PRELIMINARY ECONOMIC ASSESSMENT
NI 43-101 TECHNICAL REPORT
RED MOUNTAIN GOLD PROJECT
NORTHWESTERN BC, CANADA

55° 57’ North & 129° 42’ West

Report Date: September 3, 2014

Effective Date: July 23, 2014

Prepared for:

IDM Mining Ltd.

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This report, entitled *Preliminary Economic Assessment NI 43-101 Technical Report on the Red Mountain Gold Project, Northwestern BC, Canada* (effective as of July 23, 2014), was prepared and signed by the following authors:

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NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for IDM Mining Ltd. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions and qualifications set forth in this report.

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1.0 EXECUTIVE SUMMARY

1.1 INTRODUCTION

IDM Mining Ltd. (IDM or IDM Mining) mandated JDS Energy & Mining Ltd. (JDS) to complete a Preliminary Economic Assessment (PEA) of the Red Mountain gold project located in northwestern B.C., 18 km east of the town of Stewart. The purpose of this study is to complete a review and compilation of the resources, mining designs and preliminary economics using parameters updated to 2014 costs and IDM Mining corporate objectives.

On April 12, 2014, IDM optioned the property from Seabridge Gold Inc. (Seabridge) with the intent of initiating a PEA study and conducting further exploration work. Since acquiring the project, IDM has completed a comprehensive review and validation of the Red Mountain geological and environmental data, and JDS has carried out engineering studies in connection with the resource model, mine design, mineralised material processing, road access, environmental impacts and cost estimation.

It must be noted that this PEA is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the results of the preliminary economic assessment will be realised.

1.2 PROPERTY DESCRIPTION & OWNERSHIP

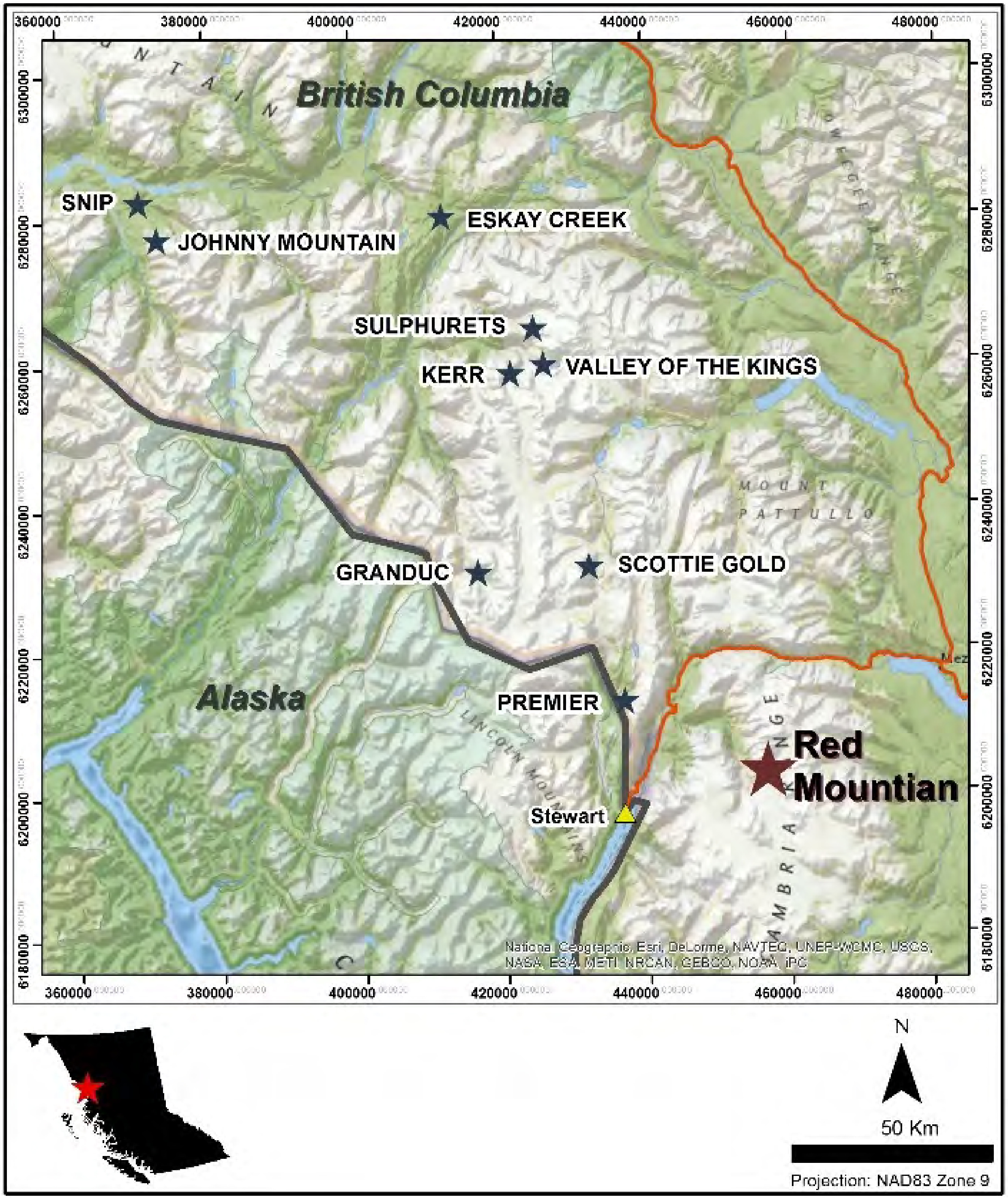
1.2.1 Property Description

The 17,125 hectare (ha) Red Mountain Gold Property 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 1,500 and 2,000 m. On NTS map sheets 103P/13 and 104A/4, the property is centred on 55º59'4"N, 129º45'37"W. The UTM coordinates are 452,450 E, 6,250,325 N in Zone 9 (NAD 83).

The area is characterised 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. An existing road extends for approximately 13 km along Bitter Creek Valley but stops approximately 7 km from the proposed mine site.

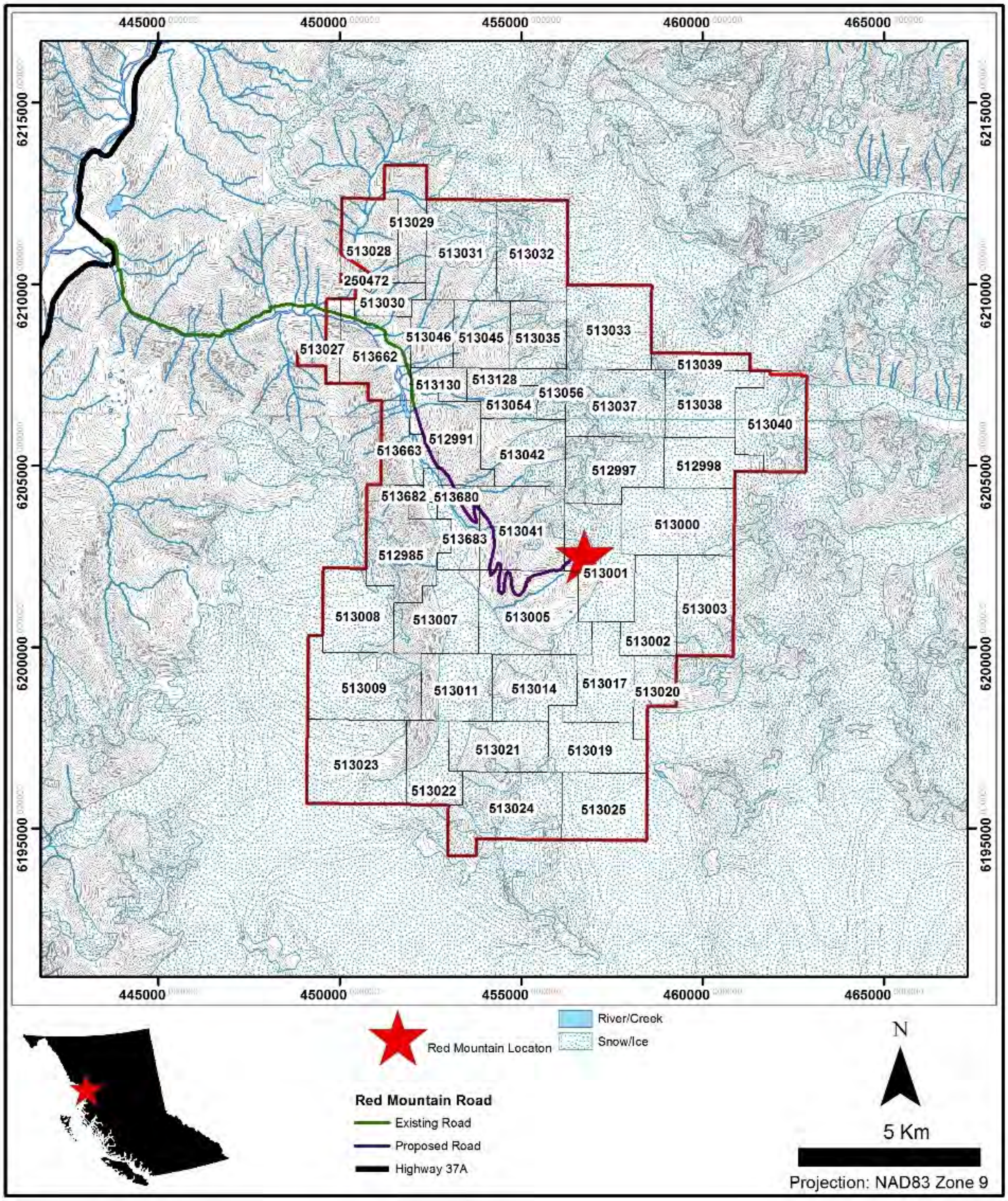
The Red Mountain Property consists of 47 contiguous mineral claims totalling 17,125 ha (Figure 1-2). No significant risks are identified which would affect access, title, or the right or ability to perform work on the property.

Figure 1-1: Red Mountain Location Map



Source: IDM (2014)

Figure 1-2: Red Mountain Project Claim Map



Source: IDM (2014)

The property falls within the Nass Wildlife Area as set out in the Nisga'a Final Agreement (NFA). Pursuant to the NFA, the Nisga'a Nation has rights to the management and harvesting of fish and wildlife within the Nass Wildlife Area.

1.2.2 Ownership

On April 15, 2014, IDM entered into an option agreement for the Red Mountain project with Seabridge through IDM's predecessor company Revolution Resources Corporation. Subsequent to the option agreement, Revolution Resources Corporation underwent a restructuring and name change to IDM Mining Ltd.

Claim title is currently under Seabridge. Upon satisfaction of the option terms, title will be transferred to IDM. Seabridge owns 100% of the property claims subject to two royalties. Barrick Gold Corporation (Barrick) 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.

Under the terms of the Option Agreement, IDM can earn a 100% interest in the Red Mountain project, subject to certain underlying royalties, by issuing to Seabridge 29,733,000 shares of Revolution, paying to Seabridge $2 million cash in staged payments ($1 million payable within 90 days, $1 million within 1.5 years), and incurring $7.5 million in exploration and development expenditures over three years ($2.5 million per year). IDM has the right to extend the deadline for expenditure of the final $2.5 million by one year upon payment to Seabridge of $250,000.

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 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 & MINERALISATION

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.

Mineralised 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 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 mineralisation 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 1.1 provides a chronological summary of recent exploration efforts on Red Mountain.

Table 1.1: Red Mountain 1988-2014 Chronological Exploration Summary

1988-89	Staking of Red Mountain by Wotan Resources Inc.
1989	Red Mountain and Wotan properties optioned to Bond. Discovery of gold-silver mineralisation 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 project acquired by Barrick following Barrick's take-over of Lac Minerals. 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.

Source: JDS (2014)

1.5 MINERAL PROCESSING & METALLURGICAL TESTING

There has been a significant amount of metallurgical testing conducted on samples from Red Mountain project. Three basic process options were explored to extract gold and silver: production of gold- and silver-bearing flotation concentrates for sale to a smelter; direct whole mineralised material cyanidation for doré production; and a hybrid flotation process with cyanide leaching of the flotation concentrate to produce doré.

Under the direction of Lac Minerals, metallurgical testing began in 1991 at Lakefield Research. A significant body of testing at Brenda Process Technology in 1994 followed this preliminary campaign. Whole cyanidation was the primary process tested during these programs.

Metallurgical testing resumed in 2001 under the direction of Dr. M. Beattie of Beattie Consulting. Production of precious metal bearing concentrates was nearly exclusively investigated in this program, conducted at PRA laboratories.

As Red Mountain is located within 35 km of the deep-sea port of Stewart that is currently shipping concentrate from two mines, extensive flotation test work was completed in 2000. Three alternatives for processing mineralised material at Red Mountain were investigated by JDS. Flotation test work indicated that ≈ 90% of the Au was recoverable in a pyrite concentrate with a concentration ratio of between 3.5:1 to 5.5:1 of the initial feed. This alternative is currently considered non-economic.

The second alternative, involved flotation followed by regrinding of the concentrate and subsequent cyanidation. Due to recovery loss and subsequent negative economic impacts, the process was not chosen for this study.

The third alternative of direct grinding to a -400 mesh grind and cyanidation would yield recoveries of 87%-90% according to historical work done by previous operators. This option was used in this PEA study.

1.6 MINERAL RESOURCE ESTIMATE

Numerous resource estimates were completed from 1989 to present. During 2000, NAMC conducted a detailed review of all data, relogged all core within a 20 m envelope of the Marc, AV and JW mineralised 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 and is the basis for the resource estimate in Table 1.2.

Table 1.2: Mineral Resource Statement at a 3 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	651,600	9.26	40.06	194,000	839,200
Indicated	10,800	9.71	30.33	3,400	10,500
Inferred	0	0.00	0.00	0	0
AV Zone					
Measured	508,200	7.14	20.88	116,700	341,200
Indicated	283,800	7.32	21.03	66,800	191,900
Inferred	1,800	10.96	39.50	600	2,300
JW Zone					
Measured					
Indicated					
Inferred	331,100	7.67	12.57	81,600	133,900
Total Measured & Indicated	**1,454,300**	**8.15**	**29.57**	**380,900**	**1,382,800**
Total Inferred	**332,900**	**7.69**	**12.72**	**82,300**	**136,200**

Source: JDS (2014). *3 g/t Au is calculated as the cut-off grade for underground longhole stoping.

Since 2000, Banks drilled two additional holes in the Marc Zone for which the QA/QC procedures were not available. The Banks holes were entered into the database and their resource effect was estimated. Although they confirmed the 2000 resource estimate, the net change to the 2000 resource was only a 0.02 g/t Au decrease in the Marc Zone average grade. As this is well within the rounding error and well below the resource estimation accuracy, and because no QA/QC data were available, the Banks holes were not included in the current resource estimate.

1.7 MINERAL RESERVE ESTIMATE

This Preliminary Economic Assessment does not support an estimate of Mineral Reserves, since a prefeasibility or feasibility study is required for reporting of Mineral Reserve estimates. This report is based on potentially mineable material (“mineable tonnes”).

Mineable tonnages were derived from the resource model described in the previous section. Measured, indicated and inferred resources were used to establish mineable tonnes.

Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

1.8 MINING METHODS

Two underground mining methods were selected based on deposit body geometry and grade of the mineralised zones:

- Longhole stoping (LH) for mining blocks steeper than 55°, which represents about 82% of mineable tonnage. This is the preferred mining method from productivity and operating cost perspective.
- Drift and Fill (D&F) for mining blocks with dips of less than 55°, which represents about 18% of mineable tonnage.

Cemented and uncemented rock fill will be used as backfill to maximise mining recovery.

The initial mine design was based on basic assumptions to generate lower limits for cut-off grades (COG) for the two planned mining methods. A value of 3 g/t Au was determined as the COG for longhole stoping and 5 g/t Au for drift and fill mining. These COG's were used to design initial mining shapes.

Mining recovery and dilution factors were applied to each mining shape based on the mining method used. The estimated mineable tonnes for the Red Mountain project are summarised in Table 1.3.

The PEA mine plan focusses on accessing and mining higher grade material early in the mine life. As such, the plan commences with mining of Marc, followed by AV, and then JW. The mine production rate is targeted at 1,000 t/d. Production in the last year of mining was slightly increased to 1,085 t/d.

Underground access will be through two portals: the existing exploration decline at 1,860 m EL and a new portal at 1,650 m EL. Access ramps will be driven at maximum grade of 15% at a 4.5 by 4.5 m profile to accommodate 20 tonne haul trucks. Level spacing is variable up to a maximum of 30 m. Mineralised zone development will be on a 4.0 x 4.0 m profile.

Mine development required over the life of the mine is summarised in Table 1.4.

Table 1.3: Mine Production Schedule

Zone	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Marc							
Tonnage	tonnes	270,000	190,000	29,000	-	27,000	**516,000**
Gold Grade	Au g/t	10.35	6.88	7.82	-	4.47	**8.62**
Silver Grade	Ag g/t	42.94	27.48	27.32	-	33.38	**35.88**
AV							
Tonnage	tonnes	-	83,000	242,000	201,000	187,000	**713,000**
Gold Grade	Au g/t	-	7.16	6.38	5.80	5.54	**6.09**
Silver Grade	Ag g/t	-	21.37	16.41	19.96	19.62	**18.83**
JW							
Tonnage	tonnes	-	-	-	70,000	79,000	**149,000**
Gold Grade	Au g/t	-	-	-	7.48	8.57	**8.06**
Silver Grade	Ag g/t	-	-	-	14.10	9.61	**11.71**
Total Mine							
Tonnage	**tonnes**	**270,000**	**272,000**	**271,000**	**271,000**	**293,000**	**1,378,000**
Gold Grade	**Au g/t**	**10.35**	**6.97**	**6.53**	**6.24**	**6.26**	**7.25**
Silver Grade	**Ag g/t**	**42.94**	**25.63**	**17.57**	**18.45**	**18.18**	**24.44**

Source: JDS (2014)

Table 1.4: Mine Development Schedule

Type	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Capex - Lateral	m	1,420	1,640	860	-	-	-	**3,920**
Capex - Vertical	m	340	2,060	1,020	1,040	160	470	**5,080**
Opex - Lateral	m	210	270	290	-	-	-	**770**
Waste Tonnage	tonnes	154,000	198,000	101,000	48,000	7,000	22,000	**531,000**

Source: JDS (2014)

1.9 RECOVERY METHODS

Gold and silver will be extracted by cyanidation from run-of-mine mineralised material delivered to the mill complex. The run of mine material will be stage crushed by a jaw and cone crusher and stored in a fine mill feed material bin. At a nominal rate of 1,000 t/d, the fine mineralised material bin will feed a conventional rod and ball mill grinding circuit followed by thickening prior to leaching. Target grind size will be 95% passing 38 microns (P_{95} 38 µm).

Gold and silver extraction will be accomplished in leach tanks with carbon in pulp adsorption, carbon elution, regeneration, electrowinning and refining. Tailings will be treated with SO_2 and air to destroy cyanide prior to discharge to the tailing management facility (TMF). Metallurgical recoveries are estimated to average 87% for gold and 80% for silver.

1.10 PROJECT INFRASTRUCTURE

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

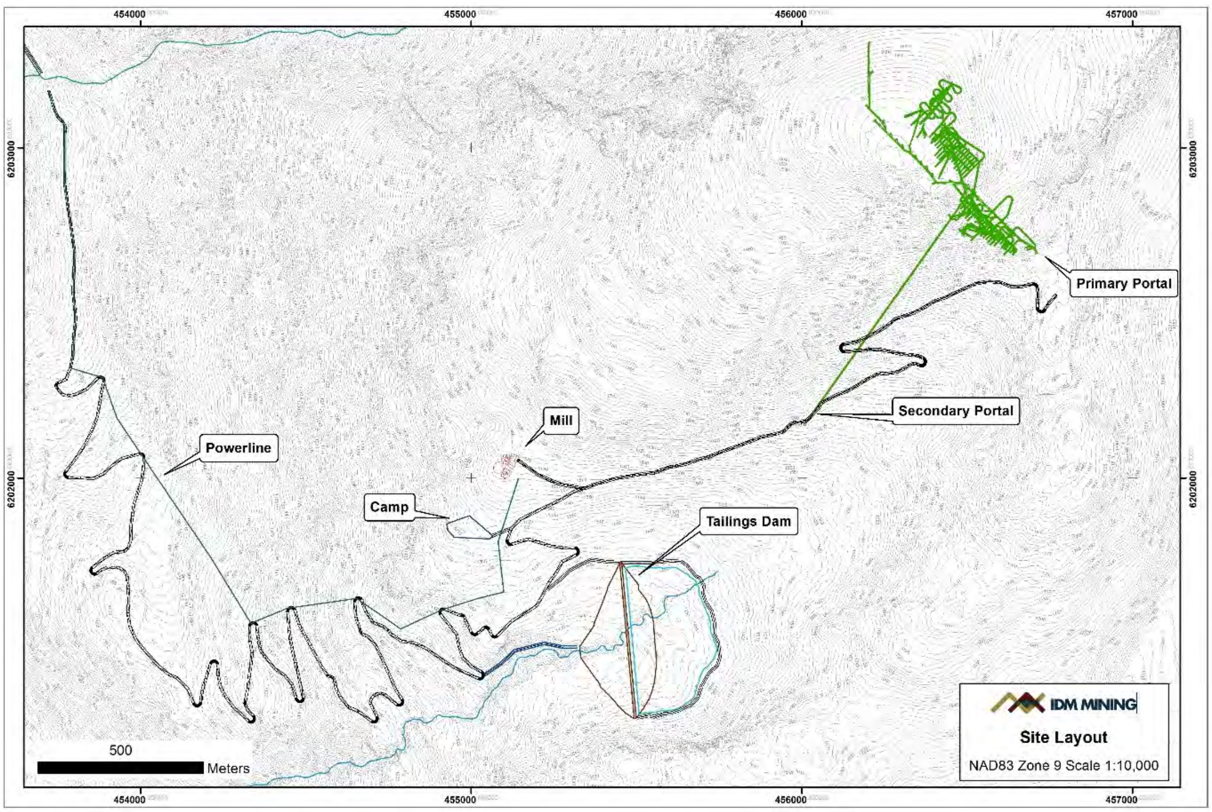
- Approximately 25 km seasonal access road from the Glacier highway to the project site.
- Approximately 5 km of on-site service roads to access the mine portals, tailings management facility and other working areas.
- Crushing and grinding circuits and gold extraction plant.
- Tailings management facility and impoundment.
- Temporary development waste storage areas (note that waste rock generated by development and mining is rehandled into the underground workings as backfill).
- Administration office, mine dry, maintenance shop, warehouse and emergency camp.
- Electrical connection to BC Hydro, transmission line adjacent to the seasonal access road and on-site substation and distribution network.
- Process and fire water storage and distribution.
- Sewage septic system.

