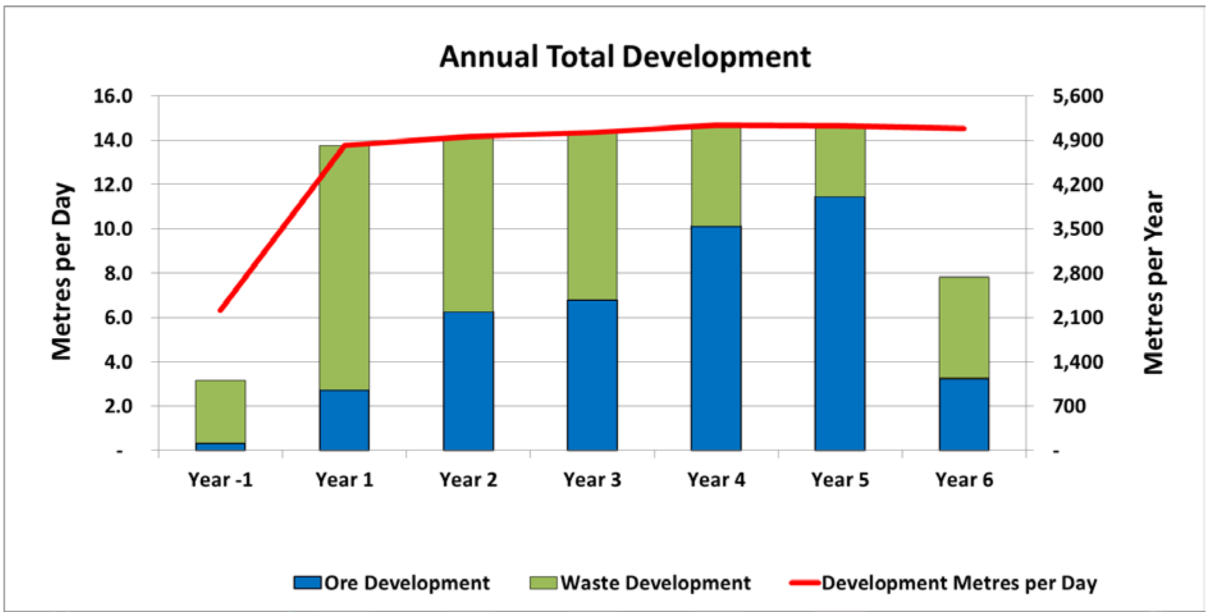
Pre-production development is scheduled to:

- Develop a second upper portal location;
- Develop a decline to the lower portion of the Marc zone;
- Development of stope crosscuts, including ore development;
- Provide ventilation and emergency egress; and
- Install mining services.

During pre-production, mining will begin on the Marc zone ramp; this access ramp is critical to getting early access to the higher-grade ore of the Marc zone and establishing the lower sublevels. Also, the mining crews will start work to excavate the second upper portal, which links to the development of the upper footwall levels of the Marc zone. Total lateral development for the pre-production period is 1,112 m, with only 50 m of Alimak vent raise required.

Once production has started, a second development crew will be concentrating on mining the lower portal and 730 m haulage ramp which connects with the internal ramp close to the 1695 level of the AV zone. This will also provide another ventilation exhaust and assist with the ventilation distribution. Annual lateral development metres and metres per day are shown in Figure 16-30.

Figure 16-30: Annual Development Schedule



Source: JDS (2017)

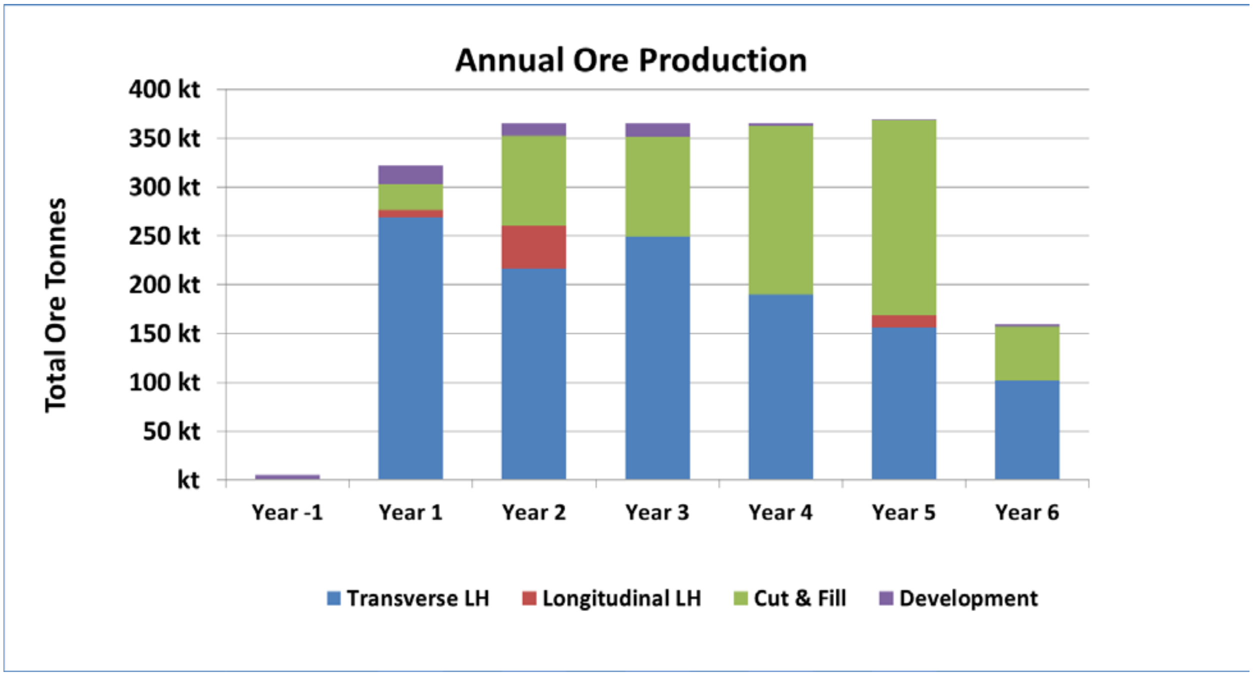
16.10.2 Mine Production

The criteria used for scheduling underground mine production at Red Mountain were as follows:

- Target the mining blocks with higher profitability in the early stages of mine life to improve Project economics;
- An average annual mill feed production rate of 365 kt/yr was scheduled, including ore from development and stopes;
- The mine will operate two 10-hour shifts per day, 350 days per year; and
- Minimize mobile equipment requirements by smoothing ore and waste development as much as possible.

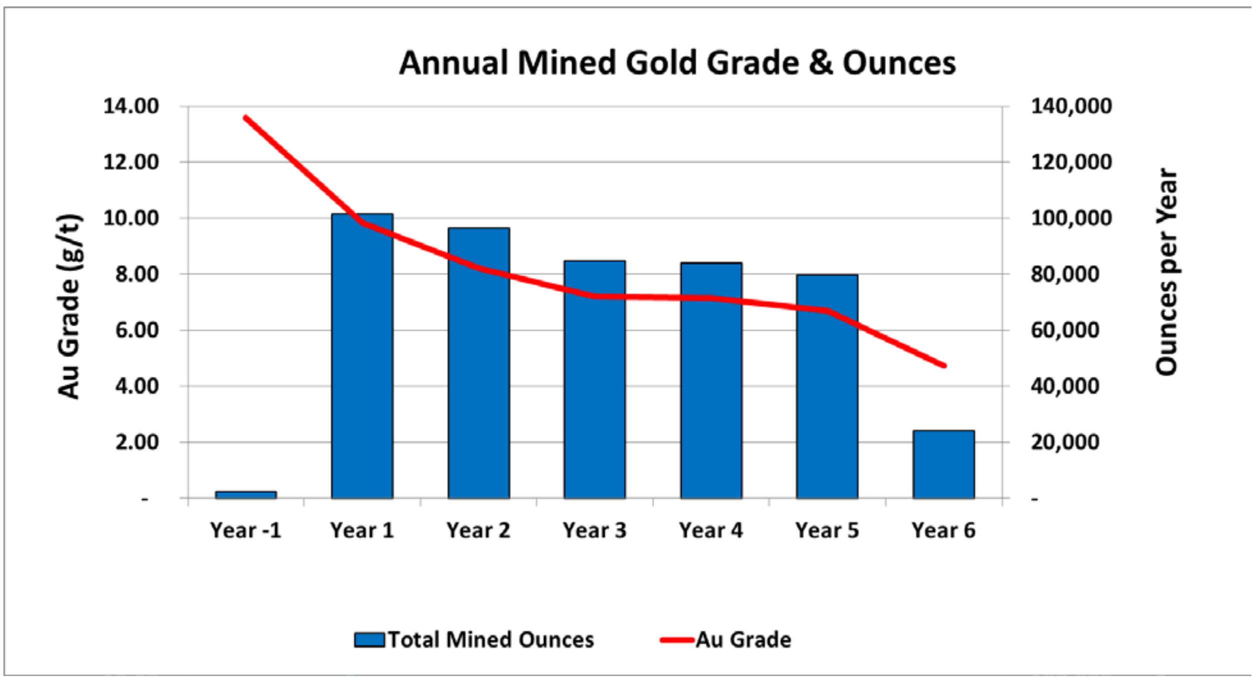
Annual production by mining method is shown in Figure 16-31. Annual gold and silver production is shown in Figures 16-32 to 16-33.

Figure 16-31: Annual Ore Production by Mining Method



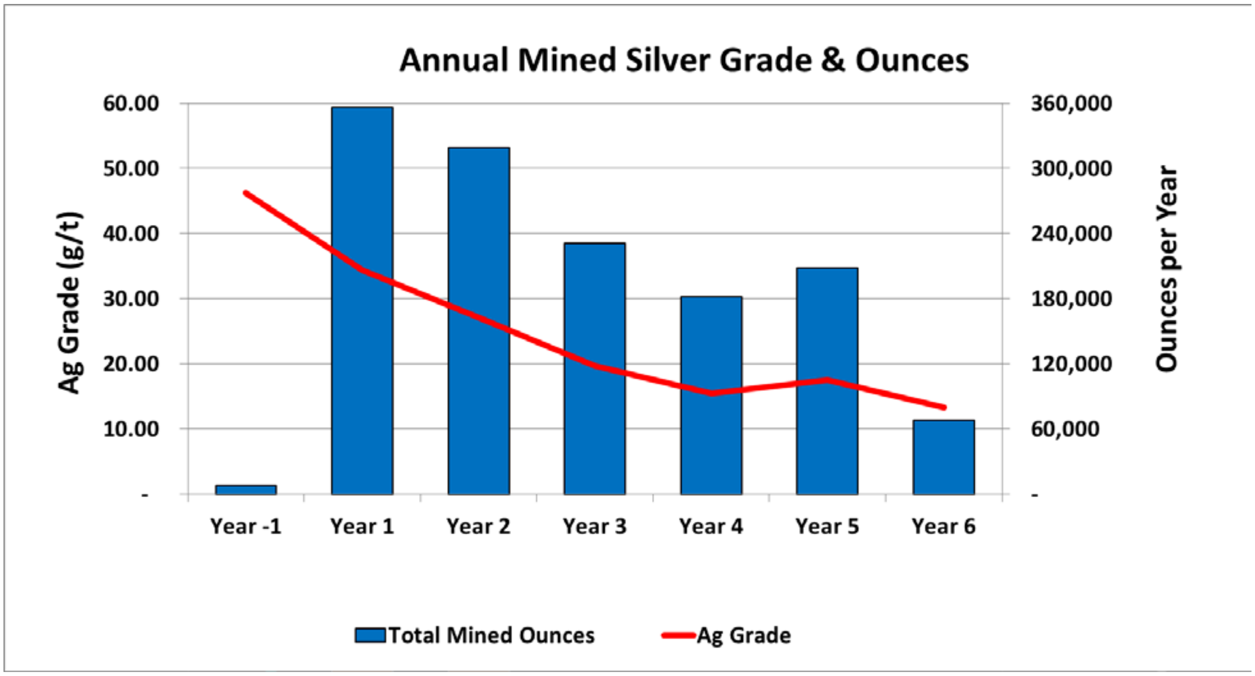
Source: JDS (2017)

Figure 16-32: Annual Mined Gold Ounces and Average Grade



Source: JDS (2017)

Figure 16-33: Annual Mined Silver Ounces & Average Grades



Source: JDS (2017)

Detailed mine planning and scheduling has been done monthly for pre-production and the first two years of the mine life, and then transitioned to annuals for the remainder of the mine life but has been summarized annually in this report. The annual mine production, development, and backfill schedules are provided in Tables 16-11 to 16-13. Ore waste and backfill tonnages have been rounded to the nearest thousand.

Table 16-11: Annual Production Schedule

Description	Unit	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Mined Waste	kt	750	54	204	143	132	81	58	79
Mined Ore	kt	1,953	5	322	366	366	366	369	159
Contained Silver	koz	1,373	8	356	319	231	182	208	68
Silver Grade	g/t	21.86	46.26	34.44	27.17	19.63	15.45	17.54	13.31
Contained Gold	koz	473	2	102	96	85	84	80	24
Gold Grade	g/t	7.53	14.20	10.28	8.56	7.46	7.34	6.94	4.90

Source: JDS (2017)

Table 16-12: Annual Development Schedule

Description	Unit	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Ore Development	km	14.30	0.12	0.95	2.18	2.34	3.54	4.00	1.14
Waste Development	km	14.59	0.99	3.86	2.77	2.65	1.60	1.12	1.59
Total Development	km	28.89	1.11	4.82	4.96	5.01	5.13	5.13	2.73
Lateral Advance Rate	m/d	13.7	6.3	13.8	14.2	14.3	14.7	14.7	14.5
Raise Development	m	675	50	210	164	99	96	-	56

Source: JDS (2017)

Table 16-13: Annual Backfill Schedule

Description	Unit	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Mined Waste to Backfill	kt	675	-	182	142	132	81	58	79
Stockpiled Waste to Backfill	kt	76	-	64	11	-	-	-	-
Historically Stockpiled Waste to Backfill	kt	90	-	-	90	-	-	-	-
Talus Quarry to Backfill	kt	617	-	-	37	144	188	206	41
Total Material Placed as Backfill	kt	1,457	-	246	281	276	270	264	120

Source: JDS (2017)

The Red Mountain schedule was sequenced and optimized using Minemax™ iGantt schedule optimizer software. To produce a balanced schedule, inputs and constraints that represent the design, mining productivities, and unit operations were included in the optimization process. Table 16-14 details the maximum daily development rates used.

Table 16-14: Development Productivities Used for Scheduling

Single Headings	Advance Rate (m/day)
Ramp	6.0
Footwall Drives/Level Access	6.0
Remucks and Sumps	6.0
Lateral Ventilation Access	6.0
Lateral Muck Pass Access	6.0
LH Ore/Waste Drift	5.0
Attack Ramp	5.0
Cut & Fill Drift	4.0
Cut & Fill Shanty	4.0

Source: JDS (2017)

Development and production ramp-up were modelled using initial periods of reduced productivities. Pre-production scheduling guidelines were established to:

- Ensure sufficient development to sustain ore production when plant construction and commissioning will be complete; and
- Include only development that was required for pre-production.

Stope productivity constraints used for scheduling purposes in iGantt can be found in Table 16-15. The ore mining rate was capped at an average of 1,045 t/d and development was constrained to limit the equipment fleet on-site and reduce early capital expenditures. A number of schedule iterations and manual adjustments to the sequence were made in order to produce a robust and realistic schedule.

Table 16-15: Stope Productivities used for Scheduling

Longhole Stope	% by Tonnage	Mining Rate (t/day)
25 m Transverse	64	550
20 m Transverse	23	340
15 m Transverse	74	400
25 m Longitudinal	4	500
20 m Longitudinal	1	400
10 m Longitudinal	<1	300

Source: JDS (2017)

Final results of the iGantt schedule were organized such that physical metres, tonnes, and ounces were broken down into different categories for direct use in the cost model. From the final schedule, cost model requirements including items such as the mining fleet, manpower, consumables, ventilation, pumping, and power were determined and used to develop costs from first principals. Reports were generated by month from the start of development (pre-production) through to the first two years of production, and annually for the remainder of the mine life. Table 16-16 shows the annual mine production summary.

Mined gold ounces will average 78,000 oz/year over the life of the mine, with the first two years of production average 100,000 oz/year when the mined ounces in pre-production are taken into account. Gold production peaks in year 1 at over 102,000 oz and decreases for the remaining mine life.

Table 16-16: Mine Production Schedule

Item	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Transverse Stoping	kt	-	269	217	249	190	157	102	**1,183**
Longitudinal Stoping	kt	-	7	44	-	-	12	-	**63**
Total Stoping Ore	kt	-	303	353	352	362	368	157	**1,895**
Cut & Fill	kt		27	92	103	172	200	55	**648**
Development Ore	kt	5	19	13	14	4	1	2	**58**
Total Development Ore	kt	5	45	105	117	176	201	58	**707**
Grade Total Ore Mined	**kt**	**5**	**322**	**366**	**366**	**366**	**369**	**159**	**1,953**
Mined Au Grade	g/t	14.20	10.28	8.56	7.46	7.34	6.94	4.90	**7.53**
Mined Ag Grade	g/t	46.26	34.44	27.17	19.63	15.45	17.54	13.31	**21.86**
Total Waste	kt	53	204	143	132	81	58	79	**750**
Total Lateral Waste Dev.	m	994	3,864	2,772	2,646	1,599	1,120	1,592	**14,587**
Total Ore Development	m	118	951	2,184	2,372	3,535	4,007	1,139	**14,306**
Grand Total Lateral Development	**m**	**1,112**	**4,815**	**4,956**	**5,018**	**5,134**	**5,127**	**2,731**	**28,893**

Source: JDS (2017)

17 Process Description/Recovery Methods

The results of the metallurgical test work described in Section 13, together with financial evaluation data, were used to select the recovery method for the Red Mountain Project. The resulting design criteria was used to design the process facility described in this report section.

The process design criteria and flowsheets were developed using both the metallurgical test work results summarized in Section 13, and industrial design factors where applicable. The 2016-2017 metallurgical test program completed by Base Met Labs indicates that Red Mountain mineralization can be treated using three stage crushing, two stage grinding, CIL, and acid wash, stripping and electrowinning for the recovery of gold and silver doré.

The plant will process material at a rate of 1,000 t/d with an average LOM head grade of 7.53 g/t gold and 21.86 g/t silver. Based on test work, the overall LOM metal recoveries are expected to be approximately 90.9% for gold and 86.3% for silver. The two-stage grinding circuit will target a product size of 80% passing (P_{80}) 25 µm, before the precious metals are recovered in the CIL, stripping and electrowinning circuits. After cyanide destruction, the tailings will be pumped to a TMF. The crushing circuit will operate at an availability of 75%, while the milling and leaching circuits will operate 24 hours per day, 365 days per year at an availability of 92%.

17.1 Introduction

The plant will consist of the following unit operations:

- **Primary Crushing** – A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product P_{80} of 103 mm;
- **Secondary/Tertiary Crushing** – Two stages of cone crushing in closed circuit with a double deck vibrating screen, producing a final product P_{80} of 8.5 mm;
- **Crushed Material Storage Bin and Reclaim** – A 5 hour live capacity bin (200 t) with two reclaim belt feeders feeding the ball mill feed conveyor;
- **Primary Grinding** – A ball mill in closed circuit with hydrocyclones, producing a transfer size T_{80} of 100 µm;
- **Secondary Grinding** – A vertical stirred mill in open circuit, producing a final product P_{80} of 25 µm;
- **Pre-leach Thickening** – A 17 m diameter high-rate thickener to achieve a 50% feed solids density for leaching;
- **Pre-oxidation** – An agitated tank sparged with oxygen to oxidize the slurry prior to leaching;
- **Cyanide Leaching and Carbon Adsorption** – Eight CIL tanks, giving 48 hours retention time to leach gold and silver into solution and adsorb the precious metals onto activated carbon;

- **Carbon Elution and Regeneration** – Acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold-rich solution, and thermal regeneration of carbon to remove organic foulants;
- **Gold and Silver Refining** – Precious metal electrowinning (sludge production), filtration, drying, and refining to produce gold and silver doré;
- **Cyanide Destruction** – Two agitated tanks, one on standby, to reduce the CN_{WAD} (weak acid dissociable) concentration in the CIL tailings to <1 ppm with sodium metabisulphite for SO_2, air and copper sulphate; and
- **Final Tailings Disposal** – Centrifugal pumps to send slurry to the TMF and a barge reclaim system to pump reclaim water back to the process plant.

17.2 Plant Design Criteria

17.2.1 Process Design Criteria

The Process Design Criteria and Mass Balance detail the annual production capabilities, major mass flows and capacities, and availability for the process plant. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 22. Key process design criteria from Section 13 are summarized in Table 17-1.

Table 17-1: Process Design Criteria

Criteria	Unit	Nominal Value	Source
General			
Daily Throughput	t/d	**1,000**	2017 mine plan
Process Plant Availability	%	**92**	Industry standard
Process Plant Throughput	t/h	**45.3**	Engineering calculation
LOM Average Gold Grade	g/t	**7.5**	2017 mine plan
LOM Average Silver Grade	g/t	**21.9**	2017 mine plan
Overall Gold Recovery	%	**90.9**	2016-17 Base Met Labs Testwork Program BL0084 and BL0184 – weighted average of zone recoveries and projected tonnages
Overall Silver Recovery	%	**86.3**	2016-17 Base Met Labs Testwork Program BL0084 - weighted average of zone recoveries and projected tonnages
Crushing			
Availability/Utilization	%	**75**	Industry standard
Crushing Plant Throughput	t/h	**55.6**	Engineering calculation
Bond Crushing Work Index	kWh/t	**11.1**	2016-17 Base Met Labs Testwork Program BL0084 - average of Marc, JW, and AV variability samples
Number of Crushing Stages	-	**3**	Vendor recommended – three stage crushing plant
Crushing System Product Size (P_{80})	mm	**8.5**	Estimated based on a final product aperture screen size of 10 mm

Table 17-1: Process Design Criteria (continued)

Criteria	Unit	Nominal Value	Source
Fine Mill Feed Storage			
Bin Capacity (live)	t	**200**	Design consideration
Bin Capacity (live)	h	**5**	Engineering calculation
Grinding			
Bond Ball Mill Work Index (overall)	kWh/t	**19.6**	2016-17 Base Met Labs Testwork Program BL0084/BL0184 - average of Marc, JW, and AV variability samples
Primary Grinding Mill Type	-	**Ball Mill**	Industry standard for primary grinding to target transfer size
Mill Diameter	m	**3.7**	Vendor recommended
Mill Length	m	**6.5**	Vendor recommended
Installed Power	kW	**1,194**	Vendor recommended
Primary Grinding Transfer Size (T_{80})	µm	**100**	Engineering calculation
Secondary Grinding Mill Type	-	**Vertical Stirred Mill**	Selected to achieve fine grinding product size
Installed Power	kW	**1,475**	Vendor recommended
Final Product Size (P_{80})	µm	**25**	Based on results from 2016-17 Base Met Labs Testwork Program BL0084 and a trade-off study summarized in Section 13
Pre-Leach Thickening			
Thickener Loading Rate	$t/h/m^2$	**0.21**	17-05-2017 Tenova D1724_Red Mountain TW_TCAN.TH.FP
Thickener Underflow Density	% w/w	**50**	Design consideration
CIL			
Pre-Oxidation	Y / N	**Yes**	2016-17 Base Metals Testwork Program BL0084 indicated improved extraction rates with pre-oxidation
Pre-Oxidation Retention Time	h	**2**	2016-17 Base Met Labs Testwork Program BL184
CIL Retention Time	h	**48**	2016-17 Base Met Labs Testwork Program BL0084
Number of CIL Tanks	#	**8**	Selected to ensure adequate recovery and carbon loading of gold and silver
Leach Slurry Flow Rate	m^3/h	**62.5**	Mass balance calculation
Carbon Loading (Gold + Silver)	g/t of carbon	**8,000**	Design consideration and vendor recommended
Carbon Processing			
Carbon Handling Capacity	t/d	**4**	Engineering calculation
Overall Gold and Silver Production, Ave.	oz/d	**1,120**	Engineering calculation
Cyanide Destruction			
Discharge Solution, CN_{WAD}	mg/L	**<1.0**	2016-17 Base Met Labs Testwork Program BL0084/BL0184
Design Residence Time	h	**1.5**	2016-17 Base Met Labs Testwork Program BL0084/BL0184
Number of Tanks	#	**2**	Selected to allow for one operating and one standby

Table 17-1: Process Design Criteria (continued)

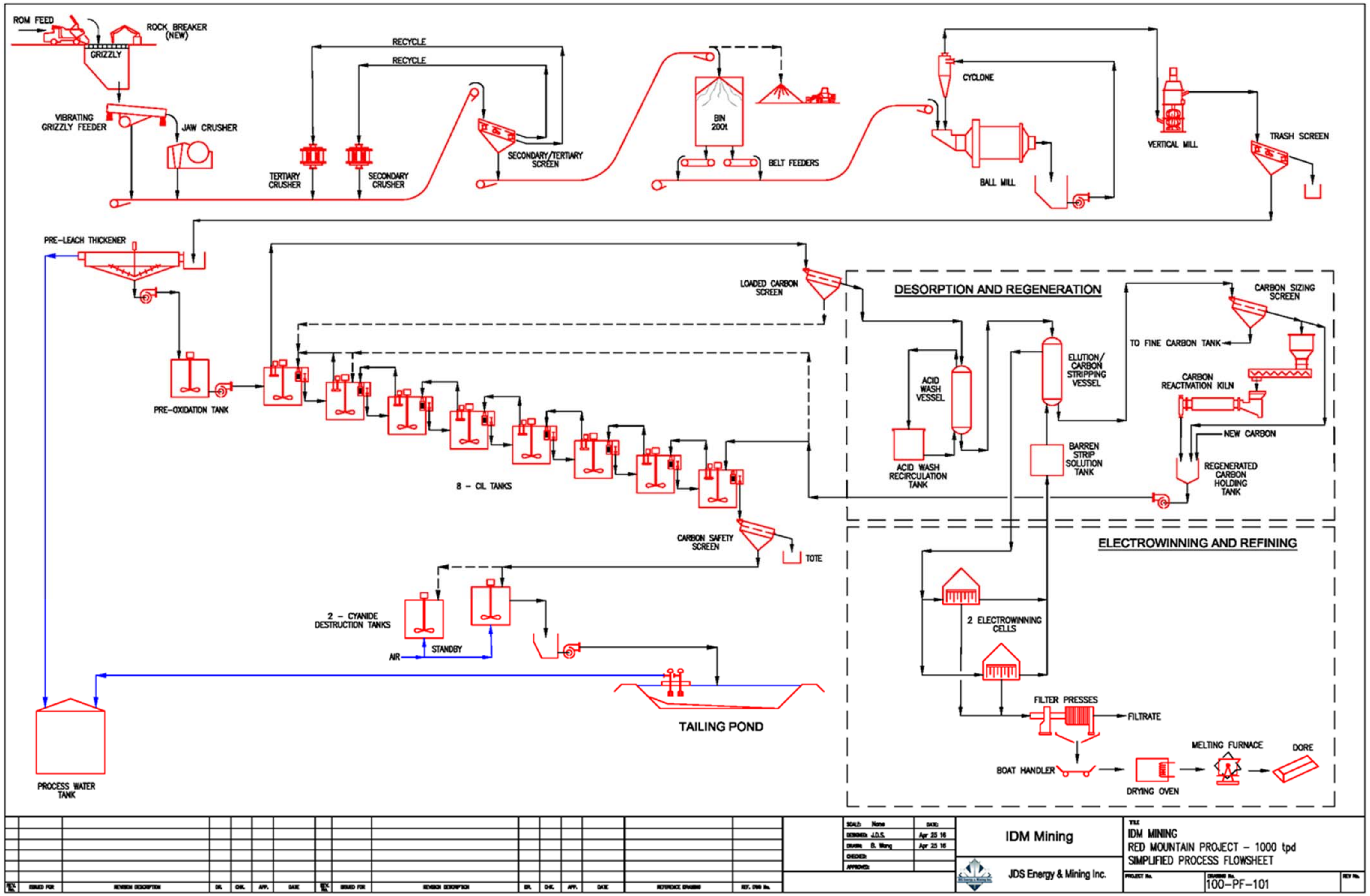
Criteria	Unit	Nominal Value	Source
SO_2 Consumption – Marc Zone	g/t	**2,447**	2016-17 Base Met Labs Testwork Program BL0084/BL0184
SO_2 Consumption – AV/JW Zone	g/t	**5,078**	2016-17 Base Met Labs Testwork Program BL0084/BL0184
CuSO4.5H2O Consumption	g/t	**300**	2016-17 Base Met Labs Testwork Program BL0084/BL0184

Source: JDS (2017)

17.3 Plant Design

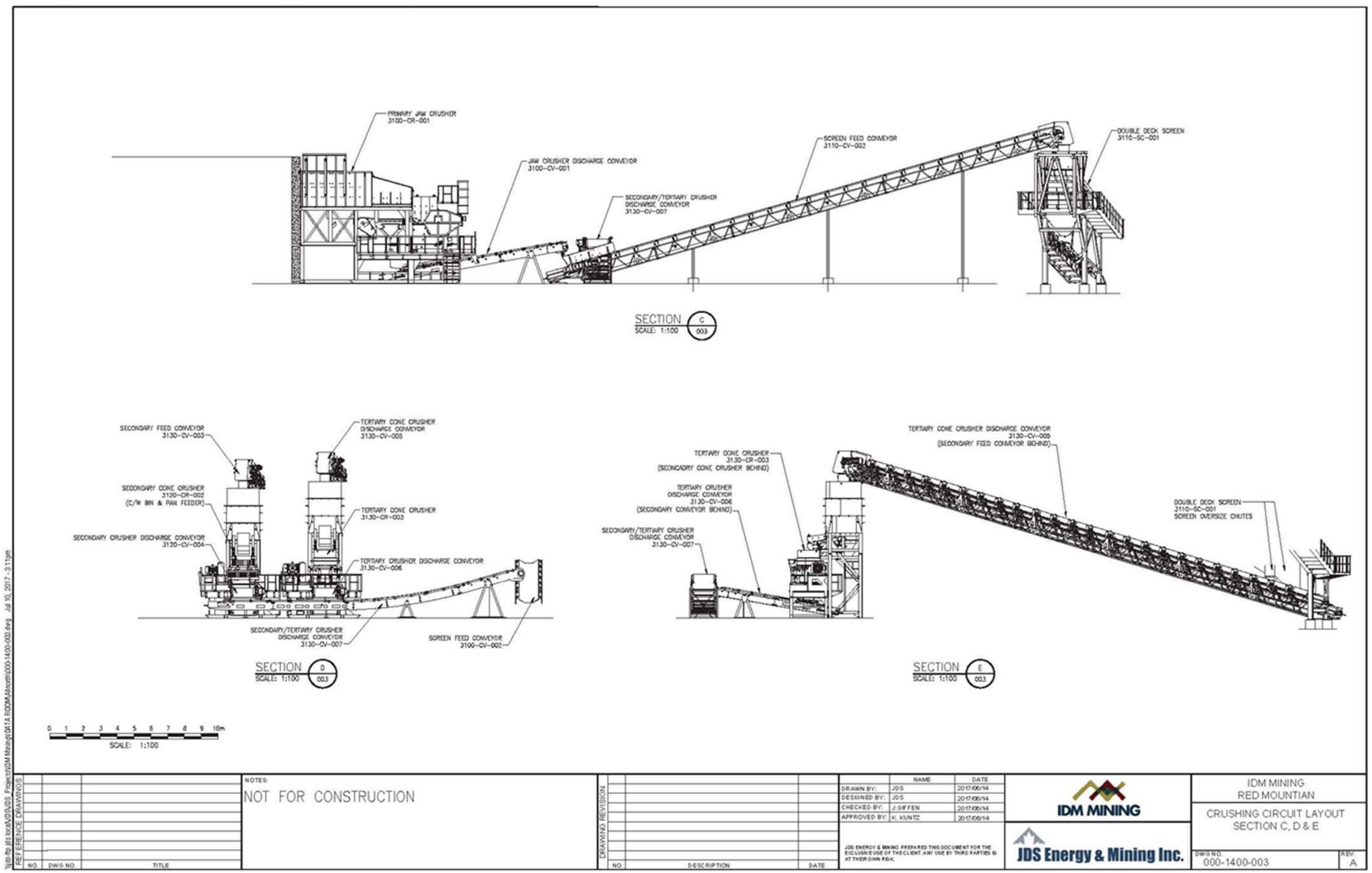
A summary of the process flowsheet is presented in Figure 17-1. Models of the crushing and process facilities are displayed in Figures 17-2 and 17-3, respectively.

Figure 17-1: Overall Process Flowsheet



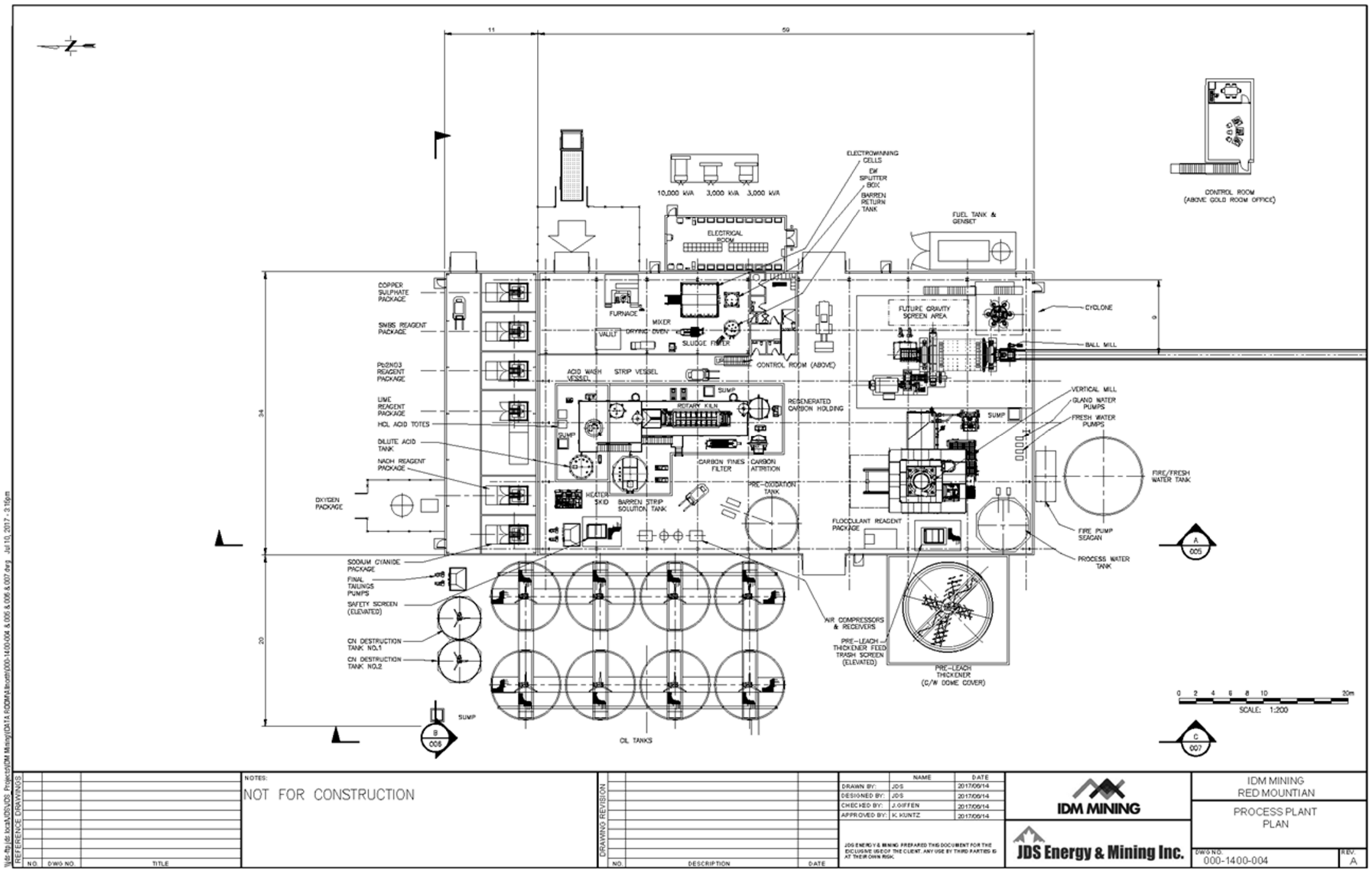
Source: JDS (2017)

Figure 17-2: Crushing and Mineralized Material Storage



Source: JDS (2017).

Figure 17-3: Process Plant



Source: JDS (2017)

17.4 Process Plant Description

17.4.1 Crushing

Material from the underground mining operations will feed a crushing plant that consists of three stages of crushing. The plant will process 56 t/h of material, operate 18 hours per day and produce a final product P_{80} of 8.5 mm.

17.4.1.1 Primary Crushing

Material will be stockpiled near the jaw crusher or direct dumped through a 500 mm static grizzly into a dump pocket. Stockpiled ROM material will be re-handled by a front-end loader and fed into the crusher. The material will discharge through a static grizzly into a 20 t live feed hopper. Oversize material from the static grizzly will be removed for later size reduction using a rock breaker.

A vibrating grizzly feeder will draw material from the dump pocket at a rate of 56 t/h. The vibrating grizzly oversized material will discharge directly into a 635 mm x 1,016 mm (25" x 40") jaw crusher with an installed power of 90 kW. The undersized -75 mm material will bypass the crusher and feed directly onto the screen feed conveyor. The primary crushing stage will produces a product P_{80} of approximately 103 mm at a crusher closed side setting (CSS) of 90 mm.

The screen feed conveyor will collect crushed product from all three stages of crushing and feed a 1,829 mm x 6,096 mm (6' x 20') double-deck vibrating screen. The top deck will have an aperture size of 45 mm, and the +45 mm material will be conveyed to the secondary crusher. The bottom deck will have an aperture size of 10 mm, and the -45 mm, +10 mm material will be conveyed to the tertiary crusher. The -10 mm final product, at an estimated P_{80} of 8.5 mm, will discharge onto the radial stacking conveyor and be transferred to the crushed material storage bin.

17.4.1.2 Secondary Crushing

Material from the secondary crusher feed conveyor will discharge into a Telsmith D 36" cone crusher with an installed power of 75 kW. The secondary crusher will reduce the material to a nominal product P_{80} of approximately 28 mm using a CSS of 25 mm. Crushed product will be transferred to the screen feed conveyor and be circulated back to the double-deck screen.

17.4.1.3 Tertiary Crushing

Material from the tertiary crusher feed conveyor will discharge into a Telsmith SBS 38" cone crusher with an installed power of 132 kW. The tertiary crusher will reduce the material to a nominal product P_{80} of 9.8 mm with a CSS of 10 mm. Crushed product will be transferred to the screen feed conveyor and be circulated back to the double-deck screen.

17.4.2 Crushed Material Storage Bin

The double-deck screen undersize, with a final product size P_{80} of 8.5 mm, will be conveyed to the crushed material storage bin. The bin will provide 200 t, or five hours, of live storage capacity. If there is a planned crusher shut down, additional material will be crushed and stored in a stockpile adjacent to the bin. As the bin capacity is depleted, a front end loader will transfer the material onto the radial stacking conveyor. Two belt feeders under the bin will be installed with variable frequency drives (VFD) to control the reclaim rate feeding the primary ball mill. Each belt feeder will be capable of providing the total throughput of 45 t/h to the plant.

17.4.3 Grinding

The grinding circuit will consist of a primary ball mill followed by a secondary vertical stirred mill. The ball mill will operate in closed circuit with a hydrocyclone cluster, while the stirred mill will operate in open circuit. The grinding circuit can process a nominal throughput of 45 t/h (fresh feed), and produce a final product P_{80} of 25 µm.

Reclaimed material from the crushed material storage bin will feed a 3.7 m diameter x 6.5 m long overflow ball mill via the ball mill feed conveyor. The mill will be installed with a 1,194 kW induction motor. A belt-scale on the feed conveyor will monitor the feed rate and the reclaim belt feeder speed can be adjusted to ensure a constant feed to the mill. Water will be added to the ball mill to maintain the slurry charge in the mill at a constant density of 70%. Slurry will overflow from the ball mill to a trommel screen attached to the discharge end of the mill. The trommel screen oversize will discharge into a trash bin for removal from the system.

Slurry from the ball mill will flow into the cyclone feed pump box and be pumped up to a cluster of six (five operating and one standby) 250 mm hydrocyclones for size classification. The coarse underflow will flow by gravity back into the primary ball mill for additional grinding, while the fine overflow, at an approximate transfer T_{80} of 100 µm, will be pumped to the vertical stirred mill. The hydrocyclones have been designed for a 300% circulating load.

Cyclone overflow will feed a VXP 5000 vertical stirred mill operating with an installed power of 1,475 kW. The mill will use 3 mm ceramic grinding media for fine attrition grinding, and will achieve a final product P_{80} of 25 µm.

17.4.4 Thickening

Stirred mill product will flow onto a 1.2 m x 2.4 m vibrating trash screen for removal of trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to a 17 m diameter pre-leach thickener. Flocculant solution (anionic polyacrylamide) will be added to the thickener feed to promote the settling of fine solids. The high-rate thickener will thicken the slurry to 50% solids. The thickener underflow will be pumped to the CIL circuit, while the thickener overflow will flow by gravity into the process water tank to be used as make-up water in the grinding circuit.

17.4.5 CIL

Pre-leach thickener underflow will be pumped to a 6 m diameter x 6 m high pre-oxidation tank prior to leaching. Oxygen will be sparged into the bottom of the agitated tank and slurry will be conditioned for two hours to oxidize sulphide minerals. Based on metallurgical testing, this step will help reduce the consumption of dissolved oxygen during cyanidation, improving metallurgical recovery. It will also reduce NaCN consumption by preventing the formation of thiocyanate by complexing some of the heavy metals such as iron. Additionally, this reduces reagent consumptions in the cyanide destruction circuit.

The slurry will then flow to the first of eight 8 m diameter x 10 m high CIL tanks. The CIL circuit is designed to provide 48 hours of retention time. Each tank includes an agitator and interstage screen pumpcell. All leach tanks will be located outside and adjacent to the main process building covered by a lean-to structure.

As the slurry flows through the eight CIL tanks, precious metals will be leached into solution and the dissolved gold and silver will be adsorbed onto activated carbon. The average carbon concentration in the CIL circuit is expected to be approximately 30 g/L, with the concentration higher at 50 g/L in the first tank to maximize adsorption. As the slurry proceeds through the circuit, metal values in the solids and solution will progressively decrease. The carbon will be transferred countercurrent to the slurry flow to maximize precious metal recovery. Once per day, loaded carbon from the first CIL tank will be pumped to the loaded carbon screen where the slurry will be separated and the carbon transferred into the acid wash vessel. The separated slurry will then flow by gravity back into the first CIL tank.

Lime slurry will be added to the first and second leach tanks at a rate of up to 1.9 kg/t to maintain protective alkalinity at a design pH of 11.0, preventing the creation of hydrogen cyanide gas (HCN). Sodium cyanide solution will be added to the circuit at a rate of up to 1.77 kg/t, while oxygen is sparged in from the bottom of each tank. This will create the conditions needed for gold and silver to dissolve into solution and adsorb onto carbon.

The tailings stream from the last CIL tank will flow onto a 1.2 m x 2.4 m stationary safety screen to capture any carbon particles that may have escaped the CIL circuit. Captured carbon particles will be collected in bins and disposed of. Safety screen undersize will then be pumped to the cyanide destruction circuit.

17.4.6 Carbon Processing

The carbon processing plant has been designed to accommodate 4 t/d of loaded carbon.

17.4.6.1 Acid Wash

Loaded carbon will be treated with hydrochloric acid solution in the acid wash tank to remove calcium deposits, magnesium, sodium salts, silica, and fine iron particles. Organic foulants such as oils and fats are unaffected by the acid and will be removed after the elution step by thermal reactivation in a kiln.

The carbon will first be rinsed with fresh water. Acid will then be pumped from the dilute acid tank to the acid wash vessel. Acid will be pumped upward through the acid wash vessel and overflow back to the dilute acid tank. The carbon will then be rinsed with fresh water to remove the acid and any mineral impurities.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into the elution vessel. Carbon slurry will discharge directly into the top of the elution vessel. Under normal operation, only one acid wash and elution will take place per day.

17.4.6.2 Carbon Stripping (Elution)

The carbon stripping (elution) process will utilize barren solution to strip the loaded carbon, creating a pregnant solution which will be pumped through the electrowinning cells before being circulated back to the Barren Solution Tank.

The strip vessel will be a carbon steel tank with a capacity to hold approximately 4 t of carbon. During the strip cycle, solution containing approximately 1% sodium hydroxide and 0.1% sodium cyanide, at a temperature of 140°C (284°F), will be circulated through the strip vessel at a pressure of 450 kPa (65 psi). Solution exiting the top of the vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold barren solution prior to passing through the solution heater. An electric boiler will be used as the primary heating source.

17.4.6.3 Carbon Regeneration

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the carbon sizing dewatering screen. The 1.5 m diameter vibratory screen doubles as a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity into the regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. Subsequently, the carbon fines will be filtered and collected into bags for disposal. A 250 kW horizontal electric kiln with residual heat dryer will be utilized to treat 4 t of carbon per day, equivalent to 100% regeneration of stripped carbon. The regenerated carbon from the kiln will flow by gravity into the carbon quench tank, cooled by fresh water and/or carbon fines water, and pumped back to the CIL circuit.

To compensate for carbon losses by attrition, fresh carbon will be added to the carbon attrition tank and mixed with fresh water to activate the carbon pores. The fresh carbon will then drain into the carbon quench tank and combine with the regenerated carbon coming from the kiln.

17.4.7 Electrowinning & Refining

Pregnant solution from the strip circuit will be pumped to the refinery for electrowinning, producing a gold and silver sludge. Pregnant solution will be pumped through two 3.54 m^3 electrowinning cells operating in parallel. Precious metals will plate on the pair of 33 stainless steel cathodes, while the barren solution will flow into the barren return tank and be pumped back to the barren solution tank for reuse. To prevent a build-up of impurities, a bleed of barren solution will periodically be sent to the CIL circuit.

Gold and silver rich sludge will periodically be washed off the stainless steel cathodes into the electrowinning sludge tank using high pressure water. Once the tank is filled, the sludge will be drained, filtered, dried, mixed with fluxes, and smelted in a 125 kW induction furnace, producing gold and silver doré. This process will take place within a secure and supervised area, and the precious metal product will be stored in a vault until shipping off site.

17.4.8 Cyanide Destruction

The cyanide destruction circuit will consist of two 5 m diameter x 6 m high mechanically-agitated tanks, each with a capacity to handle the full slurry flow for the required residence time of 1.5 hours. Cyanide will be destroyed using the SO_2/Air process. Treated slurry from the circuit will flow by gravity to a final tailings pump box and pumped to the TMF.

The cyanide destruction circuit will treat CIL tailings slurry, process spills from various contained areas, and process bleed streams.

Process air will be sparged from near the bottom of the tanks, under the agitator impeller. Lime slurry will be added, if necessary, to maintain the optimum pH of 8.0 - 8.5 and copper sulphate ($CuSO_4$) will be added as a catalyst, maintaining 73 mg/L concentration in solution. A sodium metabisulphite (SMBS) solution, at a rate of up to 5.1 kg/t, will be dosed into the system as the source of SO_2. This system has been designed to reduce the CN_{WAD} concentration to below 1.0 mg/L before the tailings is transferred to the TMF.

17.4.9 Tailings Management

Cyanide destruction tailings will be pumped from the process plant to the TMF and sub-aqueously discharged in the supernatant pond. Water from the TMF will be reclaimed by vertical pumps on a barge and returned to the plant as make-up water.

17.4.10 Reagents Handling & Storage

Reagents consumed within the plant will be prepared on-site and distributed via the reagent handling systems. These reagents include: sodium cyanide (NaCN), lime, lead nitrate (Pb_2NO_3), hydrochloric acid (HCl), caustic soda (NaOH), copper sulphate ($CuSO_4$), sodium metabisulphite (SMBS), antiscalant, flocculant, and activated carbon. All reagent areas will be bermed with sump pumps which transfer spills to the final tailings pump box, with the exception of the flocculant. Flocculant spills will be returned back to the storage tank. The reagents will be mixed, stored, and then delivered to the pre-leach thickener, CIL, acid wash, elution and cyanide destruction circuits. Dosages will be controlled by flow metres and manual control valves. The capacity of the storage tanks will be sized to handle one day of production. The reagents will be delivered in dry form, with the exception of HCl and antiscalant, which are delivered as solutions.

Table 17-2 summarizes the reagents used in the plant and their estimated daily consumption rates. The table also includes other major process consumables.

Table 17-2: Reagents and Process Consumables

Description	Delivered Form	Average Daily Usage
NaCN	1 tonne bags (dry)	1.6 t/d
Lime	2 tonne bags (dry)	1.7 t/d
Pb_2NO_3	50 kg bags (dry)	250 kg/d
HCl	208 L drums (36% liquid)	416 kg/d
NaOH	1 tonne bags (dry)	699 kg/d
$CuSO_4$	1 tonne bags (dry)	300 kg/d
SMBS	1 tonne bags (dry)	4.2 t/d
Antiscalant	1 tonne tote (liquid) or 50 kg barrels	41 kg/d
Flocculant	25 kg bags (dry)	20 kg/d
Activated Carbon	50 kg bags (dry)	120 kg/d
Ball Mill Grinding Media - 75 mm chrome steel	1 tonne bags	1.2 t/d
Stirred Mill Grinding Media - 3 mm ceramic	500 kg bags	274 kg/d

Source: JDS (2017)

17.4.11 Air Supply

An instrument and plant air system with two 15,000 Nm^3/hr compressors and associated dryers, filters, and receivers will be provided and located in a compressor room inside the plant building. Two 30 kW blowers (one operating and one standby) will be used to provide sparged air for the cyanide destruction circuit.

Oxygen will be used in the CIL circuit and will be supplied by a contracted local vendor.

17.4.12 Water Supply & Consumption

The following water types will be used in the process plant:

- **Process Water –** Overflow water from the pre-leach thickener will be used as process water. This water will have a low gold concentration and will be used predominantly in the grinding circuit to dilute slurry to the required densities;
- **Fresh Water –** Fresh water for the process plant will be pumped from a fresh water supply, such as the local water course or an impoundment which may potentially be located adjacent to the process plant. Fresh water will be used as reagent make-up water, gland water, process make-up water, and cooling water services in the strip circuit boiler. The estimated fresh water consumption in the process plant will be 16 m^3/h; and
- **Reclaim Water –** Water reclaimed from the TMF will be used as dilution water in the grinding and cyanide destruction circuits. Based on the water balance and a settled tailings density of 70%, 38 m^3/h of water will be reclaimed from the TMF.

18 Project Infrastructure & Services

18.1 Summary

The Project envisions the upgrading or construction of the following key infrastructure items:

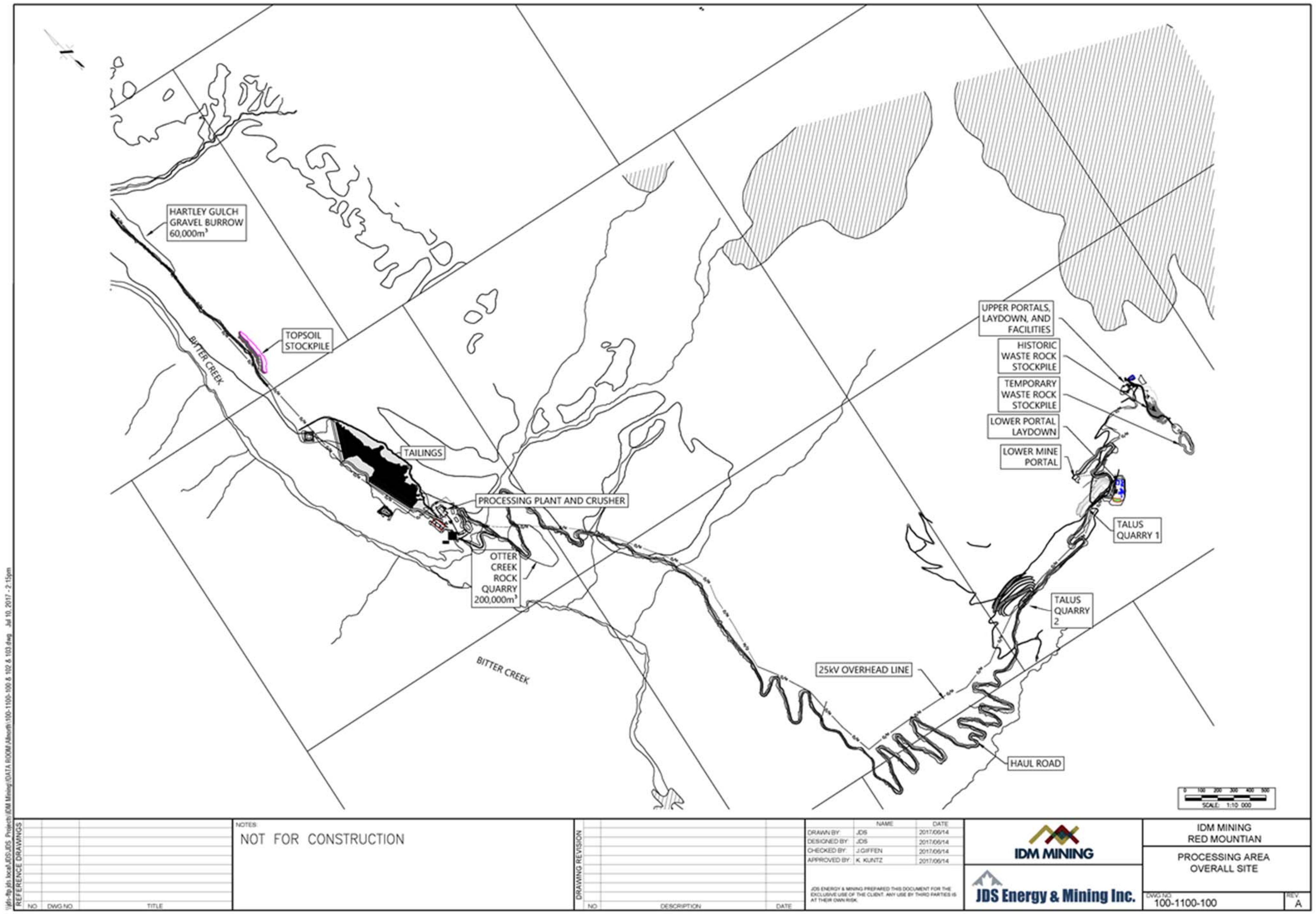
- Approximately 15 km year-round access road from Highway 37A to the processing plant site;
- Approximately 11 km year-round haul road from the processing plant site to the upper and lower mining portals near the top of Red Mountain;
- Electrical connection to BC Hydro power grid and a transmission line at 138 kV adjacent to the access road;
- Distribution powerline at 25 kV from processing plant site to the upper mine portal;
- Process plant located at Bromley Humps;
- TMF and impoundment located at Bromley Humps;
- Water management ponds to manage contact water runoff from around the Project site;
- Diversion ditches to divert non-contact water to the maximum practical extent;
- Temporary development of waste rock storage areas prior to being re-handled into the underground workings as backfill;
- Administration office, mine dry, maintenance shop, and warehouse facilities;
- Mine operations office and emergency facilities at the mine portals;
- Tailings effluent water treatment plant;
- Process and fire water storage and distribution; and
- Temporary construction camp located in Stewart.

18.2 General Arrangement

The general planned location of the plant site and portals are shown in Figure 18-1. A site plan for the plant site area is shown in Figure 18-2.

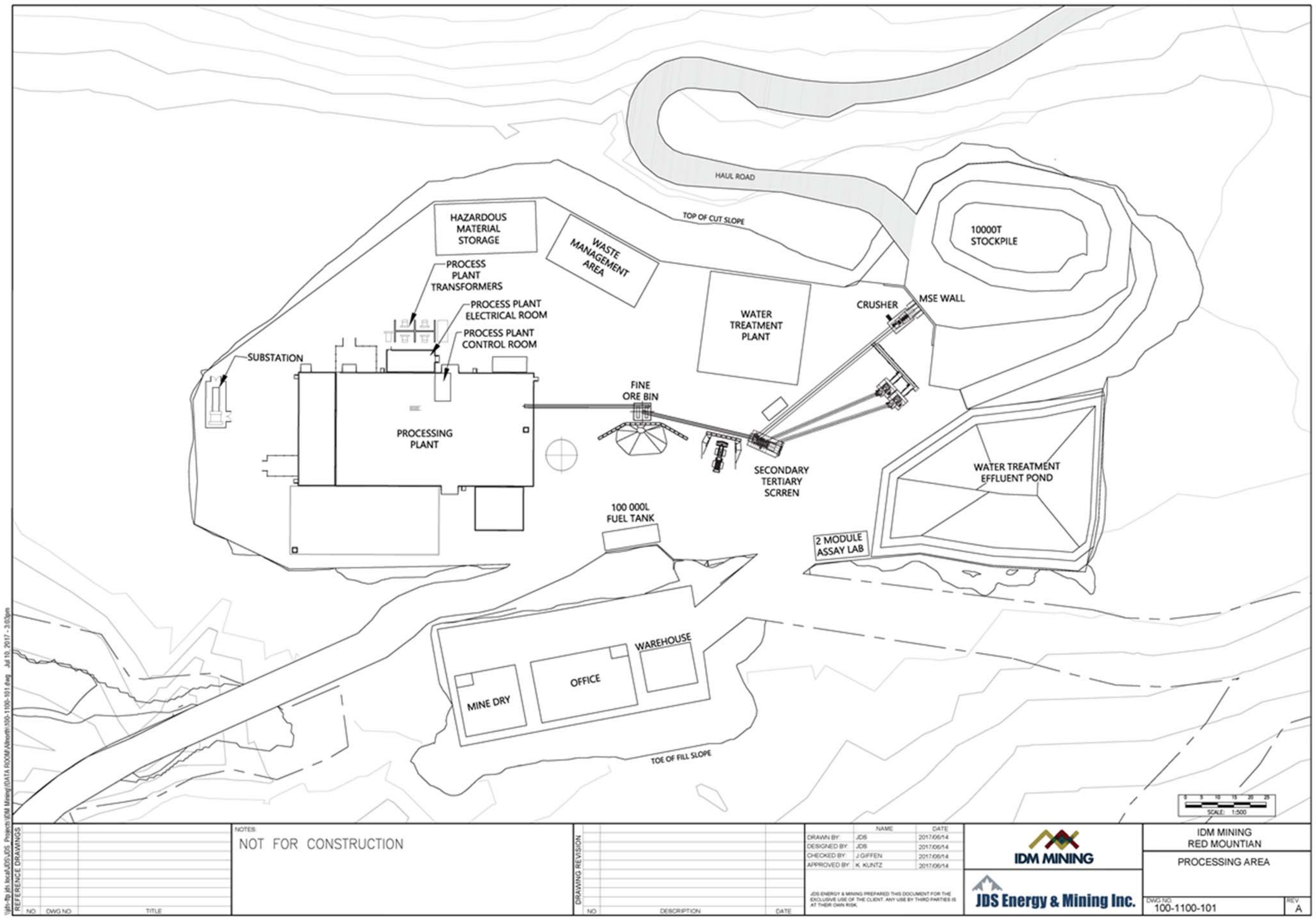
The plant facilities and TMF are envisioned to be located at a lower elevation, where the terrain is less steep and minimal earthworks are required to provide suitable footprint for the surface structures and equipment. The TMF would be located immediately north of the plant complex. The proposed location would minimize the dam footprint, construction volume and catchment area, while maximizing the storage capacity of the impoundment.

Figure 18-1: General Site Arrangement and Portal Location



Source: JDS (2017)

Figure 18-2: Plant Site Layout



Source: JDS (2017)

18.3 Site Access

18.3.1 Site Access Road

The Red Mountain Mine 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 access road will be used to transport construction material, mine equipment, operational consumables, and workers to and from the Project.

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/hr. In sections of the alignment where a 50 km/hr design speed is unfeasible due to topography, excessive earthworks, and increased cost, the design speed will be reduced to 30 km/hr. 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 will accommodate 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 18-1 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).

Table 18-1: Access Road Design Specifications

Components	50 km/hr	30 km/hr
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 (2017)

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 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.

18.3.2 Haul Road

From the mill site, 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/hr. 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 power lines will run parallel to the road and will 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 18-2 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 18-2: Haul Road Design Specifications

Components	30 km/hr
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)
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)

18.3.2.1 Steep Grade Considerations

There are sections of the haul road between the mill site 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/hr when descending grades between 5% and 17%;
- not exceed speeds of 10 km/hr on grades of 18%;
- limit payload to 36 tonnes;
- make a mandatory stop at a designated location to cool brakes; and
- 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/hr.

No hauling is to occur during periods of snow or ice cover on the road surface.

18.3.3 Avalanche Control

The combination of rugged glaciated topography, latitude, and coastal weather systems creates severe winter conditions in the Project area. The region receives some of the heaviest snowfall in North America, with settled seasonal snowpack depths ranging from 3 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 4-8 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.

18.4 Site Geotech

The TMF north embankment is located between a bedrock ridge and a steep mountain slope to the east. The TMF south embankment is located between two bedrock ridges. The process plant site is situated on a bedrock outcrop to the south of the TMF south embankment, near Otter Creek.

The surficial geology of the study area is characterized by landforms that reflect a history of glacial retreat. Drilling and mapping concluded that bedrock exposure is prevalent throughout the area. Colluvium and glacial till are the predominant surficial materials (where present) and are generally contained to the TMF basin. Surficial material thicknesses ranged from less than 1 m to approximately 6 m, with the deepest deposits encountered in the centre of the TMF north embankment footprint. Volcanic and sedimentary rocks were the prominent bedrock types encountered in the study area with some porphyry intrusive units. Intact bedrock strength data classifies the volcanic rocks, sedimentary rocks, and porphyry Intrusive rocks as strong to very strong rock with an overall FAIR Rock Mass Rating (RMR_{89}). The Mudstone unit, part of the sedimentary units, is the weakest bedrock unit and was encountered towards the south abutment of the TMF south embankment. Bedrock generally has low to moderate permeability.

The site investigation and subsequent geotechnical assessment provides specific information on the foundation characteristics for the following proposed mine infrastructure components:

- **TMF North Embankment** – Six drillholes were completed in the vicinity of the TMF north embankment during the 2016 site investigation, including two drillholes for monitoring wells. Information is also available from six drillholes from the 1996 site investigation program conducted by Golder. Overburden at the TMF north embankment varies in thickness from 0.5 to 6 m and is classified as a thin veneer of colluvium (approximately 0.5 to 2 m thick) underlain in some cases by glacial till, or occasionally thicker colluvium deposits. Intact strength testing on samples from the TMF north embankment area indicates generally strong to very strong rock with UCS test results ranging from 85 to 225 MPa and an average value of 150 MPa. The bedrock has a FAIR rock quality designation. Bedrock generally consists of low to moderate permeability. Volcanic rocks with some porphyry intrusive units to the north of a fault zone that crosscuts the TMF north embankment. An additional fault is interpreted to occur west and parallel to the TMF north embankment.
- **TMF South Embankment** – Six drillholes were completed in the vicinity of the TMF south embankment during the 2016 site investigation, including two drillholes for monitoring wells. Information is also available from five drillholes from the 1996 site investigation program conducted by Golder. Overburden at the TMF south embankment varies in thickness from 0.6 to 5 m and is classified as a thin veneer of colluvium (approximately 0.5 to 2 m thick) underlain in some cases by glacial till, or occasionally thicker colluvium deposits. Intact strength testing of samples from the TMF south embankment area indicates generally strong to very strong rock with UCS test results ranging from 80 to 205 MPa and an average value of 120 MPa. The bedrock has a FAIR rock quality designation. Bedrock generally consists of

low to moderate permeability volcanic and sedimentary rocks. Two faults were encountered that crosscut the TMF south embankment.

- **Process Plant Site** – Two drillholes were completed in the vicinity of the process plant site during this 2016 site investigation. Bedrock was encountered at or near to surface in the drillholes, with only a thin layer of topsoil present. Bedrock conditions at the process plant site area are primarily characterized by greywacke (sedimentary unit) underlain by some volcanic units (mafic and felsic dykes, gabbro, etc.) with a FAIR rock quality designation and approximately 60 MPa strength.

18.5 Foundations

The plant site area will be predominately constructed on cut pads. The plant site will have maximum cuts of up to 15 m at the uphill (east).

Critical structures that cannot tolerate differential settlements such as the process plant will be founded on competent bedrock. The currently planned cut depths are anticipated to reach fresh, competent bedrock. However, this will be further confirmed for the critical facility locations during detailed design studies.

Drilling and blasting will be required for the fresh bedrock and possibly for the upper weathered bedrock. Weathered rock will not be suitable for reuse as structural fill, and will be placed in thin lifts in the TMF.

If required, non-critical structures that are able to tolerate minor differential settlements will be designed on fill sections of the pads. Fill will consist of free-draining, coarse, granular materials, and preferably angular durable rock fill to prevent buildup of excess pore pressures. Where structural fill is to be placed on an existing natural slope, the fill will be keyed into the natural slope by excavating steps into the slope at the edge of successive lifts of structural fill. Rock fill pads will be constructed in lifts no greater than 1.5 m with the maximum rock size limited to 0.9 m. Engineered slopes constructed of structural or rock fill will be made at a gradient of 2H:1V or flatter. Buildings will be set back a minimum of 10 m from the crest of fill slopes.

18.6 Power Supply

BC Hydro will supply 138 kV power to the mine site via an 18-km overhead transmission line that will follow the incoming site access road. This power line will terminate at the Red Mountain Mine substation. 138 kV power will be transformed down to 25 kV. 25 kV will be distributed to the process plant and upper and lower mine portals.

There will be 25 kV power supplied to the process plant main transformer via underground duct bank. The transformer secondary will feed electrical room 1 (ER-1) at 4160 V also via underground ducts.

There will also be 25 kV power distributed to the upper mine portal electrical rooms (ER-2 & ER-3) via a 9 km overhead line along the access road. There will be a tap off to a 25 kV to 4160 V transformer at the upper portal and a tap off to a 25k V to 4160 V transformer at the lower portal.

18.6.1 Site Power Distribution

The total connected load for the Project is calculated at 9.9 MW, with the total operating load calculated at 8.5 MW. The anticipated load breakout is summarized by Table 18-3

Table 18-3: Electrical Load Breakdown

Operational Area	Connected Load (kW)	Operating Load (kW)
1000 – Mining	2,590	2,404
3000 – Mineral Processing	6,566	5,431
4000 – Tailings Management	500	389
5000 – Infrastructure	320	280
Total	**9,976**	**8,503**

Source: Allnorth (2017)

18.6.2 Standby Power

The permanent standby power system will consist of three (3) standby diesel generators:

- One generator will be located close to ER-1 at the process facility and will supply power to specific loads in the facility to enable purging of the process during utility supplied power outages and to keep other essential systems (agitators, reagent ventilation fans, sump pumps, controls, communications, etc.) operational. During the construction phase, this generator could be installed early in the schedule to provide temporary construction power.
- One generator will be located close to ER-2 at the upper mine portal site and will supply power to the underground area to ensure ventilation, emergency lighting, sump pumps, and refuge chambers remain operational in a power outage event. This generator will also keep other essential systems (controls, communications, etc.) operational. During construction phase, this generator could be installed early in the schedule to provide power to pit drills and temporary construction power.
- One generator will be located close to ER-3 at the lower mine portal site and will supply power to the underground area to ensure ventilation, emergency lighting, sump pumps, and refuge chambers remain operational in a power outage event. This generator will also keep other essential systems (controls, communications, etc.) operational. During construction phase this generator could be installed early in the schedule to provide power to pit drills and temporary construction power.

18.6.3 Instrumentation & Control System

The process control system will be a PLC based control system. Communication between the process plant and the mine portal PLCs will be via a fiber optic link, installed beneath the overhead power cables on the 25 KV power line from the 138 kV substation up to the mine portals.

18.6.3.1 Process Plant

There will be a PLC cabinet as well as graphical operator workstations in the main control room. It is anticipated that there will also be remote I/O cabinets in major process areas which will communicate to the main PLC via industrial Ethernet.