These key items would be constructed during a two-year pre-production period. The seasonal access road and right-of-way for the electrical power transmission line constructed in Year -2 and the remaining items constructed in Year -1.

1.11 ENVIRONMENT & PERMITTING

IDM is committed to operating the mine in a sustainable manner and every reasonable effort will be made to minimise any short and long-term environmental impacts and to ensure that the project provides lasting benefits to local communities while generating substantial economic and social advantages for shareholders, employees, and the broader community. IDM respects the traditional knowledge of the Aboriginal peoples who have historically occupied or used the Red Mountain project area. The project area watershed is relatively undisturbed by human activities with the exception of an access road that was constructed in the late 1990's but is currently decommissioned.

Figure 1-3: Red Mountain Project Site Layout Map



Source: IDM (2014)

The objective is to retain the current watershed and local ecosystem integrity as much as possible during the construction and operation of the project. Upon closure and reclamation of the project, the goal will be to return the relatively small-disturbed areas to a level 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 will require a provincial environmental assessment under the British Columbia *Environmental Assessment Act* (BC EAA).

Restoration activities are planned to consist of placing a geosynthetic liner system and 1 m thickness of granulated cover over the TMF to minimise infiltration. Covers will be graded to create natural drainage to reduce erosion. All underground development rock would be placed as backfill in the mining process. Infrastructure would be removed and disturbed sites regraded to natural slopes. The access roads would be deactivated in accordance with the Forest Practice Code. It is planned to hydrostatically seal the lower underground portal with an engineered bulkhead.

A 15-year annual monitoring program has been included in the cost estimation.

1.12 CAPITAL & OPERATING

1.12.1 Capital Cost Estimate

The capital cost (CAPEX) estimate includes all costs required to develop, sustain, and close the operation for a planned 5-year operating life. The construction schedule is based on an approximate 24-month build period. The accuracy of this estimate is ±35%.

The high-level CAPEX estimate is shown in Table 1.5. The sustaining capital is carried over operating Years 1 through 5, and closure costs are projected over Year 6.

Table 1.5: Summary of Capital Cost Estimates

Capital Cost	Pre-Production (C$M)	Sustaining/ Closure (C$M)	Total (C$M)
Crushing & Milling	23.8	0.0	23.8
Tailings Pond	3.7	11.6	15.3
Power	10.2	0.0	10.2
Mine Development	10.5	4.8	15.3
Infrastructure	2.5	0.6	3.1
Surface Equipment	1.1	0.0	1.1
Site Access Roads	5.9	0.0	5.9
Owner, Indirects, EPCM	8.6	0.0	8.6
Closure (Net of Salvage Value)	0.0	1.4	1.4
Subtotal Pre-Contingency	**66.2**	**18.4**	**84.7**
Contingency	9.9	2.8	12.7
Total Capital Incl. Contingency	**76.1**	**21.2**	**97.4**

Source: JDS (2014)

1.12.2 Operating Cost Estimate

Operating cost (OPEX) estimates for the Red Mountain project have been prepared incorporating both off-site and on-site infrastructure as related to the mine plan and processing schedule. Table 1.6 summarises the life-of-mine (LOM) OPEX estimate.

Table 1.6: Summary of Operating Cost Estimates

Operating Cost	C$/t processed	LOM C$M
Mining	66.54	91.7
Milling	27.67	38.1
G&A	10.91	15.0
Total	**105.13**	**144.9**

Source: JDS (2014)

The operating cost estimate is based on a seasonal nine-month mining operation.

1.13 ECONOMIC ANALYSIS

An 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 to approximate the true investment value. It must be noted that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations to represent an indicative value of the after-tax cash flows of the project.

The results of the economic analysis are shown in Table 1.7.

Sensitivities to metal prices, head grade, OPEX, and CAPEX were conducted by adjusting each variable up and down 15% independently of each other. As with most metal mining projects, the project is most sensitive to metal price and head grade. The project is slightly more sensitive to OPEX than CAPEX. The base case sensitivities are shown in Figure 1-4. Sensitivity based on various metal prices is illustrated in Table 1.8.

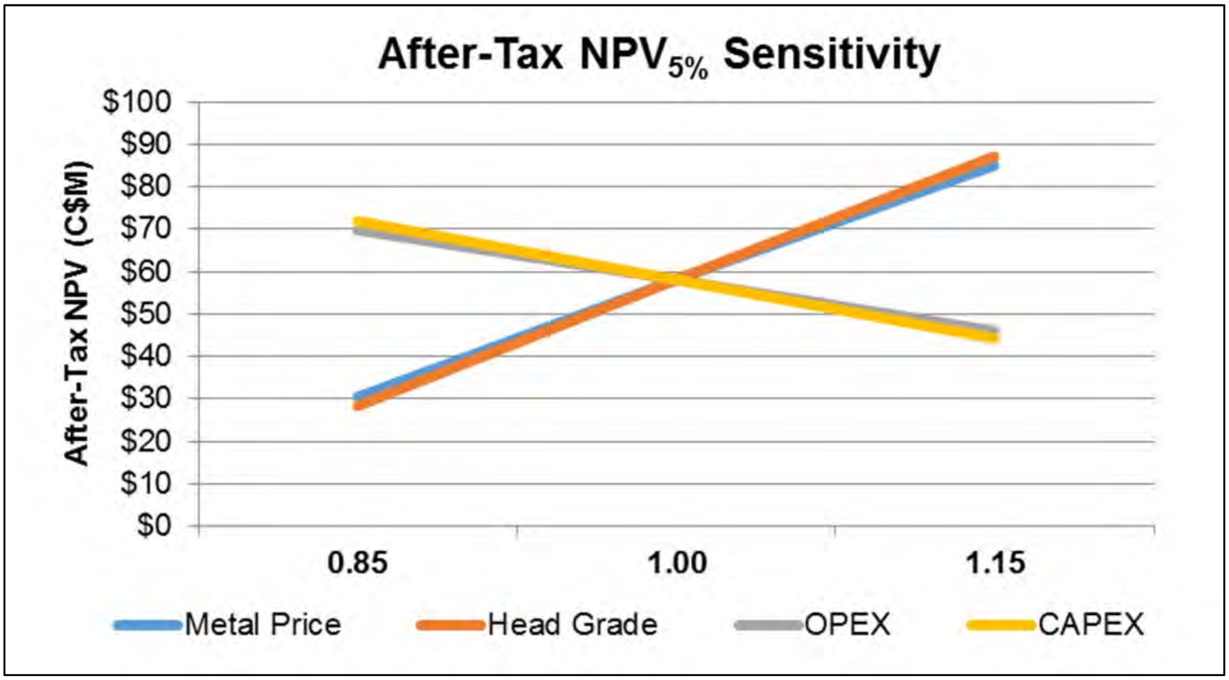
This preliminary economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

Table 1.7: Summary of Economic Assumptions & Results

Summary of Results	**Unit**	**Value**
Au Price	US$/oz	1,250
Ag Price	US$/oz	20.00
F/X Rate	USD:CAD	0.95
Mine Life	Years	5.0
Mill Feed	Mt	1.4
Throughput Rate	t/d	1,022
Average Au Head Grade	g/t	7.25
Average Ag Head Grade	g/t	24.44
Au Payable	koz	277.0
	koz/a	55.5
Ag Payable	koz	852.0
	koz/a	170.6
NSR (Net of Royalties)	LOM C$M	361.6
	$/t mined	262.37
Operating Costs	LOM C$M	144.9
	$/t mined	105.13
Au Cash Cost	US$/oz	516.23
Au Cash Cost (Net of Byproduct)	US$/oz	454.73
Pre-production Capital	C$M	66.2
Pre-production Contingency	C$M	9.9
Total Pre-production Capital	**C$M**	**76.1**
	$/t mined	**55.24**
Sustaining & Closure Capital	C$M	18.4
Sustaining & Closure Contingency	C$M	2.8
Total Sustaining & Closure Capital	**C$M**	**21.2**
	$/t mined	**15.4**
Total Capital Costs Incl. Contingency	C$M	97.4
	$/t mined	70.64
Working Capital	C$M	3.4
Pre-Tax Cash Flow	LOM C$M	119.4
	C$M/a	23.9
Taxes	LOM C$M	40.1
After-Tax Cash Flow	LOM C$M	79.2
	C$M/a	$15.9
Economic Results		
Pre-Tax Net Present Value ($NPV_{5\%}$)	**C$M**	**90.1**
Pre-Tax Internal Rate of Return (IRR)	**%**	**43.3**
Pre-Tax Payback	**Years**	**1.3**
After-Tax $NPV_{5\%}$	**C$M**	**57.6**
After-Tax IRR	**%**	**32.9**
After-Tax Payback	**Years**	**1.5**

Source: JDS (2014)

Figure 1-4: After-Tax NPV Sensitivity Graph



Source: JDS (2014)

Table 1.8: Metal Price Sensitivity Analysis

Au Price US$/oz	Ag Price US$/oz	Pre-Tax $NPV_{5\%}$ (C$M)	Pre-Tax IRR	Pre-Tax Payback	After-Tax $NPV_{5\%}$ (C$M)	After-Tax IRR	After-Tax Payback
$1,150	$18.40	$67.8	34.8%	1.7	$43.1	26.2%	2.0
$1,250	$20.00	$90.1	43.3%	1.3	$57.6	32.9%	1.5
$1,350	$21.60	$112.4	51.4%	1.1	$72.0	39.0%	1.2
$1,450	$23.20	$134.7	59.2%	0.9	$86.4	45.0%	1.0

Source: JDS (2014). Based on exchange rate of USD:CAD = 0.95.

1.14 PROJECT DEVELOPMENT

Key infrastructure items would be constructed during the two-year pre-production period. The seasonal access road and right-of-way for the electrical power transmission line would be constructed in Year -2 and the remaining items constructed in Year -1.

During Year -1, underground mine development will commence as soon as sufficient site infrastructure has been established. Initially, access to the workings will be from the tote road to the upper portal to develop access to the Year 1 production stopes at Marc. The lower portal will also be established during Year -1 to drive the incline, develop the main ventilation raise and central deposit pass.

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

1.15 CONCLUSIONS & RECOMMENDATIONS

1.15.1 Resource

A high degree of drilling and quality control work has been performed on the project by previous operators. Relogging the core to create a geological model has created confidence in the understanding of mineralised zone controls.

The Marc Zone main portion of the mineralised deposit requires no further test work.

The AV Zone is drilled at nearly a 25 x 25 m grid spacing and shows good geological and grade continuity yielding a large portion of the deposit in the measured category. It does require infill drilling for final mine stope planning on Section 1425. Both geology and gold assays align themselves well on sections either side of 1425, indicating there is a high probability that infill drilling will mimic flanking sections.

The JW zone is currently classified as inferred due to a lack of drill density. The existing drill holes display good geological continuity. The stockwork mechanism is consistent in virtually every hole and matches well with the reconstruction of the other two zones. Further infill drilling to provide assay information for mine planning and an upgrade in resource classification is recommended.

The AV Tails and 141 Zone have not been seriously evaluated for potential. Both, having scant drill data and thin horizons, require a re-examination for mineralogical controls.

1.15.2 Metallurgy

Further metallurgical test work would be beneficial to design and optimise mill feed processes including but not limited to deposit recovery variability studies, kinetic rate studies, carbon loading, pulp viscosity, comminution and thickener studies.

Samples from the deposit should be subjected to mill feed material sorting testing to determine the amenability of the mineralisation to mill feed sorting. Mill feed material sorting can provide a significant reduction to operating costs by supplying fill for underground, effectively increasing mill tonnage and reducing the generated tailings per tonne mined.

1.15.3 Infrastructure & Tailings Management Facility

There are engineering options for grid power supply that require further investigation. There is potential to reduce grid power initial capital costs.

The Tailings Management Facility may have optimisation potential through further design engineering in conjunction with detailed scheduling and tailings dam volume requirements.

1.15.4 Risks

It is the conclusion of the QPs that the PEA summarised in this technical report contains adequate detail and information to support the potentially positive economic result. The PEA proposes the use of industry standard equipment and operating practices. 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 dilution, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

1.15.5 Opportunities

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.

Pre-sorting mineralised material has had success in other mines and greatly enhanced both mining and processing efficiencies. If future test work on the Red Mountain deposit indicates pre-sorting viability, the benefits have the potential to be substantial.

1.15.6 Recommendations

Further work is recommended in two phases: completion of an exploration program, followed by a prefeasibility study (PFS). Prior to initiating a prefeasibility study, current inferred mineralised material that is considered potentially economic in the PEA study should be drilled to an indicated level for inclusion into future resource estimations in preparation for a PFS level study. Paralleling long lead-time test work and engineering with exploration resource definition drilling is recommended. Cost estimates for the recommended phases of work are included below in Table 1.9 and Table 1.10.

Table 1.9: Phase 1 Exploration & Pre-PFS Engineering Cost Estimate

Item	Cost
Underground Resource Definition Drilling	
Assay	$40,000
Labour	$280,000
Underground	$530,000
Drilling	$864,000
Camp	$456,000
Helicopter	$330,000
Subtotal	$2,500,000
Pre-PFS Engineering	
Pre-sorting Mineralised Material Test Work & Bench Test Work	$50,000
BC Hydro Impact and Facility Study	$50,000
Tailings Management Facility Design	$200,000
Access Road Detailed Design	$50,000
Total Phase 1 Estimate	**$2,850,000**

Source: JDS (2014)

Table 1.10: Phase 2 Prefeasibility Study Cost Estimate

Item	Cost
Revised Resource Estimation	$50,000
Mine Planning & Reserve Estimation	$225,000
Metallurgical Test Work	$200,000
Processing Design	$300,000
Project Infrastructure	$175,000
Tailings Management Facility	$100,000
Report Compilation	$50,000
Total Phase 2 Estimate	**$1,100,000**

Source: JDS (2014)

2.0 INTRODUCTION

2.1 BASIS OF TECHNICAL REPORT

IDM Mining Ltd. (IDM or IDM Mining) mandated JDS Energy and Mining Ltd. (JDS) to complete a Preliminary Economic Assessment (PEA) of the Red Mountain gold project located in northwestern B.C., 18 km east of the town of Stewart. The purpose of this study is to complete a review and compilation of the resources, mining designs and preliminary economics using parameters updated to 2014 costs and IDM Mining corporate objectives.

On April 12, 2014, IDM optioned the property from Seabridge Gold Inc. (Seabridge) with the intent of initiating a PEA study and conducting further exploration work.

Since acquiring the project, IDM has completed a comprehensive review and validation of the Red Mountain geological and environmental data, and JDS has carried out engineering studies in connection with the resource model, mine design, mineralised material processing, road access, environmental impacts and cost estimation.

It must be noted that this preliminary economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

2.2 SCOPE OF WORK

JDS was requested to review historical work and produce a PEA in accordance with JDS's strategy of fit for purpose design. The PEA objectives were to target a potentially economic design and operating plan that focused on design criteria specific to IDM.

JDS Energy & Mining Inc.'s scope of work included:

- Compile the technical report including historical data and information provided by other consulting companies
- Establish potentially mineable resources
- Underground mine planning
- Select mining equipment
- Processing plant design and operating parameters
- Design required site infrastructure, identify proper sites, plant facilities and other ancillary facilities
- Estimate OPEX and CAPEX for the project

- Prepare a financial model and conduct an economic evaluation including sensitivity and project risk analysis
- Interpret the results and make conclusions that lead to recommendations to improve value and reduce risks.

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 appropriate professional institutions. The QPs are responsible for specific sections as follows:

Table 2.1: Qualified Person Responsibilities

QP	Company	Report Section(s) of Responsibility
Dunham Craig, P.Geo.	JDS	1-12, 14, 19-28
Gord Doerksen, P.Eng.	JDS	15, 16
Scot Klingmann, P. Eng.	JDS	18
Tom Shouldice, P.Eng.	TS Technical Services Ltd.	13, 17

Source: JDS (2014)

2.4 SITE VISITS & INSPECTIONS

QP site visits were conducted as follows:

- Dunham Craig, as Vice President Exploration and Corporate Development, North American Metals Corp. was responsible for technical oversight of the Red Mountain project and visited the site for 30 days during 2001. Dunham Craig also visited the site on May 28, 2014.
- Gord Doerksen visited the project site on May 28, 2014.
- Scot Klingmann visited the project site on May 28, 2014.
- Tom Shouldice has not visited the project site.

2.5 UNITS, CURRENCY & ROUNDING

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

This function of this report is to provide a resource estimate and preliminary economic assessment of the Red Mountain deposit. 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.

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

Bond collected primarily exploration data. LAC continued with exploration and conducted numerous engineering studies, which culminated in a draft feasibility study. ROM conducted exploration and during the NAMC program. Detailed studies of mineralisation 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 engaged in several PEA studies as well as conducting further tailings management facility studies. Banks completed a PEA in 2013.

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.

3.0 RELIANCE ON OTHER EXPERTS

The resource estimate in this report is a compilation of historical work (1986-1996) and new work by NAMC staff during February 2000 to 2001. No new drilling was conducted by NAMC. An assay and bulk density verification program was conducted by NAMC and all appropriate drill core was re-logged. A further QA/QC report was compiled by NAMC and the large majority of the drill assays and survey locations are assumed accurate and professionally recorded by previous workers. Spot checks by NAMC indicated that this is true. Banks completed two drill holes in 2012, but these data were not included in the current resource estimate.

Metallurgical test work was conducted by LAC and NAMC staff from 1991 to 2001. Cyanidation, flotation and ABA test work was performed by previous workers and laboratories considered to be reputable but no check work has been performed by JDS on prior metallurgical or ABA work.

Underground mine and tailings management facility engineering raw data was utilised from LAC and Seabridge reports. Estimates used in this report are otherwise derived from first principles by JDS.

Wentworth Taylor, Independent CA, provided consultation to this report for taxation estimations.

Loralee Johnstone and Warren Nimchuk provided consultation to Section 4.4 and Section 19 of this report.

The QPs take responsibility for the work provided by other experts.

4.0 PROPERTY DESCRIPTION & LOCATION

4.1 PROPERTY DESCRIPTION & LOCATION

Red Mountain is situated in northwestern British Columbia, approximately 18 km east-northeast of Stewart (Figure 4-1). 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 1,500 and 2,000 m. The area is characterised 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. JDS has modified this road plan specific to the requirements of this study.