18.6.3.2 Underground Mine Portals

There will be a PLC cabinet in each mine portal electrical room as well as a graphical operator workstation. The mine PLC control system will communicate with the process plant control system via the fiber optic link.

18.6.3.3 Process Communication System

The site-wide process communication system will be an industrial Ethernet and fiber optic network, providing communication between the process controllers in the electrical rooms and the operator's workstations and graphical interface consoles in the control room. PLC configuration and historian software will also be supplied with the hardware.

18.6.3.4 Business Communication System

The site-wide process communication system will be an Ethernet cable network, providing communication between the truck shop workstation, maintenance shop workstation, data servers, email server, and other business-related computers at the site.

18.7 Water Management

18.7.1 Water Management Plans

Site water management involves controlling surface water around the Project site during the construction, operations, and closure and reclamation phases of the Project. Water in contact with mine workings or disturbed areas (groundwater inflows from the underground mine, waste rock, ore stockpile, quarry areas, tailings, laydown areas, etc.) is considered contact water. Non-contact water is runoff from undisturbed areas.

Management of surface water on site will be undertaken by the construction of systems of ditches, ponds, berms, selective grading of surfaces, and pump and pipeline systems. The major facilities for contact water management include:

- TMF;
- Seepage collection and recycle ponds;
- Water Treatment Plant;
- Borrow pit and quarry sediment ponds;
- Plant Site and ROM Stockpile runoff collection ditches; and
- Portal collection pond.

18.7.1.1 Design Storm Events

The design storm events for temporary and permanent water management structures are:

- Peak flow from 1/10 year 24-hour rainfall event for temporary structures;
- Peak flow from 1/5 year 24-hour rainfall event for the non-contact water diversion channel; and
- Peak flow from 1/200 year 24-hour rainfall event for permanent structures.

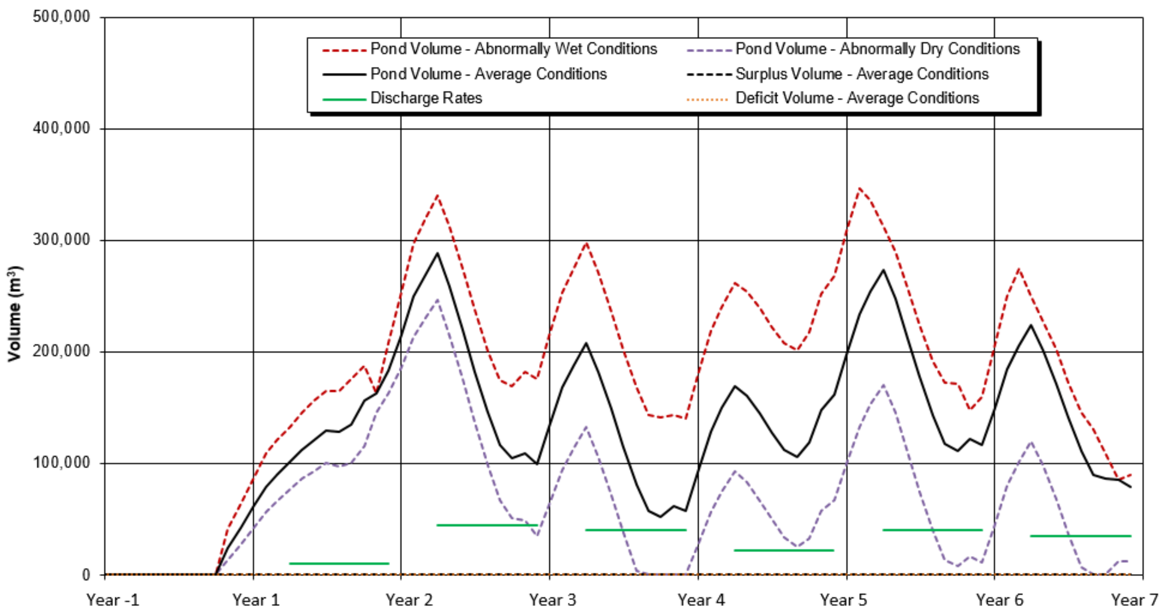
18.7.2 Water Balance

The water balance results indicate that the TMF will operate in an annual water surplus. The surplus volumes will be managed with an active water discharge. This discharge will be pumped to the water treatment plant (located at the plant site) for treatment, and then released to Bitter Creek.

The maximum allowable volume of water in the TMF assumes that a minimum storm storage capacity must be maintained at all times. The storm storage capacity for the TMF equates to the environmental design flood (EDF) volume of 160,000 m^3.

The discharge rates have been developed to provide a balance between surplus and deficit conditions under average climatic conditions (i.e., no operational surplus or deficit). However, the pond may potentially be in a deficit condition under abnormally dry conditions in some years, or in a surplus condition under abnormally wet conditions in other years. The monthly TMF pond volume and discharge rates are shown on Figure 18-3.

Figure 18-3: Water Balance Results - Base Case



Source: KP (2017b)

The water balance results indicate that the TMF is expected to be in a surplus condition that will be managed by discharging treated water up to 50,000 m^3 per month from March to October (i.e., peak flow months in Bitter Creek). Details on the water treatment process are provided in Section 18.9.3.

18.8 Waste Management

18.8.1 ARD/ML Considerations

18.8.1.1 Waste Rock, Ore and Talus

The primary rock types in the mine area include the Stuhini Group mudstones, siltstones and cherts (sedimentary rocks), the Hazelton Group clastics and volcanoclastics, and three intrusive (igneous) suites: the Hillside Porphyry, the Goldslide Porphyry, and the Biotite Porphyry. Mineralization associated with the Marc, AV, JW, and 141 zones are developed primarily in the Hillside Porphyry, and to a lesser extent in the rafts of sedimentary and volcanoclastic rocks. Pyrite is the most abundant sulphide, and is associated with the stockwork, and as an alteration mineral, although pyrrhotite and sphalerite are both locally important.

A comprehensive geochemical testing program has been completed to characterize the metal leaching/acid rock drainage (ML/ARD) potential of the waste rock, ore and talus in support of mine planning and environmental assessment. The program included laboratory based static and kinetic testing completed by MDAG (1996a) and Frostad (1999), and an extensive site monitoring program that has spanned approximately 20 years (i.e., SRK 2001, 2003, 2004 through 2012 and 2014). The laboratory program included acid-base accounting (ABA), trace element analyses, mineralogy, and kinetic testing. The site monitoring program includes two field crib tests, legacy waste rock and ore stockpiles from an underground exploration adit developed in the1990s, and more recent site water quality monitoring data. Details on the characterization program are provided in SRK 2017a, and key findings are summarized as follows.

All of the waste rock and ore samples were classified as potentially acid generating (PAG) based on neutralizing potential / acidic potential (NP/AP) and total inorganic carbon / acidic potential (TIC/AP) tests. Data from two field cribs indicate that the upper bound of onset to acidity in mudstone (intermixed with some volcanic rock) is 20 years. This lag time is longer for volcanic rocks, and may be shorter for unmixed mudstone samples.

Cadmium, zinc, cobalt, nickel and manganese leaching has been observed in neutral pH seepage from the Marc Zone legacy waste rock stockpile which is composed of a mixture of PAG sediments and volcanics from underground development in the mid-1990's. Under more acidic conditions observed in some of the laboratory tests on waste rock, ore and talus, copper and selenium were also observed. During periods of active pumping from the underground decline, arsenic levels were also somewhat elevated. Testing completed to evaluate potential for remobilization of metals during flooding of the mine indicated potential for release of arsenic, chromium, copper, lead, nickel, and zinc under reducing conditions.

The results indicate that the ore, waste rock and talus used as backfill in the mine need to be appropriately managed to prevent or minimize the potential effects of ML/ARD during operations and post-closure. Management and closure plans for these materials are described in Section 18.8.2.

18.8.1.2 Tailings

Tailings were produced for geochemical testing as part of the metallurgical testing program described previously and consisted of a slurry of fine (25 µm) tailings produced after whole-ore cyanide leaching after cyanide detoxification. Geochemical characterization of the tailings included static testing and kinetic testing, and mineralogy on the tailings solids, analysis of the process water, and, and aging tests on tailings slurry. Details on the program and its findings are documented in SRK 2017b. Key findings are summarized as follows.

The tailings from all of the ore zones were classified as potentially acid generating, solid phase concentrations of silver, gold, arsenic, bismuth, cadmium, cobalt, copper, lead, sulphur, antimony, selenium and zinc elevated relative to ten times the average crustal abundance for low and high calcium granites. Kinetic testing indicates there will be a delay to onset of acidic conditions, with further testing in progress to quantify this period. Copper, antimony, cobalt, selenium and zinc levels were elevated in the Process water. Additionally, degradation products from cyanide detoxification, including cyanate, thiocyanide and ammonia were present at high concentrations.

The testing has shown that the tailings will need to be managed to prevent or minimize ML/ARD during operations and post-closure. Additionally, water treatment will be required during operations to meet discharge criteria in the tailings effluent. Management plan and closure plans for the tailings are presented in Section 18.8.3.

18.8.1.3 Aggregate Sources & Rock Cuts

Geochemical characterization was completed on the aggregate sources and rock cuts along the access road (SRK 2017c).

Table 18-4 summarizes the geological units that will be encountered by components of the Project that are outside of the mine area, along with a summary geochemical description of each geological unit. More descriptive geochemical summaries of each geological unit and management implications for each of these follow.

Table 18-4: Geological Units Present at Aggregate Sources & Rock Cuts

Aggregate Source / Excavation Area	Geological Units					
	Bromley Area Intrusives	Hazelton Group Sediments	Coast Plutonic Complex Monzonite	Hazelton Group Volcanics	Sand and Gravel	Mine Site Talus
Geochemical Characteristics	No PAG, possible selenium ML	30%PAG; possible ML	No ML/ARD	50% PAG and ML	No ML/ARD	ML/ARD
Process Plant Foundation		X				
Access Road Rock Cuts[1]	X	X	X	X		X
Hartley Gulch Borrow					X	
Otter Creek Quarry[2]		X				
Highway 37A Quarry			X			
Roosevelt Creek Borrow					X	
Talus Quarries (2)						X

Source: SRK (2017)

Notes: 1. While each of the indicated geological units have been identified along the Access Road, the geological units at specific rock cut locations have not been reconciled.

2. Otter Creek Quarry's current location is partially within the Hazelton Group Sediments (see Figure 2-1 in Volume 7, Appendix 1-B). IDM is considering furthering the option to reposition the Otter Creek Quarry to target the gabbro in the Bromley Area Intrusives and avoid the Hazelton Group Sediments. This design will be provided as part of permit applications.

The Bromley Area Intrusives consists of mainly gabbro as well as Goldslide Porphyry, mafic dikes, and one or more unclassified dikes, and a fault zone. Each of these intrusive materials sampled were classified as non-PAG. Elemental analysis indicated enrichment of gold, cobalt, chromium, and nickel in most samples, and silver, bismuth, copper, and selenium in a few samples. Except for selenium, which was only slightly enriched in two samples, these elements are not expected to be mobile under neutral pH conditions, indicating these samples have a low metal leaching potential. The Gabbro intrusives are suitable for general use in construction, and do not require special management measures.

Approximately one-third of the Hazelton Group sediment samples are PAG or have an uncertain potential for ARD. All of the PAG samples were logged as mudstone or siltstone, whereas most non-PAG samples were identified as greywacke, suggesting that differences in lithology could be used to classify the ARD potential of these materials. However, additional samples would be required to verify this finding. Elemental analyses indicate a few of the sediment samples are enriched in silver and selenium, with selenium leaching being a potential issue for these materials. Based on these findings, the Hazelton Group sedimentary rocks require specific management measures to prevent or minimize ML/ARD.

The access road will intersect a monzonite unit that is part of the Coast Plutonic complex at 0 to 4.5 km. Additionally, this geological unit is present at the Highway 37A Quarry. The monzonite is non-PAG, and is somewhat enriched in gold, bismuth, and chromium. These elements are not expected to be mobile under neutral pH conditions, indicating that the monzonite is suitable for road construction.

The Access Road will intersect a mixture of rock types comprising the Hazelton Group volcanics at 19 to 23.4 km. The Hazelton Group volcanics are comprised of a mixture of sedimentary, volcanic, volcaniclastic, and intrusive rocks. Samples from the Hazelton Group volcanics showed a variable potential for ML/ARD, with approximately 50% of the samples classified as PAG, and a number of samples showing enrichment of gold, cobalt, chromium, nickel, sulphur and selenium, and one sample showing significant enrichment of silver, arsenic, cadmium, and zinc. Arsenic, selenium, cadmium and zinc can be relatively mobile under neutral pH conditions; these and several other trace elements as well as aluminum, iron, and manganese may be mobilized under acidic pH conditions. Based on these findings, the Hazelton Group volcanic rocks require specific management measures to prevent or minimize ML/ARD.

Section 18.8.2 describes measures to manage ML/ARD from these materials.

18.8.2 Construction Materials Management

A proportion of each of the Hazelton Group sediments and Hazelton Group volcanics are potentially ML/ARD. One or both of these materials will be encountered along the access road and at the plant site. The following mitigation measures will be implemented to address potential ML/ARD issues at each of these locations.

18.8.2.1 Process Plant Site Excavation

During drilling, blasting, and excavation of material from this location, runoff will be diverted away from the plant site to minimize contact with the broken rock.

A portion of the material to be extracted from the process plant site for use in construction of the initial TMF embankments is Hazelton Group sediments, a portion of which is potentially ML/ARD. This material will be selectively placed within the core of the initial embankment, with geochemically stable material to be placed on the outside face. At this location, the potentially ML/ARD material will be isolated from both water and air by the geomembrane liner and subsequent embankment raises.

During detailed engineering, the option of using additional gabbro rock from the Otter Creek Quarry and minimize the quantity of rock excavated at the process plant site will be evaluated.

18.8.2.2 Access Road

Both the Hazelton Group sediments and Hazelton Group volcanics are located along the road corridor. A number of strategies will be employed to minimize the amount of potential ARD/ML rock that is disturbed by road construction:

- Minimize the amount of road cut through PAG rock;
- Construct fill areas with non-PAG materials, where practical. This will be highly effective, but may require non-PAG materials to be transported larger distances at a higher cost;
- Divert runoff away from the road bed to minimize contact with broken rock. Such diversions would need to be constructed from non-PAG materials. Where runoff will cross the road, culverts can be used rather than rock drains. This option is most applicable when contact with PAG materials is unavoidable; and
- Blend high sulphide rock with rock that contains excess NP. This may be a favourable approach in certain instances.

As the road design is advanced, additional geochemical evaluations will be undertaken, and the mitigation measures will be refined at specific locations as required.

Quarries and borrow pits remaining at closure will be reclaimed in accordance with the reclamation and closure plan. This includes ensuring physical stability with final contouring.

18.8.3 Temporary Waste Rock Storage

Waste rock generated from the development of the underground mine will be stored in a temporary surface storage area located west of the upper portal. The areas will contain approximately 76,000 t of potentially acid generating (PAG) waste rock. The temporary surface storage area is expected to be depleted within two years as all waste is planned to be re-handled into the underground stopes as backfill. Extensive geochemical studies and monitoring since 1993 of waste rock stored on surface have shown that the PAG rock is stable over the short to medium term and is not anticipated to generate acid, simplifying the management of this material during operations.

The waste rock stored in temporary stockpiles will be relocated to the underground mine workings during the operations phase, and the sites will be reclaimed. Once the waste rock at the temporary surface storage areas has been used up, all future waste rock will be placed in the underground mine directly during the remainder of the operating phase. At closure, a hydraulic bulkhead will be installed and the mine will be allowed to flood. This will prevent ML/ARD from developing in the flooded portions of the mine. Water quality predictions for the underground mine indicate there is sufficient alkalinity in the groundwater to maintain neutral pH conditions within the flooded mine.

18.8.4 Tailings Management

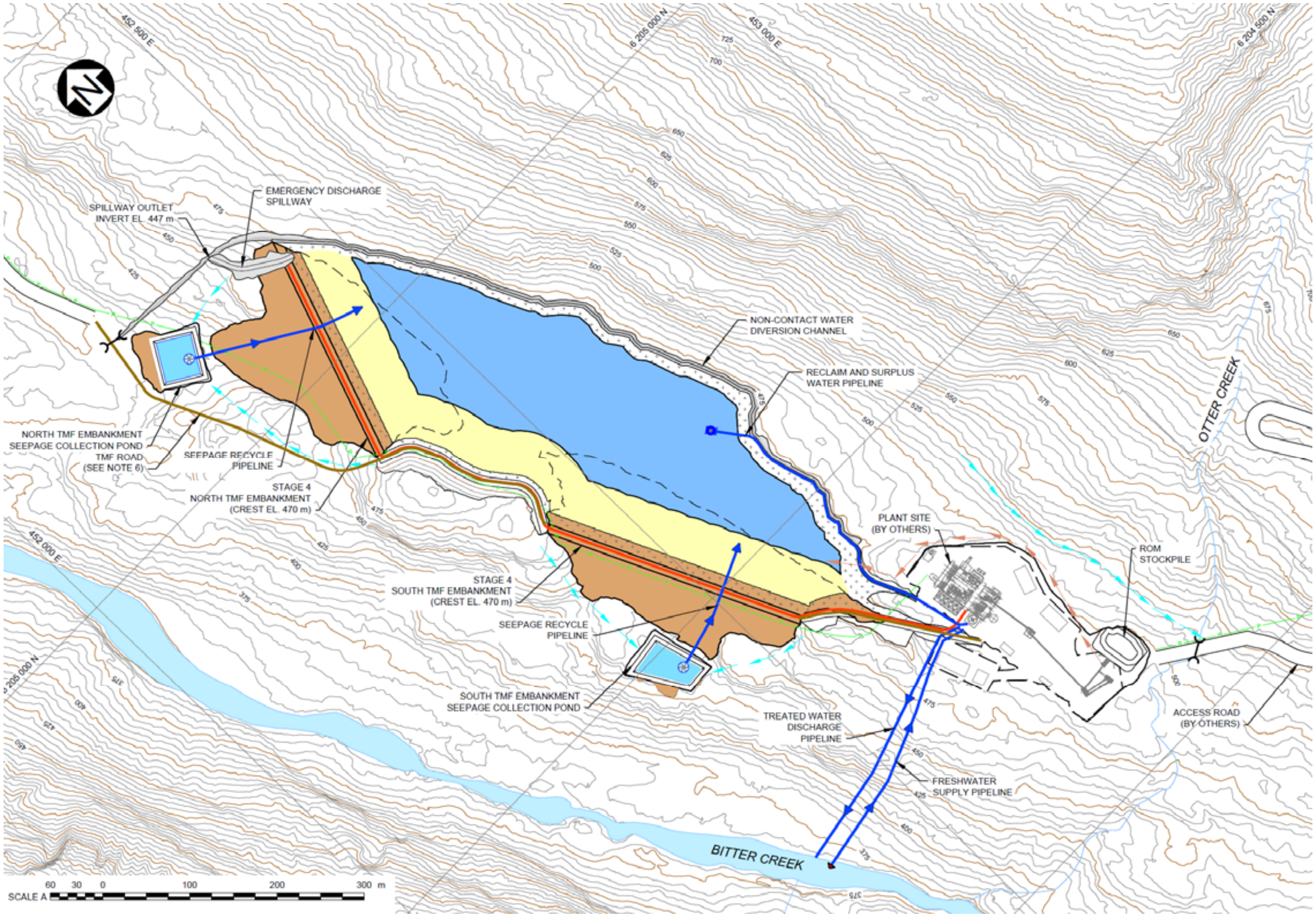
18.8.4.1 General

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

- Permanent, secure, and total confinement of all solid waste materials within an engineered disposal facility;
- Diversion of non-contact water around the TMF to the maximum practical extent;
- Control, collection, and removal of free draining liquids from the tailings during operations for recycling to the maximum practical extent;
- The inclusion of monitoring features for all aspects of the facility to ensure performance goals are achieved and design criteria and assumptions are met; and
- Staged development of the facility over the life of the Project.

The TMF will have a lined basin with two rock-fill/earth-fill dams (north TMF embankment and south TMF embankment). The embankments will be expanded during operations using the downstream method of construction. The final layout (stage 4) for the TMF is shown on Figure 18-4.

Figure 18-4: TMF General Arrangement (Stage 4)



Source: JDS (2017), KP (2017a)

18.8.4.2 TMF Geotechnical Conditions

Geotechnical investigations included 12 geotechnical drillholes at the TMF location in 2016 (KP, 2016a), logging core from four historical drillholes, and general mapping of the area.

The surficial geology of the area is characterized by landforms that reflect a history of glacial retreat. Drilling and mapping concluded that bedrock exposure is prevalent throughout the area.

Colluvium and glacial till are the predominant surficial materials (where present) and are generally contained to the TMF basin. Surficial material thickness ranges from 1 to 6 m, with the deepest deposits encountered in the centre of the north TMF embankment alignment.

Volcanic and sedimentary rocks were the predominant bedrock types encountered in the study area with some porphyry intrusive units. Intact bedrock strength data classifies the bedrock mass as strong to very strong rock with an overall FAIR Rock Mass Rating (RMR_{89}).

18.8.4.3 Design Basis & Operation Criteria

The basic design criteria for the TMF are summarized in Table 18-5. The design average throughput for the mill is approximately 1,000 t/d. The TMF impoundment is assumed to be fully lined with a geomembrane liner.

Table 18-5: Design Criteria Summary

Parameter	Units	Value
Average Mill Throughput	t/d	1,000
Design Life	yrs	6
Total Tonnes of Tailings (Design)	Mt	1.95
Year -1 (Pre-Production) – Tailings Tonnage	kt	3
Year 1 – Tailings Tonnage	kt	322
Year 2 – Tailings Tonnage	kt	366
Year 3 – Tailings Tonnage	kt	366
Year 4 – Tailings Tonnage	kt	366
Year 5 – Tailings Tonnage	kt	366
Year 6 – Tailings Tonnage	kt	158
Tailings Initial Settled Dry Density (estimated average)	t/m^3	1.21
Tailings Final Settled Dry Density (estimated average)	t/m^3	1.30
Embankment Crest Width	m	10
Embankment Upstream Slope	-	2.5H:1V
Embankment Downstream Slope	-	2H:1V
Environmental Design Flood (EDF) Volume	m^3	160,000
Probable Maximum Flood (PMF) Volume	m^3	889,000
Emergency Discharge Spillway Depth	m	2
Tailings Solids Content	%	50

Source: JDS (2017), KP (2017a)

18.8.4.4 TMF Features

The feasibility design is based on a mine life of six years and a total of 1.95 Mt of tailings, all of which will be stored in the TMF. The TMF has a storage capacity of 1.7 Mm^3 which includes 1.5 Mm^3 of tailings (1.95 Mt at an average settled density of 1.3 t/m^3), 80,000 m^3 for process water (three months of total process water), and 160,000 m^3 for the EDF as per CDA guidelines (CDA, 2014). The EDF was determined as the total runoff from the 1 in 50 year wet month plus the total runoff from a 1 in 200 year 24-hour precipitation event that bypasses the non-contact water diversion channel. Flood events exceeding the EDF event, up to and including the inflow design flood (IDF), will be passed through an emergency discharge spillway, located at the north TMF embankment.

The non-contact water diversion channel, located on the east side of the TMF, will collect runoff from the upstream catchment and safely divert it around the TMF. The channel is designed for the peak runoff from a 1 in 5 year 24-hour precipitation event, discharging to the Bitter Creek drainage area.

18.8.4.5 TMF Depth-Area-Capacity Relationship & Filling Schedule

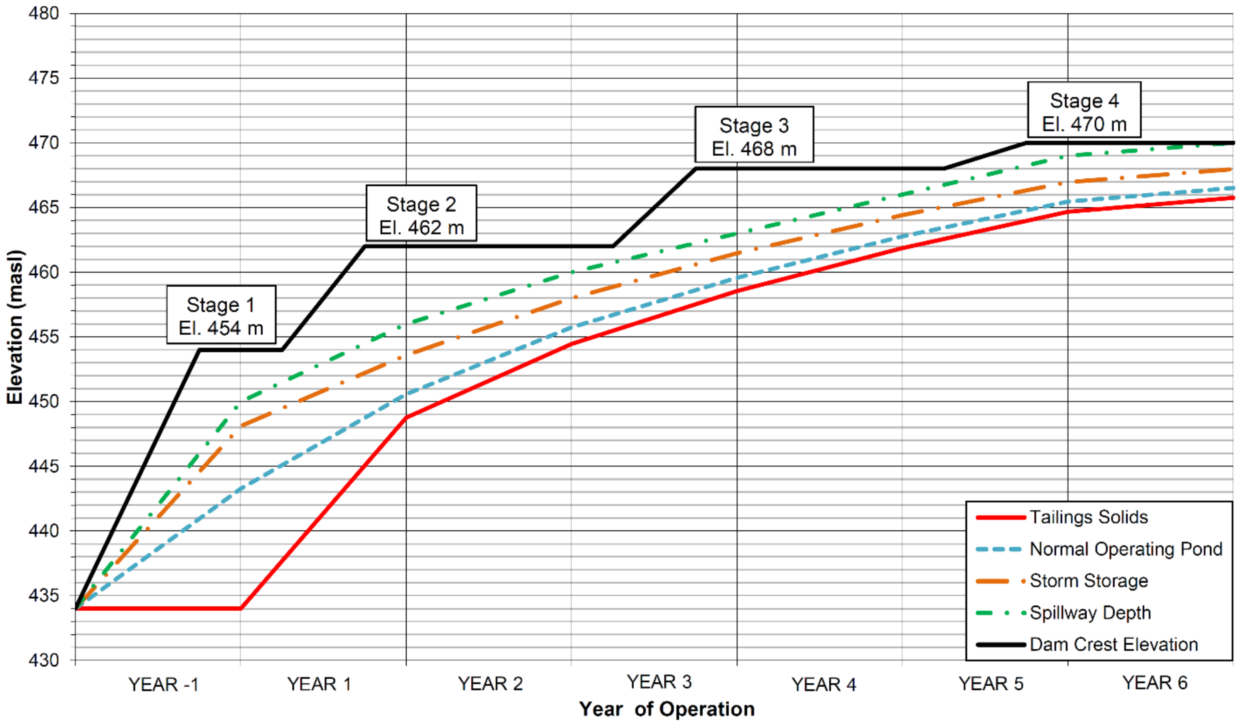
An initial starter dam (stage 1) will be constructed for the first year of operations. The TMF will be expanded in stages over the mine life. This staged approach offers the following advantages:

- The ability to refine the design, construction, and operating methodologies as experience is gained with local conditions and constraints;
- The ability to adjust plans at a future date to remain current with evolving best practices (engineering and environmental);
- To allow the observational approach to be utilized in the ongoing design, construction, and operation of the facility. The observational approach can deliver substantial cost savings and a higher level of safety. It also enhances knowledge and understanding of site-specific conditions; and
- The potential to reduce initial capital costs and defer capital expenditure relating to TMF construction until the mine is operating.

The stages are shown on the TMF filling schedule in Figure 18-5. The filling schedule and timing for staged expansions must be reviewed on an on-going basis during operations. The actual rate of filling may vary, depending on a variety of operating factors, including:

- Mill throughput;
- Settled tailings density; and
- Tailings surface slopes.

Figure 18-5: Filling Schedule



Source: KP (2017a)

18.8.4.6 TMF Embankment Construction Requirements

The TMF has two embankments: the north TMF embankment and south TMF embankment. These embankments will be constructed using material from locally borrowed sources including material generated during construction of the process plant site, TMF and haul road, or from other locally borrowed sources.

The upstream embankment slopes are 2.5H:1V to facilitate geomembrane placement. The downstream slopes are 2H:1V. The minimum embankment crest width is 10 m to allow working space for tailings, reclaim water pipelines, and traffic. The maximum embankment height is approximately 35 m.

The majority of fill for the stage 1 embankment will be general fill sourced from excavation of the process plant site platform. Subsequent embankment expansions will incorporate construction materials from local borrow pits and quarries along the haul road alignment.

The bulk embankment fill zone (zone C) will be general rock-fill/earth-fill. A layer of filter sand (zone F) will be installed across the TMF basin which will function as a geomembrane liner bedding.

18.8.4.7 TMF Seepage Control Measures

Potential seepage from the TMF will be controlled by incorporating the following measures:

- Geomembrane liner system;

- Basin underdrain; and
- Foundation drains.

These systems are described in the following sections.

18.8.4.7.1 Geomembrane Liner System

The entire TMF basin, including the upstream embankment faces, will be lined with 80-mil HDPE geomembrane. The liner system includes 12 oz/yd^2 non-woven geotextile for protection from the adjacent materials (below the geomembrane on the embankments slopes and below and above the geomembrane on the basin floor where an underdrain will be installed). The liner system also incorporates a prepared subgrade comprising processed bedding material (zone F). The geomembrane is effectively impermeable, with seepage only possible through defects that may occur during fabricate on and/or installation.

18.8.4.7.2 Basin Underdrain

An internal basin underdrain will be installed above the geomembrane on the basin floor to promote tailings consolidation. The basin underdrain will connect to an internal wet well sump and recycle pump system. Collected water will be recycled to the TMF supernatant pond.

18.8.4.7.3 Foundation Drain

Foundation drains will be installed below the geomembrane to collect groundwater flows, potential seepage, and infiltration through the TMF embankments. Collected water will drain to the seepage collection and recycle ponds downstream of the embankments. It will then be recycled to the TMF. The ponds will be lined with 80-mil HDPE geomembrane.

18.8.4.8 Tailings Distribution System

The tailings distribution system is designed to deliver the tailings to the TMF and to facilitate development of tailings beaches along the inside perimeter of the TMF embankments. The system will consist of three primary components: a tailings pump station, tailings conveyance pipeline, and discharge spigots. The tailings distribution system and the configuration of discharge spigots will evolve during operations as the TMF embankments develop and as operating procedures are refined.

Tailings will be delivered to the TMF through a single 100 mm diameter HDPE pipeline. Tailings will be discharged from the TMF embankment crests. 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, etc. Repeating this process will ensure a suitable tailings beach is established and that the pond is not against the embankments. Tailings will be selectively discharged to ensure a degree of saturation is maintained within the tailings mass to reduce ARD/ML generation potential.

18.8.4.9 Water Reclaim System

The water reclaim and surplus water management system will serve two purposes:

- To allow for the reclaim of supernatant for use in the mill; and
- To allow for the removal of surplus water for water treatment and environmental release.

The water reclaim and surplus water management system consists of a pump barge located on the TMF supernatant pond. One 150 mm diameter HDPE pipeline will extend from the barge to the reclaim water tank at the plant site. From this tank, water will be used in mill processing, or treated and discharged to Bitter Creek.

18.8.4.10 Seepage Collection & Recycle System

Seepage collection and recycle ponds will be located downstream of the north and south TMF embankments and will collect seepage from the TMF basin, runoff from the downstream TMF embankment slopes, and flow from the foundation drains. Water collected in these ponds will be recycled to the TMF supernatant pond using submersible pumps and HDPE pipelines.

The seepage collection and recycle ponds will be lined with an 80-mil HDPE geomembrane. The embankments for the ponds incorporate 2.5H:1V upstream and 2H:1V downstream side slopes. The ponds are designed to store the total runoff from a 1 in 200-year 24-hour rainfall event from their contributing catchments. Overflow from the seepage collection and recycle ponds will report to Bitter Creek should there be an event exceeding the design storm for the ponds.

18.8.4.11 Non-Contact Water Diversion Channel

The non-contact water diversion channel will collect non-contact runoff from the catchment upstream of the TMF and convey it around the TMF for discharge to the downstream environment. The diversion channel reduces the amount of runoff contributing to the TMF, in turn reducing the required capacity within the TMF for managing storm events.

The channel, and associated access berm, will be constructed out of fill material along the eastern slope of the TMF basin. The access berm is also part of the basin shaping/preparation required for geomembrane installation in the TMF basin. The channel will be lined with an 80-mil HDPE geomembrane to prevent erosion and will have a base width of 1 m and channel depth of 1 m. The channel, constructed to a 0.5% grade, will outlet to the existing drainage path, which reports to Bitter Creek downstream of the north TMF embankment.

18.8.4.12 Emergency Discharge Spillway

An emergency discharge spillway will be constructed at the north TMF embankment for each embankment stage. The TMF is designed to contain the normal operating pond level, with an additional allowance for the EDF. Flood events exceeding this volume, up to the peak runoff from a PMF event, will be conveyed from the TMF through the emergency discharge and will report to Bitter Creek. The spillway will be lined to prevent erosion and will have a base width of 5 m and channel depth of 2 m.

18.8.4.13 Instrumentation

Instrumentation installed in the TMF embankments and underlying foundations will be monitored during construction. Ongoing operations will assess performance and identify any conditions that differ from those planned for during design and analysis. Amendments to ongoing designs, operating strategies, and/or remediation work can be implemented to respond to changing conditions, should the need arise. The following types of instrumentation may be installed:

- Survey monuments: To evaluate the performance of the embankments with respect to movement, settling, etc.;
- Vibrating wire Piezometers: To monitor pore pressures within the TMF embankments to evaluate the performance of the geomembrane liner;

- Slope inclinometers: To monitor movement of the TMF embankments;
- Flow meters: To monitor effectiveness and performance of pipeline systems; an
- Pond level indicators: To monitor pond levels and assess performance and volume of supernatant pond.

18.8.4.14 Closure & Reclamation

The primary objective of the closure and reclamation initiatives will be to return the TMF site to a self-sustaining condition with pre-mining usage and capability. The reclaimed TMF will be required to maintain long-term geochemical and physical stability, protect the downstream environment, and shed surface water. Reclamation and closure will involve an active closure period and a post-closure period, in which all mine components will be prepared for permanent closure.