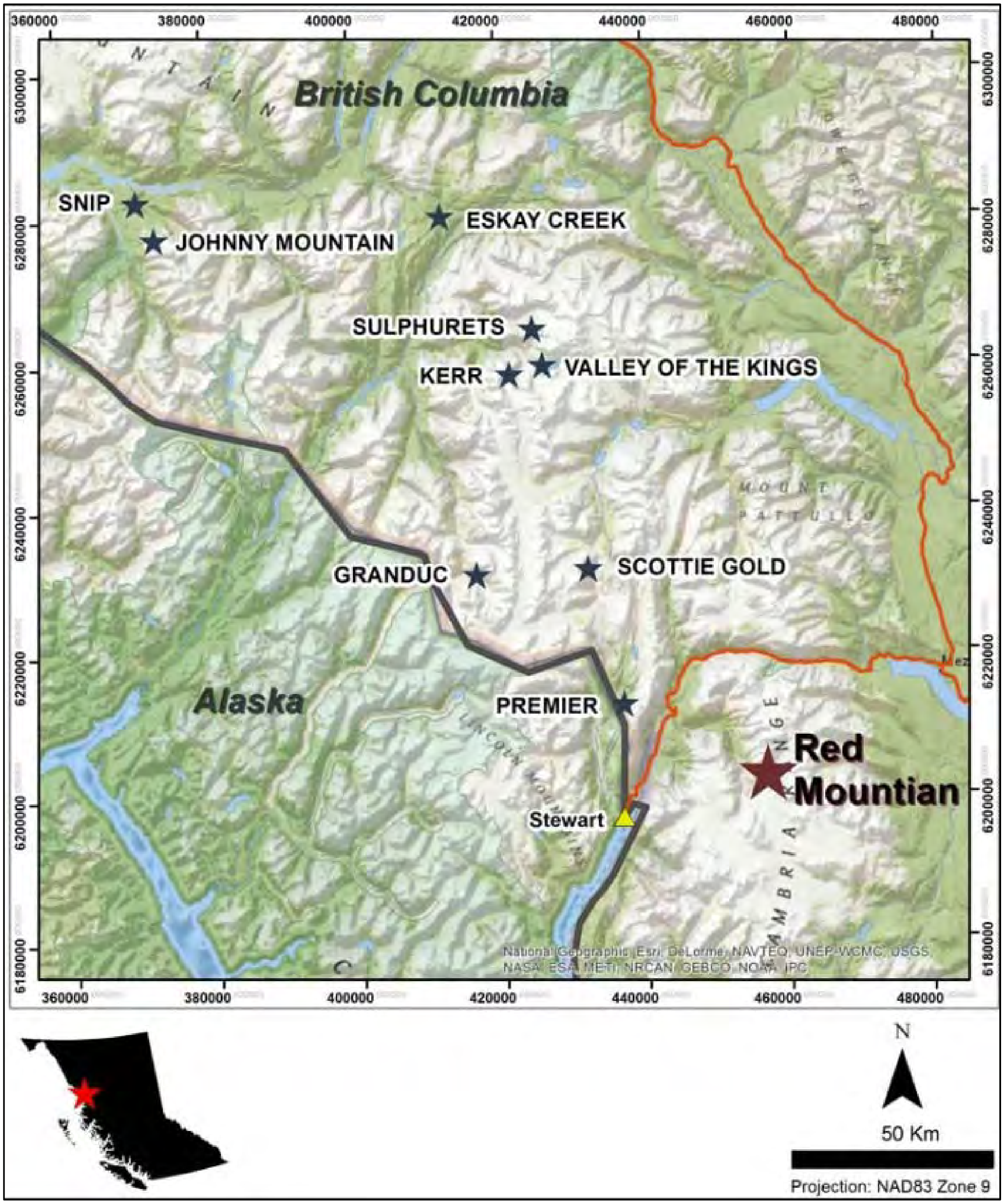
The deposit is located under the summit of Red Mountain at elevations of between 1,600 and 2,000 m. The site is drained by Goldslide Creek, which flows southwest to the flank of the Bromley Glacier and by the Rio Blanco Creek. Both of these creeks are tributaries of Bitter Creek, which in turn is a tributary 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 boundary.

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 north west of Vancouver and 180 km north of Prince Rupert. Stewart is at the end of Highway 37A, a paved all weather highway, 347 km from Smithers and 327 km from Terrace. The District of Stewart borders on the State of Alaska and extends some services to the community of Hyder, Alaska.

4.2 MINERAL TITLE

IDM has, under option agreement, the right to acquire a 100% interest in 47 contiguous claims that comprise an area of 17,125.2 ha currently owned 100% by Seabridge.

Figure 4-1: Red Mountain Project Location Map



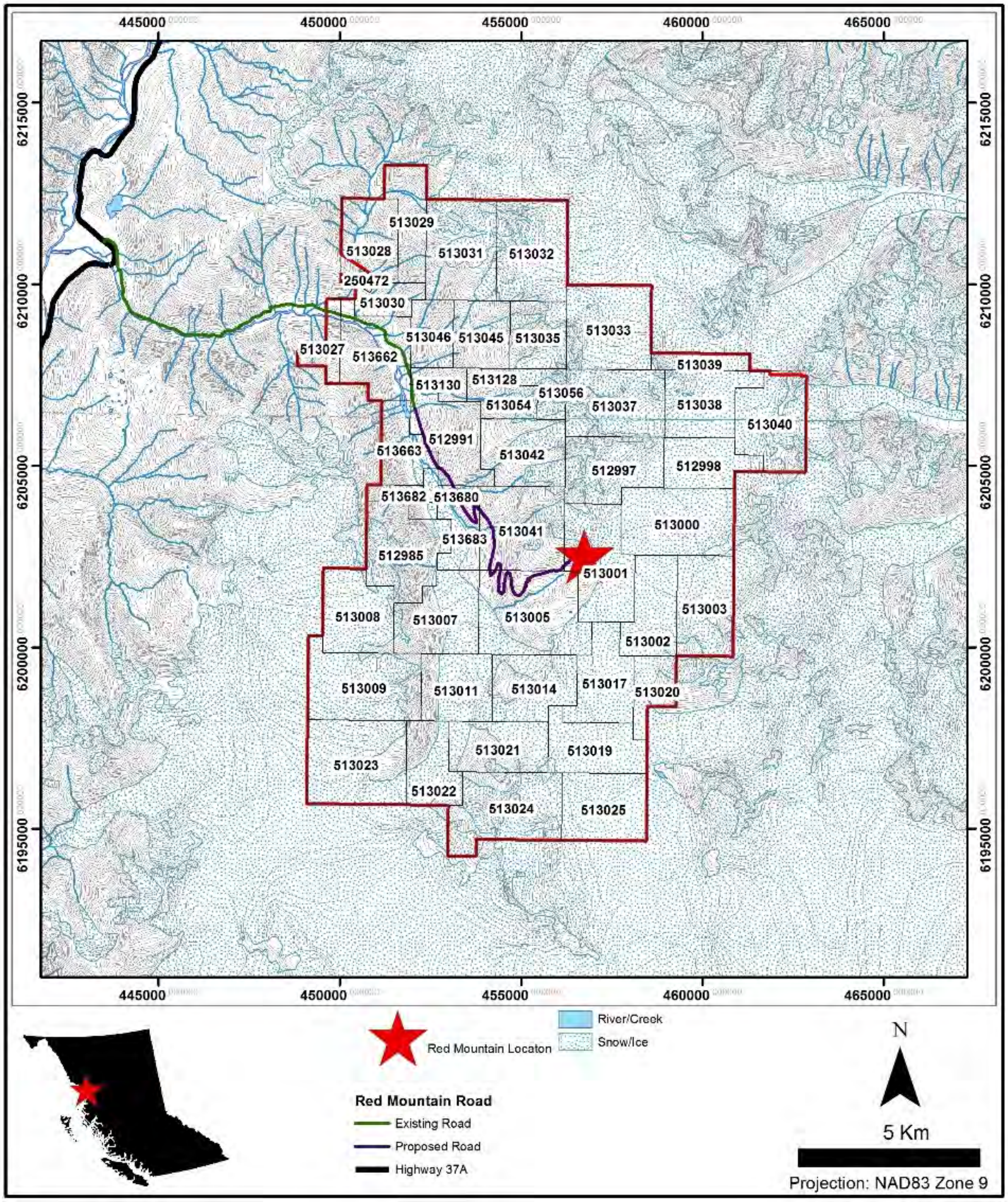
Source: IDM (2014)

Table 4.1: Red Mountain Claim Map

Tenure Number	Tenure Type		Hectares	Ownership (%)
512997	Mineral	CLAIM	452.4	100
513001	Mineral	CLAIM	525.1	100
513028	Mineral	CLAIM	361.4	100
513040	Mineral	CLAIM	470.4	100
513046	Mineral	CLAIM	217.0	100
513054	Mineral	CLAIM	180.9	100
513662	Mineral	CLAIM	434.0	100
513002	Mineral	CLAIM	362.3	100
513024	Mineral	CLAIM	580.5	100
513045	Mineral	CLAIM	289.3	100
513130	Mineral	CLAIM	108.5	100
513007	Mineral	CLAIM	452.8	100
513017	Mineral	CLAIM	380.5	100
512985	Mineral	CLAIM	488.8	100
513005	Mineral	CLAIM	670.2	100
513014	Mineral	CLAIM	398.7	100
513019	Mineral	CLAIM	380.7	100
513031	Mineral	CLAIM	542.1	100
513032	Mineral	CLAIM	542.2	100
513033	Mineral	CLAIM	542.4	100
513038	Mineral	CLAIM	398.0	100
513009	Mineral	CLAIM	597.8	100
513021	Mineral	CLAIM	380.7	100
513056	Mineral	CLAIM	144.7	100
513022	Mineral	CLAIM	308.2	100
513023	Mineral	CLAIM	634.4	100
513680	Mineral	CLAIM	90.5	100
512998	Mineral	CLAIM	307.6	100
513027	Mineral	CLAIM	126.6	100
513029	Mineral	CLAIM	289.1	100
513030	Mineral	CLAIM	162.7	100
513682	Mineral	CLAIM	108.6	100
513000	Mineral	CLAIM	579.3	100
513025	Mineral	CLAIM	435.4	100
513035	Mineral	CLAIM	289.3	100
513037	Mineral	CLAIM	506.5	100
513663	Mineral	CLAIM	253.3	100
513683	Mineral	CLAIM	181.0	100
513011	Mineral	CLAIM	362.4	100
513008	Mineral	CLAIM	416.5	100
513020	Mineral	CLAIM	199.3	100
513003	Mineral	CLAIM	434.7	100
513039	Mineral	CLAIM	126.6	100
513128	Mineral	CLAIM	36.2	100
512991	Mineral	CLAIM	416.2	100
513041	Mineral	CLAIM	543.1	100
513042	Mineral	CLAIM	416.2	100
Total Hectares			**17,125.2**	

Source: IDM (2014)

Figure 4-2: Red Mountain Claim Map



Source: IDM (2014)

IDM, under the terms of the Option Agreement, can earn a 100% interest in the Red Mountain project, 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 project is 100% owned by Seabridge and is currently optioned to IDM Mining Ltd, 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 net smelter returns (NSR), comprising a 1.0% NSR payable to Barrick and a 2.5% NSR payable to Wotan.

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. 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 Barrick and a 2.5% NSR payable to Wotan.

4.3.2 Underlying Agreements

On April 15, 2014, IDM entered into an option agreement for the Red Mountain project with Seabridge through IDM's predecessor company Revolution Resources Corporation. Subsequent to the option agreement, Revolution Resources Corporation underwent a restructuring and name change to IDM Mining Ltd.

Claim title is currently under Seabridge. Upon satisfaction of the option terms, title will be transferred to IDM. Seabridge owns 100% of the property claims subject to two royalties. Barrick holds a 1% NSR and a 2.5% royalty is payable to Wotan Resources Corp. A $50,000 annual advance royalty is payable to Wotan annually.

Under the terms of the Option Agreement, Revolution (now IDM) can earn a 100% interest in the Red Mountain project, subject to certain underlying royalties, by (1) issuing to Seabridge

29,733,000 shares of Revolution, (2) paying to Seabridge $2 million cash in staged payments ($1 million payable within 90 days, $1 million within 1.5 years), and (3) incurring $7.5 million in exploration and development expenditures over 3 years ($2.5 million per year). Revolution has the right to extend the deadline for expenditure of the final $2.5 million by one year upon payment to Seabridge of $250,000.

Upon the commencement of commercial production, Revolution 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 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.

The principal agreements governing the Red Mountain project are listed below, along with a summary of the more salient provisions and identified by claim number in Table 4.2. The 2014 mineral resource defined in his report is subject to the Barrick & Wotan Agreements only.

1. Barrick Agreement: Asset Purchase and Royalty Agreement dated August 17, 1995 between 1091064 Ontario Limited ("1091064"), Royal Oak and Barrick., Under the 1995 agreement, Royal Oak purchased its interest in Red Mountain from 1091064 (a wholly owned Barrick subsidiary) and granted 1091064 an uncapped 1.0% NSR royalty on production.1091064 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.
2. Wotan Agreement: Agreement dated July 26, 1989 between Bond, Wotan and Dino Cremonese granting Bond an option to acquire seven mineral claims., Seabridge 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: Seabridge is obligated to pay Darcy Krohman and Greg Sinitsin a 1.0% NSR royalty on production from claims 513128 and 513190. Seabridge 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. Seabridge 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			
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**				

(Seabridge, IDM 2014)

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 the remediation of certain environmental issues, 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
- removal of equipment from the site.

In 2004, the reclamation plan was filed with the BC Ministry of Energy and Mines and at that time the bond was sufficient to cover the cost of reclaiming the site, however regulators have expressed interest in updating the plan to more current costs due to general increases in fuel and contractor costs.

There is no fuel stored on site nor was there any record of any spillage of fuel while it was. Goldslide Creek downstream of the fuel storage area has been annually monitored for residual hydrocarbons, but none has been detected.

4.4.2 Required Permits & Status

Pursuant to section 3(1) of the Reviewable Projects Regulation pursuant to the British Columbia Environmental Assessment Act, the proposed production capacity for the project exceeds the criteria of 75,000 tonnes per annum (t/a) of mineral ore for a new mineral mine and will require review pursuant to the BC EAA and issuance of an Environmental Assessment Certificate (EAC). The proposed submission date of an Application for the EAC is mid-2015.

The intent of IDM is to apply for concurrent review for provincial permits pursuant to the BC EAA Concurrent Approvals Regulation. Under this regulation, provincial permits would be reviewed at the same time as the Application. No decisions on permits can be made until after a decision has been made on the EAC.

All concurrent permit applications for the project will be coordinated through the Major Projects Office (MPO) of the Ministry of Forests, Lands and Natural Resource Operations (MFLNRO). It is anticipated that the project will require approvals under the *Mines Act* (1996b), *Environmental Management Act* (2003), and *Land Act* (1996a).

IDM will meet with the appropriate provincial agencies to confirm permitting requirements related to the project.

5.0 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 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: LAC (1993)

From June 1993 to June 1994, weather data was collected for the site. Several stations were monitored but the station most relevant to this study is the Upper Tram Station. For that one-year period, 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 – Upper Tram Station

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 at the Upper Tram location are channeled by topography. 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. The Upper Tram Station is more sheltered than the top of the ridge near the portal. Wind speeds are expected to be significantly higher at the ridge where most of the projected activity is planned.

5.3.3 Precipitation

Precipitation data was collected for part of 1994 (April to August) at the Upper Tram Station; this data along with data collected at the Lower Tram Station were compared to the 1974 to 1992 Stewart Airport records. While there were insufficient data from the Upper Tram Station for an accurate correlation with the Stewart Airport, 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 (1,742 masl) is equivalent to the 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 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 was 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 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 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 is hampered during the late winter and spring by heavy snowfall and avalanche conditions. Current planning envisions a seasonal operation beginning in March and ending in mid to late December.

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 32 km from the BC Hydro sub-station north of Stewart, BC.

At the project site, a surface tote road network, basic surface structures (camp buildings, helipads, waste rock storage areas) and used mobile equipment remain from previous exploration activities. Water is readily available from both surface and underground sources. As well, mineralised zones have been bulk sampled in the Marc Zone accessed from 1,700 of existing underground decline and drift development.

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.

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

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.

6.0 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 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 tons of hand sorted ore from the Silver Tunnel. (Roosevelt #1 claim on Roosevelt Creek). Smelter returns averaged 0.26 oz/t Au, 101 oz/t Ag, 34% Pb and 8% Zn.
- **1965** Hartley Flats - 4.8 tons of hand cobbed ore from adits shipped to Trail.
- **1965** Discovery of molybdenite mineralisation 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 B.C. 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 mineralisation 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.

The following table is a recent chronological summary of exploration efforts on Red Mountain from 1988 to 2014:

Table 6.1: Red Mountain 1988-2014 Exploration Summary

1988-89	Staking of Red Mountain by Wotan Resources Inc.
1989	Red Mountain and Wotan properties optioned to Bond. Discovery of gold-silver mineralisation 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 meters of surface drilling.
1995	Red Mountain project acquired by Barrick following Barrick's take-over of Lac Minerals. 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.

Source: JDS (2014)

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 were completed in the past at a 3 g cut-off. Only the Seabridge (2002, 2008) and Banks (2013) resource estimates are considered to conform to the NI-43-101 requirements of the *BC Securities Act*.

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
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

Source: JDS (2014). (1) 0 g/t Au cut-off, (2) 6 g/t Au cut-off, (3) 3 g/t Au cut-off

6.4 HISTORIC PRODUCTION

No historical production has taken place on the property.

7.0 GEOLOGICAL SETTING & MINERALISATION

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 mineralised zones, and presents a model for deposit formation based on observed geology and gold distribution.

7.2 REGIONAL GEOLOGY

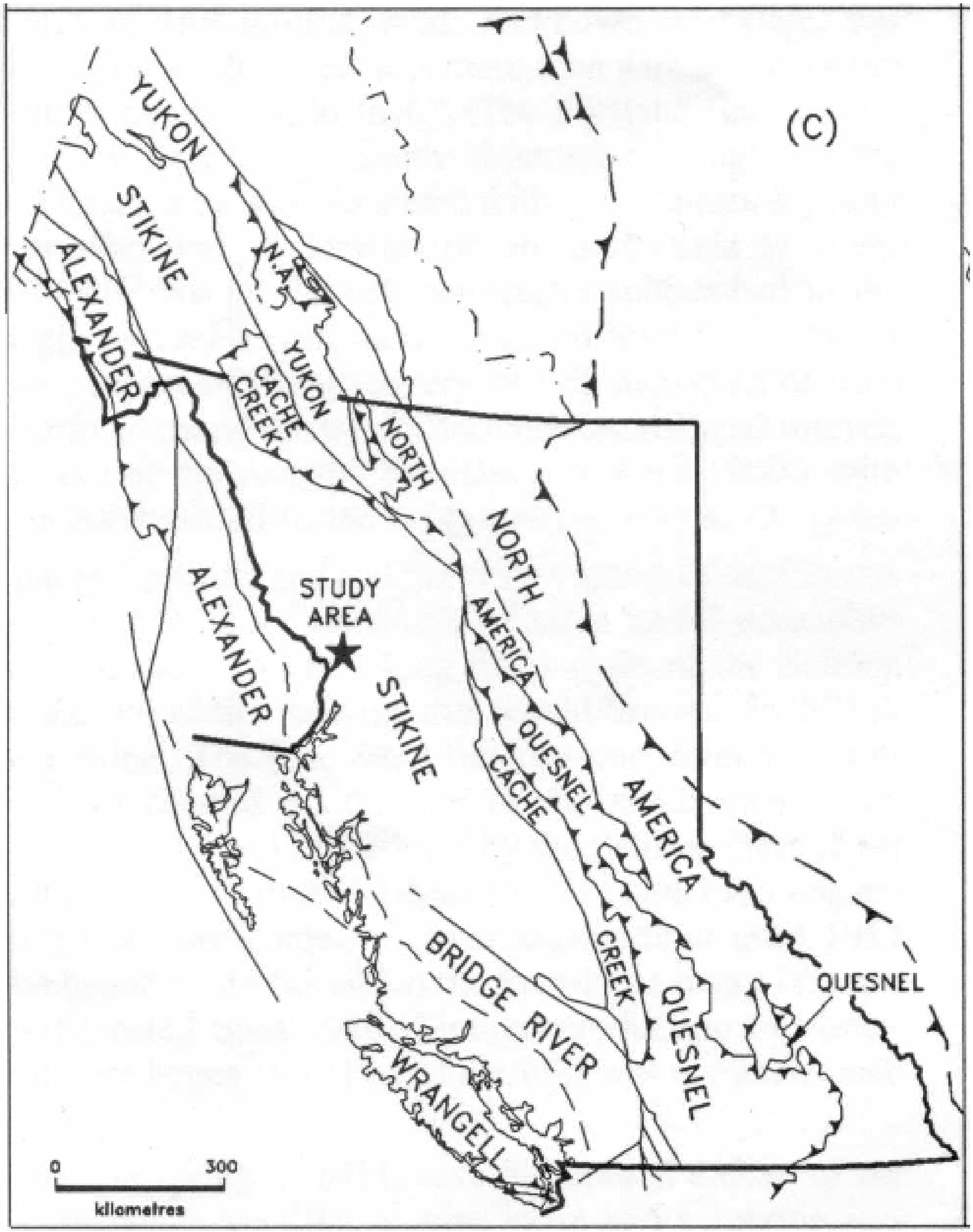
The regional geology of the Red Mountain area has been described by Greig et al (1994), Alldrick (1993) and Rhys et al (1995). The following description is drawn from these sources.

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt (see 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 form 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 mineralisation 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: JDS (2014)

7.3 LOCAL GEOLOGY

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

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

170 Ma (Middle Jurassic) – Stikinia has docked with North America. The Bowser Basin 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 backthrusting 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 folds rocks of the Stuhini, Hazelton and Bowser Lake groups.

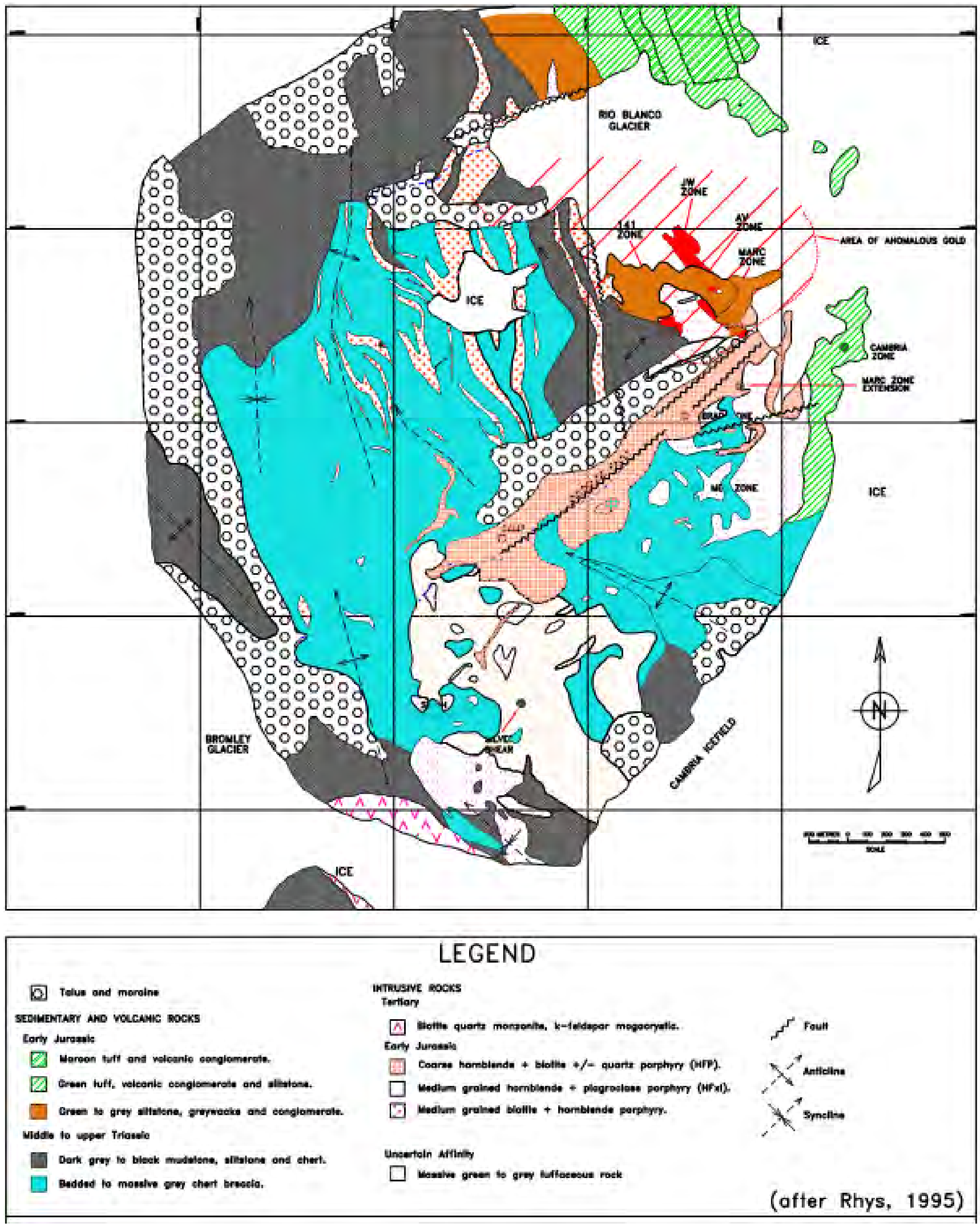
65 Ma (End of Cretaceous) – Deformation of Stikine terrain rocks is complete resulting in folded and doubly plunging structural culminations. The Red Mountain deposits have been rotated from a vertical (?) orientation to a westerly dipping, northerly plunging orientation in the eastern limb of the Bitter 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 mineralised 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: NAMC (2002)

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.

A Tertiary intrusion, the McAdam point stock, is exposed in the southern portion of the map area adjacent to the Bromley Glacier. It is a medium to coarse-grained biotite quartz monzonite dated to 45 Ma (Rhys et al, 1995). 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. There is no evidence to suggest that the intrusive units were affected by this folding event.

Brittle faulting has affected all rock units at Red Mountain. Rhys et al (1995) recognised 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 mineralised 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. Both of these sets of faults are recognised by Evenchick (personal communication, 2001) to be the result of a Miocene extensional event.

7.5 SIGNIFICANT MINERALISED ZONES

7.5.1 Mineralised Zones

The mineralised 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 mineralised 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 - 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 mineralisation was not observed during core re-logging in 2000.

7.5.2 Deposit Formation Model

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 mineralisation is similar to that of many porphyry systems. This was based on data from deep drilling that indicated mineralisation and alteration zoning common to traditional porphyry systems. Lang (2000c) suggested that while the porphyry system zonation was present the alteration and mineralisation 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.

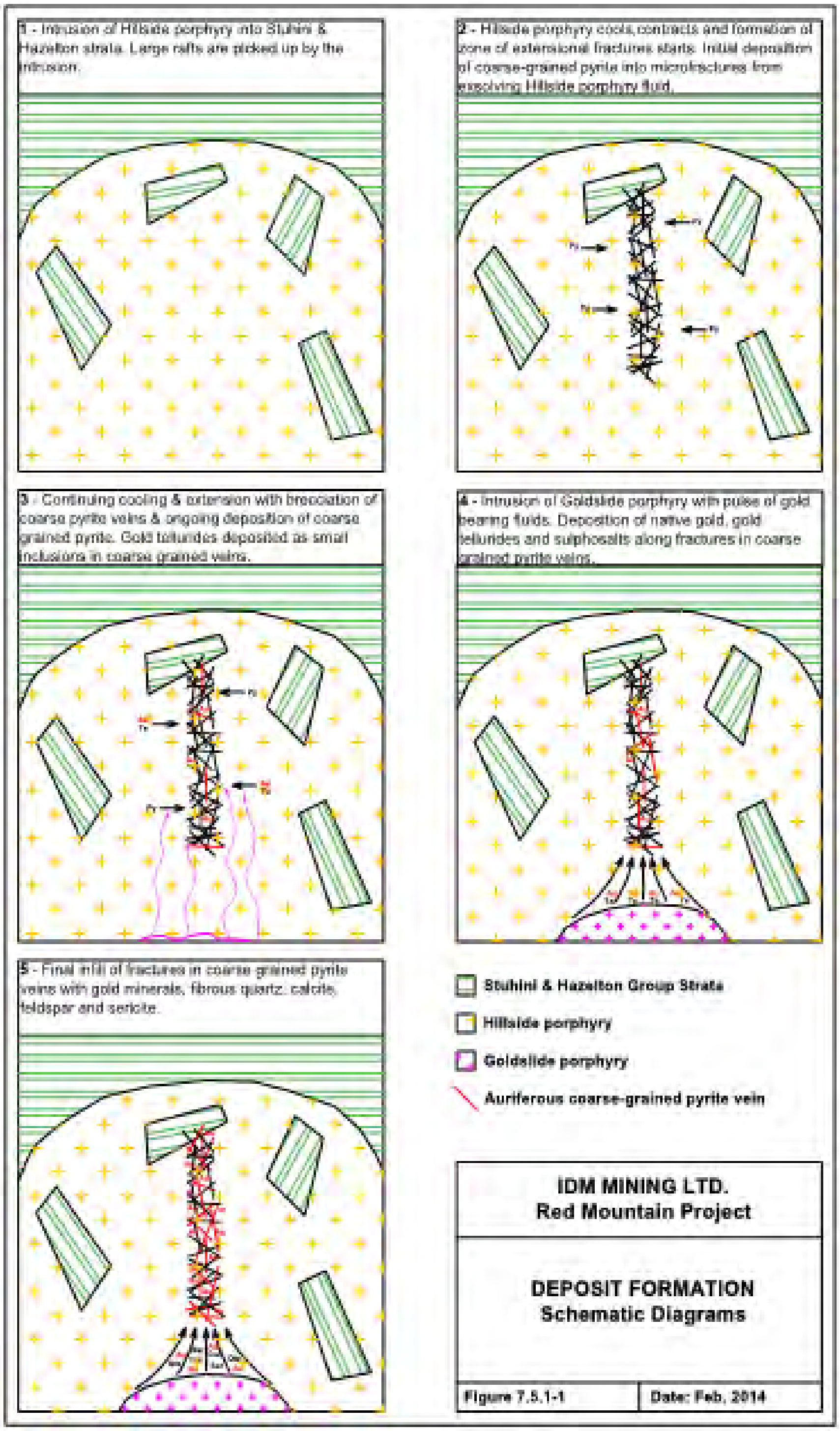
A synthesis of these models, in particular using elements of the models proposed by Lang and Barclay, appears to best fit with geological and mineralogical observations. A series of schematic diagrams illustrating this is shown in Figure 7-3 (overleaf) and a brief description of each frame is as follows:

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.
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 7-4 (after Lang, 2000c).

The model proposes a plausible origin for the structures that host sulphide and gold mineralisation, 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 7-3: Deposit Formation Models



Source: NAMC (2002)

Figure 7-4: Schematic Diagrams Showing Stages of Formation of Pyrite Veins & Timing of Au

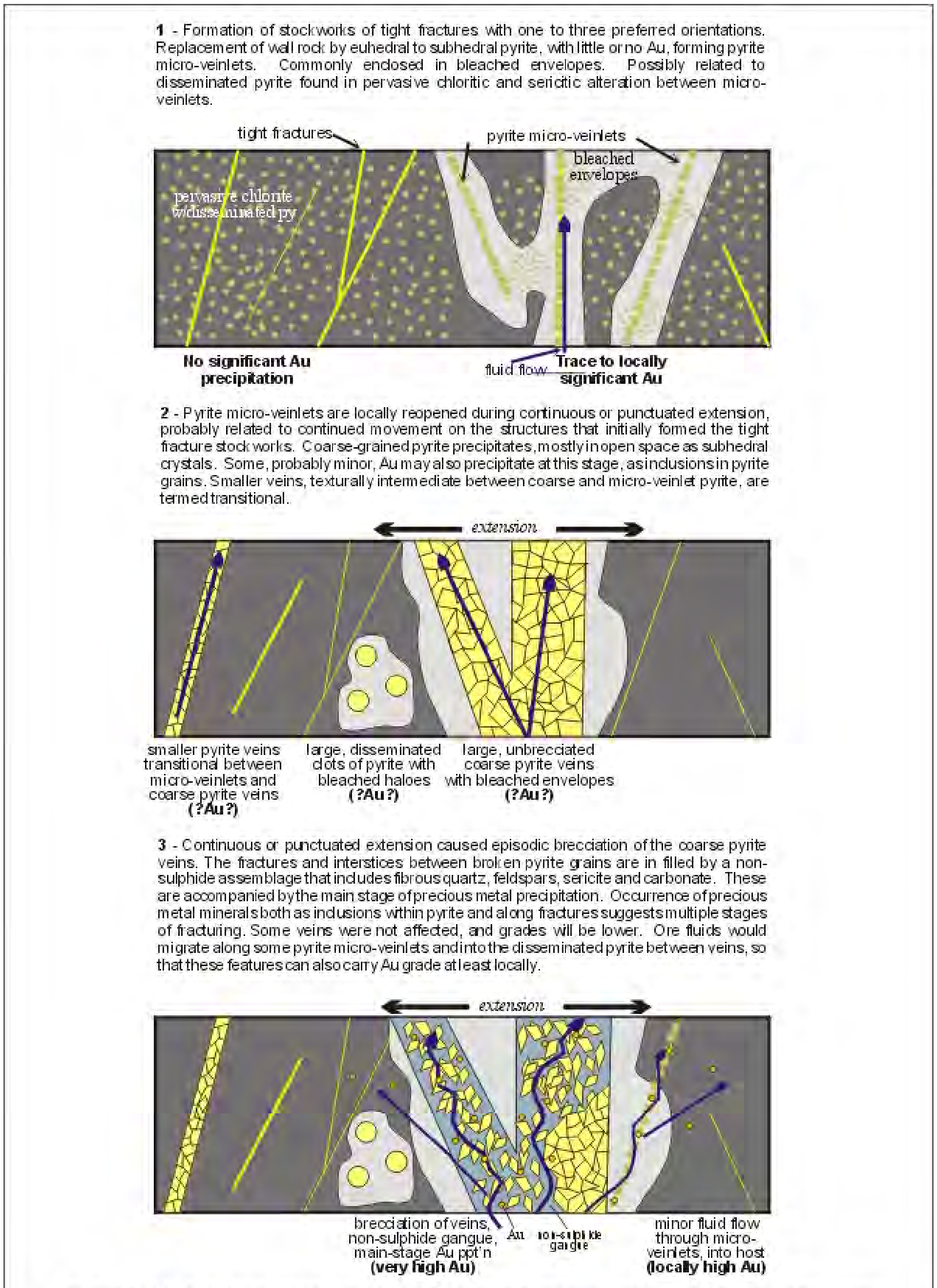


Figure 7.5.1-2: Schematic diagrams showing stages of formation of pyrite veins, and the timing of Au precipitation (from Lang, 2000C).

Source: NAMC (2001)

8.0 DEPOSIT TYPE

The Red Mountain mineral deposit is a porphyry-hosted, high-sulphidation gold/silver pyritic stockwork (see Section 7.5.1 for details).

9.0 EXPLORATION

Past exploration is detailed in Sections 6, 10 and 1. No exploration was conducted from 2001 to 2012 as the property was on care and maintenance by Seabridge. In 2012, Banks drilled three drill holes is the Marc Zone, two of which intersected the Marc mineralised zone and the third hole was abandoned prior to reaching the Mark zone. The two intercepting Banks drill holes were entered into the Gemcom database in 2014 by JDS and tested for significance to the NAMC database. The Banks holes confirmed the accuracy of the resource solid outline. When the model was re-kriged, testing the Banks drill hole influence, the change in the resource gold grade was less than 0.2% for the Marc Zone. As this was well within the accuracy limits of the 2014 resource re-estimation and there was no QA/QC available for the Banks drill holes, the Banks drill holes were not used in the 2014 resource estimation.

Exploration potential for the property is deemed as high. Since 1994, when the surface exploration was terminated, the glaciers surrounding the Red Mountain project have significantly receded exposing considerable areas 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 proximal to the known resource.

10.0 DRILLING

10.1 PRIOR DRILLING

A total of 466 surface and underground diamond drill holes (134,807.24 m) have tested a variety of targets on the Red Mountain property. Of these, 406 holes totalling 105,129.20 m were drilled by Bond and Lac between 1989 and 1994. The remaining 60 holes (29,678.04 m) were drilled by Royal Oak in 1996. No drilling was carried out by NAMC. During 2012, Banks completed two drill holes in the Marc zone. The Banks holes were reviewed, but as no QA/QC data were available, they were not used in the current resource estimate.

The majority of drilling has tested the Marc, AV, JW and AV-JW Tails mineralised zones. A total of 368 drill holes from the Bond and Lac programs, including 207 surface drill holes and 161 underground drill holes, have tested this area.

The resource estimate contained in this report encompasses the Marc, AV and JW zones and utilises data from 212 holes, including 80 surface holes and 132 underground holes, all of which were drilled during the Bond and Lac programs. None of the 1996 Royal Oak drill holes pierced the volume used in the resource model.

10.2 MINE GRID ORIENTATION

All data in the Red Mountain Gemcom database, including the drill hole orientation data, has two sets of XYZ 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 calculation has used mine grid coordinates and orientations.

10.3 SURFACE DRILLING PROGRAMS

Surface diamond drilling programs were carried out by Falcon Drilling Ltd. of Prince George, BC, from 1989 to 1991, 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 surface diamond drilling program was conducted by Britton Brothers Diamond Drilling Ltd. of Smithers, BC, using equipment suitable for production of BQTK and NQ diameter core.

Approximately 42.5% of the 80 surface holes were drilled parallel to the section lines at an orientation of 090°. Thirty-five percent were drilled at either 135° or 315° mine grid, which was parallel to the section orientation from 1989 to late 1992 when reinterpretation of the geology indicated that drilling a northerly tend to the stockwork zones. The remaining 22.5% of the surface holes were drilled at off section orientations. Inclinations for the holes ranged from -45° to -90°.

Sectional spacing for the surface drilling is 25 m for the Marc Zone and 25 to 50 m for the AV and JW zones.

10.4 UNDERGROUND DRILLING PROGRAMS

Underground drilling programs in 1993 and 1994 were carried out by J.T. Thomas Diamond Drilling Ltd. of Smithers, B.C. As with the surface drilling, they used equipment suitable for producing BQTW and NQ.

Of the underground holes, 81% were drilled parallel to the section lines; 42% oriented at 090° and 39% oriented at 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°.

Sectional spacing for the underground drilling is 25 m for the Marc Zone and 25 to 50 m for the AV and JW zones.

10.5 GENERAL DRILLING PROCEDURES

Drill control 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.

All drill holes were surveyed. For surface holes, a survey was taken for azimuth and dip, where possible, while the rods were still in the hole. If the drill was moved before such a survey, a piece of drill stem was placed inside the casing down the hole to get a collar azimuth and dip, as collars were left in place. Underground surveying was done every one to two weeks. If the drill was no longer on a site, a rod was put down the hole to get an azimuth and dip. As rock conditions underground were good, there was typically a snug fit of the rod within the abandoned hole.

Most, or all, of the pre-1993 collars were resurveyed 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.

10.6 DOWN HOLE SURVEYS

With the exception of the 1989 drill holes and a few of the 1990 drill holes, which had acid dip tests, all holes contained within the resource area have Sperry Sun surveys. From 1990 to 1992 the Sperry readings were collected every 300' (91.4 m). During 1993 and 1994 down hole orientations holes were ideally surveyed at 50 feet (15 m) depth and then every 200 ft (60 m), although variations from this did occur. Short underground holes generally had one survey near the bottom. 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. All photographs were kept. Survey readings that were suspect were not used.

10.7 DRILL HOLE ADJUSTMENTS

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

- The Sperry Sun surveys for a single 1993 underground hole had been misread. Correct readings were taken and the values entered into the database.
- For most of the 1989 drill holes and two 1990 drill holes only acid dip tests were taken, and for two 1990 drill holes no down hole survey information was collected. Average down hole deviations were calculated by using data from the Sperry Sun tests conducted on a majority of 1990 drill holes as these holes were drilled in similar orientations to the holes lacking survey data. The deviations were applied to five 1989 holes and four 1990 holes.
- 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.

10.8 GENERAL

The author considers the spacing and quality of drill hole data as reasonable for establishing the degree of confidence necessary for calculation of a mineral resource.

10.9 GEOLOGICAL 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. As there were several different people logging core, considerable time was spent trying to standardise 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.

During 2000 and 2001, NAMC relogged all core within the recognised mineralised zones including a 20 m envelope outside of the mineralised zones. The purpose of the relogging was to establish continuity of logging procedures, verify past logging entry and to determine continuity between sections. If mineralised continuity was not geologically determined between 25 m sections, the mineralisation was removed from the geological solids and excluded from resource interpolation.

10.10 PROCEDURES

Drill control 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 insured any drilling problems were noted, and helped insure good core handling practices were maintained by all drill crews.

All drill holes were surveyed. When the drill was still in place, a survey for azimuth and dip was made with the rods in the hole. Some surface holes were not surveyed before the drill was moved. On these holes, a piece of drill stem was placed inside the casing down the hole to get a collar azimuth and dip, as collars were left in place. Underground surveying was done every one to two weeks. If the drill was no longer on a site, a rod was put down the hole to get an azimuth and dip. Because rock conditions underground were good, there was typically a snug fit of the rod within the abandoned hole. Most, or all, of the pre 1993 collars were resurveyed. Collar locations from the new surveying were used in the database.

A Sperry Sun was used to get down hole orientations. Ideally, holes were surveyed at 50 feet (15 m) depth and then every 200 ft (60 m). Some variation from this occurred, especially in the early part of 1993. Short underground holes generally had one survey near the bottom.

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. All photographs were kept. Readings that were suspect were not used. Locally, pyrrhotite content is high enough that it could cause a deflection of the Sperry Sun compass. Some surface holes were intersected in the upper part of the underground workings. The results of a few pre-1993 holes do not line up well on section with newer holes. The pre-1993 hole collars were resurveyed and are correct, but the azimuth and dip must have varied from that used in the hole projections. No attempt was made to correct the down-hole data for the pre-1993 holes that do not line up well on section.