TMF closure and rehabilitation activities will be carried out progressively during the operations phase (where possible) and at the end of economically viable mining. Closure and rehabilitation activities will be conducted in accordance with international closure standards. Specifically, measures will be taken to ensure that:

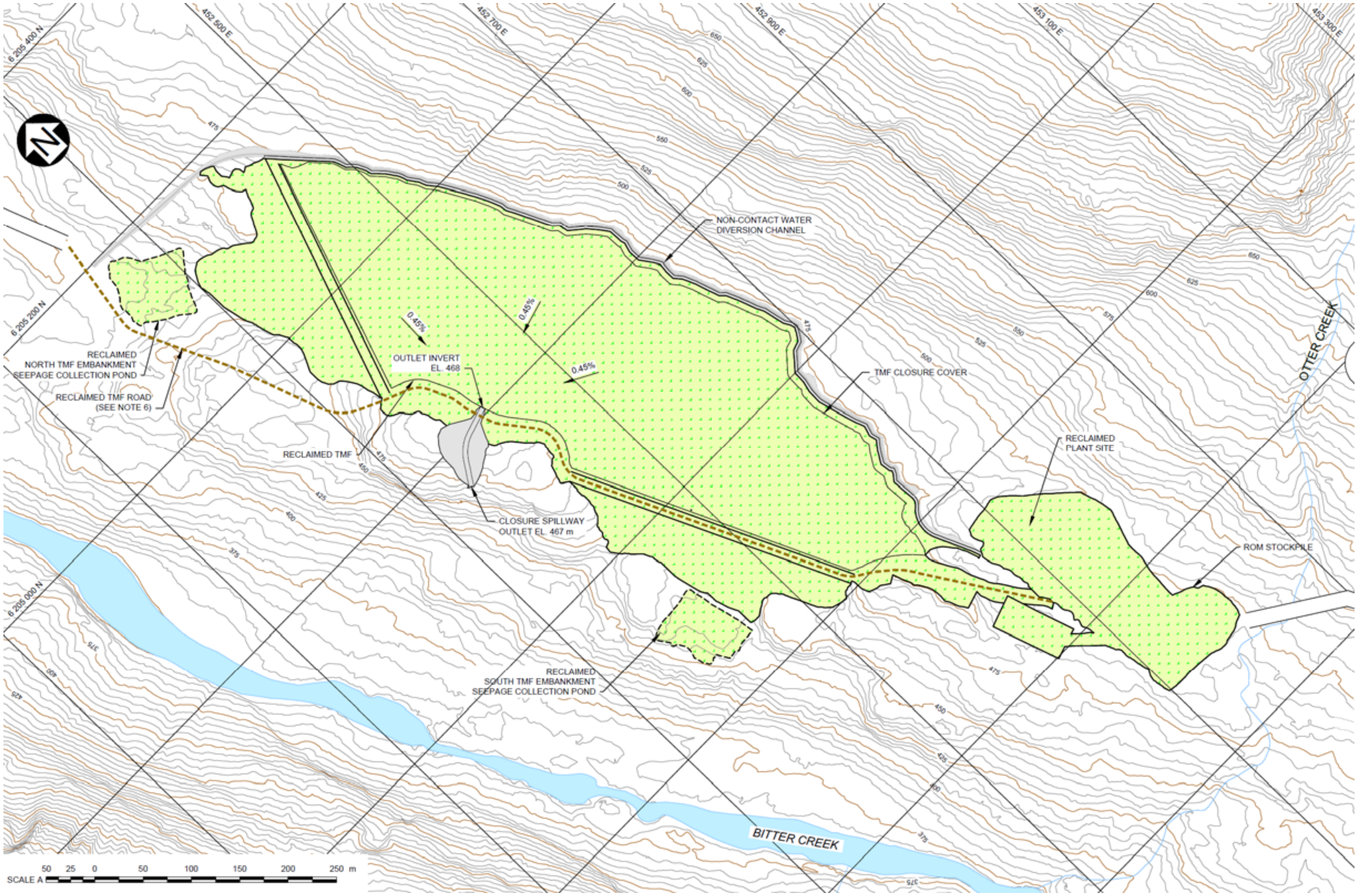
- Dust is not emitted from the facility as a result of moisture loss from the TMF surface;
- Runoff does not affect surface or groundwater;
- The TMF embankments remain stable; and
- The stored tailings remain physically and chemically stable.
- General aspects of closure will include:
- Selective discharge of tailings around the facility prior to closure to establish a final tailings beach that will facilitate surface water drainage and reclamation;
- Removal of surface water ponds and drainage of tailing waters;
- Dismantling and removal of the tailings and reclaim delivery systems and all pipelines, structures, and equipment not required beyond mine closure;
- Geomembrane capping of the tailings beaches and placement of a combined rock and soil cover that will shed runoff to a permanent spillway;
- Establishment of a permanent TMF spillway;
- Removal of the seepage collection pump-back systems at such time that suitable water quality for direct release is achieved;
- Removal and re-grading of all access roads, ponds, ditches, and borrow areas not required beyond mine closure; and
- 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 TMF. The 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 TMF and ongoing evaluation of water quality, flow rates, and instrumentation records to confirm design assumptions for closure.

Industry standard reclamation methods will be employed to close out the remainder of the Project sites. Hazardous materials will be collected for offsite disposal including hazardous components of vehicles and equipment (i.e., fuel tanks, gearboxes, and glycol-based coolant). Buildings and equipment stripped of hazardous components will be demolished and disposed in an approved landfill, located offsite. Culverts will be removed from roads and the natural drainage restored, but the roads will otherwise remain intact.

Once all buildings, facilities, and equipment have been removed, the footprints (whether bedrock or pads) will be re-contoured to allow for restoration of natural drainage to the receiving environment. The final reclaimed TMF surface is shown on Figure 18-6.

Figure 18-6: Reclaimed TMF General Arrangement



Source: JDS (2017), KP (2017a)

18.9 Plant Site Facilities

18.9.1 Process Building

The process plant will be located in a pre-engineered building. The building will measure 33 m wide by 70 m long by 21 m tall and will come equipped with a 10 t overhead crane for equipment maintenance. Figure 17-3 shows the planned layout of the process building.

18.9.2 Mine Dry & Office Complex

The 18 m x 18 m mine dry and 29 m x 18 m office complex will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will rest on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout.

The mine dry facility will service construction and operations staff during the life of the Project. It will be capable of servicing 100 workers during shift change and will contain the following:

- Male and female clean and dirty lockers; and
- Showers and washroom facilities with separate male and female sections.

The site office facility will contain the following items:

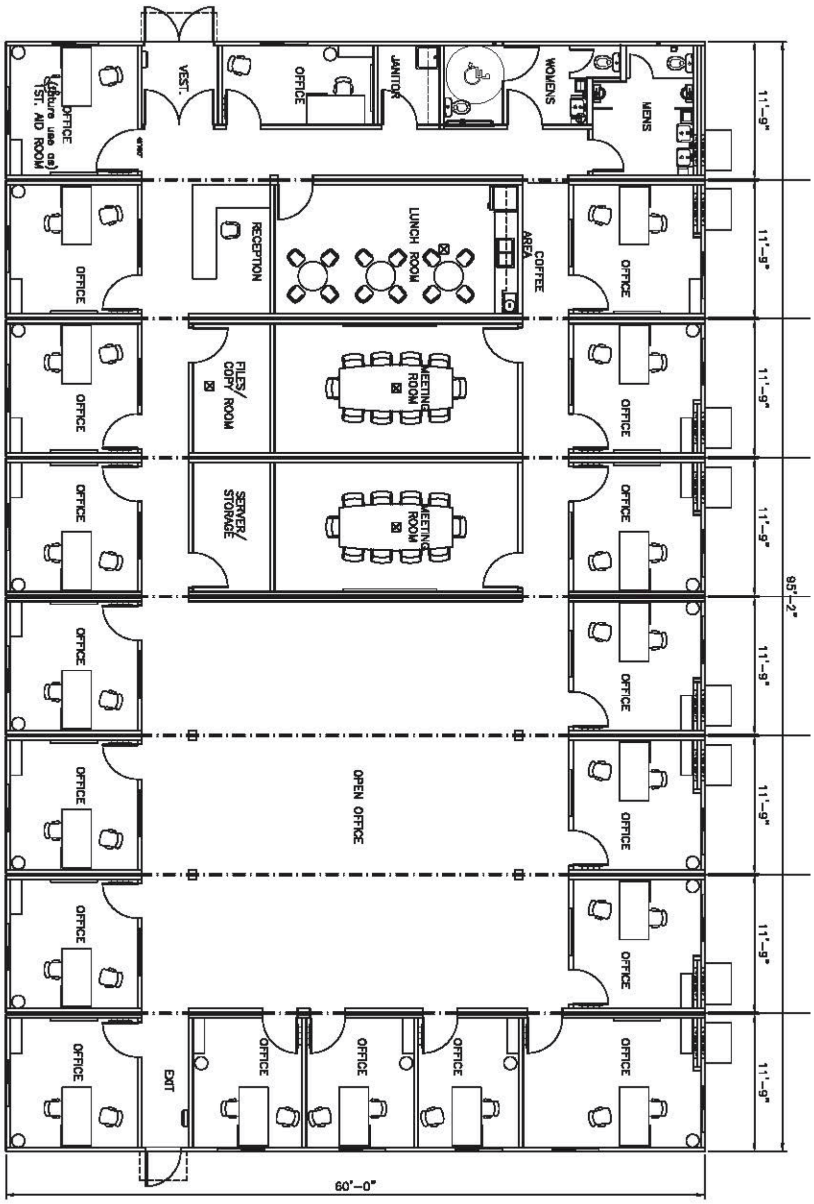
- Private offices;
- Main boardroom; and
- Mine operations line-up area.

Figures 18-7 and 18-8 show the mine dry and office complex respectively.

Figure 18-7: Mine Dry

Source: JDS (2017)

Figure 18-8: Office Complex



Source: JDS 2017

18.9.3 Water Treatment Plant

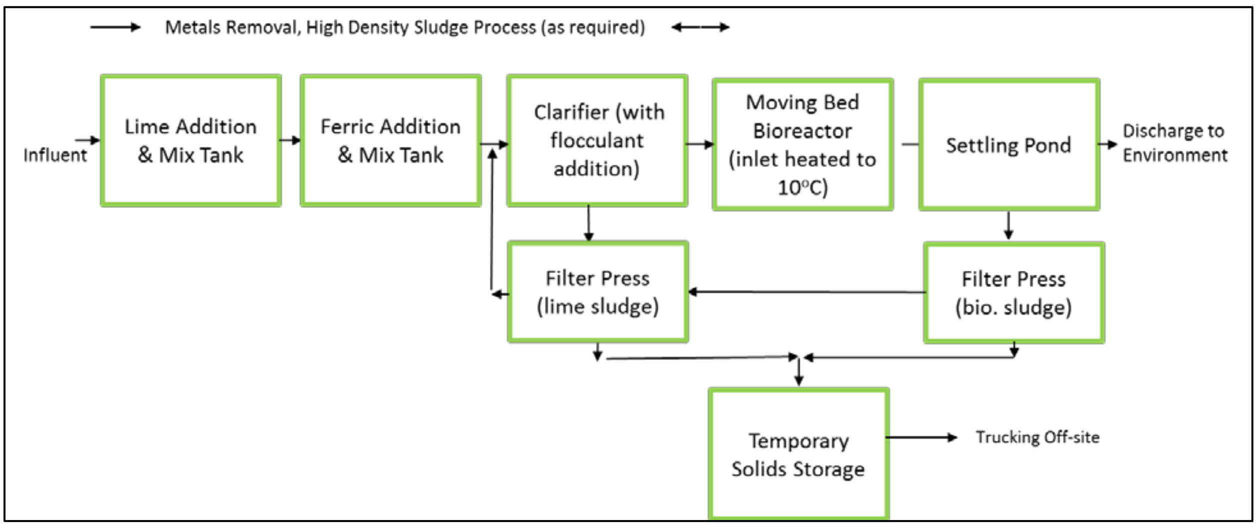
The Project requires a Water Treatment Plant (WTP) which will treat the surplus water from the TMF. Peak flows for the facility are expected to be 80,000 m^3/month. The treated water will be discharged into Bitter Creek where it will be monitored, ensuring that it meets BC water quality guidelines.

Process Overview:

- High density sludge for total suspended solids (TSS) and metals removal;
- Moving bed bio-reactor (MBBR) for ammonia removal;
- Sludge dewatering for solid residuals; and
- Water to be preheated to ensure adequate biological reaction kinetics are maintained.

A process schematic is shown in Figure 18-9.

Figure 18-9: Water Treatment Process



Source: Integrated Sustainability (2017)

18.9.3.1 TSS and Metals Treatment: Lime and Ferric Coagulation

The first stage of the proposed water treatment process is lime treatment with ferric coagulation and clarification. For this treatment stage, the influent water will be pumped into a mix tank where hydrated lime ($Ca(OH)_2$) will be added to increase the influent water pH to 9.0. At a pH of 9.0, dissolved metals such as copper, cadmium, and iron precipitate as metal hydroxide solids. The tank will be agitated mechanically to ensure proper mixing. The lime dosing system will consist of:

- Screw feeder;
- Agitated make-up tank;
- Transfer pump;
- Storage tank, metering pump; and
- Associated piping to the dosing location.

From the first mix tank, the water will flow to a second mix tank where ferric sulphate will be added. The addition of ferric sulphate serves the following purposes:

- Coagulation: ferric sulphate is a coagulant that binds precipitated metal hydroxides into larger particles; and

- Co-precipitation: the ferric added will form a ferric hydroxide precipitate ($Fe(OH)_3$) that acts as a media for adsorbing dissolved metals, including cadmium and arsenic.

The ferric sulphate reagent dosing system will consist of:

- storage tanks; and
- metering pump.

Following the ferric sulphate mix tank, the water will flow to a clarifier. Flocculant will be added to the clarifier influent pipe or clarifier centre well. The addition of the flocculant will result in the formation of larger particles and will enhance the settling of the precipitated metal hydroxide solids. The proposed flocculant dosing system will consist of a screw feeder, agitated make-up tank, transfer pump, storage tank, metering pump, and associated piping to the injection location. Flocculant reagents are used commonly in many different water treatment applications, both for drinking water and water released to aquatic environments. The WTP will use an anionic flocculant with low aquatic toxicity.

The precipitated metal solids will be collected as sludge in the clarifier. The sludge will be dewatered via a dedicated filter press to reduce the volume of treatment residuals that require disposal.

Assuming that the sludge can be dewatered to a density of 30% solids, it is estimated that approximately 16 m^3 of sludge per month will be generated in the proposed treatment process.

This sludge will mostly consist of ferric hydroxide and is expected to be non-hazardous. The dewatered sludge can either be mixed into the tailings or trucked away with other solid waste. The preliminary ferric co-precipitation sizing is presented in Table 18-6.

Table 18-6: Preliminary Ferric Co-Precipitation Tank and Clarifier Sizing

Parameter	Value	Units
Design Flow	3,000	m^3/day
Estimated Annual Treatment Requirement	500,000	m^3/year
Lime Tank		
Number of Equivalent Standard Tanks	1.0	
Residence Time	10	minutes
Standard Tank Volume (each)	20.8	m^3
Height to Diameter Ratio	1.0	
Tank Diameter	3.0	m
Tank Wetted Height	3.0	m
Tank Headspace	0.5	m
Total Tank Height	3.5	m
Lime Dosing Rate	300	kg/day, dry lime
Lime Dosing Rate	2,000	L/day, 14% solution
Ferric Sulphate Mix Tank		
Number of Equivalent Standard Tanks	1.0	
Residence Time	5	minutes
Standard Tank Volume (each)	10.4	m^3
Height to Diameter Ratio	1.0	

Table 18-6: Preliminary Ferric Co-Precipitation Tank and Clarifier Sizing (continued)

Parameter	Value	Units
Tank Diameter	2.4	m
Tank Wetted Height	2.4	m
Tank Headspace	0.5	m
Total Tank Height	2.9	m
Ferric Sulphate Dosing Rate	215	kg/day, dry
Ferric Sulphate Dosing Rate	135	L/day, 40% solution
Clarifier		
Rise Rate	1.0	m/hr
Clarifier Surface Area	125	m^2
Diameter	12.6	m
Flocculant Dosing Rate	6	kg/day, dry
Flocculant Dosing Rate	2,400	L/day, 0.25% solution

Source: SRK (2017)

The clarifier overflow will be treated in the MBBR biological system.

18.9.3.2 Ammonia Treatment: MBBR

The clarifier overflow will be pumped to the proposed MBBR process, which is an aerobic biological treatment system for removal of cyanide, cyanate, thiocyanate, and ammonia. The MBBR technology utilizes attached growth media that is circulated in an aerated reactor. Autotrophic microorganisms convert reduced ammonia species to nitrogen in a process known as nitrification. Air supplied to the MBBR by blowers and an air diffuser grid located on the bottom of the reactor vessel will keep the reactor aerated and will also mix the tank.

The pH of the reactors will be maintained at near neutral using lime. Micronutrients, such as phosphate, will be added as required. The micronutrients will be stored in a tote, and dosed to the MBBR tanks via a metering pump.

To treat the ammonia loading anticipated for the Red Mountain Project, a two stage MBBR system is proposed. The first stage would consist of two tanks operated in parallel, followed by a third tank in series. The MBBR media and tank preliminary sizing assumes that influent water is pre-heated to approximately 10^{o} Celsius, to allow suitable reaction kinetics for the biological attenuation. The MBBR media and tank preliminary sizing is summarized in Table 18-7.

Table 18-7: MBBR Media and Tank Preliminary Sizing

Parameter	Value	Units
MBBR Media		
Unit Media Area	700	m^2/m^3
Overall Performance	0.9	g/m^2/day @ 10^oC
Design Feed Ammonia (as N) Loading	300	kg/day
Average Feed Ammonia (as N) Loading	0.9	g/m^2/day @ 10^oC
Media Volume, Total	476	m^3
Media Surface Area, Total	333,333	m^2
MBBR Tanks		
Number of Tanks	3	Two in parallel followed by one in series
Media Occupation	60%	% of total tank volume
Tank Volume (per tank)	794	m^3
Reactor Height: Diameter	1.0	
Tank Diameter	7	m
Tank Wetted Height	7	m
Tank Headspace	1	m
Total Tank Height	8	m
Lime Dosing Rate	3,300	kg/day, dry lime
Lime Dosing Rate	21,800	L/day, 14% solution
Monopotassium Phosphate Dosing Rate	85	kg/day, dry
Monopotassium Phosphate Dosing Rate	1,630	L/day, 5% solution
Flocculant Dosing Rate	3	kg/day, dry
Flocculant Dosing Rate	1,200	L/day, 0.25% solution

Source: SRK (2017), Integrated Sustainability (2017)

18.9.3.3 Effluent Pond or Clarifier

Following the second stage of the MBBR system, the treated effluent will be piped to an effluent pond or clarifier prior to discharge to the receiving environment. The effluent pipe from the MBBR system will be dosed with flocculant to enhance settling of the biomass generated from the MBBR system. The effluent pond or clarifier will serve as a final settling stage for removal of biomass. From the effluent pond, the treated water will be discharged to the receiving environment.

18.9.3.4 Water Treatment Design Basis

18.9.3.4.1 Influent Flow Rate and Water Quality

The design basis for the proposed water treatment system is based on results of the water balance and water quality model developed for the Project.

The potential requirement to capture and treat mine contact water is driven by the concentrations of ammonia, total suspended solids, copper, and iron predicted by the water quality model. Model predictions of water treatment plant influent water chemistry is summarized in Table 18-8.

Table 18-8: WTP Design Basis

Parameter	Influent Concentration to Water Treatment System
Physical Parameters	
Conductivity	10400
pH	7.85
Anions and Nutrients	
Acidity (as CaCO3)	20
Alkalinity, Total (as CaCO3)	102
Ammonia, Total (as N)	122
Ammonia + Degradation Products (as N)	285
Bromide (Br)	1
Chloride (Cl)	23
Fluoride (F)	0.4
Nitrate (as N)	12
Nitrite (as N)	0.79
Sulfate (SO4)	1970
Cyanide	
Cyanide, Weak Acid Diss.	0.038
Cyanide, Total	0.89
Cyanate	381
Thiocyanate (SCN)	151
Dissolved Metals	
Aluminum (Al)-Dissolved	0.055
Antimony (Sb)-Dissolved	2.4
Arsenic (As)-Dissolved	0.014
Barium (Ba)-Dissolved	0.10
Beryllium (Be)-Dissolved	0.0005
Bismuth (Bi)-Dissolved	0.00025
Boron (B)-Dissolved	1.1
Cadmium (Cd)-Dissolved	0.0013
Calcium (Ca)-Dissolved	167
Cesium (Cs)-Dissolved	0.00023
Chromium (Cr)-Dissolved	0.00075
Cobalt (Co)-Dissolved	0.084
Copper (Cu)-Dissolved	0.46
Iron (Fe)-Dissolved	0.78
Lead (Pb)-Dissolved	0.0019
Lithium (Li)-Dissolved	0.023
Magnesium (Mg)-Dissolved	12
Manganese (Mn)-Dissolved	0.23

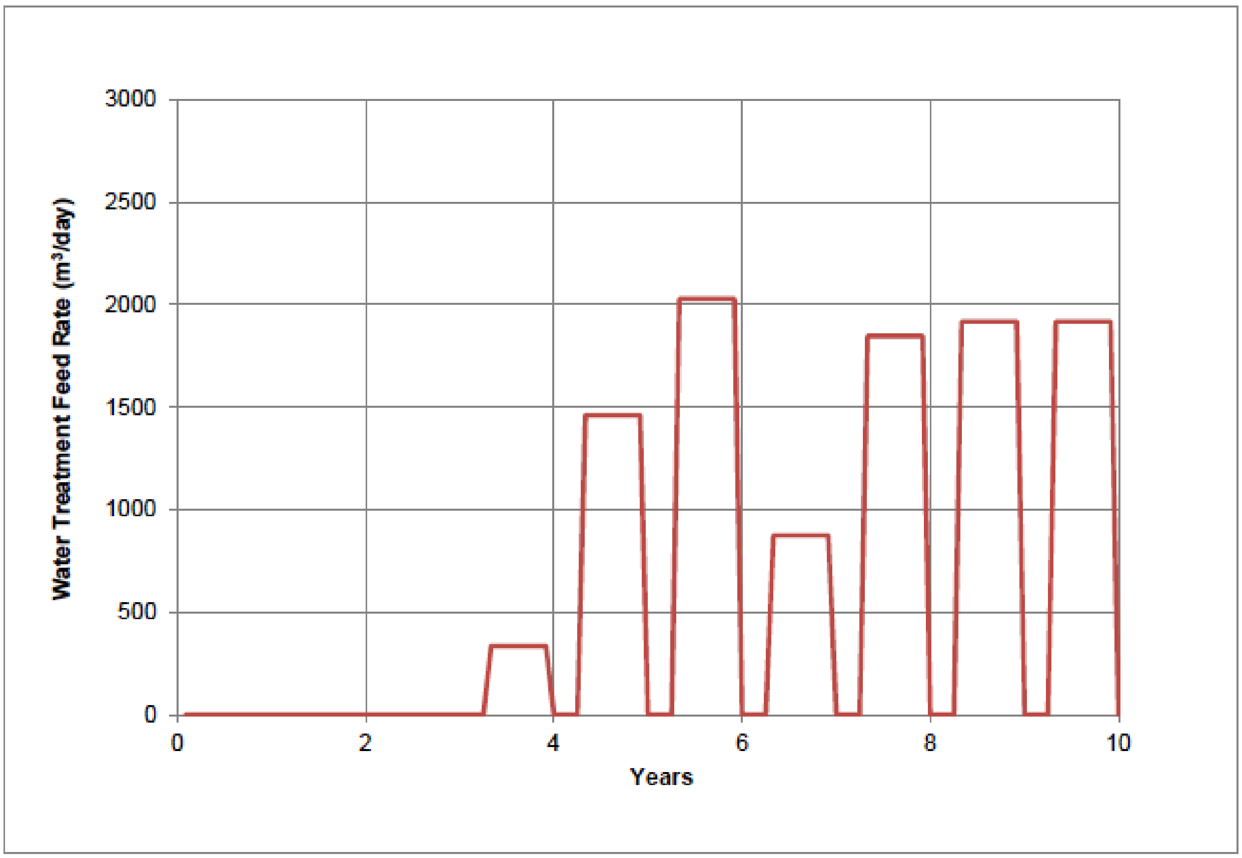
Table 18-8: WTP Design Basis (continued)

Parameter	Influent Concentration to Water Treatment System
Mercury (Hg)-Dissolved	0.00015
Molybdenum (Mo)-Dissolved	0.16
Nickel (Ni)-Dissolved	0.004
Phosphorus (P)-Dissolved	0.25
Potassium (K)-Dissolved	120
Rubidium (Rb)-Dissolved	0.026

Source: SRK (2017)

The estimated water treatment design capacity is based on the results of the site wide water balance. Predicted water treatment influent flow rates are shown in Figure 18-10.

Figure 18-10: Design Influent Flow Rate to WTP (Model Years)



Source: SRK (2017)

18.9.3.4.2 Effluent Quality

Concentrations of TSS, ammonia, and copper in the TMF pond is expected to exceed the Metal Mining Effluent Regulations (MMER) limits. Treatment is also required for iron to meet receiving water guidelines in Bitter Creek, which include British Columbia Ministry of Environment Water

Quality Guidelines for Freshwater Aquatic Life (BC WQGs) and Canadian Council of Minister of the Environment (CCME).

A summary of the expected post-treatment concentrations of the parameters described above is provided in Table 18-9.

Table 18-9: Expected WTP Treated Effluent Chemistry

Parameter	Post-Treatment Expected Concentration (mg/L)	Notes on Expected Effluent Concentration
Ammonia (as N)	5	MEND, 2014: typical precious metal sector treated effluent (average concentration)
Dissolved Iron	0.1	MEND, 2014: typical precious metal sector treated effluent (average concentration)
Dissolved Copper	0.03	Based on analog treated effluent of similar treatment plants and influent water chemistry
Total Suspended Solids	15	Required to meet MMER

Source: SRK (2017)

18.10 Ancillary Facilities

18.10.1 Truck Shop & Warehouse

The surface maintenance shop at site will consist of a 30 m long by 20 m wide insulated fabric structure to accommodate repair and maintenance of mining equipment and light vehicles. More extensive repair work will be conducted off-site. The total floor area of the truck shop will consist of four truck bays.

Equipment will be washed underground. Tire repair will be done outside, weather permitting. In poor weather, tire repair will be done in the shop with the appropriate safety measures, such as personnel access control and clearances.

The warehouse consists of a 15 m x 12 m insulated fabric structure that will house spare parts, consumables, and other materials.

18.10.2 Assay Lab

An assay laboratory will be located adjacent to the process plant. This facility will serve the plant's assay, environmental, and metallurgical requirements. The laboratory will consist of pre-fabricated modules and ancillary equipment, such as drying ovens, dust and fume control, and heating equipment.

18.10.3 Fuel Storage & Distribution

Diesel will be trucked to the Project site from Stewart on an as-needed basis and stored in a 100,000 L Enviro-Tank, including an integrated dispensing system. Surface mobile equipment would fuel-up at the storage tank and fixed equipment would be supplied by the fuel and lube truck. Fuel will be shuttled via fuel truck to a 20,000 L Enviro-Tank located at the upper portal.

18.10.4 Potable Water

Water will be pumped from Bitter Creek to the potable water treatment plant. The plant will be contained in a 20 ft shipping container assembled prior to shipment to site. It will contain the complete treatment system including filtration, UV disinfection, and chlorine disinfection.

Treated water from the potable water plant will be stored in an insulated and heated potable water storage tank which will accommodate the potable water demand variances and then be distributed to the facilities.

18.10.5 Sewage Collection & Treatment

Sewage will be stored in the tanks, and will be pumped out and trucked to Stewart for disposal.

18.11 Off-Site Infrastructure

18.11.1 Camp Accommodations

The construction camp will be located in Stewart, BC where it will utilize an existing camp complex of 49 beds with the temporary addition of 44 new beds to accommodate the 93 required beds at Red Mountain. It has been assumed that 25% of the labour force will be local and will not be staying in the camp.

The kitchen/dining/recreation complex will include the following:

- Kitchen complete with cooking, preparation, and baking areas, dry food storage, and walk-in freezer/cooler. The kitchen will be provided with appropriate specialized fire detection and suppression systems;
- Dining room with serving and lunch preparation areas;
- Mudroom complete with coat and boot racks, benches, and male-female washrooms;
- Housekeeping facilities;
- Reception desk and lobby; and
- Recreation area.

The camp will be constructed from modular units manufactured off-site in compliance with highway transportation size restrictions. Camp modules will rest on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout.

19 Market Studies and Contracts

19.1 Market Studies

At this time, no market studies have been completed as the gold to be produced at Red Mountain can be readily sold in the open market. Gold refining charges were estimated to be US$5.00/payable oz. Silver refining charges were estimated to be US$0.50/payable oz.

19.2 Contracts

No contractual arrangements have been made for the sale of gold doré at this time.

19.3 Royalties

The Project was evaluated utilizing the following royalties:

- 1.0% NSR royalty to Franco-Nevada;
- 2.5% NSR royalty to Wotan; and
- 10% of annual payable gold is sold to Seabridge at a discount price of US$1,000 (capped at a total of 50,000 oz).

19.4 Metal Prices

The base and precious metal markets benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong) and fluctuate on an almost continuous basis. Historical metal prices for gold and silver are shown in Figure 19-1 and Figure 19-2 respectively, and demonstrate the change in metal prices from 2000 through to May 2017.

Metal pricing used in the economic analysis are similar to other recently published technical report values and represent the approximate 18-month trailing averages. See Figure 19-3 for exchange rate trends.

The reader is cautioned that the metal prices and exchange rate used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. Metal prices and exchange rates are based on many complex factors and there is no reliable method of predicting them.

Figure 19-1: Average Gold Cash Price as at May 2017

Source: Kitco.com (2017)

Figure 19-2: Average Silver Cash Price as at May 2017

Source: Kitco.com (2017)

Figure 19-3: Average F/X Rate as at May 2017

Source: Bank of Canada (2017)

Table 19-1 summarizes the metal prices and exchange rates assumed for the economic analysis. A review of recent Technical Reports was conducted and the rates in Table 19-1 were found to be representative of current market conditions.

Table 19-1: Metal Price & Foreign Exchange Rates used in Economic Analysis Scenarios

Parameter	Unit	Base Case
Gold Price	US$/oz	1,250
Silver Price	US$/oz	17.00
Exchange Rate	US$:C$	0.76

Source: JDS (2017)

20 Environmental Studies, Permitting & Social or Community Impact

20.1 Overview

Major mining projects in BC are subject to environmental assessment and review prior to certification and issuance of permits to authorize construction and operations. Environmental assessment is a means of ensuring the potential for adverse environmental, social, economic, health, and heritage effects or the potential adverse effects on Aboriginal interests or rights are addressed prior to project approval. Depending on the scope of a project, assessment and permitting of major mines in BC will proceed through the BC EA process pursuant to the BCEAA and the CEAA (2012).

Pursuant to section 3(1) of the Reviewable Projects Regulation, the proposed production capacity for the Project exceeds the threshold criterion of 75,000 t/a (or 205 t/d) of mineral material for a new mineral mine. The Project will thus require a provincial Environmental Assessment Certificate, because its proposed production rate exceeds the specified threshold. At a federal level, proposed gold mine developments (other than placer mines) that exceed a threshold criterion of 600 t/d, as specified under the Regulations Designating Physical Activities, are required to complete an EIS pursuant to the CEAA (2012). Thus, completion of an EIS will be necessary for the Project since the proposed production rate exceeds the specified threshold. IDM has formally entered both the provincial and federal assessment process in October 2015 with the filing of a Project Description Report.

Since that time, a number of steps in the process have been undertaken successfully and IDM has filed a Project Application Report in July 2017 that will fulfill the requirements of the federal and provincial environmental assessment processes. Provincial and federal decisions for the Project under BCEAA and CEAA are expected in the second quarter of 2018. Provincial approval of the Environmental Assessment Certificate and federal approval of the EIS will then allow for the issuance of the necessary statutory permits and authorizations to commence construction of the Project. Permitting for the Project is being pursued in a synchronous manner with the environmental assessment process.

20.2 Environmental Studies

Environmental studies at the Red Mountain Gold Property were completed at various times by different operators. In general, data collection occurred between 1990 and 1992 by Hallam Knight and Piésold for Bond Gold, in 1993 and 1994 by Rescan for LAC Minerals, and in 1996 and 1997 by Royal Oak. Subsequently, many engineering and environmental studies have utilized this data. The historic environmental database was utilized for initiating an environmental assessment in 1996 by Royal Oak. The environmental studies included sampling and assessment of water quality, climate, hydrology, hydrogeology, wildlife and vegetation, fisheries, ARD/ML, terrain stability, socio-economics, and culture and heritage. The available information indicates that the effects of the Project on the environment can be mitigated to meet regulatory requirements.

IDM has completed a gap analysis of all previously available baseline studies; this resulted in additional studies being conducted in 2015 – 2017, which are summarized in Table 20-1. These additional studies were conducted in order to update the baseline to current environmental conditions, to address refinement of the Project design, and to reflect current regulatory requirements. The gap analysis indicated that additional information was required for the Project area atmosphere/climate, surface hydrology, aquatics, water quality, sediment quality, terrestrial wildlife, and fish habitat. IDM has completed comprehensive studies of rock geochemistry, archaeology and heritage resources, land use, cultural, and socio-economic baseline to characterize the regional human environment. Where available, traditional ecological knowledge functions were supplied as additional information for the assessment of the Project's effects. Further, IDM Mining is currently working with NLG to finalize the ecological and socio-economic assessments that are required under Chapter 10, paragraphs 8(e) and 8(f) of the NFA. Mitigation measures are being developed in consultation with NLG through the environmental assessment process.

Table 20-1: Priority Studies Currently Underway

Baseline Component	Additional Information
Terrain and Physiography	Mapping to reflect changes as the glaciers of the Cambria Icefield have retreated. Natural hazards assessments in the Bitter Creek Valley proximal to the proposed mine and access route.
Water Quality	Water quality data collection to address gaps and Project design refinements.
Climate	Meteorological monitoring to extend historical data.
Hydrology	Monitoring of stream flow to support Project design, fisheries and water quality assessments.
Hydrogeology	Monitoring of groundwater to support Project design, fisheries, and water quality assessments.
Wildlife and Vegetation	Baseline studies of wildlife for the Bitter Creek Valley to support the assessment of Project effects.
Fisheries	Baseline studies of fisheries for the Bitter Creek and Bear River to support the assessment of Project effects.
ARD/ML	Testing of tailings sample to assess the effects of possible ARD/ML and mitigate the effects to support wastewater quality assessment and water quality predictions associated with various disposal options.
Terrain Stability Assessment	Terrain stability assessment along road corridor and near Project facilities.
Socio-economics	Baseline assessment of the socio-community and economic characteristics of the area to support an assessment of Project effects.
Culture and Heritage	Archaeological overview assessment pursuant to the BC Heritage Act, and to address the interests and concerns of Aboriginal Groups.
Treaty and Aboriginal Interests	Consultation with Aboriginal Groups in the Project area - specifically Nisga'a Nation - are ongoing.