Core was halved for analytical sampling. Samples were marked and tagged by the logging geologist. A sample sheet was also made with sample numbers and sample from-to to insure sample numbers corresponded with the right intervals when samples were collected. Within and around the potential ore zones, samples were generally of standard length, though they were broken at major lithological/ alteration/ mineralisation changes. This was standardised to 1.0 m sample lengths, though some holes were sampled in 1.5 m lengths in the earlier part of the program. In areas well outside the potential ore zones, sample length was based on geology and mineralisation. Samples generally had an assay and an ICP analysis. Whole rock samples were taken every 20 to 30 m, or with every major lithological change, whichever came first. Proximal, or within the ore zone, whole rock samples were taken every 10 m. Samples, which were half core, were at a minimum 0.5 m long. All whole rocks also had an ICP analysis. Many whole rock analysis, especially near mineralised zones, were on samples already chosen for assay.

Analytical samples were cut by a rock saw to get half core samples. 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 insure that this process worked efficiently and to insure good quality control.

Data was entered into the computer by data entry personnel. All 1993 data was checked in January and February of 1994. In 1994, a system where data was entered and checked by different people as soon as possible after logging was instituted. The geologist who logged a hole was responsible to insure all data was entered and 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.

10.11 INTERPRETATION & RELEVANT RESULTS

Drilling procedures and core logging were conducted to an above average industry standard. Core storage and data control were conducted to a high standard. Re-logging of all core within the mineralised zones by NAMC provided geological logging continuity and verification of historical work in preparation for creating geological solids for resource interpolation.

11.0 SAMPLE PREPARATION, ANALYSES & SECURITY

11.1 SAMPLING METHODS & PROCEDURES

Standard procedures for sample collection and preparation were applied to all drill core collected on the property since 1989. A summary of these procedures, grouped by year and operator is provided below.

11.1.1 1989 - 1992 Drill Core Samples

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.

11.1.2 1993 - 1994 Drill Core Samples

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, British Columbia.

11.2 DRILL CORE RECOVERY

Core recovery was recorded for a portion of the Marc, AV and JW zones by previous operators. Within the geological solids used for the resource calculation, 746 sample intervals were recorded for recovery, with the recovery percentage distribution outlined in Table 11.1.

No sample recovery bias with respect to gold grade was observed.

Table 11.1: Core Recovery

Core Recovery %	Frequency	Cumulative %
10%	0	0.00%
20%	0	0.00%
30%	0	0.00%
40%	1	0.13%
50%	1	0.27%
60%	1	0.40%
70%	2	0.67%
80%	15	2.68%
85%	9	3.89%
90%	22	6.84%
95%	58	14.61%
100%	637	100.00%
n=	746	

Source: NAMC (2001)

11.3 1993-1994 UNDERGROUND CHIP SAMPLES

During the 1993 and 1994 programs, every underground round was sampled. This included face chip samples and muck samples.

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.

11.4 1993-1994 BULK SAMPLES

A muck sample was collected from every underground round, either from the main decline or from the cross cuts designed to assess the Marc Zone mineralisation. 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 than2.0 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.

11.5 SECURITY MEASURES

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 facility in Stewart where sampling under supervision and delivery directly to the Eco-Teck laboratory located in Stewart was completed. NAMC samples were collected by a staff professional geologist and delivered to the Chemex laboratory under the direct supervision of the geologist.

11.6 SAMPLE PREPARATION

The primary assay laboratory used by Bond and Lac prior to 1993 for the assaying of all drill core samples collected from the Red Mountain project was Mineral Environments Laboratories of North Vancouver, British Columbia (Min-En). Min-En Labs followed routine sample preparation techniques. Pulps were analyzed for gold and silver using a fire assay technique with an atomic absorption (AA) finish on a one assay ton (1 AT or 30 g) weight. If the gold assay result was greater than 0.5 oz/t (approximately 17 g/t) the pulp was re-assayed with a gravimetric finish.

During the main drill programs in 1993 and 1994, carried out by Lac Minerals Ltd., the primary assay laboratory used for all surface and underground samples was Eco-Tech Laboratories located in Stewart, British Columbia (Eco-Tech). Eco-Tech performed routine gold and silver fire assays with an Atomic Absorption finish. The gold assay technique used by the lab was dependent upon the grades obtained by FA-AA. Samples grading >10 g/t Au were re-analyzed with a gravimetric finish and metallics were performed on material grading >30 g/t Au.

11.7 ANALYTICAL PROCEDURES - CHECK ASSAY LABORATORY

11.7.1 1989 - 1992

The check assay laboratory used by Bond and Lac prior to 1993 was Bondar-Clegg Laboratories, of North Vancouver, B.C. During this period, 13,668 samples were submitted to the primary laboratory (Min-En) for analysis (Table 11.2). A total of 1,470 of these, or 10.8% have been checked assayed by Bondar-Clegg. The pulps submitted were analyzed for gold and silver using a standard fire assay techniques with an atomic absorption (AA) finish on a one assay ton (1 AT or 30 g) weight. If the gold assay result was greater than 0.2 oz/t (approximately 7 g/t), the pulp was re-assayed with a gravimetric finish.

11.7.2 1993 - 1994

For the 1993 and 1994 programs, the Chemex Labs of North Vancouver, B.C. was used as the check assay laboratory (Table 11.2). During this time, 30,834 samples were submitted to the primary lab, Eco-Tech. A total of 3,963 of these, 349 pulps and 3,614 rejects, corresponding to 12.9%, were submitted for check assay.

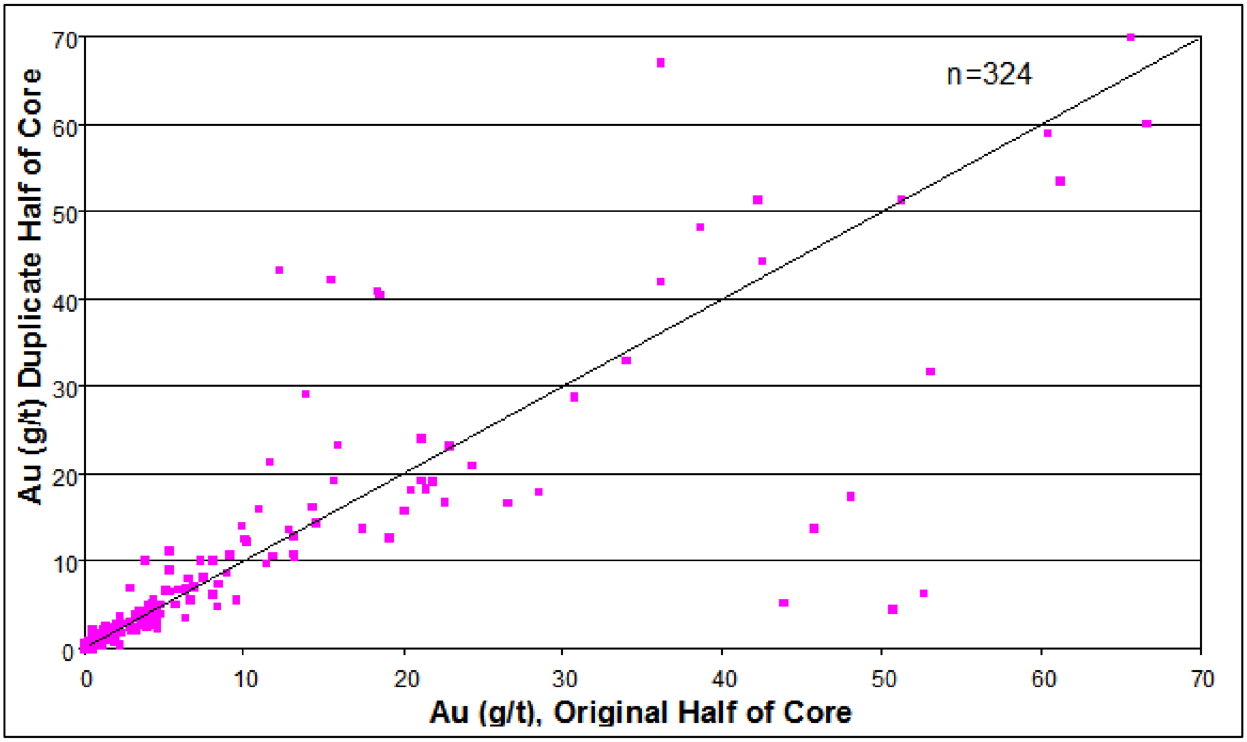
The samples submitted were analyzed for gold and silver using a standard fire assay techniques with an atomic absorption (AA) finish on a one assay ton (1 AT or 30 g) weight. If the gold assay result was greater than 10 g/t, the pulp was re-assayed with a gravimetric finish. If the gold assay result was greater than 30 g/t, metallics were performed.

Table 11.2: Sample Preparation & Analytical Procedures for Primary & Check Assay Labs

Sample Preparation Procedure	Min-En Labs 1989-1992	Bondar Clegg 1989-1992 (Check Samples)	Eco-Tech 1993-1994	Chemex 1993-1994 (Check Samples)
Dry sample	@ 95°C		@ 60°C	X
Jaw crush	¼ inch		X	X
Roll/Cone crush	rolled to -1/8 inch		-10 mesh	-10 mesh
Jones Riffle Splitter	X		X	X
Bag/store coarse reject	X		X	X
Ring pulverise sub-sample	500 gram split to 95% -120 mesh		250-400 gram split to 85% -140 mesh	200-350 gram split to 90% -150 mesh
Rolled and homogenised	X		X	X
Lab insertion of standards	1 in every 24 samples	1 in every 21 samples	1 in every 37 samples	
Lab insertion of blank	1 in every 24 samples		1 in every 37 samples	
Lab insertion of random duplicate		2 in every 21 samples	1 in every 37 samples	
Total samples in batch	26	24	40	n/a
Weight of sample	1 assay ton (15 grams for high sulfide)	1 assay ton (30 grams)	1 assay ton	1 assay ton
Fluxed and inquarted with silver	X	X	X	X
Cupelled to precious metal bead	X	X	X	X
Bead digestion by aqua regia	X	X	X	X
Analyze by Atomic Absorption	X	X	X	X
Quality Control:	if Standard not within ±3 std dev re-assay entire batch		if Standard not within ±3 std dev re-assay entire batch	
Gravimetric finish	if Au > 0.5 oz/t (approx. 17 g/t)	if Au > 0.2 oz/t (approx. 7 g/t)	if Au > 10 g/t	if Au > 10 g/t
Metallics			if Au > 30 g/t	if Au > 30 g/t

Source: NAMC (2001)

Figure 11-1: Comparison of Au Grades Original vs. Duplicate Halves of Core for Variance Holes



Source: NAMC (2001)

11.7.3 2000

In 2000, NAMC collected 197 check samples with particular emphasis on the Marc and AV Zones. Representative samples were chosen throughout the deposit from areas where check assay coverage was deemed low relative to other areas. The majority of samples were taken from 1993 and 1994 underground and surface drill holes. Of the samples, 167 were pulps and 30 were rejects. The samples were sent to Chemex for gold analysis by fire assay with a gravimetric finish.

11.8 QUALITY ASSURANCE ACTIONS

Sampling procedures and quality assurance have been conducted in conformance with NI 43-101 guidelines. There are no actions or recommendations required on drill data at the time of this report.

11.9 OPINION ON ADEQUACY

The data generated by Bond and Lac is reliable. Samples that were used in the mineral resource were assayed at both Eco-Tech and Chemex. A series of statistical analyses were carried out to compare the Eco-Tech and Chemex results in 1993; no statistical differences were found between them.

In 1993, many of the pulps and rejects from core samples within the Marc, AV and JW zones were sent to Chemex after being assayed by Eco-Tech. A much smaller number were sent in 1994. In general, there is a good correlation between results from the two labs. However, on average, Eco-Tech tends to have slightly higher gold results in ore material than Chemex (1 to 3%). No apparent reason could be found for the difference, and it is still uncertain which lab is giving the more accurate results. When the standards created from Marc Zone material were sent for round robin assaying to various labs, Eco-Tech did tend to have a slightly higher average than most other labs but the significance to the resource estimation is considered immaterial.

11.10 ANALYTICAL PROCEDURES

Most sample preparation and assaying in 1993 and 1994 was done in a lab in Stewart owned by Eco-Tech Labs of Kamloops, BC. Using a lab in Stewart reduced sample freight costs and provided security of sample delivery. Chemex Labs in Vancouver was the main check assay lab.

11.10.1 1993 - Assaying

In 1993, sample preparation and gold and silver assays were performed by Eco-Tech in Stewart. Samples were dried, crushed to -10 mesh and an approximately 350 g split taken using a riffle splitter. The split was pulverised to 85% -140 mesh prior to assaying.

All samples were assayed for gold using a conventional 1 AT (30 g) assay with an AA finish. Samples that contained more than 10 g/t Au were reassayed using gravimetric procedures, and samples containing more than 30 g/t Au were reassayed using a metallic procedure. Some less than 30 g/t samples were analyzed by metallic procedures as well. In metallic assays, the entire 250 to 400 g pulp was screened with the +150 mesh fraction assayed in its entirety for gold. The gold content of the coarse fraction was then weight averaged with assay results from the –150 fraction. Duplicate assays were performed on the -150 fraction to ensure sample homogeneity.

Silver was assayed by digestion with aqua regia of a 2 g sample followed by atomic absorption analysis. Samples at Eco-Tech were processed in batches of 40. Each batch included as a minimum the following quality control insertions:

- One reagent blank
- One in house (or CANMET) standard inserted by the lab
- One blind repeat inserted in the bucking room
- One repeat from a batch of samples analyzed during a previous shift.

Quality control data is included with most of the sample assay certificates.

Lac instituted a QA/QC program that consisted of:

- One standard was inserted in every 10 samples.
- For every 1 in 10 samples, the sample pulp was sent to Chemex
- For every 1 in 20 samples, the coarse reject was sent to Chemex.

In addition, the rejects for most samples from ore zones were sent to Chemex for reassay.

CANMET standards were used for standard material by Lac. Approximately 30 to 40 grams were sent per sample. The CANMET standards used were:

Gold ore MA-2b .... ± 2.39 g/t
Gold ore MA-3 .... ± 7.49 g/t
Gold ore MA-1b .... ± 17.0 g/t

11.10.2 1993 - Other Analysis

All samples analyzed for gold were also subjected to ICP analysis. In 1993, ICP analyses were performed in Eco-Tech's lab in Kamloops on a split from the assay pulp. Chemex also did an ICP analysis on samples sent to Vancouver. Whole rock analyses were done by X-RAL Labs in Don Mills, Ontario on a separate pulp portion.

Specific gravity determinations were made by Eco-Tech in Stewart for core samples that were visually thought to be in the ore zone. Halved pieces of core were used, before samples were crushed for sample preparation. Determinations were made by weighing the piece of core in air and then immersed in water. A deep fryer basket was used to immerse the sample. Generally, the larger pieces of core in a sample were used for specific gravity determinations.

11.10.3 1994 - Assaying

In 1994, Eco-Tech in Stewart was the primary sample preparation and assay lab. Sample preparation and assaying continued to be done in separate buildings. Sample preparation, gold assay and lab QA/QC procedures (excepting repeats and duplicates) were the same as in 1993. A comparison between silver assay results and silver in ICP results in 1993 showed no statistical difference. In 1994, samples that contained greater than 30 g/t Ag in the ICP were assayed for silver.

Additional sampling of 1993 core was undertaken in the early part of 1994 when holes were relogged by LAC. Few, if any, results from this work will be incorporated in an ore resource.

When drilling recommenced in April 1994, a more stringent standard insertion program was instituted. Four standards were created by CDN Resource Laboratories of Delta, BC, using material from the Marc Zone bulk samples. Material was crushed, pulverised to –200 mesh and then homogenised. 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.

Acceptable levels for the standards, based on the round-robin analysis, are presented in the table below.

Table 11.3: Acceptable Assay Standard Levels

Standard Number	Mean -2 Std. Dev.	Mean g/t Au	Mean +2 Std. Dev.
1	1.59	1.90	2.20
2	2.85	3.19	3.53
3	5.68	6.34	7.00
4	13.86	14.15	15.43

Source: NAMC (2001)

One standard was inserted for every 20 samples. Analytical results were tracked and if standard results were out of acceptable limits, the lab was asked to reassay the samples that were analyzed in the same batch as the standard.

In addition to a systematic standard program, at least 1 in 40 samples had 2 assay splits from the coarse (-10 mesh) sample taken and 1 in 40 samples had a duplicate assay done on the assay pulp. This replaced Eco-Tech's sample repeat procedure used in 1993. When a duplicate assay was done 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 (resplit), the assay from the original pulp was given as the sample result with the resplit result at the end of the certificate.

The Stewart assay lab was closed in October 1994 and thereafter assaying was done in Eco-Tech's Kamloops, BC lab. Sample preparation remained in Stewart.

11.10.4 1994 - Other Analysis

In 1994, ICP analysis was done by Eco-Tech in Kamloops. Whole rocks were done by Chemex using XRF. Comparisons between Chemex and X-Ral showed that Chemex had similar analytical quality. Specific gravity was done by Eco-Tech using the same procedures as in 1993.

11.11 CONCLUSIONS & RECOMMENDATIONS

The data generated by Lac in 1993 and 1994 is reliable. Samples used in the mineral resource were assayed at both Eco-Tech and Chemex. A series of statistical analysis to compare the Eco-Tech and Chemex results in 1993 was completed and results could find no statistical differences between them.

In 1993, many of the pulps and rejects from core samples within the Marc, AV and JW zones were sent to Chemex after being assayed by Eco-Tech. A much smaller number were sent in 1994. In general, there is a good correlation between results from the two labs. However, on average, Eco-Tech tends to have slightly higher gold results in ore material than Chemex (1 to 3%). Lac addressed this situation and no apparent reason could be found for the difference, and it is still uncertain which lab is giving the more accurate results. When the standards created from Marc Zone material were sent for round robin assaying to various labs, Eco-Tech did tend to have a slightly higher average than most other labs but the variance is considered immaterial to the accuracy of the resource estimate.

A full QA/QC report is described in Dunham L. Craig, P.Geo., Red Mountain Project Stewart, B.C. Canada 2001 Resource Estimate, May 2001 (North American Metal Corp).

12.0 DATA VERIFICATION

Data verification was completed by Bond, LAC and NAMC. In 2000, NAMC cross-referenced and catalogued all data from previous operators. Data that could not be verified were removed from the database. In addition, NAMC conducted metallurgical composites from drill core and cross-referenced the assay results to the original Bond and LAC data.

12.1 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 pulverised to >90% -150 mesh. The pulps were then sent to IPL Laboratories in Vancouver, B.C. 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 devise.

Table 12.1: DDH Composite Assays vs. Metallurgical Composites

Metallurgical Composite	DDH Comp Average Au g/t	Met Comp Average Au g/t	DDH Comp Average Ag g/t	Met Comp Average Ag g/t
Composite 1 – Section 1220	9.03	8.60	26.17	28.0
Composite 2 - Section 1200	7.77	8.14	52.8	62.3
Composite 3 - Section 1100	8.99	8.31	44.6	45.7
Stage 2 - Marc Zone	13.51	12.87	24.0	51.4
Stage 2 - AV Zone	16.8	14.84	16.0	22.0

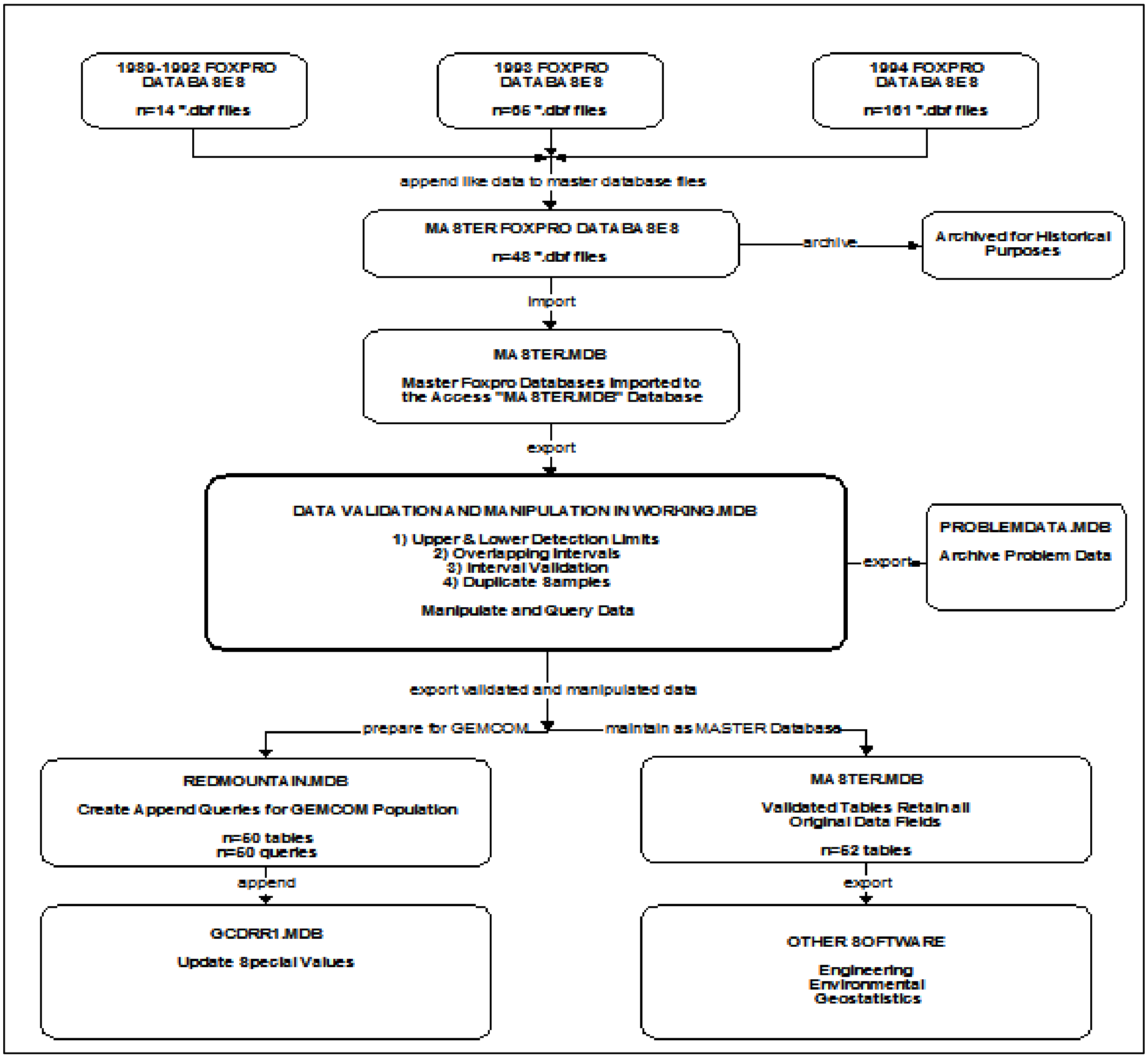
Source: NAMC (2001)

12.2 ELECTRONIC DATA VERIFICATION

Lac Minerals Inc. collected and organised over one gigabyte of electronic information during 1993-1994. As the project was under fast track conditions by LAC management, the data were never compiled into a cohesive database that was accessible by a single program. NAMC, upon receiving the data from Price Waterhouse Coopers, undertook to create a Microsoft Access database that held all of the site exploration and environmental work.