Source: IDM (2017)

20.3 Land Capability & Use

Gold was first discovered at Red Mountain in 1965 and mineral exploration in the area dates back to the late 19th century. The region has a rich history of mining that includes past and present operations such as the historic Premier Gold Mine, Dolly Varden Silver, Eskay Creek, Snip, and current major deposits and projects including Galore Creek, Red Chris, Brucejack, and KSM.

Forestry production in the Red Mountain 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-Alice Arm area. Agriculture potential in the study area is also limited by poor soil conditions, marketing restrictions, and short growing seasons (Royal Oak, 1996).

Other resource interests overlapping with the Project area include one guide outfitter concession (601036) and two traplines (TR0614T101, TR0614T094), as well as one commercial recreation licence (910116) for a heli-ski operation.

20.3.1 Vegetation

The Red Mountain site lies within the alpine area above the local treeline, which occurs at approximately 1300 m in the Coastal Mountain-heather Alpine biogeoclimatic zone. The Bitter Creek Valley contains two major biogeoclimatic zones, namely the Coastal Western Hemlock along the valley floor and the Mountain Hemlock at mid-elevations (BC Ministry of Forest and Range Kalum Wall Map, 2008).

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 shrubs (BC Ministry of Forest and Range, 2006).

The Coastal Western Hemlock landscape, at low elevations in the Bitter Creek Valley (Bromley Humps area), 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 (BC Ministry of Forest Bro `31, 1999).

The Mountain Hemlock 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 (BC Ministry of Forest Bro 51, 1997).

20.3.2 Wildlife

The larger wildlife species present within the proposed Project area and 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.

20.3.3 Fisheries & Aquatic Resources

The Project area lies within the Bitter Creek drainage basin, a tributary of Bear River with a confluence near Highway 37A, downstream of the Project site. Bear River contains a majority of the known fisheries resources in the Project area, which include coho salmon, steelhead, pink salmon, chum salmon, sockeye salmon, and trout.

Fish resource and fish habitat studies conducted in the 1990's and 2014 to 2016 indicate that there is limited usage and available fish habitat in Bitter Creek. Dolly Varden is the main fish species to use Bitter Creek and the lower reaches of Roosevelt Creek. There is a series of fish barriers about midway up the Bitter Creek valley, and downstream of the proposed mill site and tailings management area. Thus, no major Project infrastructure is located near fish-bearing reaches of Bitter Creek.

20.4 Economic Impacts

The Project is expected to provide economic benefits to the local communities as a result of direct training and employment opportunities, as well as indirect employment. The company expects to provide seasonal employment for up to 249 people during the 14-month construction phase. During the six year mine operational phase, full-time employment of up 230 people is expected.

The overall economic impacts to the District of Stewart, approximately 20 km from the proposed mine site, as well as nearby communities and the province are expected to be beneficial. The larger and more distant communities of Terrace and Smithers have adequate facilities and infrastructure to absorb potential impacts of Project development, particularly as these are expected to revolve around company and employee purchases of goods and services. Stewart is likely to experience the most direct economic impacts from Project development as a result of the expected increase in employee and company expenditures.

Additional indirect employment opportunities such as goods and services contracts will increase, creating growth in the local, regional, and provincial economies. The Project will also generate annual revenues associated with property tax, licensing fees, royalties, and income tax for local, provincial, and federal governments.

20.4.1 Social Community

The workforce for operations is expected to live in Stewart, which has sufficient facilities and infrastructure to accommodate the potential increase in residents during operation of the Project. Services provided by government agencies, communication and media, commercial operations and transportation would continue to adequately serve the increased population.

Power, water supply, solid waste management services, and community services and infrastructure currently available in Stewart are adequate to provide for the population increase associated with the Project. Stewart is served by an elementary and a secondary school, both of which are operating below capacity. The Stewart Health Care Centre provides complete health services and is designed to accommodate a community of up to 5,000 residents. According to Statistics Canada, the population of the District of Stewart was 494 in 2011.

20.4.2 Aboriginal Land Use

The Red Mountain property falls within the Nass Area and Nass Wildlife Area, as set out in the NFA. Pursuant to the NFA, Nisga'a Nation, as represented by NLG, has Treaty rights to the management and harvesting of fish, wildlife, and migratory birds within the Nass Area and the Nass Wildlife Area.

Nisga'a citizens are proud of their close ties to the land and practice cultural activities, including seasonal resource harvesting of terrestrial and marine plants, hunting and trapping wildlife and migratory birds, and fishing. NLG's website outlines the importance of the land to Nisga'a Nation's culture, governance, and survival. NLG stresses the importance of the Nisga'a system of land ownership that "sets the economic rules" and the "social foundation" for their society. They write:

The system clearly laid out the rules of access to the rich economic resources of the Nisga'a lands — who has right to go where — and thereby protected against internal strife. People knew the rules for using an area and proper behaviour on the land; access to particular land areas and its resources weren't a matter of battles, you simply had to ask. This is the kind of control and laws which are, in reality, the essence of government. For the Nisga'a the laws of government and property are integral to the structure of society and family relation. (Nisga'a Lisims Government, a)

NLG also writes about the importance of the land as the staging ground of Nisga'a Nation's cultural histories (adaawak), as the territory of the matrilineal houses (wilp), and the direct relationship of Nisga'a citizens to the land and animals by way of the four tribes (pdeek): Raven/Frog (Ganada), Wolf/Bear (Laxgibuu), Killer Whale/Owl (Gisk'aast), and Eagle/Beaver (Laxsgiik). Traditionally, under the system of Ango'oskw (resource holding), a Nisga'a hunter or fisherman would seek permission of the chief of the wilp to use the natural resources found in their territory. This process was "an important display of kwhlixhoosa'anskw (respect)."

Pursuant to the Nass Area Strategy, implemented in 2008 in response to resource development in the Nass Area, "only environmentally sound resource development projects that are consistent Nisga'a Nation Treaty rights will proceed" (Nisga'a Lisims Government, b). The Nisga'a Nation strives for sustainable prosperity and self-reliance, working with partners to build:

- Forest products;
- Fish and seafood products;
- Telecommunications;
- Hydroelectric power generation;
- Mineral resource development;
- Land lease; and
- Tourism.

In addition to the cultural importance of the land, Nisga'a Nation is active in modern economic resource development including forestry tenures, commercial recreation, angling licences, and trap lines.

20.4.3 Government

The District of Stewart covers a large area around the town of Stewart. The District is governed by an elected Mayor and Council. The local government is generally in favour of responsible commercial and resources development in the area.

The Project is outside the boundaries of the District and it is unlikely that the District would obtain permission from the provincial government for expansion of the District to include the Project site.

Regionally, the District of Stewart is a part of the Regional District of Kitimat-Stikine, which provides government services to the area.

20.5 Environmental Approvals

The Project will require a review by the British Columbia EAO pursuant to the BCEAA and the Canadian Environmental Assessment Agency pursuant to the Canadian Environmental Assessment Act 2012 (CEAA, 2012) to determine whether the Project can be issued an Environmental Assessment Certificate. A Mines Act Permit, from the BC Ministry of Mines, and an Environmental Management Permit, from the BC Ministry of Environment is required for commercial production. Various baseline studies to support the assessment by the EAO have been undertaken by several former property operators; an extensive program has been ongoing, undertaken by IDM, since 2014.

No technical or policy issues are anticipated for obtaining the required permits and approvals for this Project. A reclamation bond must be deposited with the government on the issuance of the Mines Act permit. It is anticipated that the cost of the bond will increase from the $1 M that is already held by the Ministry of Energy and Mines.

20.6 Anticipated Provincial Permits & Authorizations

Provincial permitting and licencing (statutory permit processes) will proceed in a synchronous manner with the environmental review pursuant to the BCEAA and CEAA. No permits for the commercial development of the Project will be issued before an Environmental Assessment Certificate (EAC) is obtained. Consequently, IDM will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process following issuance of the EAC and reduce the time to start construction.

Table 20-2 presents a list of provincial authorizations, licences, and permits required to develop the Project. The list includes only the major permits and is not intended to be comprehensive.

Table 20-2: List of Anticipated Provincial Permits & Authorizations

Permits	Agency	Legislation
Environmental Assessment Certificate	BC Environmental Assessment Office	Environmental Assessment Act
Licence of Occupation	Ministry of Forests, Lands and Natural Resource Operations	Land Act
Licence to Cut	Ministry of Forests, Lands and Natural Resource Operations	Forestry Act
Burning Reference Number	Ministry of Forests, Lands and Natural Resource Operations	Wildfire Act
s.9 Approval or Authorization for Changes in and About a Stream	Ministry of Forests, Lands and Natural Resource Operations	Water Act
s.8 Water Use Approval	Ministry of Forests, Lands and Natural Resource Operations	Water Act
Mining Lease	Ministry of Energy and Mines	Mineral Tenure Act
Mines Act Permit	Ministry of Energy and Mines	Mines Act
Mining Right of Way Permit	Ministry of Energy and Mines	Mining Right of Way Act
Food Premises Permit	Northern Health Authority	Public Health Act-Food Premises Regulation and DWP Act
Filing of Certification Letter	Northern Health Authority	Public Health Act - Sewage Disposal Regulation
Operating Permit	Northern Health Authority	Drinking Water Protection Act and Regulation
Utility Permit	Ministry of Transportation and Infrastructure	Transportation Act, Motor Vehicle Act
Hazardous Waste Registration	Ministry of Environment	Environmental Management Act - Hazardous Waste Regulation
Fuel Storage Permit	Ministry of Environment	Environmental Management Act
Effluent Discharge Permit	Ministry of Environment	Environmental Management Act
Operating Permit	Ministry of Energy and Mines	Mines Act

Source: JDS (2017)

20.7 Anticipated Federal Permits & Authorizations

Federal authorizations and permits that may be required are listed in Table 20-3.

Table 20-3: List of Anticipated Federal Permits & Authorizations

Permits	Agency	Legislation
Explosives Permit	NRCan	Explosives Act
EA Approval	CEAA	Canadian Environmental Assessment Agency

Source: JDS (2017)

20.8 Mine Closure

A detailed closure and reclamation plan has been developed as part of the EA and will be refined as part of the provincial permitting process. In summary, the mine closure concept is to meet water quality objectives without ongoing treatment for acid rock drainage. This will be achieved by placing all of the potentially acid generating waste rock underground. Waste rock will either be mixed with lime or placed as cemented rock backfill. Following closure, the three underground portals will be hydrostatically sealed with engineered bulkheads to allow the mine to flood. Any potentially acid generating mine tailings left on surface will be stored in an engineered lined TMF. The TMF supernatant pond will be drained and a geosynthetic liner installed over the surface of the exposed tailings beach. A graded earthfill/rockfill cover will be constructed on top of the liner and revegetated to facilitate runoff from the surface of the reclaimed TMF towards a permanent closure spillway and to minimize infiltration.

The structures on the Project 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.

Concrete slabs, footings, and retaining walls will be taken apart by drilling and blasting or with a hydraulic excavator outfitted with a rock breaker. Concrete fragments will be placed underground.

After removal of the process building, equipment, and foundations, a soil sampling program will be conducted to determine if there are any contaminates in the immediate vicinity.

Bridges will be removed from the mine roads. Additionally, all culverts will be removed from the roads and cross-ditched for drainage. Organic material will be spread on the road surface and the road will be re-vegetated as required.

The cost of closure and reclamation has been estimated in this report and is detailed in the Capital Costs Section 21.1.

20.9 Site Management & Monitoring

IDM will design, construct, operate, and decommission the Project to meet all applicable BC environmental and safety standards and practices. Some of the provincial legislation that establishes or enables these standards is as follows:

- Mine Act (BC);
- Land Act (BC);
- Environmental Management Act (BC);
- Health Act (BC);
- Forest Act (BC);
- Forest and Range Practices Act (BC);
- Fisheries Act (BC);
- Soil Conservation Act (BC);
- Water Act (BC); and
- Wildlife Act (BC).

IDM has developed and will implement 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.

Water management will be a critical component of the Project, as the most likely avenue for transport of any contaminants into the natural environment will be through surface or groundwater. As such, IDM has developed a site water management plan that addresses mining activities undertaken during all phases of the Project. The goals of this management plan are as follows:

- Provide and retain water for mine operations;
- Provide a basis for management of the freshwater on the site;
- Avoid harmful impacts on fish and wildlife habit; and
- Manage water to ensure that discharges comply with the applicable water quality levels, guidelines.

20.10 Waste Rock & Tailings Disposal

The main waste management issue for the Project is the prevention and control of metal leaching/acid rock drainage (ML/ARD) from the tailings, and any potentially acid generating rock that is produced during mine development or operation.

The Project will create waste rock from mine development and tailings as a byproduct of mineral processing. Existing stockpiles and all development waste will be placed underground during mining as stope backfill.

There is an existing pile of mine development waste on the ridge near the upper portal. Project data indicates that 90,000 t are currently stored there; specifically, there are 5,000 t adjacent to the portal and 85,000 t stored 250 m south of the portal. SRK verified that the amount is in general agreement with the volume of the existing underground excavation (SRK, 2003).

In 2000, SRK visually inspected the waste rock storage pile. The waste rock was very fresh in appearance, with little sign of oxidation or secondary mineral accumulation. From acid base accounting data, the waste rock contained high amounts of sulphide.

Carbonate veining was also observed in many of the rocks. Field tests completed on the waste rock pile indicate that the cold climatic conditions at Red Mountain site provide an important control on the rate of sulphide oxidation. Leachate from crib tests constructed in 1996 had neutral pH's and moderate sulphate levels. Paste pH's in the seven-year-old waste pile were also neutral. In contrast, humidity cell tests completed on similar materials produced acidic leachate within several weeks of testing (SRK b, 2000).

In March 2002, NAMC submitted a revised reclamation plan to the BC Ministry of Energy and Mines. The revisions from the original reclamation plan, filed by Royal Oak in 1996, proposed treatment of the 90,000 tonnes of waste material by in-place re-contouring rather than placing the material underground (NAMC, 2002). The NAMC revised reclamation plan was approved by the BC Ministry of Energy and Mines in April 2002 (BC MEM, 2002).

Tailings and waste rock have been characterized as having potential for ML/ARD. Tailings process water is expected to contain residual metals and ammonia from destruction of cyanide solutions. The

Project incorporates appropriate design features and mitigation measures consistent with best practices for waste and water management to address these issues including:

- Fully lined TMF with seepage collection and pump back systems;
- Water treatment plant to treat effluent from the tailings pond during mine operations;
- Water collection ponds to control suspended sediment concentrations in seepage and runoff associated with the waste rock stockpiles and groundwater discharged from the mine; and
- Backfilling of all underground development rock into the underground mine as part of the mining process.

20.11 Site Monitoring

As a condition of the Mineral and Coal Exploration Activities & Reclamation Permit No. MX-1-422 (BC MEM, 2002), transferred from Seabridge to IDM, IDM is required to complete annual monitoring activities to document conditions resulting from exploration activities at the Red Mountain site, including:

- Collection and analysis of seep and crib drainage samples;
- Monitoring of dump weathering; and
- Documenting general site conditions.

An environmental monitoring program has been implemented by IDM.

A detailed monitoring program has been provided as a component of the EMS and will ultimately be implemented through all phases of the mine life per conditions to be specified in future operational permits.

20.12 Water Management

Potential water sources consist of underground mine development drainage, reclaim from the TMF, and freshwater from Bitter Creek.

Non-contact water from the upstream catchment above the TMF will be routed around the TMF through the non-contact water diversion channel.

Water stored in the TMF will be reclaimed to the plant site for mill processing water through a reclaim barge in the pond. The results of the water balance indicated that the TMF will operate in surplus conditions which will be managed by removing surplus supernatant water to the water treatment plant and then discharging to Bitter Creek after treatment. Seepage will be collected in the foundation drains and the seepage collection ponds downstream of the TMF embankments and recycled to the TMF supernatant pond.

Contact water at the plant site will be routed to the TMF via a system of diversion ditches. Contact water in borrow pits and quarries will be collected in sediment ponds and discharged to either Bitter Creek or Goldslide Creek after sediment settling.

Underground dewatering at the upper portal will discharge directly on to the Cambria Icefield, while dewatering at the lower portal will be pumped to a portal collection pond for attenuation and then discharged to Goldslide Creek.

21 Capital & Operating Costs

21.1 Capital Cost Estimate

21.1.1 Summary and Estimate Results

LOM Project capital costs total $202 M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to a 1,000 t/d production. Initial capital costs total $136 M and are expended over a 15-month pre-production construction and commissioning period;
- Sustaining Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $57 M and are expended in operating years 1 through 6; and
- Closure Costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $10 M (net of equipment salvage value), and are primarily incurred in year 7, with costs extending into year 11 for active closure activities. Costs beyond year 11 include ongoing monitoring and are excluded from the economic analysis.

The capital cost estimate was compiled using a combination of quotations, database costs, and data base estimations. Once compiled, the overall cost estimate was benchmarked against similar operations.

Table 21-1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q2 2017 dollars with no escalation.

Table 21-1: Capital Cost Summary

Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
Mining	11.3	38.3	-	49.6
Site Development	9.0	2.2	-	11.2
Mineral Processing	37.7	0.4	-	38.0
Tailings Management	7.2	10.9	-	18.1
Infrastructure	23.7	-	-	23.7
Off-site Infrastructure	2.8	-	-	2.8
Project Indirects	9.3		-	9.3
EPCM	13.0	-	-	13.0
Owners Costs	9.1	-	-	9.1
Closure	-	-	12.4	12.4
Salvage Value	-	-	(3.8)	(3.8)
Subtotal Pre-Contingency	**123.0**	**51.7**	**8.6**	**183.3**
Contingency	12.7	5.2	1.2	19.1
Total Capital Costs	**135.7**	**56.9**	**9.9**	**202.4**

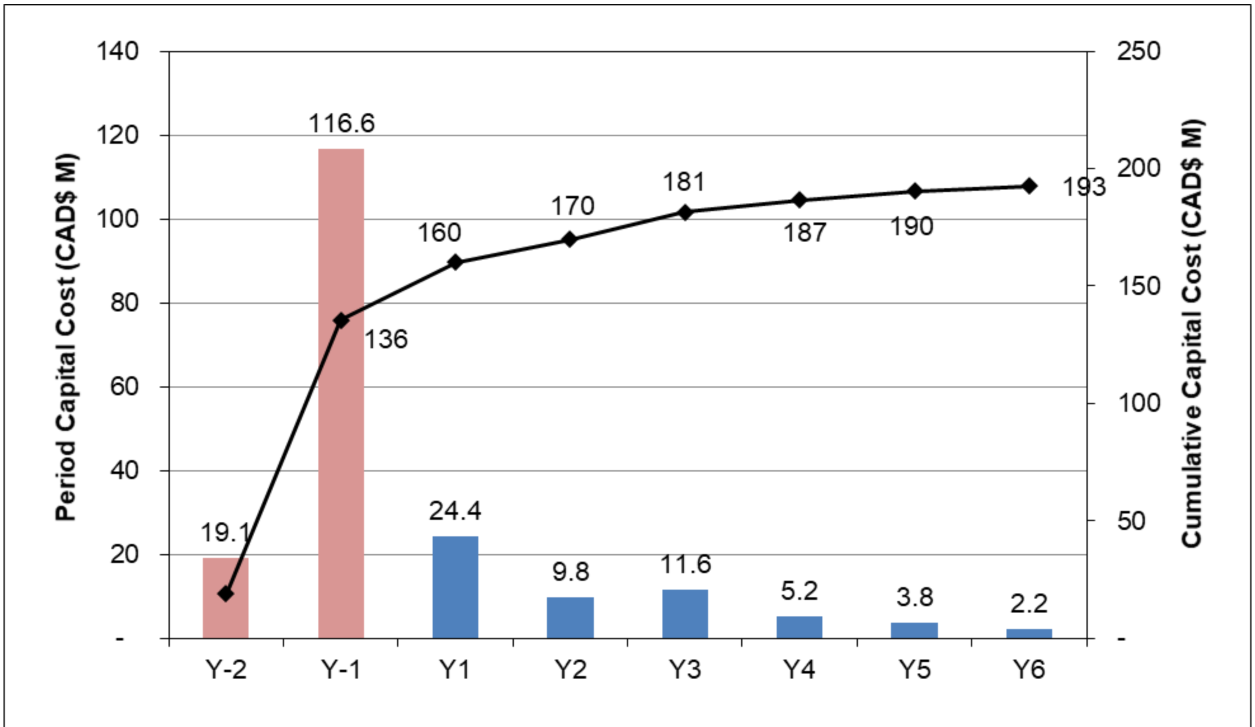
Source: JDS (2017)

Figure 21-1 presents the capital cost distribution for the pre-production and sustaining phases. As typical with underground operations, the majority of sustaining capital costs relate to underground lateral and vertical development, and tailings storage facility expansion.

21.1.2 Capital Cost Profile

All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21-1 presents an annual LOM capital cost profile (excluding closure years).

Figure 21-1: Capital Cost Profile (Closure Years not Shown)



Source: JDS (2017)

21.1.3 Key Estimate Assumptions

The following key assumptions were made during development of the capital estimate:

- The capital estimate is based on the contracting strategy, execution strategy, and key dates described within the Project execution schedule;
- Underground mine development activities will be performed by the owner's teams; and
- All surface construction (including earthworks) will be performed by contractors.

21.1.4 Key Estimate Parameters

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered AACE Class 3 estimates. The overall Project definition is estimated to be 30%;
- Estimate Base Date: The base date of the estimate is June 26, 2017. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Currency: All capital costs are expressed in Canadian Dollars (C$). Portions of the estimate were estimated in US Dollars (US$) and converted to Canadian Dollars at a rate of US$0.76:C$1.00.

21.1.5 Mine Capital Costs

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases and similar mines in western Canada. Table 21-2 summarizes the underground mine capital cost estimate.

Table 21-2: Mine Capital Costs

Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
Underground Mobile Equipment	1.3	2.8	4.1
Underground Infrastructure	3.5	4.4	7.9
Capital Lateral Development	4.3	27.6	31.9
Capitalized Vertical Development	2.1	3.4	5.5
Total Mining (excl. Contingency)	**11.3**	**38.3**	**49.6**

Source: JDS (2017)

21.1.5.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. A portion of the equipment was leased, reducing the initial capital costs.

21.1.5.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.1.5.3 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes, and underground infrastructure.

- Lateral development fuel, equipment usage, power, and consumables requirements were developed based on the mine plan requirements. Manufacturer database equipment usage rates were applied to the required operating hours; and
- Lateral development labour requirements were determined by the required equipment fleet in operation. Supervision and support services were pro-rated to the development costs, based on the mix of underground activities occurring.

21.1.5.4 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, mineralized material extraction, mine maintenance, and mine general costs) incurred prior to the introduction of feed to the processing facilities and the commencement of Project revenues. They are included as a pre-production capital cost.

The basis of these costs is described in Section 22, Operating Costs, as they are estimated in the same manner. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.1.6 Surface Construction Costs

Surface construction costs include site development, the mineral processing plant, TMF, and site infrastructure. These cost estimates are primarily based on quotes, quantity takeoffs and database costs. Table 21-3 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Table 21-3: Surface Construction Basis of Estimate

Commodity	Basis
Access Roads	Quantities were developed from 3D model layouts and design calculations with specific design basis. Completed by On Site Engineering. Bridges calculated based on specifications and database unit rates.
Bulk Earthworks	Quantities developed from 3D grading model and design drawings. Database and local contractor unit rates for bulk excavation and fill. Database unit rates for surface drainage, temporary roads, and slope stabilization.
Concrete	Quantities were developed from 3D model layouts and design calculations. Budgetary unit rates were obtained from local contractors.
Structural Steel	Quantities were developed from 3D model layouts and design calculations. Budgetary unit rates were obtained from local contractors.
Pre-Engineered Buildings	Quantities were developed from 3D model layouts and design calculations. Budgetary unit rates were obtained from design firm and database unit rates.
Modular Buildings & Warehouses	Quotations were provided from local suppliers based on specifications supplied by JDS.
Mechanical Equipment	A combination of quoted and database costs from general arrangement drawings. Database factor applied against mechanical equipment costs for installation.
Piping	Piping quantities were developed based on design drawings. Database unit rates were applied based on similar projects.
Power Transmission Line and Major Sub-Stations	Detailed estimate provided by experienced line contractor operating in the area of the Project.
Electrical and Instrumentation	Electrical material quantity take-offs for wire, cable tray, junction boxes, etc., were based on single line diagrams and GA drawings. Database unit pricing for installation man-hours were applied to the engineered quantities.

Source: JDS (2017)

21.1.6.1 Surface Construction Sustaining Capital

Sustaining capital costs are included in the estimate for continued construction of the TMF, lower portal laydowns, and site water management. Pre-production capital costs allow for construction of the impoundment to accept one year's production of tailings from the mineral processing plant. The balance of the facility is progressively constructed between year 1 and year 5 of operations.

21.1.7 Indirect Costs

Indirect costs are those that are not directly accountable to a specific cost object. Table 21-4 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-4: Indirect Cost Basis of Estimate

Commodity	Basis
Construction Support Services	Time based cost allowance for general construction site services, including a crew of four (temporary power, heating and hoarding, contractor support, etc.) applied against the surface construction schedule of 11 months.
Temporary Buildings	Temporary office requirements have been estimated based on the development schedule and contracting strategy for the Project. Modular style offices will be utilized to house the supervisory and administration staff from the PM group and construction contractors.
Temporary (Construction) Power	Temporary power required to service construction activities and temporary buildings was estimated from experience at other similar sized projects. Three large generators will be located to provide construction power to the underground mine, construction office complexes, and the process plant site. Several smaller generators will be rented to provide construction power at ancillary areas. Costs are included for the rental of the generators, fuel and parts for operations, and a generator mechanic on a callout service. Small generators (less than 20 kW) will be provided by contractors; the costs for the acquisition and usage of these are included in the construction equipment costs.
Contractor Mobilization	Factored allowance (1%) of contractor direct labour costs.
Logistics & Freight	Freight costs have been developed based on the estimated number of loads which has been completed in detail including sea and land. Fuel, cement, and explosives commodity pricing includes delivery to site.
Start-up & Commissioning	First fills and vendor assistance costs have been factored based on similar projects with first principles estimate for initial mill charges. Factored allowance (2%) for the provision of vendor services for commissioning support.
Detailed Engineering	Engineering costs within the estimate are based on budgetary quotations received from entities involved in the development of the FS.
Project & Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration database unit (hourly) rates.

Source: JDS (2017)

21.1.8 Owner's Costs

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements listed below are described in more detail within Section 21.2.

- Pre-production milling: Costs of the owner's processing labour, power, and consumables incurred before declaration of commercial production;
- Surface support: Costs of the owner's surface support labour, maintenance, and equipment usage; costs for contract water supply and waste removal prior to commercial production; and
- Pre-production general and administration: Costs of the owner's labour and expenses (safety, finance, security, purchasing, management, etc.) incurred prior to commercial production.

21.1.9 Closure Costs & Salvage Value

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine in western Canada. Activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF;
- Closure of the underground mine portals;
- Access road closure;
- Power transmission line and substation removal;
- Re-vegetation and seeding; and
- Ongoing site monitoring.

Due to the short mine life, a salvage value was assumed at an average of 12% of the initial supply cost of all mobile equipment, process equipment, and ancillary buildings.

The majority of closure costs are incurred immediately following completion of operations (year 6). Monitoring activities are anticipated to extend to year 20. For the purpose of the economic model, all costs occurring beyond year 10 are expressed as a discounted (5% per annum) lump sum in year 10. Table 21-5 shows the estimated costs for closure and salvage value.

Table 21-5: Closure and Salvage Estimate Summary

Item	Estimated Cost (M$)
Demolition and Waste Removal	3.6
Tailings Management Facility Closure	5.6
Underground Mine Closure	0.6
Monitoring, Maintenance, and Re-Vegetation	0.4
Access Road Closure	0.4
Power Line and Substation Removal	1.1
Indirect Costs	0.5
Contingency	1.2
Subtotal	13.6
Salvage Value	(3.8)
Total Closure (Net Salvage Value)	**9.9**

Source: JDS (2017)

21.1.10 Cost Contingency

An overall contingency of 10.2% was applied to the LOM capital costs of the Project. LOM Project contingency amounts to $17.9 M.

21.1.11 Capital Estimate Exclusions

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

- Working capital (included in the financial model);

- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any predevelopment and Project sunk costs (engineering fieldwork and studies, exploration programs, permitting etc.);
- Provincial sales tax;
- Closure bonding; and
- Escalation cost.

21.2 Operating Cost Estimate

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined proven project execution strategies. The estimate was developed using first principles and applying directly applicable project experience, and avoiding the use of general industry factors.

The target accuracy of the operating cost is consistent with an AACE Class 3 estimate.

The operating cost estimate is broken into four major components:

- Underground mining;
- Processing;
- Site services; and
- General and administrative expenses (G&A).

The operating cost is based on a leased mining equipment fleet and minimal use of permanent contractors, except where value is provided through expertise and/or efficiencies/skills. The down-payments for the leased equipment are captured in the capital estimate either as initial or sustaining capital. The lease payments are accounted for in the mine equipment operating costs and make up $14.3 M over the LOM, or $7.33/t processed.

Operating costs incurred during the construction phase (pre-production years -2 and -1) are capitalized and form part of the capital cost estimate as owner's costs. Underground lateral and vertical waste development after the pre-production period have been capitalized and will not appear as an operating cost (refer to Section 21.1.1.5 – Sustaining Capital Cost). Capital waste development represents the mine's permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, mill feed pass accesses, and permanent explosive storage cut-outs, as well as main ventilation raises.

Some of the costs incurred during the pre-production period relate to the costs to purchase items such as consumables required for the following year of production. The timing of these costs has been accounted for in the economic analysis as working capital in year -1.

Operating costs are presented in 2017 Canadian dollars on a calendar year basis with a fixed exchange rate of US$:C$ of 0.76. No escalation or inflation is included.

Operating costs over the LOM are estimated to be $274 M or $140.02/t processed exclusive of ocean transportation. LOM operating costs and total unit costs are summarized in Table 21-6 and the breakdown is illustrated in Figure 21-2.

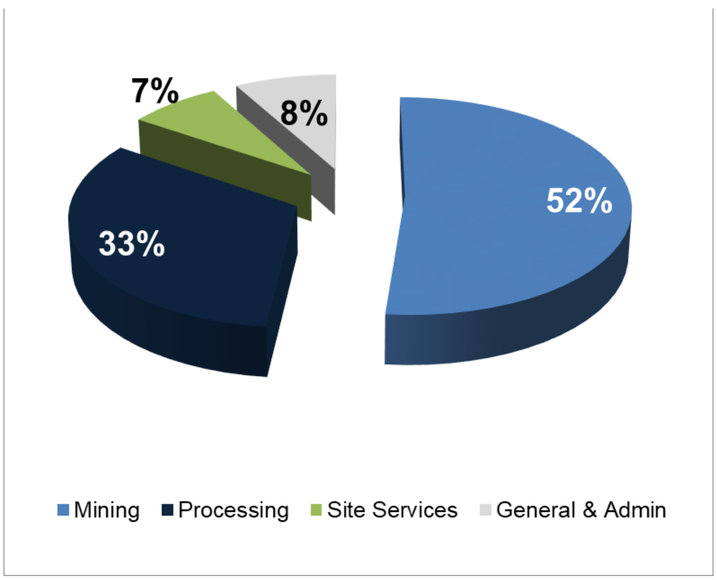
Table 21-6: LOM Average Operating Costs

Operating Cost	Unit Cost ($/t processed)	LOM Cost (M$)
Mining	72.30[1]	141.2
Processing	45.96	89.8
Site Services	10.40	20.3
G&A	11.36	22.2
Total	**140.02**	**273.5**

Source: JDS (2017)

Note: *includes 5 kt mined in pre-production but processed in year 1. Average unit cost for tonnes mined exclusively during the production period is $72.50/t.

Figure 21-2: Operating Cost Breakdown



Source: JDS (2017)

21.2.1 Operations Labour

This section provides an overview of total workforce and the methods used to build the labour rates.

Table 21-7 summarizes the total planned workforce during Project operations.

Table 21-7: Summary of Peak Employment by Area

Department	Total Persons Employed (Peak)
Mining	130
Processing	64
Site Services	7
G&A	23
Total	230

Source: JDS (2017)

Labour base rates were determined by reference to other northern Canadian operations and benchmarked against Costmine (Canadian Mine Salaries, Wages, Benefits 2016 Survey Results). Labour burdens were assembled using first principles. The following items are included in the burdened labour rates:

- Scheduled overtime costs based on individual employee rotation;
- Unscheduled overtime allowance of 10% for hourly employees;
- CPP, EI, WCB as legislated;
- Statutory holiday allowance of 6.9% of scheduled hours;
- Vacation pay allowance of 4% of scheduled hours; and
- Health benefits package of $6,534/employee/year.

21.2.2 Underground Mining Operating Costs

Mine operating unit costs, reported in $/t processed, are summarized below in Table 21-8 and include the following functional areas:

- Waste development – costs related to the drilling, blasting, mucking, and hauling of non-capital development;
- Production – costs related to the drilling, blasting, mucking, and hauling of ore;
- Backfill – costs related to CRF and RF backfill operations, including the CRF plant and the talus quarry;
- Mine Maintenance – maintenance labour costs that support all other sectors; and
- Mine General – costs related to mine support activities, such as technical services, shared infrastructure, support equipment, and definition drilling.

Table 21-8: Underground Mine Operating Costs

Mining Category	Unit Cost ($/t processed)	LOM Cost (M$)
Lateral Waste Development	13.13	25.6
Production	27.89	54.3
Backfill	16.20	31.5
Mine Maintenance	7.13	13.8
Mine General	8.15	15.9
Total	**72.50**	**141.2**

Source: JDS (2017)

Costs are averaged over the life of the mine, and range from a high of $78/t in year 1, to a low of $65/t in year 4. The fluctuation in cost is mostly due to a decrease in mine development costs in year 2.

Mine operating costs have been built up using a combination of first principle engineering and equivalent project scaling.

21.2.2.1 Mining Labour Requirements

Table 21-9 summarizes all mining labour requirements.

Table 21-9: Mining Labour Requirements

Position	Manpower Complement	Staff/ Hourly	Base Salary ($)	Burden (%)	Total Salary ($)	Annual Cost ($)
Management						
Mining Superintendent	1	Staff	160,000	12	180,912	180,912
Maintenance Superintendent	1	Staff	160,000	12	180,912	180,912
Mine Shift Foreman	1	Staff	120,000	16	139,312	139,312
Technical Services Superintendent	1	Staff	160,000	12	180,912	180,912
Mine Clerk	1	Staff	60,000	27	75,954	75,954
Total	**5**					
Operations						
Mine Supervisor/Shift Boss	4	Staff	100,000	26	125,688	502,752
Safety/Trainer/Mine Rescue	2	Hourly	73,000	51	110,380	220,760
Blaster	7	Hourly	51,100	77	90,350	632,450
Development Services	6	Hourly	60,225	78	107,471	644,826
Jumbo Operator	7	Hourly	60,225	78	107,471	752,297
Production Drill Operator	3	Hourly	54,750	78	97,208	291,624
LHD Operator	7	Hourly	51,100	77	90,350	632,450
Truck Driver	15	Hourly	51,100	77	90,350	1,355,250
Ground Support/Bolter/Shotcrete	7	Hourly	60,225	78	107,471	752,297
Mine Helper	3	Hourly	47,450	65	78,486	235,458

Table 21-9: Mining Labour Requirements (continued)

Position	Manpower Complement	Staff/ Hourly	Base Salary ($)	Burden (%)	Total Salary ($)	Annual Cost ($)
Utility Vehicle Operator/Nipper	4	Hourly	41,975	63	68,383	273,532
Total	**66**					
Mine Services						
CRF Plant Operators	3	Hourly	51,100	77	90,350	271,050
Backfill Miner	3	Hourly	47,450	65	78,486	235,458
Electrician	4	Hourly	73,000	47	107,059	428,236
Utility Vehicle Operator/Nipper	3	Hourly	41,975	63	68,383	205,149
Total	**13**					
Mine Maintenance						
Mine Maintenance Supervisor	1	Staff	100,000	19	118,512	118,512
Maintenance Planner	1	Staff	85,000	21	102,912	102,912
HD Mechanic	7	Hourly	73,000	47	107,059	749,413
Mechanic Helper	3	Hourly	45,625	56	71,156	213,468
Welder	2	Hourly	73,000	47	107,059	214,118
Electric/Hydraulic Mechanic	7	Hourly	73,000	47	107,059	749,413
Total	**21**					
Technical Services						
Senior Mine Engineer	1	Staff	125,000	16	144,512	144,512
Geotechnical Engineer / Backfill	1	Staff	105,000	18	123,712	123,712
Chief Geologist	1	Staff	120,000	16	139,312	139,312
Mine Ventilation/Project Engineer	1	Staff	105,000	18	123,712	123,712
Surveyor/Mine Technician	2	Staff	75,000	31	97,894	195,788
Production Geologist	1	Staff	100,000	19	118,512	118,512
Technician/Sampler	1	Staff	80,000	22	97,712	97,712
Sr. Mine Technician	1	Hourly	90,000	20	108,112	108,112
Total	**9**					
Total Labour	**114**					**11,390,799**

Source: JDS (2017)

21.2.3 Processing Operating Costs

Processing operating costs were estimated to include all gold and silver recovery activities to produce unrefined gold and silver doré on-site. The crushing and process plants are designed for a throughput of 1,000 t/d. Labour rates and benefit loadings are based on information supplied by JDS. All reagent cost estimates are detailed in Section 21.2.3.4. The process operating costs are summarized in Table 21-10.

Table 21-10: Process Operating Costs

Processing Category	Unit Cost ($/t processed)	LOM Cost (M$)
Labour	19.47	38.0
Power and Oxygen	5.03	9.8
Maintenance and Consumables	20.28	39.6
Support Equipment	1.17	2.3
Total Processing OPEX	**45.96**	**89.8**

Source: JDS (2017)

21.2.3.1 Processing Labour Requirements

Table 21-11 summarizes all processing labour requirements.

Table 21-11: Processing Labour Requirements & Costs

Position	Manpower Complement	Staff/ Hourly	Base Salary ($)	Burden (%)	Total Salary ($)	Annual Cost ($)
Operations						
Mill Operations Superintendent	1	Staff	160,000	12	180,912	180,912
Operations Shift Foreman	2	Staff	100,000	20	125,688	251,377
Mill Admin Assistant	1	Staff	60,000	21	75,954	75,954
Control Room Operator	4	Hourly	72,800	35	112,510	450,038
Crusher Operator	4	Hourly	64,480	36	101,310	405,240
Grinding/ADR	4	Hourly	72,800	35	112,510	450,038
CIL/Tailings/Detox Operator	4	Hourly	72,800	35	112,510	450,038
EW/Gold Room Operator	2	Hourly	72,800	35	112,510	225,019
Reagent Helpers/Operator	4	Hourly	56,160	37	89,847	359,388
Mill Labourer	8	Hourly	47,840	39	78,058	624,460
Total	**34**					
Maintenance						
Mill Maintenance Foreman	1	Staff	100,000	20	125,688	125,688
Maintenance Planner	1	Staff	80,000	18	97,712	97,712
Electrician Apprentice	2	Hourly	60,320	37	95,710	191,420
Electrician	2	Hourly	83,200	34	126,509	253,018
Instrumentation Technician	1	Hourly	83,200	27	114,020	114,020
Millwright	8	Hourly	83,200	34	126,509	1,012,074
Welder	2	Hourly	83,200	34	126,509	253,018
Total	**17**					
Technical Services						
Sr. Metallurgical Engineer	1	Staff	125,000	14	144,512	144,512
Metallurgical Engineer (Process Control)	1	Staff	110,000	15	128,912	128,912
Metallurgical Technician	2	Staff	70,000	24	92,189	184,379
Chief Assayer	1	Staff	110,000	15	128,912	128,912

Table 21-11: Processing Labour Requirements & Costs (continued)

Position	Manpower Complement	Staff/ Hourly	Base Salary ($)	Burden (%)	Total Salary ($)	Annual Cost ($)
Assay Technician	6	Hourly	76,960	35	118,109	708,657
Laboratory Technician	2	Hourly	56,160	37	89,847	359,388
Total	**13**					
Total Labour	**64**					**6,994,483**

Source: JDS (2017)

21.2.3.2 Processing Fuel & Power

Table 21-12 presents the processing oxygen and power costs.

Table 21-12: Processing Fuel & Power Cost

Description	Usage	Unit Cost ($/X)	Annual Cost (M$)	Unit Cost ($/t)
Power	24,596,776 (kWh)	0.061/kWh	1.5	4.18
Oxygen Supply (OTF)	2.1 (t/d)	400	0.3	0.85
Total - Power and Oxygen			**1.8**	**5.03**

Source: JDS (2017)

21.2.3.3 Processing OPEX Maintenance & Consumables

Table 21-13 presents the processing consumable costs.

Table 21-13: Processing Consumables

Consumables	Annual Cost (M$)	Unit Cost ($/t)
Grinding Media	0.80	2.24
Liners	0.54	1.51
Maintenance Labour and Supplied	0.79	2.18
Reagents	5.15	14.35
Total	**7.28**	**20.28**

Source: JDS (2017)

21.2.3.4 Reagent Costs

Reagent costs were based on recent quotes including freight to site. Table 21-14 summarizes the reagent and chemical requirements.

Table 21-14: Reagent Requirements & Costs

Reagents	Usage (t/a LOM Average)	Unit Cost of Reagent ($/t)	Annual Cost (M$)
Lime	575	545	0.31
Cyanide	530	3,534	1.87
Pb_2NO_3	91	3,271	0.30
Flocculant	7	4,850	0.04
Carbon	44	4,192	0.18
Caustic	255	1,429	0.36
HCL	146	695	0.10
Anti-scalant	15	3,645	0.05
SMBS	1,373	1,100	1.51
CuSO4.4H2O	103	4,061	0.42
Total			**5.15**

Source: JDS (2017)

21.2.4 Site Services

Site services operating costs account for costs to cover surface support, avalanche control, maintenance, ancillary power, and water treatment.

Table 21-15 summarizes the infrastructure and site services operating costs.

Table 21-15: Summary of Site Services Costs

Site Services Categories	Unit Cost ($/t processed)	LOM Cost (M$)
Surface Support	4.47	8.7
Avalanche Support	2.61	5.1
Maintenance	0.25	0.5
Ancillary Power	0.83	1.6
Water Treatment	2.24	4.4
Total Site Services Costs	**10.40**	**20.3**

Source: JDS (2017)

21.2.4.1 Surface Support

Surface support costs include:

- Labour costs for surface services and maintenance of infrastructure facilities;
- Surface mobile equipment operations and maintenance; and
- Material for resurfacing of road.

The manpower required to perform surface support work is listed in Table 21-16.

Table 21-16: Surface Support Labour Rates & Quantities

Position	Manpower Complement	Staff/ Hourly	Base Salary ($)	Burden (%)	Total Salary ($)	Annual Cost ($)
Surface Foreman	1	Staff	85,000	21%	102,912	102,912
Facilities Maintenance-Tradesman	1	Hourly	83,200	37%	114,020	114,020
Mobile Equipment Operator - Site	2	Hourly	60,320	43%	86,210	172,420
Mobile Equipment Operator - Access Road	4	Hourly	60,320	59%	95,710	382,840
Labourers/Apprentices	1	Hourly	52,000	46%	75,735	75,735
Total	**9**					**847,927**

Source: JDS (2017)

Surface mobile equipment operations and maintenance costs include fuel and maintenance for each piece of support equipment shown in Table 21-17. Costs are based on an allowance for operating hours per year.

Table 21-17: Surface Support Equipment Quantities

Equipment Description	Equipment Quantity (#)
Truck - Ford F150	5
Sand Truck/Plow truck*	1
Water Truck*	1
72 Passenger Bus*	3
Tool Carrier - Cat 966K (c/w Attachments)*	1
Skid Steer Loader (1Cu.M) †	1
3 T Forklift - Warehouse - CAT P5000*	1
Grader - Cat 160M*	1
Fuel Truck (10,000 L)*	1
Portable Diesel Light Plant	3
Portable Diesel Heater †	3
Manlift - 60ft *	1
Snowcat †	1
Avalauncher	1
Dozer - Cat D7 †	1

*Used equipment

†Existing

Source: JDS (2017)

The estimated LOM surface support cost is $4.47/t processed or approximately $1.6 M per year. The cost estimates are shown in Table 21-18.

Table 21-18: Surface Support Operating Costs

Surface Support Category	Unit Cost ($/t processed)	LOM Cost (M$)
Equipment Maintenance	1.21	2.4
Fuel	0.86	1.7
Materials	0.09	0.2
Labour	2.31	4.5
Total	**4.47**	**8.7**

Source: JDS (2017)

21.2.4.2 Avalanche Support

Avalanche support costs include labour and equipment costs associated with:

- Labour costs for avalanche techs and snowcat operators;
- Helicopter rentals;
- Snowmobile and snowcat equipment hours and maintenance; and
- Miscellaneous travel expenses and supplies (including explosives).

The manpower required to perform avalanche support work is listed in Table 21-19.

Table 21-19: Avalanche Support Labour Rates and Quantities

Position	Manpower Complement	Shift Rotation	Staff/Hourly	Loaded Annual Cost (M$)
Lead Avalanche Tech - Zone A	1	2x2	Contractor	0.18
Lead Avalanche Tech - Zone B	1	2x2	Contractor	0.18
Assistant Avalanche Tech - Zone B	1	2x2	Contractor	0.15
Snowcat Operator	1	4x4	Contractor	0.04
Total	**4**			**0.55**

Source: JDS (2017)

Avalanche support costs include fuel and maintenance for each piece of support equipment shown in Table 21-20. Fuel for the helicopters is included in the rental. Costs are based on estimated operating hours per year.

Table 21-20: Avalanche Support Equipment Quantities

Equipment Description	Equipment Quantity
Helicopter (Rental) - Zone A	1
Helicopter (Rental) - Zone B	1
Snowmobile	2
Snowcat	1

Source: JDS (2017)

The estimated LOM avalanche support cost is $2.61/t processed or $1 M per year. The cost estimates, broken down by component, is shown in Table 21-21.

Table 21-21: Avalanche Support Operating Costs

Avalanche Support Category	Unit Cost ($/t processed)	LOM Cost (M$)
Equipment Maintenance	0.61	1.2
Fuel	0.03	0.1
Labour	1.49	2.9
Misc. Costs	0.50	0.9
Total	**2.61**	**5.1**

Source: JDS (2017)

21.2.4.3 Maintenance

Site services infrastructure maintenance totals $0.5 M over the LOM. It includes maintenance of the following facilities:

- TMF tailings distribution and reclaim system;
- Off-site power transmission line;
- On-site power transmission line;
- Emergency power generation;
- Power substations and distribution;
- Potable water system;
- Mine dry/administration office;
- Mine maintenance shop; and
- Mine/plant warehouse.

The estimated LOM maintenance cost is $0.25/t processed or approximately $0.1 M per year.

21.2.4.4 Ancillary Power

Ancillary power costs total approximately $2 M over the LOM and include costs to power the:

- Reclaim barge at the TMF; and
- Ancillary buildings including (mine dry, admin, and office buildings).

The estimated LOM ancillary power cost is $0.83/t processed or approximately $0.3 M per year.

21.2.4.5 Water Treatment

The water treatment operating cost estimate includes costs associated with the following:

- System maintenance;
- Power;
- Consumables (chemicals); and
- Sludge disposal.

The estimated LOM water treatment cost is $2.24/t processed or approximately $0.8 M per year. The cost estimate breakdown by component is shown in Table 21-22.

Table 21-22: Water Treatment Operating Costs

Water Treatment Category	Unit Cost ($/t processed)	LOM Cost (M$)
System Maintenance	0.28	0.5
Power	1.61	3.1
Consumables (chemicals)	0.34	0.7
Sludge Disposal	0.01	0.2
Total	**2.24**	**4.4**

Source: JDS (2017)

21.2.5 G&A Costs

General and administrative costs comprise the following categories:

- Labour;
- On-site items comprised of health and safety, medical and first aid, environmental, human resources, legal, external consulting, communications, and office supplies; and
- Off-site items comprised of dismantling and demobilization of the construction camp.

The total G&A unit operating cost is estimated at $11.36/t processed. Table 21-23 summarizes the G&A operating costs. A detailed breakdown of the G&A operating costs are shown in Table 21-24.

Table 21-23: Summary of G&A Costs

G&A Category	Unit Cost ($/t processed)	LOM Cost (M$)
Labour	7.49	14.6
Off-Site Items	0.21	0.4
On-Site Items	3.66	7.2
Total G&A Costs	**11.36**	**22.2**

Source: JDS (2017)

Table 21-24: G&A Detailed Costs

Area	Unit Cost ($/t processed)	LOM Cost (M)
G&A Labour		
General Management	0.92	1.8
Human Resources	1.06	2.1
Community Relations	0.18	0.4
IT/OT Support	0.31	0.6
Administration	2.38	4.6
Health and Safety	1.06	2.1
Environmental	1.03	2.0
Security	0.55	1.1
Subtotal G&A Labour	**7.49**	**14.6**
G&A Items – Offsite		
Dismantling of Construction Camp	0.10	0.2
Demobilization	0.11	0.2
Subtotal G&A Offsite	**0.21**	**0.4**
G&A Items – Onsite		
Health and Safety, Medical, and First Aid	0.71	1.4
Environmental	0.41	0.8
Human Resources	0.82	1.6
Insurance and Legal	0.88	1.7
External Consulting	0.14	0.3
IT and Communications	0.62	1.2
Office and Miscellaneous Costs	0.08	0.2
Subtotal G&A On-Site Items	**3.66**	**7.2**
Total G&A Costs	**11.36**	**22.2**

Source: JDS (2017)

21.2.5.1 Labour

General and administrative labour includes all on-site and off-site positions. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations.

Table 21-25 summarizes the G&A workforce labour rates and quantities.

Table 21-25: G&A Labour Complement and Rates

Position	Manpower Complement	Staff/ Hourly	Base Salary ($)	Burden (%)	Total Salary ($)	Annual Cost ($)
General Management						
General Manager	1	Staff	230,000	10%	253,712	253,712
Administrative Assistant	1	Staff	55,000	28%	70,480	70,480
Accounting						
Controller/Accountant	1	Staff	150,000	14%	170,512	170,512
Payroll Supervisor	1	Staff	90,000	20%	108,112	108,112
Payroll Clerk	1	Staff	55,000	28%	70,480	70,480
Human Resources						
HR Superintendent	1	Staff	160,000	13%	180,912	180,912
Human Resources Coordinator	1	Staff	70,000	24%	86,902	86,902
Trainer	1	Staff	90,000	20%	108,112	108,112
Community Relations						
Community Relations Coordinator	1	Staff	50,000	30%	65,007	65,007
IT/OT Support						
IT/Telecom Technician	1	Staff	90,000	20%	108,112	108,112
Health and Safety						
Health and Safety Superintendent	1	Staff	160,000	13%	180,912	180,912
Health and Safety Coordinator	2	Staff	75,000	31%	97,894	195,788
Environmental						
Environmental Superintendent	1	Staff	160,000	13%	180,912	180,912
Environmental Coordinator	1	Staff	80,000	22%	97,712	97,712
Environmental Monitors	1	Staff	65,000	33%	86,338	86,338
Logistics & Warehousing						
Sr. Purchasing/Contracts	1	Staff	100,000	19%	118,512	118,512
Procurement/ Contracts Agent	1	Staff	80,000	22%	97,712	97,712
Warehouse Supervisor	1	Staff	80,000	22%	97,712	97,712
Warehouse Clerk	2	Hourly	56,160	60%	89,847	179,694
Security						
Protective Services Officers	2	Staff	75,000	31%	97,894	195,788
Total	**23**					**2,653,421**

Source: JDS (2017)

(*) Total may not match due to timing of maximum quantity by year

22 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the Project. Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, recovery, US$:C$ exchange rates, operating costs, and capital costs to determine their relative importance as Project value drivers.

This Technical Report contains forward-looking information regarding projected mine production rates, construction schedules, and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21 of this report (presented in 2017 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

Table 22-1 outlines the planned LOM tonnage and grade estimates.

Table 22-1: Life of Mine Plan Summary

Parameter	Unit	Value
Mine Life	Years	5.4
Resource Mined	Mt	1.95
Average Throughput Rate	t/d	1,000
Average Au Head Grade	g/t	7.53
Average Ag Head Grade	g/t	21.86
Au Payable	koz	425
	koz/a	78
Ag Payable	koz	1,173
	koz/a	215

Source: JDS (2017)

Other economic factors include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated);
- Closure cost of $13.7 M (including 10% contingency) and salvage value of $3.8 M were considered;

- Nominal 2017 dollars;
- No inflation;
- No provincial sales taxes or duties;
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Working capital calculated as one month of operating costs (mining, processing, site services, and G&A) in year 1 totalling $4 M considered in the pre-production period;
- Results are presented on 100% ownership basis and do not include management fees or financing costs; and
- Exclusion of all pre-development and sunk costs up to the start of detailed engineering (i.e., exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Table 22-2 outlines the key inputs and assumptions used in the economic analysis. Metal pricing and exchange rates used in the economic model are approximate 18-month trailing averages and represent similar values to recent technical reports.

The reader is cautioned that the gold prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 22-2: Key Inputs and Assumptions

Parameter	Unit	Value
Gold Price	US$/oz	1,250
Silver Price	US$/oz	17.00
Exchange Rate	US$:C$	0.76
Mine Operating Days	days/a	350
Plant Operating Days	days/a	365
Franco-Nevada Gold Royalty	% NSR	1.0
Wotan Royalty	% NSR	2.5

Source: JDS (2017)

22.2 Revenues & NSR Parameters

Mine revenue is derived from the sale of doré into the international marketplace. No contractual arrangements for refining exist at this time. Details regarding the terms used for the economic analysis can be found in the market studies (Section 19) of this report. Doré production and sale are assumed to begin in year 1, Q1 and continue for 5.4 years.

Table 22-3 outlines the NSR parameters and recoveries by zone that were used in the economic analysis.

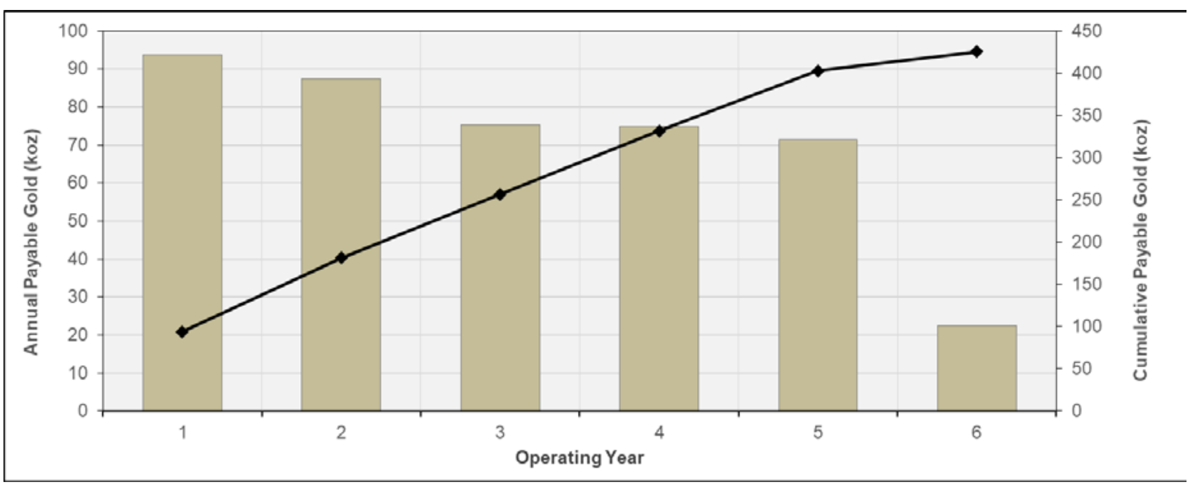
Table 22-3: Recoveries & NSR Parameters

Parameter	Unit	Value
Recoveries		
Marc Zone		
Au	%	92.8
Ag	%	90.1
AV Zone		
Au	%	88.1
Ag	%	78.3
JW Zone		
Au	%	92.1
Ag	%	90.3
141 Zone		
Au	%	89.9
Ag	%	84.9
NSR Assumptions		
Au Payable	%	99.0
Au Refining Charge	US$/pay oz	5.00
Ag Payable	%	99.0
Ag Refining Charge	US$/pay oz	0.50

Source: JDS (2017)

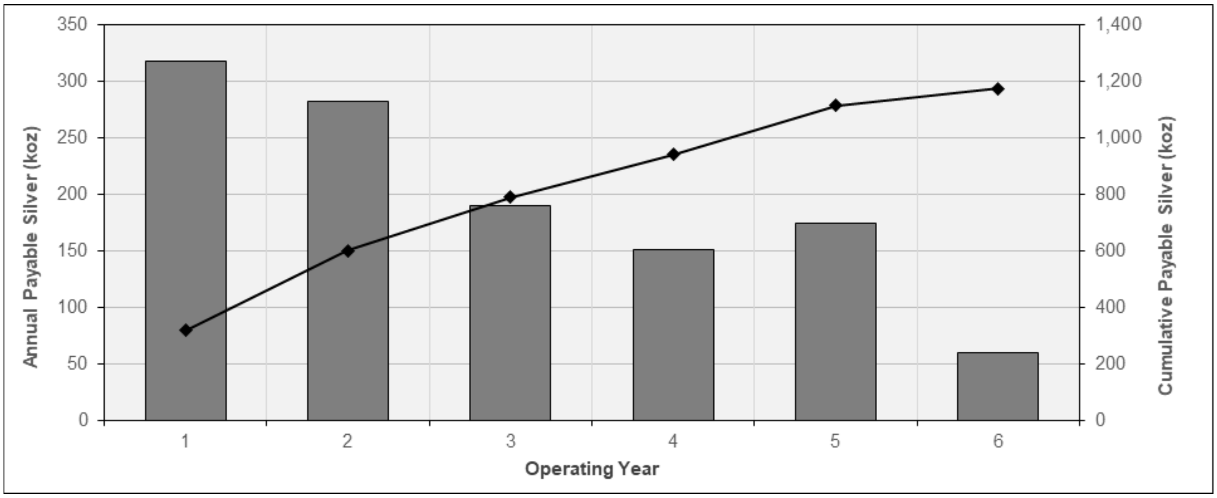
Figure 22-1 and Figure 22-2 show breakdowns of the amount of gold and silver recovered during the mine life and the amount of payable metal for the Project. A total of 425 koz of gold and 1,173 koz of silver are projected to be produced during the mine life. Gold accounts for about 96% of net Project revenues with silver accounting for the remaining 4%.

Figure 22-1: Payable Gold Production by Year



Source: JDS (2017)

Figure 22-2: Payable Silver Production by Year



Source: JDS (2017)

22.3 Taxes

The Project has been evaluated on an after-tax basis in order to provide a more indicative, but still approximate, value of the potential Project economics. Both BC Mineral Taxes and Federal and Provincial Income Tax were applied to the Project. A tax model was prepared by Wentworth Taylor, a specialized mining tax accountant with applicable British Columbia mineral tax regime experience. The tax model contains the following assumptions:

- 15% federal income tax rate; and
- 11% British Columbia provincial tax rate.

Mineral Property Tax Pools

- Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with appropriate opening balance to calculate income taxes.
- Capital Cost Allowance (CCA)
- Specific capital cost class CCA rates were applied and used to calculate the appropriate CCA the Company can claim during the entire life of the Project.

The model also contains British Columbia mineral taxes which include:

- 2% net current proceeds tax; and
- 13% net revenue tax.

Total taxes for the Project amount to $63 M.

22.4 Third Party Royalties

- The economic analysis for the Project accounts for the following royalties:
- 1.0% NSR royalty to Franco-Nevada;

- 2.5% NSR royalty to Wotan; and
- 10% of annual payable gold is sold to Seabridge at a discount price of US$1,000, up to a maximum of 50,000 oz.

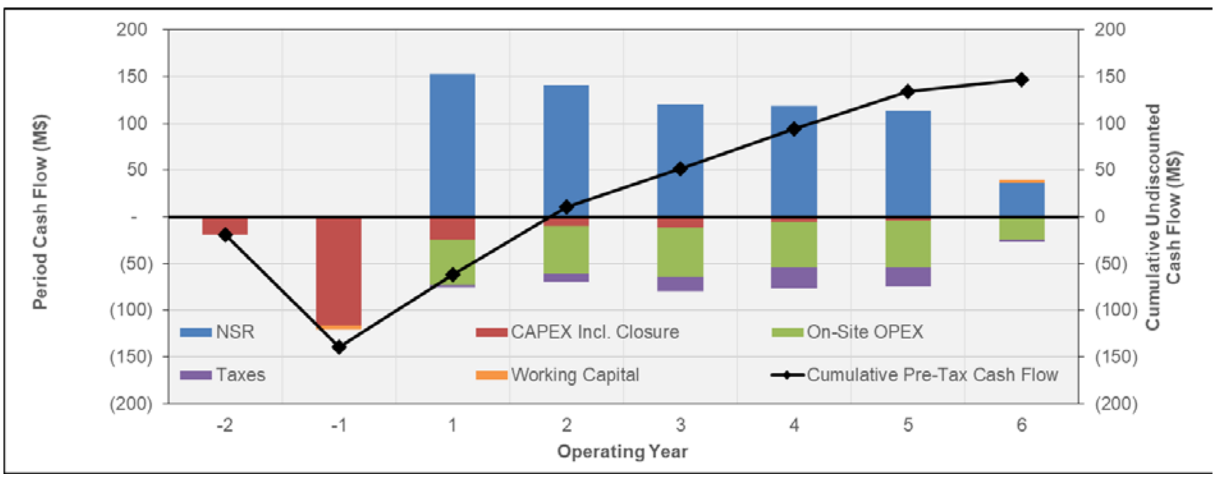
A total of $24 M of third party royalties are payable over the LOM, and 43 koz of gold are sold to Seabridge at the discount price (resulting in a US$10 M reduction to NSR).

22.5 Economic Results

The Project is economically viable with an after-tax internal rate of return (IRR) of 32% and an NPV at 5% ($NPV_{5\%}$) of $104 M. Figure 22-3 shows the projected cash flows from the economic analysis and Table 22-4 summarizes the detailed results of this evaluation.

The after-tax break-even gold price for the Project is approximately US$887/oz, based on the LOM plan presented herein and a silver price of US$17/oz.

Figure 22-3: After-Tax Annual Cash Flows



Source: JDS (2017)

Table 22-4: Summary of Results

Parameter	Unit	Value
AISC (excluding by-product)*	US$/payable oz Au	658
AISC (net of by-product) **	US$/ payable oz Au	611
Capital Costs		
Pre-Production Capital	M$	123.0
Pre-Production Contingency	M$	12.9
Total Pre-Production Capital	M$	**135.7**
Sustaining and Closure Capital	M$	60.4
Sustaining and Closure Contingency	M$	6.4
Total Sustaining & Closure Capital	M$	66.8
Total Capital Costs Incl. Contingency	M$	**202.4**
Working Capital	M$	4.0
Pre-Tax Cash Flow	LOM M$	208.0
	M$/a	39
Taxes	LOM M$	63.2
After-Tax Cash Flow	LOM M$	144.8
	M$/a	27
Economic Results		
Pre-Tax $NPV_{5\%}$	M$	155
Pre-Tax IRR	%	40
Pre-Tax Payback	Years	1.7
After-Tax $NPV_{5\%}$	M$	104
After-Tax IRR	%	32
After-Tax Payback	Years	1.9

Source: JDS (2017)

*(Operating Costs + Sustaining Costs + Closure Costs + Refining Costs + Royalties)/Payable Au Oz

**(Operating Costs + Sustaining Costs + Closure Costs + Refining Costs + Royalties – Payable Ag Value)/ Payable Au Oz

The annual cash flow model is shown in Table 22-5.