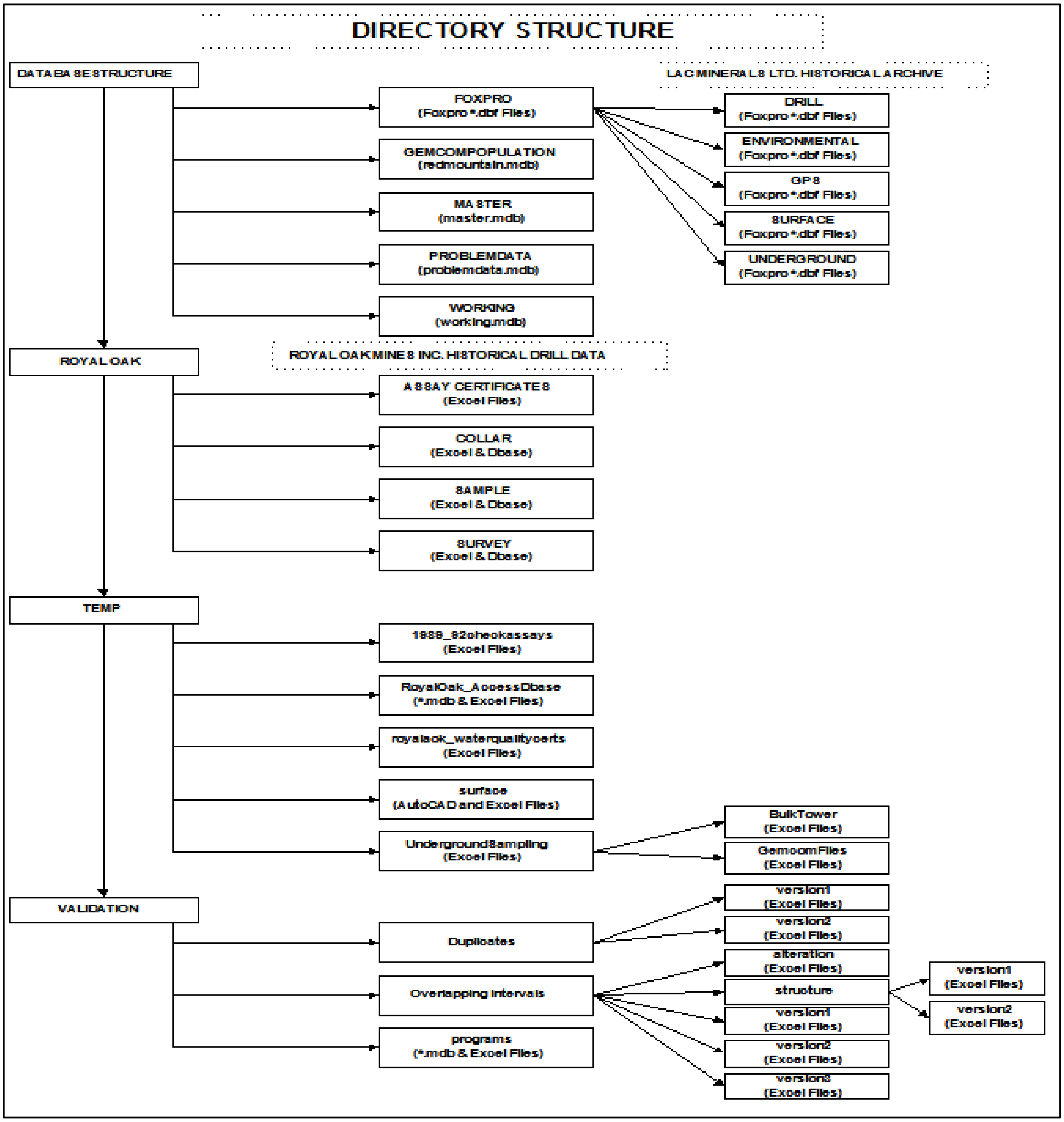
During 2000, NAMC hired Adrian Bray, who worked on the project for both Bond and LAC, to assist in compiling the database and perform queries for exactness and correctness. Items under question were researched to the original paper copies for correctness.

Figure 12-1: Data Validation Flowchart



Source: NAMC (2001)

Figure 12-2: Directory Structure



Source: NAMC (2001)

12.3 DRILL ASSAY CROSS VALIDATION

During 1994, four short drill holes were drilled on section 1275 N from collar points 1 m apart in a square pattern. The two primary objectives of the test were (1) to saw the core in half and assay both sides of the core to test variance in the selection of core halves, and (2) to test for variance within the stockwork zone over a spatial difference of 1 m. Table 12.2 summarises the weighted assay averages for the higher-grade intervals in the four drill holes from 13 to 29 m.

Table 12.2: 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

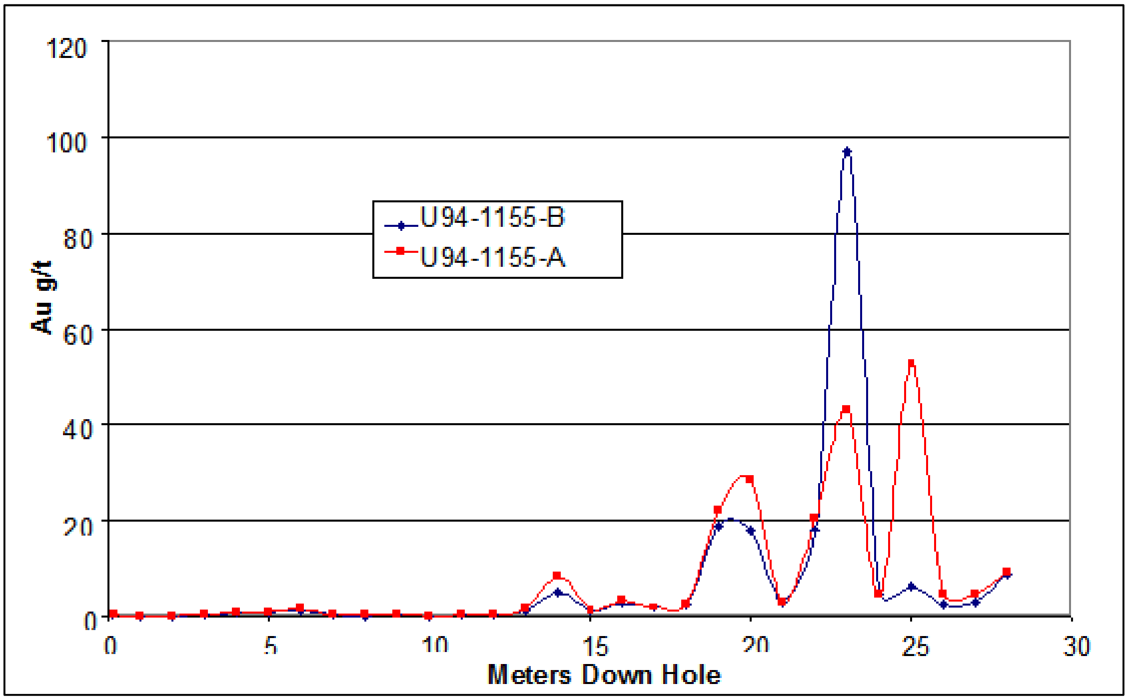
Source: NAMC (2001)

Figures 12-3 to 12-5 display the down hole assay comparisons for each half of the core for holes U94-1155 to U94-1157. Figure 12-6 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 comparison is relatively normal for a gold deposit and affects almost all ranges of assays. This would be expected in the Red Mountain style of stockwork. Stereonet analysis of the stockwork veining show that only 20% of the veins have a consistent trend within the stockwork envelope (Barclay, 2000) with the balance being relatively random. This randomness and rapid thickening and thinning over sub-metre and sub-centimetre distances was observed in both core and cross cut and is an explanation for the variance in grade as gold is associated with pyrite systems.

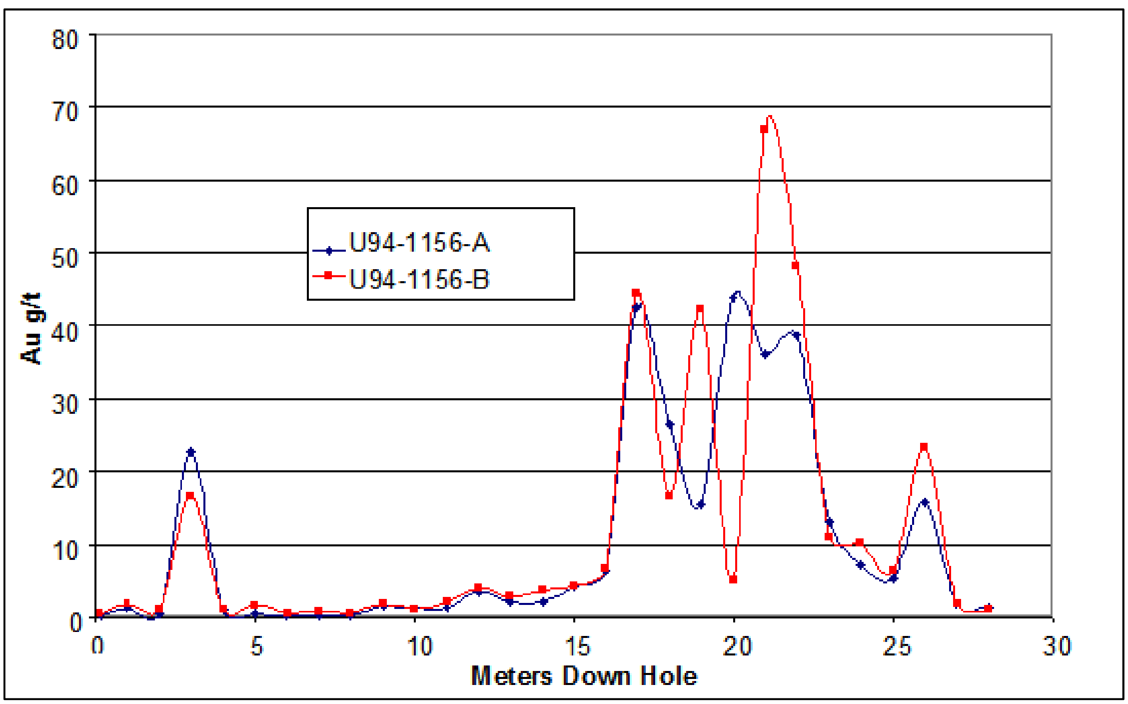
This variance is also evident in the four individual drill holes (Figure 12-3 to Figure 12-5) and is a product of the same effect. Smoothing of data by compositing removes the inherent volatility of assay data and variance of the smoothed data in Table 12.2 is well behaved.

Figure 12-3: U94-1155 Au Assays on Both Halves of Core



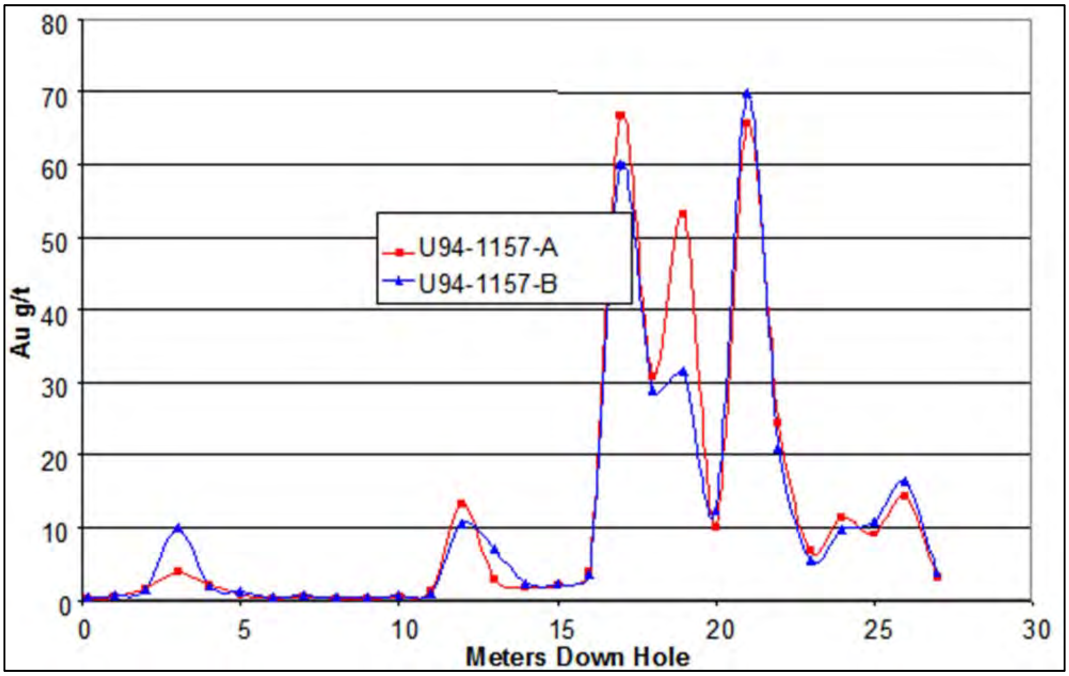
Source: NAMC (2001)

Figure 12-4: U94-1156 Au Assays on Both Halves of Core



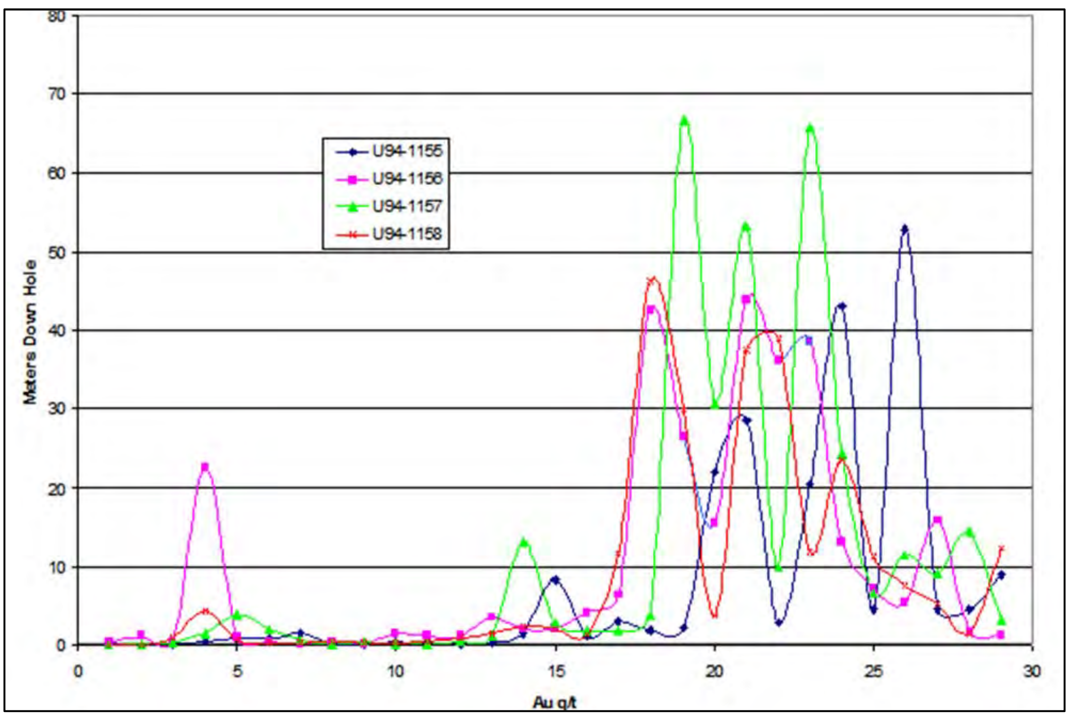
Source: NAMC (2001)

Figure 12-5: U94-1157 Au Assay of Both Halves



Source: NAMC (2001)

Figure 12-6: Au Assay Comparison for DDH U94-1155, -1156, -1157 & -1158



Source: NAMC (2001)

13.0 MINERAL PROCESSING & METALLURGICAL TESTING

13.1 TESTING & PROCEDURES

Previous metallurgical testing was performed by Lakefield Research (1991), Brenda Process Research (1994) and International Metallurgical and Environmental (1997), a derivative of Brenda Process Research. 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, another test program was conducted at Process Research Associates (PRA) under the direction of Dr. Morris Beattie, P.Eng.

All of the metallurgical test procedures are documented in reports appended to this report.

13.1.1 Metallurgical Testing Lakefield 1991

Initial test work by Lakefield Research appears to have been conducted on a composite of material from the Marc Zone. The head assays were as follows:

Au .... 12.8 g/t
Ag .... 46 g/t
Te .... 34 ppm
As .... 450 ppm
Fe 9.72%

Cyanidation testing indicated that the gold and silver recovery were both grind sensitive across the range from 80% passing 93 µm down to 23 µm. Across this range, the gold extraction increased from 76.8% to 87.1% and the silver extraction increased from 74.2% to 90.1%. Extraction was not improved by increasing the cyanide level above 0.5 g/L nor by increasing the pH above 12. At the fine grind, the cyanide consumption was reported to be 2.24 kg/t.

Lakefield performed one flotation test at a relatively coarse grind of 80% passing 140 µm. They recovered 92.3% of the gold and 90.5% pf the silver to a rougher concentrate representing 36.7% of the feed weight and containing 31.8 g/t Au and 112 g/t Ag. Attempts to upgrade the concentrate by cleaning resulted in a dramatic loss in recovery.

13.1.2 Brenda Process Technology 1994 - 1997

A significant program of testing was performed under Brenda Process Technology from 1994-1997. The program focused on cyanide leaching, but flotation and leaching of the flotation concentrates were also examined.

Initially, a series of 12 variability style composites were tested. Most to the variability samples originated from the Marc zone. Several larger composites of the Marc, AV and JW zones were constructed for more detailed flow sheet development work. The majority of the samples tested originated from drill core or drill core rejects. One large bulk sample was also tested. The composites constructed were well documented, identifying drill hole numbers and interval lengths. The grade of the samples tested covered a range from 3.07 g/t Au to 63.3 g/t Au. Table 13.1 displays the composite identification and the head assay data.

Table 13.1: Composites Tested in Brenda Process Technology Test Program

Identification	Zone	Au Grade (g/t)	Ag Grade (g/t)
Avg Grade	Marc	11	43.5
Avg Grade	AV	9.9	29.5
High Grade	Marc	32.2	57
Low Grade	Marc	3.1	19
Telluride Rich	?	57.9	347.5
High Arsenic	Marc	11.6	57.5
Low Arsenic	Marc	8	32
Low Antimony	Marc	10.8	9
High Zinc	Marc	7.9	56
Low Zinc	Marc	16.6	51
Black Bedded	?	8.2	22
Tetrahedrite	Marc	11.8	48
Marc Composite	Marc	12.4	48
AV Composite	AV	7.9	25
JV Composite	JW	10.7	21

Note: Details of the composite construction can be found in the original Brenda Process Technology Report – Appendix 1.