Table 22-5: Cash Flow Model

Item	Source	Unit	Pre-Production Total	Production Total	Life of Mine Total	Y -2	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
KEY PARAMETERS																		
Metal Prices																		
Gold (Au)	link	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Silver (Ag)	link	US$/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
Exchange Rates																		
USD : CAD	link	x:x	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
UNDERGROUND MINING																		
Total Production																		
Ore	calc	ktonnes	5	1,948	1,953	-	5	322	366	366	366	369	159	-	-	-	-	-
Au Grade	calc	g/t	13.58	7.51	7.53	-	13.58	9.82	8.20	7.20	7.14	6.70	4.72	-	-	-	-	-
Contained Au	calc	troy koz	2	470	473	-	2	102	96	85	84	80	24	-	-	-	-	-
Ag Grade	calc	g/t	46.25	21.79	21.86	-	46.25	34.43	27.14	19.63	15.45	17.54	13.31	-	-	-	-	-
Contained Ag	calc	troy koz	8	1,365	1,373	-	8	356	319	231	182	208	68	-	-	-	-	-
Operating Days	input	days	176	1,938	2,114	-	176	350	350	350	350	350	188					
Average Mining Rate	calc	tpd	30	1,005	924	-	30	920	1,045	1,045	1,045	1,056	847	-	-	-	-	-
MINERAL PROCESSING																		
Total Mineral Processing																		
Ore Processed	calc	ktonnes	-	1,953	1,953	-	-	322	366	366	366	369	165	-	-	-	-	-
Au Grade	calc	g/t	-	7.53	7.53	-	-	9.87	8.21	7.24	7.14	6.72	4.76	-	-	-	-	-
Contained Au	calc	koz	-	473	473	-	-	102	96	85	84	80	25	-	-	-	-	-
Ag Grade	calc	g/t	-	21.86	21.86	-	-	34.49	27.25	19.81	15.52	17.49	13.60	-	-	-	-	-
Contained Ag	calc	koz	-	1,373	1,373	-	-	357	320	233	183	208	72	-	-	-	-	-
Operating Days	input	days	-	1,988	1,988	-	-	365	365	365	365	365	163	-	-	-	-	-
Average Plant Throughput	calc	tpd	-	983	983	-	-	882	1,002	1,002	1,002	1,012	1,010	-	-	-	-	-
RECOVERED METAL																		
Total Recovery																		
Au	link	%	-	90.9%	90.9%	-	-	92.8%	91.6%	89.3%	90.1%	90.4%	89.9%	-	-	-	-	-
	calc	koz	-	430	430	-	-	95	88	76	76	72	23	-	-	-	-	-
Ag	link	%	-	86.3%	86.3%	-	-	90.1%	88.8%	82.1%	83.4%	84.5%	83.5%	-	-	-	-	-
	calc	koz	-	1,185	1,185	-	-	321	284	191	152	176	60	-	-	-	-	-
SALES & NSR																		
Payable Metals																		
Au Payable	link	%	-	99.0%	99.0%	-	-	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	-	-	-	-	-
	calc	koz	-	425	425	-	-	94	87	75	75	71	22	-	-	-	-	-
	calc	US$M	-	531.6	531.6	-	-	117.3	109.4	94.1	93.6	89.3	28	-	-	-	-	-
Ag Payable	link	%	-	99.0%	99.0%	-	-	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	-	-	-	-	-
	calc	koz	-	1,173	1,173	-	-	318	282	189	151	174	60	-	-	-	-	-
	calc	US$M	-	19.9	19.9	-	-	5.4	4.8	3.2	2.6	3.0	1.0	-	-	-	-	-
Total Payable Metals	**calc**	**US$M**	**-**	**551.6**	**551.6**	**-**	**-**	**122.7**	**114.2**	**97.3**	**96.1**	**92.2**	**29.1**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**-**	**725.7**	**725.7**	**-**	**-**	**161.4**	**150.2**	**128.1**	**126.5**	**121.4**	**38.2**	**-**	**-**	**-**	**-**	**-**
Refining Charges																		
Au Refining Charge	link	US$/pay oz	-	5.00	5.00	-	-	5.00	5.00	5.00	5.00	5.00	5.00	-	-	-	-	-
	calc	US$M	-	(2.1)	(2.1)	-	-	(0.5)	(0.4)	(0.4)	(0.4)	(0.4)	(0.1)	-	-	-	-	-
Ag Refining Charge	link	US$/pay oz	-	0.50	0.50	-	-	0.50	0.50	0.50	0.50	0.50	0.50	-	-	-	-	-
	calc	US$M	-	(0.6)	(0.6)	-	-	(0.2)	(0.1)	(0.1)	(0.1)	(0.1)	(0.0)	-	-	-	-	-
Total Refining Charges	**calc**	**US$M**	**-**	**(2.7)**	**(2.7)**	**-**	**-**	**(0.6)**	**(0.6)**	**(0.5)**	**(0.4)**	**(0.4)**	**(0.1)**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**-**	**(3.6)**	**(3.6)**	**-**	**-**	**(0.8)**	**(0.8)**	**(0.6)**	**(0.6)**	**(0.6)**	**(0.2)**	**-**	**-**	**-**	**-**	**-**
Seabridge Agreement																		
Au Sold to Seabridge at Discount	calc	koz	-	43	43	-	-	9	9	8	7	7	2	-	-	-	-	-
Remaining Payable Au	calc	koz	-	383	383	-	-	84	79	68	67	64	20	-	-	-	-	-
Seabridge Royalty Buy-Out Option	calc	C$M	-	-	-			-										
Net Smelter Return																		
Net Smelter Return *(before Royalties)*	**calc**	**US$M**	**-**	**538.2**	**538.2**	**-**	**-**	**119.7**	**111.4**	**95.0**	**93.8**	**90.0**	**28.4**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**-**	**708.2**	**708.2**	**-**	**-**	**157.5**	**146.6**	**125.0**	**123.4**	**118.4**	**37.3**	**-**	**-**	**-**	**-**	**-**
Royalties																		
Franco Nevada NSR Royalty	link	%	-	1.00%	1.00%	-	-	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	-	-	-	-	-
	calc	US$M	-	(5.5)	(5.5)	-	-	(1.2)	(1.1)	(1.0)	(1.0)	(0.9)	(0.3)	-	-	-	-	-
	calc	C$M	-	(7.2)	(7.2)	-	-	(1.6)	(1.5)	(1.3)	(1.3)	(1.2)	(0.4)	-	-	-	-	-
Wotan NSR Royalty	link	%	-	2.50%	2.50%	-	-	2.50%	2.50%	2.50%	2.50%	2.50%	2.50%	-	-	-	-	-
	calc	US$M	(0.1)	(12.9)	(13.0)	(0.0)	(0.0)	(2.2)	(2.8)	(2.4)	(2.4)	(2.3)	(0.7)	-	-	-	-	-
	calc	C$M	(0.1)	(17.0)	(17.1)	(0.1)	(0.1)	(2.9)	(3.7)	(3.2)	(3.1)	(3.0)	(1.0)	-	-	-	-	-
Total Royalties	**calc**	**US$M**	**(0.1)**	**(18.4)**	**(18.4)**	**(0.0)**	**(0.0)**	**(3.4)**	**(4.0)**	**(3.4)**	**(3.3)**	**(3.2)**	**(1.0)**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**(0.1)**	**(24.2)**	**(24.3)**	**(0.1)**	**(0.1)**	**(4.5)**	**(5.2)**	**(4.5)**	**(4.4)**	**(4.2)**	**(1.3)**	**-**	**-**	**-**	**-**	**-**
Net Smelter Return																		
Net Smelter Return *(after Royalties)*	**calc**	**US$M**	**(0.1)**	**519.8**	**519.8**	**(0.0)**	**(0.0)**	**116.3**	**107.4**	**91.6**	**90.5**	**86.8**	**27.3**	**-**	**-**	**-**	**-**	**-**
	calc	*US$/t*	-	*266*	*266*	-	-	*361*	*294*	*250*	*247*	*235*	*166*	-	-	-	-	-
	calc	**C$M**	**(0.1)**	**684.0**	**683.9**	**(0.1)**	**(0.1)**	**153.0**	**141.3**	**120.5**	**119.0**	**114.2**	**36.0**	**-**	**-**	**-**	**-**	**-**
	calc	*C$/t*	-	*350*	*350*	-	-	*475*	*386*	*329*	*325*	*309*	*218*	-	-	-	-	-

Table 22-5: Cash Flow Model (continued)

Item	Source	Unit	Pre-Production Total	Production Total	Life of Mine Total	Y -2	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
OPERATING COSTS																		
Mining	calc	C$/t processed	-	72.30	72.30	-	-	78.14	75.50	76.01	65.23	68.04	70.80	-	-	-	-	-
	link	C$M	-	(141.2)	(141.2)	-	-	(25.1)	(27.6)	(27.8)	(23.9)	(25.1)	(11.7)	-	-	-	-	
Processing	calc	C$/t processed	-	45.96	45.96	-	-	47.34	44.22	46.71	48.18	47.41	37.31	-	-	-	-	-
	link	C$M	-	(89.8)	(89.8)	-	-	(15.2)	(16.2)	(17.1)	(17.6)	(17.5)	(6.1)					
Site Services	calc	C$/t processed	-	10.40	10.40	-	-	11.18	9.91	10.17	10.17	10.07	11.70	-	-	-	-	-
	link	C$M	-	(20.3)	(20.3)	-	-	(3.6)	(3.6)	(3.7)	(3.7)	(3.7)	(1.9)	-	-	-	-	
General & Administration	calc	C$/t processed	-	11.36	11.36	-	-	13.91	10.79	10.73	10.72	10.58	12.26	-	-	-	-	-
	link	C$M	-	(22.2)	(22.2)	-	-	(4.5)	(3.9)	(3.9)	(3.9)	(3.9)	(2.0)	-	-	-	-	
Total Operating Costs	**calc**	**C$/t processed**	**-**	**140.02**	**140.02**	**-**	**-**	**150.56**	**140.43**	**143.63**	**134.29**	**136.09**	**132.07**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**-**	**(273.5)**	**(273.5)**	**-**	**-**	**(48.5)**	**(51.4)**	**(52.5)**	**(49.1)**	**(50.3)**	**(21.7)**	**-**	**-**	**-**	**-**	**-**
INCOME																		
Net Operating Cash Flow	**calc**	**C$M**	**(0.1)**	**410.5**	**410.4**	**(0.1)**	**(0.1)**	**104.5**	**90.0**	**68.0**	**69.9**	**63.9**	**14.2**	**-**	**-**	**-**	**-**	**-**
	calc	**C$/t**	**-**	**210.15**	**210.10**	**-**	**-**	**324.68**	**245.98**	**185.84**	**191.13**	**172.99**	**86.39**	**-**	**-**	**-**	**-**	**-**
Au Cash Cost	*calc*	*US$/oz*	-	*538*	*539*	-	-	*436*	*498*	*582*	*549*	*586*	*788*	-	-	-	-	-
Au Cash Cost (Net of By-Product)	*calc*	*US$/oz*	-	*491*	*492*	-	-	*378*	*443*	*539*	*515*	*545*	*743*	-	-	-	-	-
Operating Margin	*calc*	*%*	-	*57%*	*57%*	-	-	*65%*	*60%*	*53%*	*55%*	*53%*	*37%*	-	-	-	-	-
CAPITAL COSTS																		
Initial & Sustaining Capital Costs																		
Mining	link	C$M	(11.3)	(38.3)	(49.6)	-	(11.3)	(16.7)	(8.0)	(6.1)	(4.7)	(0.8)	(2.0)					
Site Development	link	C$M	(9.0)	(2.2)	(11.2)	(1.7)	(7.2)	(1.3)	(0.9)	(0.0)	(0.0)	(0.0)	(0.0)					
Mineral Processing	link	C$M	(37.7)	(0.4)	(38.0)	-	(37.7)	(0.4)	-									
Tailings Management	link	C$M	(7.2)	(10.9)	(18.1)	(1.3)	(5.9)	(3.8)	(0.0)	(4.4)	(0.0)	(2.6)	(0.0)					
Infrastructure	link	C$M	(23.7)	-	(23.7)	(3.2)	(20.5)	-	-									
Off-Site Infrastructure	link	C$M	(2.8)	-	(2.8)	(2.8)	-	-	-									
Project Indirects	link	C$M	(9.3)	-	(9.3)	(0.6)	(8.7)	-	-									
EPCM	link	C$M	(13.0)	-	(13.0)	(6.5)	(6.5)	-	-									
Owner's Costs	link	C$M	(9.1)	-	(9.1)	(1.1)	(7.9)	-	-									
Subtotal - Pre-Contingency	**calc**	**C$M**	**(123.0)**	**(51.7)**	**(174.7)**	**(17.3)**	**(105.7)**	**(22.2)**	**(8.9)**	**(10.5)**	**(4.7)**	**(3.4)**	**(2.0)**	**-**	**-**	**-**	**-**	**-**
Contingency	link	C$M	(12.7)	(5.2)	(17.9)	(1.8)	(10.9)	(2.2)	(0.9)	(1.1)	(0.5)	(0.3)	(0.2)					
Contingency (% of Total Costs)	calc	%	10.3%	10.0%	10.2%	10%	10%	10%	10%	10%	10%	10%	10%	0%	0%	0%	0%	0%
Total	**calc**	**C$M**	**(135.7)**	**(56.9)**	**(192.6)**	**(19.1)**	**(116.6)**	**(24.4)**	**(9.8)**	**(11.6)**	**(5.2)**	**(3.8)**	**(2.2)**	**-**	**-**	**-**	**-**	**-**
Salvage & Closure Costs																		
Closure Costs	input	C$M	**-**	**(12.4)**	**(12.4)**	-	-	-	-					(11.8)	(0.1)	(0.1)	(0.1)	(0.4)
Closure Contingency	input	C$M	**-**	**(1.2)**	**(1.2)**	-	-	-	-					(1.2)	(0.0)	(0.0)	(0.0)	(0.0)
Salvage Value	input	C$M	**-**	**3.8**	**3.8**	-	-	-	-					3.8				
Total - Salvage & Closure Costs	**calc**	**C$M**	**-**	**(9.9)**	**(9.9)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**(9.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.5)**
Total Capital Costs																		
Total Capital Costs	**calc**	**C$M**	**(135.7)**	**(66.8)**	**(202.4)**	**(19.1)**	**(116.6)**	**(24.4)**	**(9.8)**	**(11.6)**	**(5.2)**	**(3.8)**	**(2.2)**	**(9.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.5)**
WORKING CAPITAL																		
Working Capital & Reserve Accounts																		
Working Capital	calc	C$M	**(4.0)**	**4.0**	**-**	-	(4.0)	-	-	-	-	-	4.0	-	-	-	-	-
CASH FLOWS																		
Pre-Tax																		
Net Pre-Tax Free Cash Flow	calc	C$M	(139.8)	347.8	208.0	(19.1)	(120.7)	80.1	80.2	56.4	64.7	60.1	16.1	(9.1)	(0.1)	(0.1)	(0.1)	(0.5)
Cumulative Pre-Tax Free Cash Flow	calc	C$M			208.0	(19.1)	(139.8)	(59.7)	20.5	76.9	141.6	201.7	217.8	208.7	208.6	208.5	208.4	208.0
Post-Tax																		
Income Taxes	link	C$M	-	(39.3)	(39.3)	-	-	-	(6.0)	(13.6)	(12.8)	(12.1)	(1.3)	5.4	0.6	0.4	-	-
BC Mineral Taxes	link	C$M	-	(23.9)	(23.9)	-	-	(2.2)	(1.9)	(2.0)	(9.0)	(8.4)	(1.7)	1.2	0.0	0.0	0.0	0.1
Net After-Tax Free Cash Flow	calc	C$M	(139.8)	284.6	144.8	(19.1)	(120.7)	78.0	72.3	40.7	43.0	39.7	13.1	(2.6)	0.5	0.3	(0.1)	(0.4)
Cumulative After-Tax Free Cash Flow	calc	C$M			144.8	(19.1)	(139.8)	(61.8)	10.4	51.2	94.2	133.9	147.0	144.4	144.9	145.3	145.2	144.8
ECONOMIC INDICATORS																		
Pre-Tax																		
Pre-Tax IRR	**calc**	**%**			**40.2%**													
Pre-Tax Payback	**calc**	**Years**			**1.7**													
Pre-Tax NPV @ 0%	**calc**	**C$M**			**208.0**													
Pre-Tax NPV @ 5%	**calc**	**C$M**			**154.7**													
Post-Tax																		
After-Tax IRR	**calc**	**%**			**31.5%**													
After-Tax Payback	**calc**	**Years**			**1.9**													
After-Tax NPV @ 0%	**calc**	**C$M**			**144.8**													
After-Tax NPV @ 5%	**calc**	**C$M**			**103.8**													

Source: JDS (2017)

22.6 Sensitivities

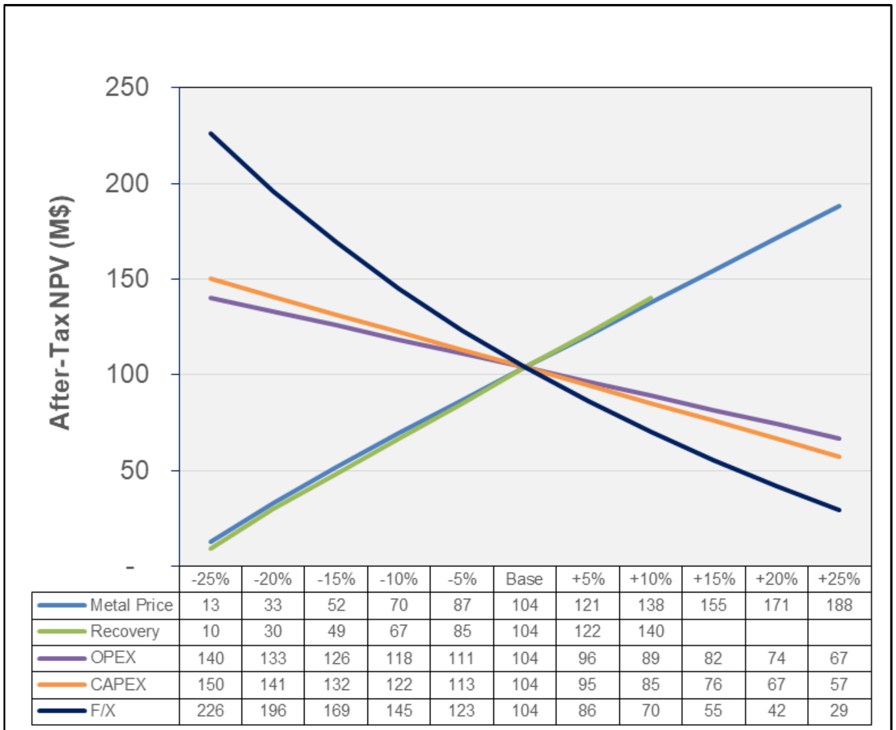
A sensitivity analysis was performed to test Project value drivers on the NPV using a 5% discount rate. The results of this analysis are presented in Table 22-6. The analysis revealed that the Project is most sensitive to recovery, then metal prices, followed by FX rate, and finally capital costs. The Project showed the least sensitivity to operating costs. Figure 22-4 shows the results of the sensitivity tests.

Table 22-6: Sensitivity Results

Variable	After-Tax $NPV_{5\%}$ (M$)		
	-25% Variance	0% Variance	+25% Variance
Metal Price	13	104	188
Recovery	10	104	N/A
OPEX	140	104	67
CAPEX	150	104	57
FX Rate	226	104	29

Source: JDS (2017)

Figure 22-4: After-Tax NPV @ 5% Sensitivity Results



Source: JDS (2017)

23 Adjacent Properties

There are no adjacent properties relevant to the scope of this report.

24 Other Relevant Data and Information

24.1 Project Execution & Development Plan

24.1.1 Introduction

The purpose of the Red Mountain Project Execution Plan (PEP) is to provide a framework for managing the engineering, procurement, and construction phases of the Red Mountain Project, and describes the Project development strategies that were considered to form the basis for the capital cost estimate and Project schedule within the Red Mountain Project FS.

The PEP for the FS is based on the following principles:

- Promote safety in design, construction, and operations to succeed;
- Use fit-for-purpose designs, constructions, and operations;
- Establish permanent infrastructure early, to the extent practical, to minimize costs of temporary construction facilities; and
- Negotiate contracts with suppliers, contractors, and engineers with proven track records in mine developments.

24.1.2 Project Execution Locations

IDM currently operates a corporate office in Vancouver, BC. It is not expected that any significant volume of Project work will be performed in this office; however, it will be used as a hub for Project meetings as required. Detail design will be performed by firms specializing in engineering for mining and milling projects.

In the field, IDM will initially maintain a small office at the construction camp location in Stewart, BC, during the start of Project development until infrastructure is available at site.

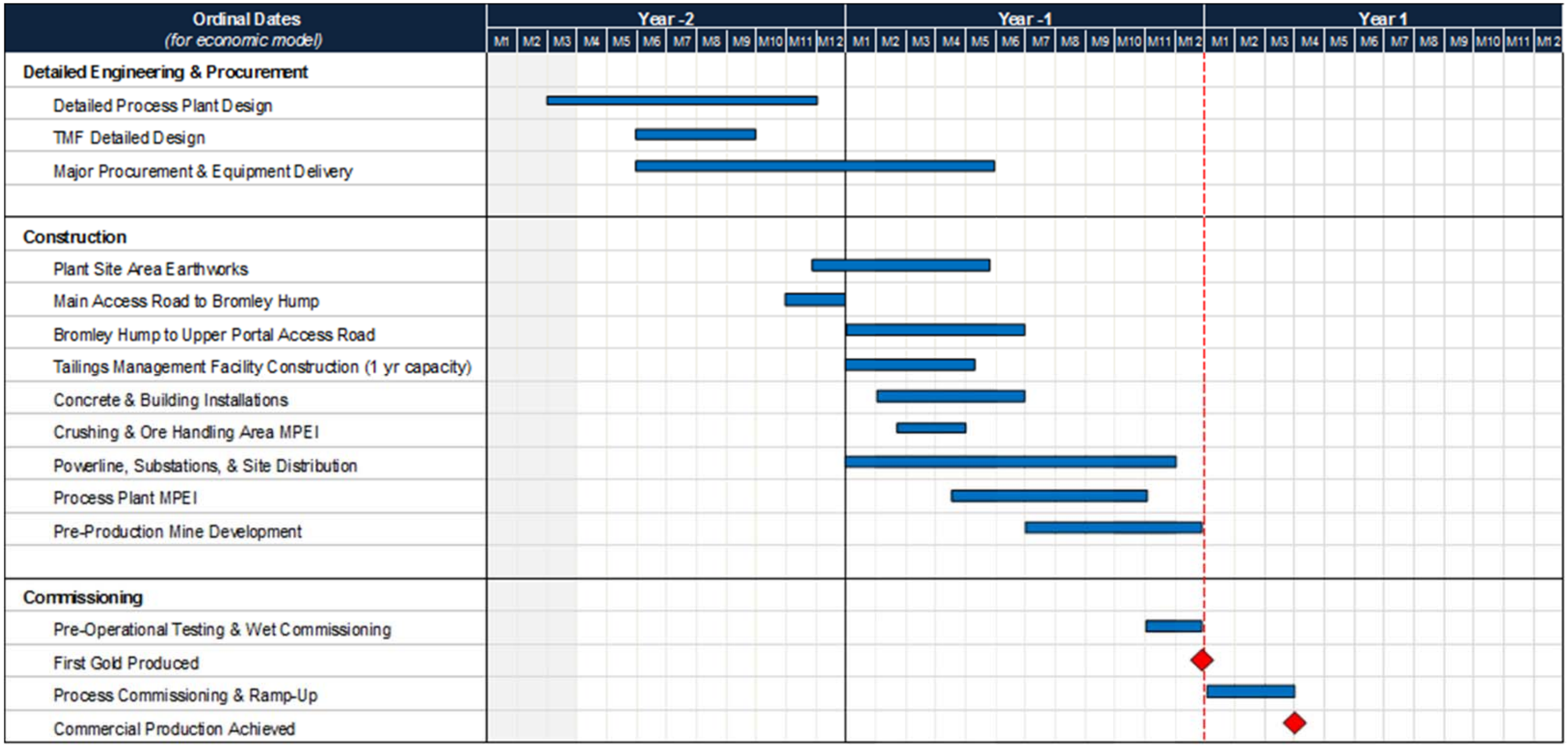
Once access is established to the Bromley Humps area, the centre of activity will shift to the Project site as major construction activities kick-off and major procurement slows down. The Project strategy is to have the majority of personnel based at the Project site to avoid the need for large satellite offices.

24.1.3 Project Development Schedule Overview

A labour-resource loaded level 3 Project schedule was developed for the Project, using the capital cost estimate as the basis for on-site man-hours to drive activity durations. Figure 24-1 presents a summary schedule for the development of the Red Mountain Project.

The critical path for the Project runs through the detailed engineering and construction activities related to the processing facilities. Construction activities are dependent on receiving the access road permit and the Mines Act and Environmental Management Act (MAPA/EMA) permit. Other near-critical activities include construction of the main access road, site preparations (earthworks and temporary facilities), and establishment of the TMF.

Figure 24-1: Project Schedule



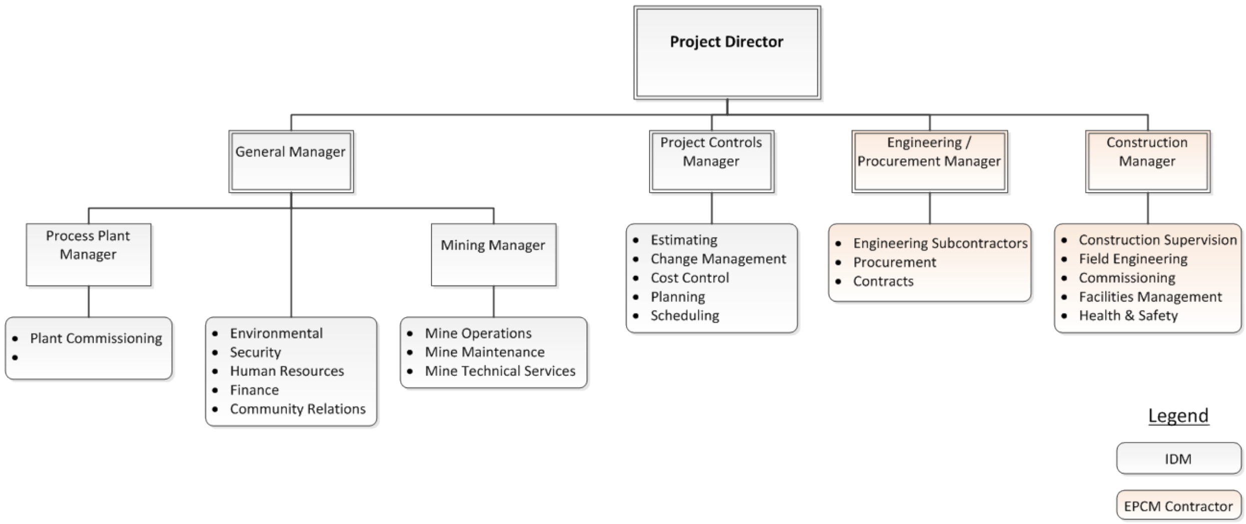
Source: JDS (2017)

24.1.4 Project Management

24.1.4.1 Organization & Responsibilities

The Project Management Team (PM Team) will be an integrated team comprised of 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 24-2 presents a representative preliminary organization chart

Figure 24-2: Preliminary Project Management Team Organization Chart



Source: JDS (2017)

24.1.4.1.1 Senior Project Management

Overall delivery of the Project to the defined metrics will be the responsibility of the IDM Project Director. The Project Director will provide high level direction to the PM Team, with support from the contractor and the owner's Pre-Operational Team to manage Project activities.

The Project Director will be responsible for the execution of Project activities, including detailed engineering, procurement, logistics, construction, commissioning, and Project controls.

24.1.4.1.2 Owners Operations Team

A portion of the owner's Operations Team will be mobilized during the Project development phase for functions required over the LOM (i.e., not limited to construction support):

- Mining operations, including maintenance;
- Environmental;
- Security;
- Accounting;
- Community Relations;

- Human Resources; and
- Site Services (Construction Manager responsible during construction).

24.1.4.1.3 Engineering Team

The Engineering/Procurement Manager will oversee, coordinate, and integrate engineering activities. The Engineering Team will consist of various engineering sub-contractors, who will develop the detailed designs and specifications for the Project, and then transition to the field to provide QA, field engineering, and commissioning support.

24.1.4.1.4 Procurement Team

The Engineering/Procurement Manager will oversee and manage procurement activities undertaken by engineering contractors (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 vendors and construction contractors to provide the appropriate goods and services.

24.1.4.1.5 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 progresses on schedule and within budget.

24.1.4.1.6 Commissioning Team

The Construction Manager will oversee the commissioning team, and be responsible for the timely handover of process and infrastructure systems to the owner once construction activities have been substantially completed. The commissioning team will be supported by disciplined engineering resources to complete pre-commissioning activities and to obtain technical acceptance and transfer care, custody, and control of completed systems to the owner.

24.1.4.1.7 Project Controls Team

The Project Controls Manager will oversee the Project controls team, and be responsible for the development, implementation, and administration of the processes and tools for Project estimating, cost control, planning, scheduling, change management, progressing, and forecasting.

24.1.4.2 Project Procedures

During the Project setup phase (immediately upon Project approval), a Project procedures manual will be developed, which will outline standard procedures for site construction. This document will focus on the interfacing between the owner and engineering contractors, and will address delegation of authority, change management, procurement workflows, QA, and reporting standards.

24.1.5 Engineering

24.1.5.1 Engineering Execution Strategy

The general engineering execution strategy for the Project will be to utilize multiple engineering firms with specialized knowledge of their assigned scope. Coordination of engineering interfaces and overall management of engineering schedule and deliverables will be the responsibility of the

Engineering/Procurement Manager. The following major engineering contract packages have been identified for the Project:

- Detailed engineering and procurement of process facilities, and select on-site infrastructure and field engineering support;
- Detailed engineering of the TMF and associated water diversion structures;
- Detailed engineering of the site access and haul roads;
- Site water management design;
- Water treatment plant design; and
- Power transmission line detailed design.

24.1.5.2 Engineering Management

24.1.5.2.1 Baseline Engineering Data

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 Project change management processes.

24.1.5.2.2 Design Criteria Approval

The Project critical path includes timely completion of engineering activities. To prevent delays or late changes in engineering deliverables and to keep efforts focused, a formal engineering approval procedure will be enacted for the Project.

24.1.5.2.3 Engineering Progress & Performance Monitoring

Each engineering contractor will provide a deliverables list as part of their services proposal. Deliverables (and their associated budgets) will be grouped into logical Engineering Work Packages (EWPs), which will be used as the metric for tracking engineering progress for the Project.

24.1.6 Procurement & Contracting

24.1.6.1 Procurement Execution Strategy

The general procurement execution strategy for the Project will involve utilizing known suppliers, with a preference for Nisg̱a'a Nation, local and regional suppliers and construction contractors. The Engineering Subcontractor Procurement Manager (under the direction of the Engineering/Procurement 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.

24.1.6.2 Construction Contracting Strategy

Table 24-1 presents a listing of the major contract packages identified for the Project. For the purpose of the FS, all mechanical, piping, electrical, and instrumentation (MPEI) works have been identified as performed by multiply entities within the Project estimate and schedule. During Project execution, a minimum of two MPEI contractors will be engaged to avoid reliance on the performance of a single entity. During contractor pre-qualifications, if multi-discipline contractors cannot be sourced, then a horizontal contracting strategy will be employed (separate contractors for each trade, i.e., mechanical, piping, electrical, and instrumentation).

The strategy for the underground mining activities on the Project is to use the owner's labour forces, with select seconded trainers from contract labour providers, if required.

Table 24-1: Major Construction Contracts (Capital Phase)

Package Breakdown Structure	Contract	Estimated Man-Hours	Estimated Value (C$M)	Contract Type
CC001	Bulk Earthworks	55,941	11.7	Unit Rate
CC002	Tailing Facility & Diversion Channel Construction	22,802	7.2	Unit Rate
CB001	Concrete Installations	12,952	2.1	Unit Rate
CE001	HV Power Transmission Line Installation	24,000	11.0	Lump Sum
CE003	Underground Electrical Distribution	10,000	1.2	Lump Sum
CA001	Plant Site Architectural Works	19,071	3.9	Lump Sum
CS001	Structural Steel Supply & Erection	1,499	0.7	Unit Rate
CS002	Ancillary Buildings Supply/Install	8,514	3.5	Lump Sum
CG001	Mechanical/Piping/Electrical/Instrumentation #1	95,148	14.8	T&M

Source JDS (2017)

24.1.6.3 Procurement Schedule & Critical Activities

Procurement activities will be prioritized to schedule critical items, both due to fabrication/delivery time of the equipment (such as the grinding mill package and underground mining equipment), and due to the necessity to obtain certified vendor data to complete structural and foundation designs.

Tendering and award of the following packages are considered time critical:

- Main access road construction;
- Detailed engineering packages, particularly for the TMF and processing packages;
- Site earthworks; and
- Concrete installations.

24.1.6.4 Selection of Suppliers & Contractors

A competitive bidding process will be applied to achieve the best commercial and technical results from the procurement effort. During the Project setup phase, any preferred vendors will be identified and sole source strategies implemented into the procurement plan. Nisg̱a'a Nation and local involvement will form part of the bid evaluation scoring criteria in order to give preference for suppliers and contractors that maximize benefits for Nisg̱a'a Nation and the Stewart area.

The level of vendor quality surveillance/inspection (VQS) required for each package will be established during bid evaluations, and will be determined by evaluating a supplier's ability to achieve suitable quality according to specifications and Project QA requirements.

24.1.7 Logistics & Material Management

24.1.7.1 Logistics Execution Strategy

The general logistics strategy for the Project is as follows:

- Ensure expediting activities achieve the Project schedule requirements;
- Manage freight movement on a global basis to maximize leveraging the freight tonnage/volume to optimize cost associated with the movement of freight; and
- Identify and optimize various aspects such as logistics, customs clearance, and local content.

24.1.7.2 Shipping Routes

Materials and equipment will be brought to site via BC Highway 37A from southern BC and other parts of Canada. Equipment originating in Asia could be shipped via sea freight through the Stewart World Port. All road transport is on paved road, with the exception of the 15 km site access road, which will be constructed with a gravel surface.

24.1.7.3 Freight Quantities

Table 24-2 presents the estimated international and domestic freight quantities for the pre-production period of the Project.

Table 24-2: Freight Quantities

Grouping	Loads	Tonnes
Underground Mining / Site Support Equipment	46	795
Contractor Equipment & Materials	90	1,931
Infrastructure & Utilities	145	2,759
Processing Equipment	140	2,795
Tailings & Water Management	27	334
Explosives	25	616
Consumables & Supplies	12	223
Total	**485**	**9,453**

Source: JDS (2017)

24.1.7.4 Pre-Assembled Equipment

Pre-assembly strategies reduce overall site man-hours and the associated indirect costs, but require more careful engineering and logistics planning. The following goods have been identified for pre-assembly within the FS estimate:

- Secondary / tertiary crushers;
- Electrical houses;
- Fuel tanks;
- Water and process tanks (up to 5 m diameter);

- Fuel loading/unloading station;
- Conveyors (shipped in pre-fabricated lengths); and
- Transfer towers, braced frames, and stair towers.

24.1.7.5 Site Materials Strategy

The general strategy for site materials control is as follows:

- Control and supervise materials movement at site through materials/inventory control from receiving, preservation, inventory, and free-issue to contractor to meet the Project requirements for equipment and materials procured by the construction team or IDM (process equipment, as an example);
- Leverage contractor methods and procedures for receipt, storage, and retrieval of materials procured within their scope of work;
- Utilize a common labour pool for warehouse and laydown staff (equipment operators and labourers) for the management and movement of freight, except for items requiring special handling or rigging (such as structural steel); and
- Utilize a single temporary warehouse to be used for the receipt and storage of all equipment requiring climate controlled indoor storage. Equipment and material that do not require climate controlled storage will be stored in laydown areas within the construction site. Use of sea containers and/or temporary shelters will be required to store goods that need to be protected during construction.

24.1.7.6 Construction Execution Plan Overview

The main objectives of the construction execution strategy for the Red Mountain Project include:

- Execute all activities with a goal of zero harm to people, assets, the environment, or reputation;
- Strive to eliminate process, operational, and maintenance safety hazards;
- Meet or exceed environmental regulatory and permit requirements to minimize impact;
- Deliver a high-quality facility that meets or exceeds the defined Project goals;
- Establish and maintain a high level of motivation by providing a positive working environment for all personnel;
- Identify and remove barriers that affect Project progress;
- Cultivate an atmosphere of positive social impact in the Nisg̲a'a Nation communities and surrounding communities; and
- Identify outstanding achievements during construction and commissioning of the Project.

The path of construction for the Red Mountain Project is driven by two main critical paths:

- Key infrastructure development early in the schedule to support the start of pre-production mining activities for the development of underground workings and ore production; and

- Concurrent construction of the processing and ancillary areas to allow early operations.

The overall construction duration, from the start of the main access road construction to the commissioning of first gold production, will be approximately 15 months.

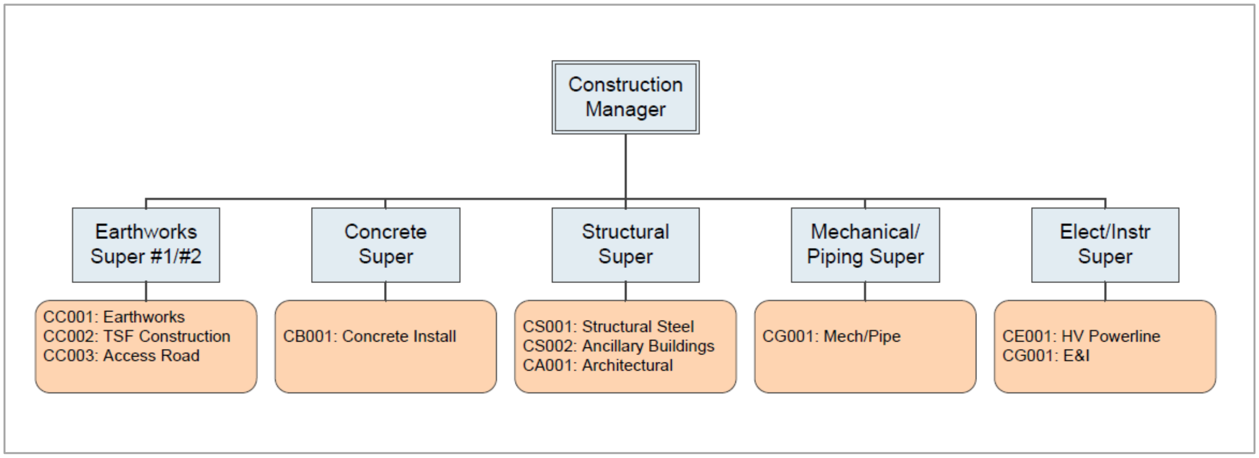
24.1.7.7 Site Management

During the construction phase, the Construction Manager and the Mine General Manager (or their designate) will carry overall responsibility for the Project site. A division of area responsibility will be established between the construction manager and mine general manager.

24.1.7.8 Construction Management

The Construction Manager will be responsible for construction contractor oversight. Figure 24-3 shows the Construction Manager's responsibility for the superintendents.

Figure 24-3: Construction Management Responsibilities



Source: JDS (2017)

24.1.7.9 Safety Management

A comprehensive Safety Management Plan (SMP) will be developed prior to site mobilization. The SMP will address overall safety policies, procedures, and standards for the Project, including standard operating practices and emergency response plans.

24.1.7.10 Quality Management

Construction quality will be managed through the implementation of a Site Quality Management Plan (SQMP), which will detail the site quality management systems to be used for all construction activities. The SQMP encompasses all activities of the Project, including design, procurement and construction. Site QA is the responsibility of the field engineering team, and is verification that QC is being performed by the contractor, subcontractor, laboratory, and third-party inspection services.

24.1.7.11 Construction Quantities

Table 24-3 presents the estimated major commodity quantities for the Project. Quantities are based on the FS engineering take-offs and capital estimate.

Table 24-3: Major Construction Quantities

Discipline	Commodity Quantity	
	Unit	Quantity
Power Transmission Line	km	21.3
Bulk Earthworks – Total Cut	m^3	362,500
Bulk Earthworks – Total Fill	m^3	369,200
Bulk Earthworks – Crushed Material	m^3	51,300
HDPE Liners	m^2	125,000
Site Wide Concrete (including lean-concrete)	m^3	1,360
Structural Steel (excluding buildings)	Tonne	995
Process Buildings	m^2	2,346
Ancillary Buildings	m^2	10,494
Mechanical (Crushing and Plant Area Only)	# tagged equip	229
Cable Tray	km	1,900
Power Cable	m	8,100

Source: JDS (2017)

24.1.7.12 Construction Milestones

Table 24-4 presents the major construction milestones for the Project.

Table 24-4: Major Construction Milestones

Milestone	Date
Road Permit Received	Month 1
MA/EMA Permit Received	Month 3
Site Access Road Complete	Month 2
Site Development - Plant Pad (Earthworks) Completed	Month 4
Transmission Line (138 kV) complete	Month 6
Mine Dry / Administration - Construction Completed	Month 5
TMF - Phase 1 Construction Completed	Month 6
Maintenance Shop / Warehouse - Construction Complete	Month 6
Site Haul Roads Complete	Month 9
Process Facilities - Construction Completed	Month 13
Transmission Line (25 kV) Complete	Month 13
Pre-Operational Testing & Wet Commissioning Complete	Month 15
First Gold	Month 15

Source: JDS (2017)

24.1.8 Commissioning

24.1.8.1 Commissioning Methodology

Progressive commissioning for the Red Mountain Project will be performed by subsystems. A system will be defined as a logical grouping of equipment or systems that can be placed in service more or less by itself and that contribute to a common purpose or functionality. A system may also be a facility or a building. Subsystems will be defined during detailed engineering.

24.1.8.2 Commissioning Safety & Training

The Health, Safety, and Environmental Plan (HSE Plan) developed during execution will address specific safety procedures that will apply during the commissioning stage of the Project. The commissioning and turnover phase presents significant and unique safety risks. A comprehensive lock-out tag-out program is an effective control to manage these risks.

24.1.8.3 Commissioning Stages

- Construction Release (Stage 1): Construction contractor completes a system subject to agreed punch list items.
- Pre-Operational Equipment Testing (Stage 2): Energize and test individual equipment within subsystems to ensure functionality; includes equipment functionality tests controlled by the plant control system (signed-off loop diagrams).
- Pre-Operational Systems Testing (Stage 3): Systems tested with water, air, and insert materials, and capable of continuous and safe operation with all instrumentation connected, the control system is operational, and all interlocks functional.
- Ore Commissioning (Stage 4): Plant ready to accept ore and all operating and maintenance staff are fully trained to operate and maintain the plant; individual systems operate successfully under load for a defined period of time.
- Ramp-Up (Stage 5): Increase ore feed to design throughput rate.

25 Interpretations and Conclusions

Results of this FS demonstrate that the Red Mountain Project warrants development due to its positive, robust economics.

It is the conclusion of the QPs that the FS summarized in this Technical Report contains adequate detail and information to support a feasibility level analysis. Standard industry practices, equipment, and design methods were used in this Feasibility Study. With the exception of those outlined in this section, the report authors are unaware of any unusual or significant risks or uncertainties that would affect Project reliability or confidence based on the data and information made available.

For these reasons, the path going forward must continue to focus on obtaining environmental permit/approvals, while concurrently advancing key activities that will reduce Project execution time.

Most mining projects are exposed to risks that might impact the economics of the project to varying degrees. Most risks are external and largely beyond the control of the project proponents. They can be difficult to anticipate and mitigate although, in many instances, some reduction in risk might be achieved by regular reviews and interventions over the life of the project. Certain opportunities that can enhance the project economics might also be identified during the early years of construction, particularly with respect to conservative engineering and design parameters applied during the engineering stage of project development.

25.1 Risks

External risks are things such as the political situation in the project region, metal prices, exchange rates, and government legislation. These external risks are generally applicable to all mining projects.

Feasibility engineering formulates design and engineering solutions to reduce that risk common to every mining project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, environmental and social impact, political risks, schedule and cost overruns, and labour sourcing. JDS is of the opinion that these risks have been identified and mitigation measures have been considered.

Potential risks associated with the Red Mountain Project include:

- **Environmental Permitting –** There is risk to the Project related to potential delays in receiving the required environmental and construction permits to construct the mine. IDM will need to continue working closely with the regulatory authorities and Nisg̱a'a Nation to provide detailed information to prove the effectiveness of mitigation measures developed to manage the various impacts. IDM has invested in developing constructive and mutually respectful relationships with NLG as well as other Project-affected communities. Through consultation processes, NLG stakeholders have raised normal concerns about the Project. These are being addressed in the Project design and management plans as much as possible. Ongoing consultation with Nisg̱a'a Nation and community stakeholders will be required during the life of the Project to ensure that any concerns are swiftly resolved.

- **Groundwater –** The extent to which the groundwater inflow estimates are realized require that the development plan addresses mitigating variation. Increases in the actual amount of groundwater encountered would impact development costs. Drilling for drainage, and operational definition drilling included in the mine plan, will help to identify specific water bearing zones with higher than expected flows and to then establish control and/or management procedures. As well, initiating certain development earlier in the mine life to allow more time for dewatering may prove cost effective.
- **Mining Dilution –** Unplanned dilution from unsatisfactory mining practices, ground control issues or deposit geometrical variability can lead to waste rock shipped with ore resulting in higher operating costs.
- **Cost Escalation –** The capital and operating costs described in this report are achievable under current economic conditions and Project fit-for-purpose philosophy. Should any of these factors change to the negative, the Project economics would be impacted unfavourably.
- **Geotechnical Conditions –** Geotechnical studies were done to estimate anticipated ground conditions in the proposed underground mine. There is a risk that a larger percentage of the ore must be extracted using C&F in certain areas rather than the longhole method resulting in higher costs. This risk can be managed by completing additional geotechnical investigations and studies on those areas identified in the FS as having potentially weaker ground conditions.
- **Metallurgy –** While the sampling regime and metallurgical testing programs were sufficient for a FS, recovery variability may occur during operations. A reduction in gold recovery would have a negative impact on Project economics.
- **TMF Capacity –** The TMF proposed in the FS has the capacity to store the mine plan ore volume. However, any significant changes to the assumed consolidated tailings density or consolidation rates has the potential to reduce the capacity of the TMF resulting in additional engineering being required to determine additional storage options.
- **Weather Conditions –** Extreme winter conditions will impact personnel and equipment productivities during construction and operations. Snow fall in the region is very high and is coupled with steep alpine terrain, which leads to significant avalanche risk. Snow fall and avalanche potential have the ability to close the access and haul roads for extended periods of time, resulting in construction and operating delays. This risk can be managed by completing detailed avalanche mapping risk assessment in order to identify all avalanche terrain presenting a hazard to Project workers, facilities, and infrastructure. Such work is also necessary to establish and evaluate appropriate avalanche risk mitigation options for the Projects and to develop an Avalanche Safety Plan.
- **Retention of Skilled Personnel** – A relatively short mine life coupled with a potential lack of housing in Stewart might make attracting and retaining skilled personnel difficult. This could lead to increased payroll cost, operational limitations, and being unable to fill positions with local individuals. This risk can be mitigated by continuous labour market monitoring, providing attractive remuneration packages and incentives, and creating positive workplace conditions. It will also be important to develop and provide training programs for local individuals.

25.2 Opportunities

The FS has highlighted several opportunities to increase mine profitability and Project economics, as well as reduce identified risks.

Potential opportunities associated with the Red Mountain Project include:

- Inferred Resources are not included in the production schedule; however, a plan to infill drill specific areas could significantly improve the Project economics. Operational definition drilling will test Inferred Resources as part of the production sequence. The conversion of existing Inferred Resources and the identification of additional Inferred Resources that can be converted into Measured and Indicated classes, and potentially into Proven and Probable Reserves;
- 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 sulphidation system;
- The sediment-porphyry contact that controls mineralization in the Marc/AV/JW zones can be traced in the SF zone for a further 800 m along strike to the north through sparse drilling with isolated gold intercepts. Further drilling could potentially expand resources both up and down-dip from the AV and JW Zones, and along strike from the 141 Zone and Marc Zone (to the south);
- Optimization of mine plans and production schedules to include results of additional resources and geotechnical investigations may add more economic value to the Project;
- The increased use of used processing equipment, which is presently available from several sources, would reduce the Project's equipment cost and overall Project CAPEX, and potentially reduce the engineering, procurement, and construction schedules;
- In the spring of 2016, IDM signed an MOU with Bridge Power Corp., an independent power producer with run-of-river hydroelectric generation rights to Bitter Creek. The companies are committed to sharing environmental baseline data, and potentially sharing capital costs for construction of the access road and power line. This would potentially result in substantial cost reductions to the capital and operating costs at Red Mountain;
- With road access in mineral-rich northwestern British Columbia, toll-treatment of nearby deposits could add value to the Project;
- The recent drilling program has discovered additional mineralization that is not part of the current mineral resource. If this mineralization can be upgraded to Indicated Mineral Resource and incorporated into the mine plan, then the mine life could be extended; and
- Additional geotechnical investigation of the deposit, especially in areas identified as having relatively weaker ground conditions in the FS, will assist in optimizing the mine design and ground support recommendations in these areas. Potential exists, if proven, to shift the mining method in these areas from higher cost C&F to lower cost longhole stoping, leading to an improvement in mining operating costs.

26 Recommendations

Due to the positive, robust economics, it is recommended to expediently advance the Red Mountain Project to construction and development, followed by production. The recommended development path is to continue efforts to obtain the BC MA/EMA permit approval. It is also recommended that IDM concurrently advance key activities that will reduce Project execution time. This includes initiating basic and detailed engineering work to finalize engineering designs, prepare work packages for procurement and the procurement of long lead time equipment.

The estimated budget to complete engineering designs and studies required for the MAPA and EMA permit submission is provided in Table 26-1.

Table 26-1: Estimated MAPA/EMA Permit Budget

Description	Cost (M$)
Engineering / Permit Management	1.8
Geotech / Water Quality Design	0.3
Tailings Management Facility Design	1.1
Infrastructure Design	0.5
Additional Metallurgical Testing	0.1
Avalanche / Terrain Stability Hazard Assessments	0.1
BC Hydro Studies	0.6
Geotechnical Investigations and Drilling	1.8
Third Party Review	0.3
Total	**6.6**

Source: IDM (2017)

The estimated budget to complete detailed engineering and procurements activities in advance of construction is provided in Table 26-2.

Table 26-2: Estimated Detailed Engineering and Procurement Budget

Description	Cost (M$)
Detailed Engineering	5.5
Procurement	0.5
Total	**6.0**

Source: JDS (2017)

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28 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
'	Minute (Plane Angle)
"	Second (Plane Angle) Or Inches
°	Degree
°C	Degrees Celsius
3D	Three-Dimensions
A	Ampere
a	Annum (Year)
ABA	Acid Base Accounting
ac	Acre
Acfm	Actual Cubic Feet Per Minute
ACS	ARSENEAU Consulting Services
ADR	Adsorption, Desorption And Refining
AISC	All in Sustaining Cost
ALT	Active Layer Thickness
amsl	Above Mean Sea Level
AN	Ammonium Nitrate
AP	Acid Potential
APEGBC	Association Of Professional Engineers And Geoscientists Of BC
ARD	Acid Rock Drainage
Au	Gold
B	Billion
BCEAA	British Columbia Environmental Assessment Act
BCEAO	BC Environmental Assessment Office
BD	Bulk Density
BMa	Bulk Mineral Analysis
Bt	Billion Tonnes
BTU	British Thermal Unit
BV/h	Bed Volumes Per Hour
bya	Billion Years Ago
C$	Dollar (Canadian)
Ca	Calcium
CAPEX	Capital Cost Estimate
CCA	Canadian Cost Allowance
CDE	Canadian Development Expense
CEAA	Canadian Environmental Assessment Act

Symbol/Abbreviation	Description
CEE	Canadian Exploration Expense
cfm	Cubic Feet Per Minute
CIL	Carbon in Leach
CIM	Canadian Institute Of Mining And Metallurgy
CIP	Carbon in Pulp
cm	Centimetre
cm^2	Square Centimetre
cm^3	Cubic Centimetre
COG	Cut-Off Grades
cP	Centipoise
Cr	Chromium
CRF	Cemented Rock Fill
CS	Arseneau Consulting Services
CSS	Closed Side Setting
Cu	Copper
d	Day
D&F	Drift And Fill
d/a	Days Per Year (Annum)
d/wk	Days Per Week
dB	Decibel
dBa	Decibel Adjusted
DGPS	Differential Global Positioning System
dmt	Dry Metric Ton
DO	Dissolved Oxygen
DWT	Dead Weight Tonnes
EA	Environmental Assessment
EAC	Environmental Assessment Certificate
EAO	Environmental Assessment Office
EIS	Environmental Impact Statement
ELC	Ecological Land Classification
EPCM	Engineering, Project & Construction Management
ERD	Explosives Regulatory Division
EWP	Engineering Work Package
FEL	Front-End Loader
FS	Feasibility Study
ft	Foot
ft^2	Square Foot
ft^3	Cubic Foot
ft^3/s	Cubic Feet Per Second

Symbol/Abbreviation	Description
FX	Exchange Rate
g	Gram
G&A	General And Administrative
g/cm^3	Grams Per Cubic Metre
g/L	Grams Per Litre
g/t	Grams Per Tonne
Ga	Billion Years
gal	Gallon (US)
GJ	Gigajoule
GPa	Gigapascal
gpm	Gallons Per Minute (US)
GSC	Geological Survey Of Canada
GW	Gigawatt
h	Hour
h/a	Hours Per Year
h/d	Hours Per Day
h/wk	Hours Per Week
ha	Hectare (10,000 M2)
ha	Hectare
HG	High Grade
HLEM	Horizontal Loop Electro-Magnetic
hp	Horsepower
HSE	Health, Safety & Environmental Plan
Hz	Hertz
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IDM	IDM Mining Ltd.
in	Inch
in^2	Square Inch
in^3	Cubic Inch
IRR	Internal Rate Of Return
JDS	JDS Energy & Mining Ltd.
K	Hydraulic Conductivity
k	Kilo (Thousand)
kg	Kilogram
kg/h	Kilograms Per Hour
kg/m^2	Kilograms Per Square Metre
kg/m^3	Kilograms Per Cubic Metre
km	Kilometre
km/h	Kilometres Per Hour

Symbol/Abbreviation	Description
km^2	Square Kilometre
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
kVA	Kilovolt-Ampere
kW	Kilowatt
kWh	Kilowatt Hour
kWh/a	Kilowatt Hours Per Year
kWh/t	Kilowatt Hours Per Tonne
L	Litre
L/min	Litres Per Minute
L/s	Litres Per Second
LDD	Large-Diameter Drill
LG	Low Grade
LHD	Load Haul Dump
LOM	Life Of Mine
m	Metre
M	Million
m/min	Metres Per Minute
m/s	Metres Per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metres per Day
m^3/h	Cubic Metres Per Hour
m^3/s	Cubic Metres Per Second
Ma	Million Years
MA/EMA	Mines Act and Environmental Management Act
MAAT	Mean Annual Air Temperature
MAE	Mean Annual Evaporation
MAGT	Mean Annual Ground Temperature
mamsl	Metres Above Mean Sea Level
MAP	Mean Annual Precipitation
MAPA	Mine Act Permit Application
masl	Metres Above Mean Sea Level
Mb/s	Megabytes Per Second
mbgs	Metres Below Ground Surface
Mbm^3	Million Bank Cubic Metres
Mbm^3/a	Million Bank Cubic Metres Per Annum
mbs	Metres Below Surface

Symbol/Abbreviation	Description
mbsl	Metres Below Sea Level
MELP	Ministry Of Environment, Lands And Parks
mg	Milligram
mg/L	Milligrams Per Litre
min	Minute (Time)
mL	Millilitre
ML/ARD	Metal Leaching/Acid Rock Drainage
mm	Millimetre
Mm^3	Million Cubic Metres
MMER	Metal Mining Effluent Regulations
mo	Month
MPa	Megapascal
MPC	Mobile Power Centre
MPEI	Mechanical, Piping, Electrical, and Instrumentation
MS	Microsoft
MSU	Mobile Substation Unit
Mt	Million Metric Tonnes
MVA	Megavolt-Ampere
MW	Megawatt
NaCN	Sodium Cyanide
NAD	North American Datum
NAMC	North American Metals Corporation
NFA	Nisg̱a'a Final Agreement
NG	Normal Grade
NLG	Nisga'a Lisims Government
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm^3/h	Normal Cubic Metres Per Hour
NNP	Net Neutralization Potential
NP / AP	Neutralizing Potential / Acidic Potential
NPV	Net Present Value
NQ	Drill Core Diameter Of 47.6 mm
NRC	Natural Resources Canada
NSR	Net Smelter Return
OP	Open Pit
OPEX	Operating Cost Estimate
OSA	Overall Slope Angles
oz	Troy Ounce
P.Geo.	Professional Geoscientist

Symbol/Abbreviation	Description
Pa	Pascal
PAG	Potentially Acid Generating
PAX	Potassium Amyl Xanthate
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFS	Preliminary Feasibility Study
PMF	Probable Maximum Flood
ppm	Parts Per Million
PRA	Process Research Associates
PSD	Particle Size Distribution
psi	Pounds Per Square Inch
QA/QC	Quality Assurance and Quality Control
QP	Qualified Person
RC	Reverse Circulation
RF	Rock fill
RMR	Rock Mass Rating
ROM	Run Of Mine
ROQ	Run of Quarry
RoW	Right of Way
rpm	Revolutions Per Minute
RQD	Rock Quality Designation
s	Second (Time)
SG	Specific Gravity
Scfm	Standard Cubic Feet Per Minute
SEDEX	Sedimentary Exhalative
SFD	Size Frequency Distribution
SG	Specific Gravity
SMP	Safety Management Plan
SQMP	Site Quality Management Plan
SRK	SRK Consulting Services Inc.
t	Tonne (1,000 Kg) (Metric Ton)
t/a	Tonnes Per Year
t/d	Tonnes Per Day
t/h	Tonnes Per Hour
TCR	Total Core Recovery
TIC / AP	Total Inorganic Compound / Acidic Potential
TMF	Tailings Management Facility
TOC	Total Organic Compound
ts/hm^3	Tonnes Seconds Per Hour Metre Cubed

Symbol/Abbreviation	Description
UCS	Unconfined Compressive Strength
UG	Underground
US	United States
US$	Dollar (American)
UTM	Universal Transverse Mercator
V	Volt
VEC	Valued Ecosystem Components
VFD	Variable Frequency Drives
VMS	Volcanic Massive Sulphide
VQS	Vendor Quality Surveillance
VSEC	Valued Socio-Economic Components
w/w	Weight/Weight
wk	Week
wmt	Wet Metric Ton
WOL	Whole Ore Leach
WRSF	Waste Rock Storage Facility
µm	Microns

APPENDIX A –

QP CERTIFICATES

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
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jdsmining.ca

CERTIFICATE OF AUTHOR

I, Gord Doerksen, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017, (the "Technical Report") prepared for IDM Mining Ltd.;

2. I am currently employed as VP- Engineering with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Mining Engineer (P.Eng. #32273) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Mining Engineer in Yukon Territory. I am a Member of the Canadian Institute of Mining and Metallurgy and a Registered Member of the Society of Mining Engineers of the AIME.

4. I am a graduate of Montana Tech with a B.Sc. in Mining Engineering (1990). I have been involved in mining since 1985 and have practiced my profession continuously since 1990. I have held senior mine production and mine technical positions in mining operations in Canada, the US and in Africa. I have worked as a consultant for over eight years and have performed mine planning, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person, for a significant number of engineering studies and technical reports many of which were located in Latin America.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I visited the Red Mountain project on May 28, 2014 and August 27, 2016;

7. I am responsible for Sections 1, 2, 3, 18 (except 18.4, 18.7, 18.8.1, 18.8.4), 19, 20, 21, 22, 23, 24, 25, 26, 27, 28 of this Technical Report;

8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

9. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for "NI 43-101 Preliminary Economic Assessment Technical Report for the Red Mountain Project, British Columbia, Canada" dated July 12, 2016;

10. As of the effective date of this Technical Report, to the best of 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; and

11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) "Gord Doerksen, P. Eng."

Gord Doerksen, P. Eng.

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CONSENT OF QUALIFIED PERSON

FILED BY SEDAR

I, Gord Doerksen, P. Eng., consent to the public filing by IDM Mining Ltd. of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017 and a report date of August 10, 2017 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the "News Release").

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017.

(Original signed and sealed) "Gord Doerkson, P. Eng"

Gord Doerksen, P. Eng.

JDS Energy & Mining Inc.
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Vancouver, BC V6C 2W2
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jdsmining.ca

CERTIFICATE OF AUTHOR

I, Michael Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017, (the "Technical Report") prepared for IDM Mining Ltd.;

2. I am currently employed as Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over ten years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I visited the Red Mountain project on March 25, 2016;

8. I am responsible for Sections 15, 16 (except 16.2) of this Technical Report;

9. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

10. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for "NI 43-101 Preliminary Economic Assessment Technical Report for the Red Mountain Project, British Columbia, Canada" dated July 12, 2016;

11. As of the effective date of this Technical Report, to the best of 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; and

12. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) "Michael Makarenko, P. Eng."

Michael Makarenko, P. Eng.

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CONSENT OF QUALIFIED PERSON

FILED BY SEDAR

I, Michael Makarenko, P. Eng., consent to the public filing by IDM Mining Ltd. of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017 and a report date of August 10, 2017 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the "News Release").

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017.

(Original signed and sealed) "Michael Makarenko, P. Eng."

Michael Makarenko, P. Eng.

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CERTIFICATE OF AUTHOR

I, Kelly McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada” with an effective date of June 26, 2017, (the “Technical Report”) prepared for IDM Mining Ltd.;

2. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 10 years consulting in the mining industry in metallurgy and process design engineering;

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I have not personally visited the Red Mountain project site;

6. I am responsible for Sections 13 & 17 of this Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for “NI 43-101 Preliminary Economic Assessment Technical Report for the Red Mountain Project, British Columbia, Canada” dated July 12, 2016;

9. As of the effective date of this Technical Report, to the best of 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; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) “Kelly McLeod, P. Eng.”

Kelly McLeod, P. Eng.

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CONSENT OF QUALIFIED PERSON

FILED BY SEDAR

I, Kelly McLeod, P. Eng., consent to the public filing by IDM Mining Ltd. of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017 and a report date of August 10, 2017 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the "News Release").

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017.

(Original signed and sealed) "Kelly McLeod, P. Eng."

Kelly McLeod, P. Eng.

CERTIFICATE OF AUTHOR

I, Dr. Gilles Arseneau, P. Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017, (the "Technical Report") prepared for IDM Mining Ltd.;

2. I am currently the President of ARSENEAU Consulting Services Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. 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 have practiced my profession continuously since 1995. I have worked in exploration in North and South America and have extensive experience modelling gold mineralization similar to the Red Mountain deposit.

   I am Professional Geoscientist registered as a member, in good standing, with the Association of Professional Engineers & Geoscientists of British Columbia (no. 23474).

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I visited the Red Mountain project on March 25, 2016;

5. I am responsible for Sections 14 of this Technical Report;

6. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for "NI 43-101 Preliminary Economic Assessment Technical Report for the Red Mountain Project, British Columbia, Canada" dated July 12, 2016 and the principal author of a mineral resource update dated March 1, 2017;

7. As of the effective date of this Technical Report, to the best of 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; and

8. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) "Dr. Gilles Arseneau, P. Geo."

Dr. Gilles Arseneau, P. Geo.

FILED BY SEDAR

I, Dr. Gilles Arseneau, P. Geo., consent to the public filing by IDM Mining Ltd. of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017 and a report date of August 10, 2017 (the "Technical Report"). I also consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the "News Release").

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017.

"Original signed by "

____

Dr. Gilles Arseneau, P. Geo.

Certificate of Qualified Person

I, Andrew P. Hamilton, P.Geo., do hereby certify that:

1. That I am a consulting geologist with a street address of 1339 East 18th Street, North Vancouver, B.C., V7J 1M2;

2. This certificate applies to the Technical Report entitled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017, (the "Technical Report") prepared for IDM Mining Ltd.;

3. I am a Registered Professional Geoscientist (P.Geo. #24873) registered with the Association of Professional Engineers, Geoscientists of British Columbia;

4. I am a graduate of the University of British Columbia with a B.Sc. degree in Geology in 1991. I have practiced my profession continuously since graduation;

5. I have held technical positions with exploration and development companies in Canada and Central America during which I was responsible for program design, data collection and management, QAQC and resource modelling. As a consultant I have conducted data and QAQC audits for projects in late stage development and operating mine environments;

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI-43-101. I am independent of the Issuer and related companies applying all tests in Section 1.5 of the NI 43-101;

7. I visited the Red Mountain project site between July 20 to 24, 2016, May 8 to 11, 2017 and was on a work rotation basis from April to October 2000;

8. I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11 and 12 of the Technical Report;

9. I have prior involvement of the property that is the subject of this Technical Report. I worked at the Red Mountain project site on a work rotation basis in 2000 for the purpose of determining geological interpretation and resource estimation criteria. I directly participated in this work and supervised staff performing in these determinations. During this time, I was employed as a Project Geologist with North American Metals Corp., a previous owner of the Red Mountain Project. I am co-author of the technical report titled "Mineral Resource Update for the Red Mountain Gold Project, Northwestern BC, Canada" dated February 24, 2017 with an Effective Date of January 20, 2017 (the "Technical Report") prepared for IDM Mining Ltd.

10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to make the technical report not misleading;

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI43-101 and Form NI 43-101F1.

Dated this 10th day of August, 2017 in Vancouver, British Columbia.

*[Original "Signed and sealed"]*_________
Andrew P. Hamilton, P.Geo.

Consent of Qualified Person

To:
British Columbia Securities Commissions
Alberta Securities Commissions
Ontario Securities Commissions
Nova Scotia Securities Commissions
TSX Venture Exchange

August 10, 2017

I, Andrew Hamilton, do hereby consent to the public filing of the technical report entitled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" dated August 10, 2017 with and effective date of June 26, 2017 ("the technical report") and to extracts from the technical report filed in the news release filed by IDM Mining Ltd. on June 26, 2017 ("the new release").

I confirm that I have read the disclosure document and that it fairly and accurately represents the information in the technical report being filed.

[Original "Signed and Sealed"]

Andrew Hamilton, P. Geo.

CERTIFICATE OF AUTHOR

I, Ken Embree, P.Eng., do hereby certify that:

1. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project, British Columbia, Canada", with an effective date of June 26, 2017, (the "Technical Report") prepared for IDM Mining Ltd. ;
2. I am employed as Managing Principal of Knight Piésold Ltd. with an office at 1400-750 West Pender Street, Vancouver, BC, V6C 2T8;
3. I am a Professional Engineer in good standing with the Association of Professional Engineers and Geoscientists of British Columbia in the area of geological engineering (Registration No. 17439). I am also registered as a Professional Engineer in Ontario (#100040332).
4. I am a graduate of the University of Saskatchewan with a B.Sc. in Geological Engineering (1986). I have practiced my profession continuously since 1986. My experience includes tailings, waste and water management for mine developments in Canada, the US and South America.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Red Mountain Project site on July 12 and 13, 2016.
7. I am responsible for Section number 18.4, 18.7 and 18.8.4 of the Technical Report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have not had prior involvement with the property that is the subject of the Technical Report;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
11. As of the effective date of the Technical Report and the date of this certificate, to the best of 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;

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) "Ken Embree, P.Eng."

Ken Embree, P.Eng.

CONSENT OF QUALIFIED PERSON

FILED BY SEDAR

I, Ken Embree, P.Eng., consent to the public filing by IDM Mining Ltd. of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project, British Columbia, Canada" with an effective date of June 26, 2017 and a report date of August 10, 2017 (the "Technical Report"). I consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the "News Release").

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017

(original signed and sealed) "Ken Embree, P.Eng."

Ken Embree, P.Eng.

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F: +1.604.687.5532
vancouver@srk.com
www.srk.com

CERTIFICATE OF AUTHOR

I, Kelly Sexsmith, P. Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017, (the "Technical Report") prepared for IDM Mining Ltd.;

2. I am currently employed as Principal Consultant with SRK Consulting (Canada) Inc. with an office at Suite 2200 – 1066 West Hastings Street, Vancouver, British Columbia, V6E 3X2;

3. I am a graduate of the University of British Columbia with a B.Sc. in Geology, 1989, and a graduate of the Colorado School of Mines with an M.S. in Environmental Sciences, 1996. I have practiced my profession continuously since 1990;

4. I have been an consultant for over 25 years providing technical expertise in the characterization and prediction of acid rock drainage and metal leaching for various new, developed, and closed mining properties, management of environmental studies and permitting for new mine developments, and third party review and audits of geochemistry, and water management activities associated with mining projects.

5. I am a Registered Professional Geologist in British Columbia (#21397) and Saskatchewan (#13977);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I visited the Red Mountain project in August 2000 and August 2003 in support of previous work at this project site;

8. I am responsible for Sections 18.8.1 of this Technical Report;

9. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

10. I have had prior involvement with the property that is the subject of this Technical Report. I was a QP for “Red Mountain Project Engineering Study” prepared for Seabridge Gold Inc. dated August 2003, for “Preliminary Economic Assessment Red Mountain Gold Project, British Columbia prepared for Seabridge Gold Inc. dated January 31, 2008. Since 2000, I have also contributed to ongoing monitoring, baseline studies, and environmental assessment work at the site – all in the areas of waste and water management;

11. As of the effective date of this Technical Report, to the best of 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; and

12. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) “Kelly Sexsmith, P. Geo.”

Kelly Sexsmith, P. Geo.

SRK Consulting (Canada) Inc.
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www.srk.com

Project number: 2CI017.001

August 10, 2017

FILED BY SEDAR

CONSENT OF QUALIFIED PERSON

I, Kelly Sexsmith, P. Geo., consent to the public filing by IDM Mining Ltd. of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada" with an effective date of June 26, 2017 and a report date of August 10, 2017 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the "News Release").

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017.

(Original signed and sealed) "Kelly Sexsmith, P. Geo."

Kelly Sexsmith, P. Geo.

SRK Consulting (Canada) Inc.
2200–1066 West Hastings Street
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CERTIFICATE OF QUALIFIED PERSON

I, Bruce Murphy P.Eng., residing in North Vancouver, British Columbia do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200, 1066 West Hastings Street, Vancouver, British Columbia, Canada.
2) I am a graduate of University of the Witwatersrand, Johannesburg, South Africa with a M.Sc. degree in Mining Engineering. I have practiced my profession continuously since graduation (1989) working in the rock engineering field on operating mines till 2002 and then in the consulting field.
3) I am a registered Professional Engineer in British Columbia (APEGBC 44271), and Manitoba (APEGM 41987).
4) I have personally visited Red Mountain project on 27 September 2016.
5) As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
6) I have read National Instrument 43-101 and the definition of Qualified Person set out in the Instrument and certify that by virtue of my education, affiliation to a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
7) I am responsible for report Section 16.2.
8) SRK followed CIM “Best practices” and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding Technical Report is based on a site visit, a review of project files and discussions with IDM personnel.
9) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the subject property or securities of IDM Mining (IDM).
10) That, as of the date of this certificate, to the best of 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.

Effective Date: June 26, 2017
Signing Date: August 10, 2017

(Original signed and sealed) “Bruce Murphy P. Eng.”

Bruce Murphy, P.Eng

U.S. Offices:
Anchorage 907.677.3520
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vancouver@srk.com
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CONSENT OF QUALIFIED PERSON

FILED BY SEDAR

I, Bruce Murphy, P.Eng., consent to the public filing by IDM Mining Ltd. of the technical report titled “NI 43-101 Feasibility Study Technical Report for the Red Mountain Project British Columbia, Canada” with an effective date of June 26, 2017 and a report date of August 10, 2017 (the “Technical Report”).

I also consent to any extracts from, or a summary of, the Technical Report in the June 26, 2017 news release of IDM Mining Ltd. (the “News Release”).

I certify that I have read the News Release filed by IDM Mining Ltd. and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

DATED this 10th day of August, 2017.

(Original signed and sealed) “Bruce Murphy P. Eng.”

Bruce Murphy, P.Eng
Principal Consultant