The twelve variability samples were tested using a cyanide bottle roll test. The conditions for the tests were standardized at 48 hours leaching with 1 g/L NaCN and a pH of 10.5 to 11.0. The nominal primary grind sizes of the tests were 90% passing 200 mesh (75 µm). The tetrahedrite sample achieved the lowest extraction but this may have been due to increased copper dissolution and cyanide consumption by the sample. Of the remaining samples, the extraction ranged from 83% to 94.3%. The telluride rich sample achieved an extraction of 93.5%. Table 13.2 displays the results of the initial tests. Tests 113 to 117 were replicate tests and used finer primary grind and in some cases higher pH. The alteration of conditions in tests 113 to 117 only resulted in minor improvements in metallurgical performance.

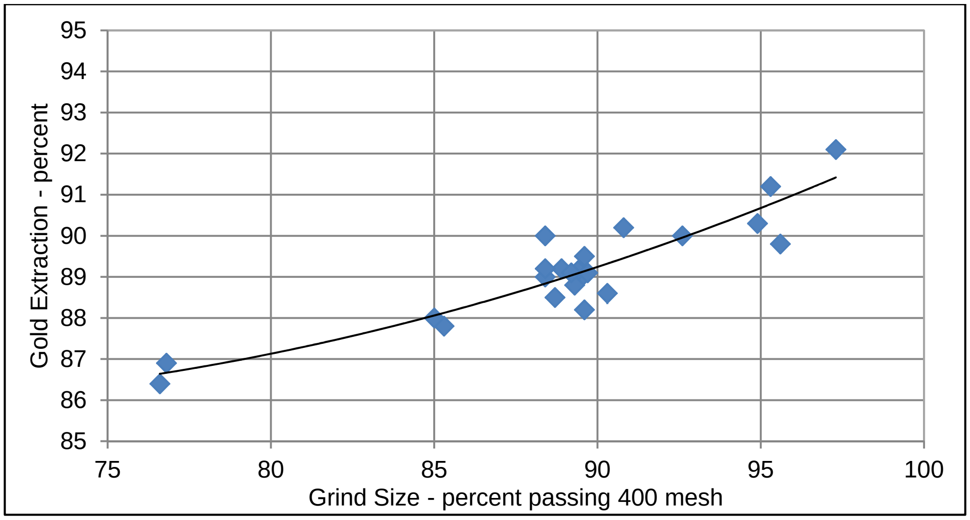
Table 13.2: Variability Cyanide Leach Results

			Feed Grade (g/t)		Extraction (%)	
Test	**Lithology**	**Zone**	**Au**	**Ag**	**Au**	**Ag**
101	Avg Grade	Marc	11.2	43	83.4	74.2
102	Avg Grade	AV	10.6	30	81.1	70.1
103	High Grade	Marc	32.2	57	91.5	86.1
104	Low Grade	Marc	3.1	19	83.7	67.7
105	Telluride Rich		63.3	365	93.5	89.7
106	High Arsenic	Marc	12.4	60	83.1	78.4
107	Low Arsenic	Marc	8	32	94.4	72
108	Low Antimony	Marc	10.8	9	92.6	78.2
109	High Zinc	Marc	7.9	56	94.3	83.8
110	Low Zinc	Marc	16.6	51	89.8	84.2
111	Black Bedded		8.2	22	90.2	81.7
112	Tetrahedrite	Marc	11.7	48	78.6	53.7
113	Avg Grade	Marc	10.8	44	86.1	79.7
114	Avg Grade	AV	9.2	29	82.7	72.6
115	High Arsenic	Marc	10.8	55	84.2	81.8
116	Telluride Rich		52.5	330	93.5	87
117	Tetrahedrite	Marc	11.9	53	77.3	66

There was no noted relationship between leach extraction and feed grade for either gold or silver. The average cyanide and lime consumptions for the tests were 1.3 and 3.0 kg/t of cyanide and lime, respectively.

Detailed test work was conducted on Marc Zone composite sample prepared mostly from drill holes completed during 1993. The composite was shown to be sensitive to grind across the range of 75% to 95% passing 400 mesh (37 µm) with the gold extraction being increased about 6% at the finest grind compared to the coarsest. Figure 13-1 displays the effect of grind size on extraction rate for the Marc Composite. An increase in gold extraction was noted with increased cyanide concentration but this was accompanied by an increase in cyanide consumption. The average cyanide consumption was 1.2 kg/t and the average lime consumption was 1.4 kg/t.

Figure 13-1: Effect of Grind Size on Gold Extraction – Marc Zone

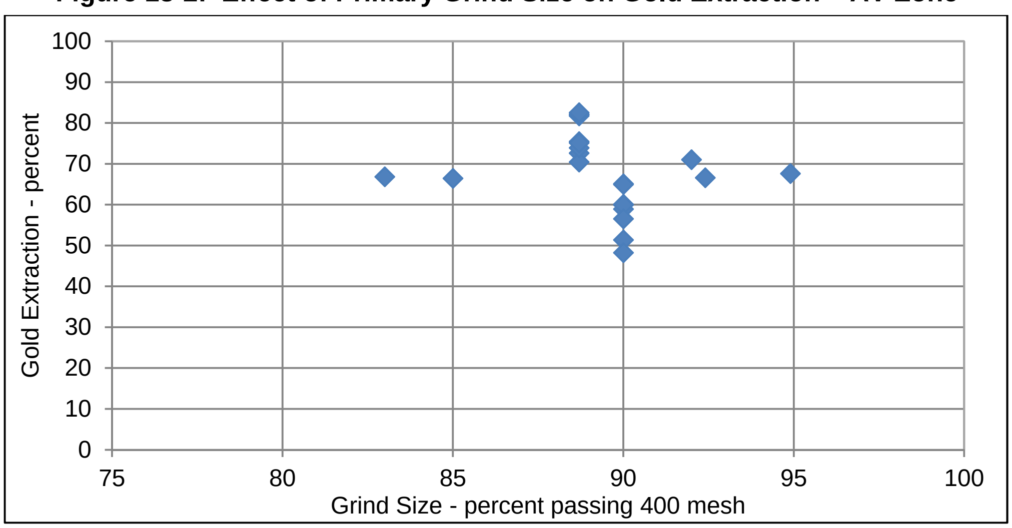


Note: Data taken from Brenda Process Technology Report tests 124-137, 140, 141, 144-157.

The extraction of silver for the Marc composite average 85% and showed little variation to the parameters tested.

Cyanide leach tests were conducted on the composite from the AV Zone. This sample responded very differently from the Marc Zone sample. The sample showed no grind sensitivity and resulted in a gold extraction of 68% under the standard conditions. Figure 13-2 displays the effect of primary grind on gold extraction.

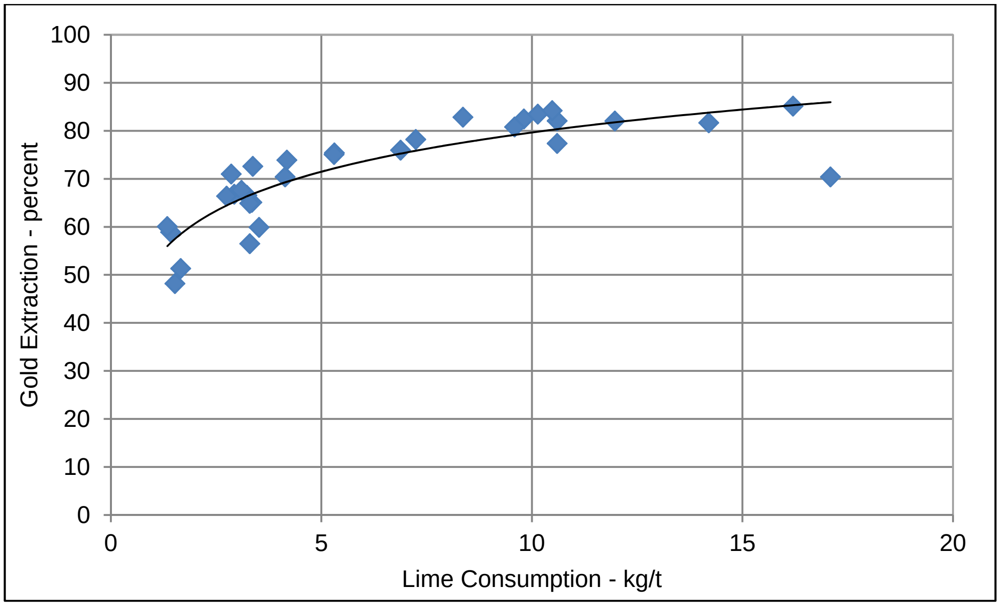
Figure 13-2: Effect of Primary Grind Size on Gold Extraction – AV Zone



Note: Data taken from Brenda Process Technology Report tests 164-187, 196-204.

The use of a higher cyanide concentration, oxygen enrichment, pH greater than 12 and the addition of activated carbon to the leach improved the gold extraction to 82.5%. The more aggressive test conditions resulted in a cyanide consumption of 2.5 kg/t. Figure 13-3 displays the relationship between lime consumption and gold extraction.

Figure 13-3: Relationship Between Lime Consumption & Gold Extraction – AV Composite



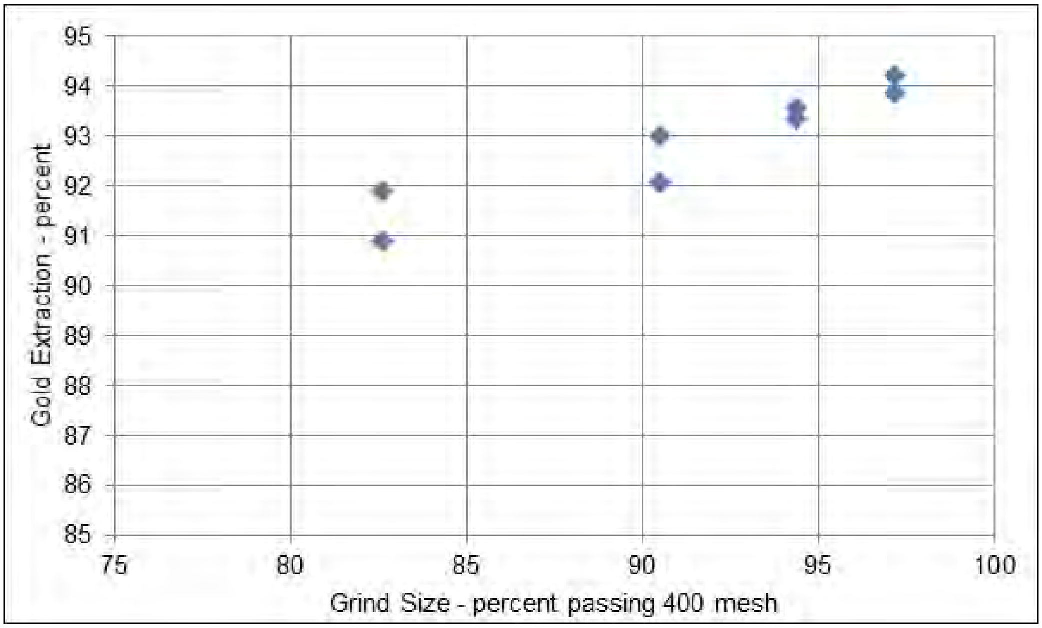
Note: Data taken from Brenda Process Technology Report tests 164-187, 196-204.

Similarly, silver extractions for the AV composite ranged from 56% to 77%. The conditions that improved gold recovery for this sample, also improved the silver extraction rates. The average silver extraction rate at the altered conditions was 71%.

The Brenda Process Report author reached the following conclusion: the reason for the different response was that the Marc Zone and AV samples were differing gold mineralogy. The Marc composite contained a high proportion of native gold while the AV zone contained a greater proportion of gold-tellurides alloys. Further investigation would be confirmed this hypothesis. It would also be prudent to consider other factors like the presence of copper sulphosalts and organic carbon as potential causes of poor performance. Exact sample locations need to be reviewed to establish whether the observed differences in extraction are the result of localized zoning rather than absolute differences between the major zones. It may not be appropriate to apply the characteristics of single composites to entire zones.

The JW zone, which was subjected to a similar set of standard tests, produced the best gold extraction results of all the zone composite samples. Gold in the feed was between 90.9% and 94.2% extracted using the standard conditions. As with the Marc zone, gold extraction increased as the primary grind size became finer. Figure 13-4 displays the relationship between primary grind size and gold extraction.

Figure 13-4: Effect of Primary Grind Size on Gold Extraction – JW Zone



Note: Data taken from Brenda Process Technology Report tests 188-195

Silver extractions for the JW zone averaged 86% and showed practically no variance to the parameters investigated during testing. The average consumption of cyanide and lime was 2.0 and 2.7 kg/t, respectively. The tests were completed at a high cyanide concentration (1,000 ppm).

Testing was conducted on ancillary processes such as thickening of the slurry, and gold adsorption on activated carbon. This test work did not encounter any complications and provided design parameters that would be considered normal. Cyanide destruction test work was also completed for the INCO SO_2/air process.

During 1997, IME conducted additional tests on two composite samples to establish the effectiveness of sulphur reduction of the tailings by flotation of all the contained sulfides. The samples graded 5 g/t Au and 13.6% S and 3.9 g/t Au and 10.6% S. The samples were subjected to cyanidation followed by cyanide destruction and then bulk sulfide flotation. The gold extraction in these samples was in the range of 77% to 79%. The low extraction was attributed to the low head grade.

Flotation of the detoxified tailings was successful in producing a final tailing with a positive NNP and a sulphur content of 0.2% to 0.3% sulphur. The concentrate had a mass representing 30% of the feed weight. The samples tested had a sulphur content somewhat higher than the average sulphur content for the deposit.

13.1.3 Metallurgical Testing PRA 2000

Five composites were tested in the program representing samples from the Marc and AV zones. The majority of the meaningful testing was completed on two global composites representing the Marc and AV zones. The global composites were constructed from quartered drill core and were constructed from seven and five drill holes from the Marc and AV zone, respectively. A summary of the composite head assay data are displayed in Table 13.3.

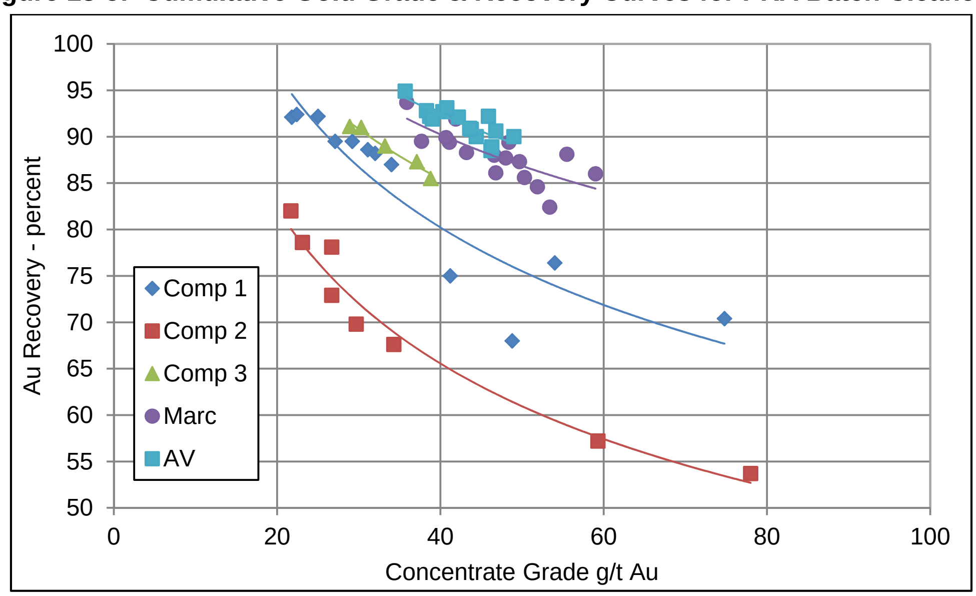
Table 13.3: Head Assay Data for PRA Test Composites

Composite	Assay – % or g/t		
	Au	Ag	S
Composite 1	8.6	28	11.5
Composite 2	8.1	62	7.8
Composite 3	8.3	46	8.8
Marc	**12.9**	**51**	**10.4**
AV	**14.8**	**22**	**13.6**

Note: All assays are in g/tonne except S, which is in percent.

The process flow sheet objective was to produce a gold- and silver-bearing sulphide concentrate. The concentrate would be marketed and sold on the basis of the gold and silver value. Rougher and flotation tests were conducted to determine the grade and recovery curves for this process. A summary of the flotation grade and recovery data for the batch cleaner tests is shown in Figure 13-5.

Figure 13-5: Cumulative Gold Grade & Recovery Curves for PRA Batch Cleaner Tests

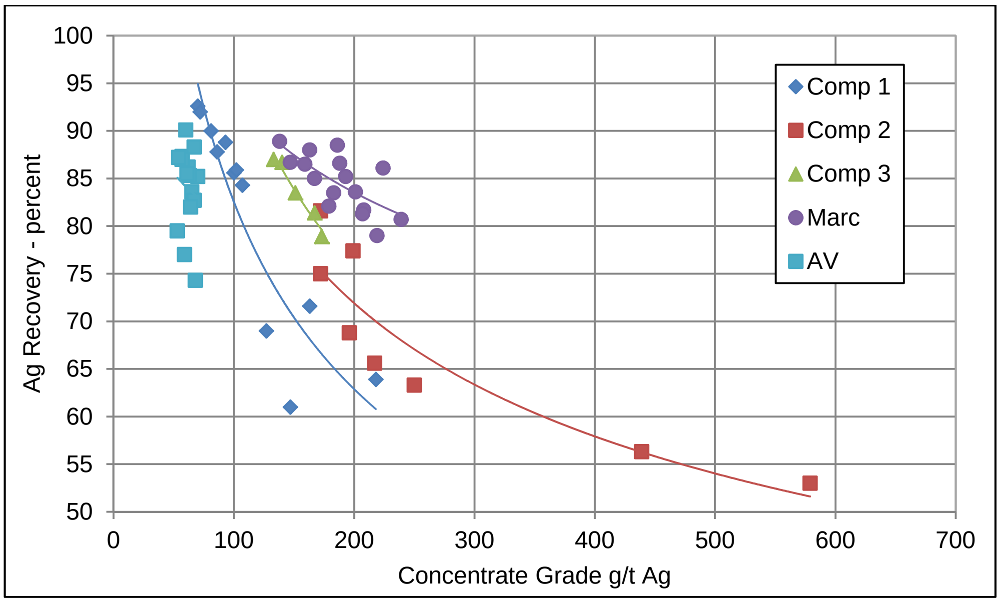


As shown in the graph, the concentrate grades in the samples ranged from about 20 to 80 g/t. Each sample displayed an inversely proportional relationship between gold concentrate grade and recovery. Based on this data set, concentrate gold grade would vary between 25 to 58 g/t at gold recoveries of 90%.

The gold upgrading ratio (the ratio of gold concentrate grade to feed grade) at high gold recovery, ranged between 3.5 and 5 and was a function of the sulphur grades in the feed.

The metallurgical performance of silver was very similar to gold. Figure 13-6 displays the cumulative batch cleaner test results for the composites tested at PRA.

Figure 13-6: Cumulative Silver Grade & Recovery Curves for PRA Batch Cleaner Tests



The data set for silver would indicate that the concentrate silver grade would range between 60 and 200 g/t at a silver recovery of 85%.

Bond Ball mill tests performed on composite samples indicated an average energy consumption of 18.5 kWh/t.

Thickener settling tests were performed on flotation tailings ground to a nominal size of 175 µm K_{80}. The thickener area requirement averaged 0.14 m^2/t/d.

13.2 RELEVANT RESULTS

A cyanide leach process was selected as the best method to extract gold and silver from Red Mountain ore. While this process has higher capital and operating cost, the production of gold and silver bearing flotation concentrates for sale was less economically favourable due to the lower metal recoveries and high cost of transportation and marketing low grade concentrates.

The mill process design and metallurgical projections were based on the test work completed at Brenda Process Technology from 1994 and 1997. Testing was performed on three composites: the Marc zone, the AV zone and JW zone. The metallurgical response of the samples was different by zone, therefore metallurgical performance estimates were based on these initial zone composite response characteristics.

The comminution data was very limited. The crushing circuit sizing was based on other similar operations of equivalent tonnage. These data were cross-referenced with nominal nameplate ratings with equipment selected for the capital cost estimates. The rod and ball mill grinding circuit sizing was based on Bond ball mill determinations (19 kWh/t), cross-referenced with equipment suppliers nameplate ratings. The target grind size of 95% passing 400 mesh or 38 µm would be on the finest end of the operating range for this type of equipment. Additional crushing and grinding tests to more accurately assess the grinding energy requirements are recommended when the project moves to the next engineering phase.

Finally, the thickener performance data was based on the settling tests performed at Brenda Process Technology and industry standard factors. Due to the very fine grind size, reported sericite mineralization and slow settling rates, a factor of 0.7 $t/m^3/d$. Ultimate densities from the settling tests were used to estimate thickener underflow targets of 45% solids by weight.

13.3 RECOVERY ESTIMATE ASSUMPTIONS

The recovery estimate assumptions were based on the Brenda Process Technology testing completed between 1994 and 1997. The grind sensitivity of the Marc and JW zones was taken into consideration and a grind size of 95% passing 400 mesh or 38 µm was used for recovery estimates. The recoveries used for each zone were estimated from the composite samples. Table 13.4 displays the recovery estimates and reagent consumptions assumed for the economic projections.

Recovery data for the 48-hour leach test results were used and the leach circuit were design on a nominal 48-hour residence time.

Table 13.4: Recovery & Reagent Consumption Assumptions

Recovery	Au %	Ag %	NaCN kg/t	Lime kg/t
Marc	90.5	85.0	0.8	1.4
AV	82.0	71.0	0.8	10
JW	93.0	86.0	0.8	2.7

14.0 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

In 2014, JDS received the NI 43-101 2001 Red Mountain Resource Estimate completed by Dunham Craig, P.Geo. for Seabridge both in report form and in a Gemcom software electronic database. The database was verified as accurate and the resource was re-estimated using the same parameters as 2001 to verify the accuracy of the 2001 resource and geological model. In 2014, the resource and model were deemed accurate to the model produced in 2001 through cross-validation using Gemcom software.

One change was made in the 2014 resource estimate: 2001 tonnage was previously estimated at a specific gravity (SG) of 3.0 for all resources. In 2014, tonnage was estimated using a SG of 3.0 for resource blocks grading > 5 g/t and a SG of 2.89 for resource blocks grading < 5 g/t.

The 2014 Red Mountain Resource is stated in Table 14.1.

Table 14.1: Mineral Resource Statement at a 3 g/t Cut-off Grade

Zone	Tonnage (tonnes)	In-situ Gold Grade (g/t)	In-situ Silver Grade (g/t)	Contained Gold (troy oz)	Contained Silver (troy oz)
Marc Zone					
Measured	651,600	9.26	40.06	194,000	839,215
Indicated	10,800	9.71	30.33	3,400	10,477
Inferred	0	0.00	0.00	0	0
AV Zone					
Measured	508,200	7.14	20.88	116,700	341,202
Indicated	283,800	7.32	21.03	66,800	191,935
Inferred	1,800	10.96	39.50	600	2,308
JW Zone					
Measured	-	-	-	-	-
Indicated	-	-	-	-	-
Inferred	331,100	7.67	12.57	81,600	133,900
Total Measured & Indicated	**1,454,300**	**8.15**	**29.57**	**380,900**	**1,382,800**
Total Inferred	**332,900**	**7.69**	**12.72**	**82,300**	**136,200**

Source: JDS (2014)

In-situ resource blocks within the Marc, AV and JW zones are broken out in the grade groups shown in Table 14.2.

Table 14.2: Marc, AV & JW Zone Grade Groups

Variable Density (>5 g/t = 3.0 <5 g/t = 2.89)						
ZONE	**Grade Group**	**Volume km³**	**Density t/m³**	**Tonnage kt**	**Au Grade g/t**	**Ag Grade g/t**
MARC	**>30 g/t**	0.384	3.000	1.152	31.66	156.77
	>25 g/t	3.269	3.000	9.807	27.53	113.15
	>20 g/t	10.049	3.000	30.147	23.95	91.02
	>15 g/t	28.146	3.000	84.438	19.46	69.72
	>10 g/t	75.079	3.000	225.238	14.85	54.56
	>9 g/t	91.878	3.000	275.633	13.87	51.18
	>8 g/t	110.651	3.000	331.952	12.96	48.78
	>7 g/t	132.185	3.000	396.556	12.07	46.17
	>6 g/t	153.799	3.000	461.398	11.28	44.23
	>5 g/t	178.869	3.000	536.608	10.47	42.29
	>4 g/t	202.828	2.987	605.849	9.79	40.93
	>3 g/t	222.105	2.979	661.560	9.26	40.00
	>2 g/t	231.953	2.975	690.019	8.98	39.73
	>1 g/t	234.026	2.974	696.011	8.92	39.62
	>0 g/t	234.278	2.974	696.738	8.91	39.57
	Total	**234.278**	**2.974**	**696.738**	**8.91**	**39.57**
Variable Density (>5 g/t = 3.0 <5 g/t = 2.89)						
ZONE	**Grade Group**	**Volume km³**	**Density t/m³**	**Tonnage kt**	**Au Grade g/t**	**Ag Grade g/t**
AV	**>30 g/t**	0.000	0.000	0.000	0.00	0.00
	>25 g/t	0.263	3.000	0.789	26.50	15.09
	>20 g/t	3.009	3.000	9.028	22.14	16.44
	>15 g/t	13.208	3.000	39.625	18.20	19.38
	>10 g/t	47.347	3.000	142.042	13.69	19.74
	>9 g/t	60.880	3.000	182.640	12.75	19.63
	>8 g/t	80.487	3.000	241.462	11.71	19.45
	>7 g/t	105.936	3.000	317.809	10.69	18.90
	>6 g/t	137.698	3.000	413.094	9.72	18.79
	>5 g/t	179.639	3.000	538.916	8.73	18.96
	>4 g/t	223.972	2.978	667.039	7.92	19.71
	>3 g/t	267.839	2.964	793.816	7.22	20.98
	>2 g/t	284.751	2.959	842.690	6.95	21.33
	>1 g/t	286.848	2.959	848.752	6.91	21.28
	>0 g/t	286.848	2.959	848.752	6.91	21.28
	Total	**286.848**	**2.959**	**848.752**	**6.91**	**21.28**

Variable Density (>5 g/t = 3.0 <5 g/t = 2.89)						
ZONE	**Grade**	**Volume**	**Density**	**Tonnage**	**Au Grade**	**Ag Grade**
JW	**Group**	**km³**	**t/m³**	**kt**	**g/t**	**g/t**
	>30 g/t	0.000	0.000	0.000	0.00	0.00
	>25 g/t	0.152	3.000	0.457	26.98	5.55
	>20 g/t	0.940	3.000	2.819	22.02	10.38
	>15 g/t	9.412	3.000	28.236	17.46	14.63
	>10 g/t	26.360	3.000	79.080	14.10	14.05
	>9 g/t	30.955	3.000	92.865	13.42	13.82
	>8 g/t	36.557	3.000	109.672	12.67	13.77
	>7 g/t	44.039	3.000	132.117	11.79	14.37
	>6 g/t	54.321	3.000	162.962	10.78	14.47
	>5 g/t	76.351	3.000	229.052	9.22	13.46
	>4 g/t	98.329	2.975	292.569	8.21	12.78
	>3 g/t	111.666	2.965	331.112	7.67	12.57
	>2 g/t	113.825	2.964	337.354	7.58	12.39
	>1 g/t	115.389	2.963	341.872	7.50	12.26
	>0 g/t	115.389	2.963	341.872	7.50	12.26
	Total	**115.389**	**2.963**	**341.872**	**7.50**	**12.26**

Source: JDS (2014)

This section summarises the procedure used by JDS to calculate the resource for three mineralised zones deemed potentially mineable in the Red Mountain deposit: the Marc, AV and JW zones.

From the period February 2000 to February 2001, the previous owners of the Red Mountain project, NAMC, focused on creating a resource estimate to be used in a future reserve estimate. Core within a 20 m envelope of the current resource was re-logged in 206 drill holes. QA/QC was checked and compiled for the drill database. Historical electronic data were compiled into a MS Access database and validated. Outside consultants were brought in to assist in ore genesis modelling, structural modelling, regional geology and geostatistics.

14.2 DESIGN OF MODELLING CRITERIA

To establish geologically sound modelling criteria, a significant amount of time and effort was invested during the 2000 field season into detailed investigations of the geology and mineralisation at Red Mountain, prior to the relogging of any drill core. Areas of investigation included general lithology, nature of sulphide occurrences, relationship of pyrite to gold grade and structural control on mineralisation.

The results of the studies outlined above suggest that the following are important modelling criteria:

1. Basic lithology, including major structural features, with appropriate textural modifiers.
2. The limits of pyrite, and more rarely pyrrhotite, stockworking. These limits are often, but not always coincident with a 1 g/t gold assay outline. Inside this outline, sulphide occurs as disseminations, microveinlets, 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 microveinlets 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.

14.3 DATA COLLECTION

Diamond drill core from 206 diamond drill holes that intersect the Marc, AV and JW zones were relogged by NAMC staff during the 2000 field season at the Red Mountain project offices in Stewart. For each hole, an interval of core was selected for logging that extended approximately 20 m into both the hangingwall and footwall. A total of 12,830.72 m were relogged.

14.4 SOLID MODELLING

Three-dimensional solids were created for both the geologic units and the pyrite stockwork 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.
- Geology and stockwork outlines were drawn onto respective sets of sections using the guidelines described below.

14.4.1 Stockwork Outlines

The stockwork outlines were based on the limits of pyrite and pyrrhotite. The limit of the stockwork is very abrupt in some places and gradational into the wall rock in others. If the latter was the case, a subjective call was made as to where the limit was based on sulphide vein density. The stockwork outlines often, but not always, corresponded to areas of intense quartz sericite alteration that give the rock a bleached appearance.

The following areas were excluded from the stockwork outlines:

- Areas where no stockworking was noted to be associated with gold values. Although such occurrences are spatially associated with the auriferous sulphide stockworks, they are also very erratic and cannot be modeled with confidence.
- Isolated stockwork zones with no demonstrable continuity. These consist of a drill hole intersection that could not reasonably be joined on section or in plan view to another intersection. As such, they could only be modeled as a small disk or spherically shaped body.
- Single intersections that could only be interpreted as small lobes of stockworking on a single section. These lobes could not be modeled in any direction with confidence.
- The cross-section information was transposed onto level plans that were plotted every 10 m. Both geologic units and stockwork zones were interpreted on the level plans based on sectional information, as well as information from underground and surface mapping where applicable.
- Data on both the cross-sections and the level plans were compared to ensure consistency of the model.
- The geology and stockwork outlines from both the vertical sections and level plans were digitized into 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. The level plans were digitized as closed polylines. The splined vertical section outlines and level plan outlines (both are 3D rings) were then assigned rock codes as follows:

Zone	Rock Code
Marc	101
AV	201
JW	301

- Solids were created from the 3D rings using the method (two sets of rings) available in Gemcom software. Rather than simply stitching the sections together, this method utilises points common to the splined vertical section outlines and the level plan outlines, allowing the level plan interpretation to be honoured. Additional surfaces were created to model known fault structures across which the displacement of geological units and stockwork zones could be demonstrated.

14.4.2 Geology Outlines

For the geology outlines, it was first determined whether the unit being modeled could be interpreted to be continuous between sections or between drill holes on the same sections. If a geologic unit was intersected only in a single drill hole on a section and could not be modeled with intersections on an adjacent section, the width of the intersection was examined. The minimum dimension for inclusion in the model was 4.0 m. Statistical evaluation of the geology and stockwork models indicated that gold and silver mineralisation cross cut geology and host rock type had little or no effect on gold or silver emplacement. Resource interpolation was performed using stockwork outlines only.

14.5 BULK DENSITY

The bulk density of the Red Mountain gold deposits has been tested by two sampling programs. During 1993 and 1994, Lac Minerals had 4,225 specific gravity 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 14.3.

Table 14.3: 1993-1994 Specific Gravity Sample Results

Zone	# Samples	Range of Values	Avg. SG	Pyrite %
All samples	4283	1.44 - 4.12	2.86	N/A
All within 2000 solids	1290	1.85 - 4.04	2.95	6.11
2000 Marc solid	1058	2.03 - 4.04	2.95	6.43
2000 AV solid	194	1.85 - 3.85	2.99	5.83
2000 JW solid	38	2.67 - 3.12	2.90	1.91
2000 solids - > 5.0 g/t Au	667	2.48 - 4.04	3.01	8.66
2000 solids - < 5.0 g/t Au	623	1.85 - 3.58	2.89	3.43

Source (NAMC 2001)

14.6 ASSAY & COMPOSITE STATISTICS

14.6.1 Introduction

The Red Mountain drill hole database consists of the following:

Drill Holes..446
Au Assays..49,771
Ag Assays..39,192
ICP Assays..44,350
Whole Rock Assays..4,665

Of the above data set, 224 drill intersects penetrate the solids used for the resource estimate and are divided into the following zones. A few drill holes intersect the solids more than once, as shown in Table 14.4.

Table 14.4: Drill Intercepts Intersecting Solids

Zone	Drill Intercepts
Marc Zone (101)	169
AV Zone (201)	34
JW Zone (301)	19

Source (NAMC 2001)

14.6.2 Assay Statistics

The original assays, as opposed to subsequent check assays or duplicates, were accepted for the resource calculation. As outlined in Section 12-1, the gold assay technique used was dependent on the value obtained in the initial assay by FA-AA. If the initial result was over 10 g/t, the sample was re-assayed using a gravimetric finish. If the result was over 30 g/t, a metallics assay was performed. For resource estimation purposes, metallics results were given precedence over gravimetric results, which in turn were given precedence over AA results.

Table 14.5 displays the population statistics of the Marc and AV combined drill hole gold and silver assays. These assays are extracted from within the resource estimate solids and are used for preparing composites for the measured and indicated categories. The JW Zone is separate as the zone is considered inferred.

Table 14.5: Summary Assay Statistics for the Marc & AV Zones

Statistic	Au Assays	Ag Assays
Mean	11.33871	46.13685
Standard Error	0.82243	2.723983
Median	4.6	18.6
Mode	1.88	0.05
Standard Deviation	36.99109	118.5793
Sample Variance	1368.341	14061.05
Kurtosis	798.5323	130.4959
Skewness	24.24847	9.602271
Range	1320.65	2151.95
Minimum	0.01	.05
Maximum	1320.66	2152
Sum	22938.21	87429.33
Count	2023	1895
Confidence Level (95%)	1.6129	5.342319

Source (NAMC 2001)

Composites were set at 1.5 m down hole starting at the upper elevation intercept of the solid. This distance was chosen because 28% of the assay interval lengths were greater than 1.5 m (see Table 14.6). Residuals were considered to have a statistically insignificant effect.

Table 14.6: Assay Intervals

Assay Interval Length	# Assays in solids
0 - <0.5 m	10
≥.5 - <1.0	29
≥1.0 - <1.5	1422
≥1.5 - <2.0	562
≥2.0	0

Source (NAMC 2001)

14.6.3 Composite Statistics

The resource model was designed with the concept that the post-mineral fault offsets of the deposit would be reconstructed for interpolation. Therefore, variography and interpolation represents the combined data set for the Marc and AV zones. The JW zone, due to a lack of data, was interpolated as an independent data set.

Gold population characteristics for the Marc and AV zones are very similar, which supports the combining of the data sets. The only significant difference is a decrease in both the mean and mode for the AV zone. The JW zone, due to its lack of data (n = 82) does not have well defined characteristics.

Silver does not correlate well with gold (correlation coefficient = .24). Consequently, silver has had a separate set of variography and population statistics applied to the interpolation.

14.6.4 Top Cut Applied to Composites

Gold values used in the interpolation runs were top cut to 44 g/t Au. This 44 g/t Au top cut was used in the interpolation runs for the Marc, AV and JW 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 the Marc, AV and JW zones.

14.7 GEOSTATISTICS

Dr. A. J. Sinclair, P. Eng, P. Geo., evaluated the semivariogram modelling and top cut statistics for North American Metals Corporation. The following Sections (14.7.1 to 14.7.4) are excerpts and paraphrases from his report entitled, *Summary Report Re. Data Analysis and Semivariogram Modelling (OK) on Marc and AV Zones*.

A summary of geostatistics is provided below:

1. Gold assay distributions for the Marc and AV zones are very nearly lognormal with small amounts (less than 1%) of upper (anomalous) and lower subpopulations.
2. The combined data for Marc and AV zones indicate that an upper subpopulation of anomalous values (greater than a threshold of 44 g/t Au) representing about 0.88% of the data must be treated specially during estimation (perhaps cut to a lower value).
3. Contoured grades for the Marc zone show a tendency for values greater than the average to be oriented preferentially in the direction of greatest geological continuity with the main body of the Marc deposit. Very high-grade values have much less tendency toward anisotropy in their spatial distribution.
4. A semivariogram model has been defined for Au (1.5 m composites) for the Marc zone that is consistent with the available directions for which closely spaced assay data exist (directions of preferential drilling), and with the known/interpreted geological continuity model. A limited number of directional semivariograms for the AV zone are consistent with the Marc model. This model is adequate as a basis for ordinary kriging.
5. Semivariogram models have been obtained for Ag for both the Marc and AV zones.

14.7.1 Data Analysis

Dr. Sinclair examined the data distributions for Marc, AV and combined Marc and AV data sets (1.5 m composites). Each of these data sets is remarkably close to a single lognormal population although when viewed as probability plots, each demonstrates small percentages of additional subpopulations on each tail. The case is well represented by the combined data, shown as a three component, lognormal model in the complete report. The statistics for these three lognormal subpopulations are given in Table 14.7.

Table 14.7: Statistics for a Three-Component Lognormal Model for Combined

Sub Population	Average[1]	(Avg+s)*	(Avg-s)*	Percentage
1	137.70	437.90	43.30	0.88
2	4.88	43.90	0.54	98.42
3	0.07	0.50	0.01	0.70

Source: NAMC (2001). [1] All reported grades are antilogs of the original log-value and thus are in ppm.

For estimation purposes, the lower population (#3) can be ignored. The upper, anomalous population (#1) is conveniently separated from the bulk of data at a threshold of 44 g/t. For estimation purposes, values higher than 44 must be treated specially. One alternative is to cut these high values to a lower value.

Dr. Sinclair also examined contoured plots of values in order to understand the spatial distribution of values. For the Marc zone, high values tend to cluster in small groups, most of which are well within the mass of data but a few of which are marginal to the data concentration. There appears to be a general trend of above-average values along the general length of the zones. However, very high values show lower tendency for anisotropy on the contoured plots. The implication is that ranges and perhaps anisotropy ratio decrease as grade increases.

14.7.2 Semivariogram Modelling – Gold

Experimental semivariograms (pairwise, relative) 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 semivariogram (the part most important for ordinary kriging). It was possible to obtain experimental semivariograms for a number of orientations in an east-west vertical section, for several orientations in a vertical section oriented at 135^{o} azimuth, and for a northerly horizontal direction. These experimental semivariograms by themselves are insufficient to entirely define a 3D model; geology must also be incorporated.

The experimental semivariograms for the east-west section clearly define a geometric anisotropy with the long axis plunging westerly at about 75^{o} (there is some latitude in the selection of the exact plunge). The anisotropy ratio in this section is in the range 3 to 5, and a tentative value of 4 has been accepted for the initial model. The less abundant data for the 135^{o} azimuth section are very consistent with the interpretation for the east-west section. The northerly, horizontal experimental semivariograms are in the direction of greatest geological continuity — the semivariograms are consistent with the direction of greatest continuity in the east-west section and so are assigned the same range. Consequently, the 3D model is defined in a manner that is both consistent with the known/interpreted geology and honours the experimental semivariograms. The final model for use in ordinary kriging is:

$C_o = 0.24$

$C_1 = 0.40$

a (270^o azimuth, 75^o plunge) = 40 m

a (90^o, 15^o plunge) = 10 m

a (northerly azimuth, 0^o plunge) = 40 m.

The semivariogram model is reasonably well known for purposes of ordinary kriging, with the exception that more data could refine the ranges in the plane of greatest geological continuity (north-striking, westerly dip of 75^o). In addition, there are certainly parts of the deposit (e.g., upper and lower extensions to the main body of the deposit) where the orientation of principal geological continuity changes (on sections one can see substantial flattening and thinning of the upper and lower extremities of the deposit). Semivariogram models for these extremities are apt to be different from the main body of the deposit but insufficient data are available to define such models. The principal difference in SV models for the extremities will be in the orientation of the plane of principal geological continuity. The use of a general SV model for these extremities is a reasonable first approximation because data are relatively widely spaced for blocks between drill holes, whereas blocks near drill holes will be more greatly influenced by the nearby values.

Semivariogram modelling of the AV zone is difficult with the available data. The approach used here is to examine the limited direction(s) available in comparison with the same orientation for the Marc. This was found to be consistent and the semivariogram for the Marc was accepted for the AV. From a geological standpoint, this is reasonable because the AV is almost certainly the faulted northern extension of the Marc. In addition, the data distributions are very similar, with the minor exception of the very highest (anomalous) values.

14.7.3 Semivariogram Modelling – Silver

Pairwise relative semivariograms were determined separately for various drill hole directions (as described for Au) in the Marc and AV zones separately. In this case, the experimental data for the AV zone were not consistent with that for the Marc zone so separate models were developed for each zone as follows:

Marc: Isotropic, spherical model $C_o = 0.25$
C1 = 0.40
a = 12 m.

AV: Isotropic, spherical model $C_o = 0.2$
$C_1 = 0.3$
a = 6 m.

A centered plot of ranges in the vertical east-west plane indicate a hint of anisotropy in the major direction of geological continuity (75^o dip to the west). Considering the uncertainty of range determination in each individual experimental semivariogram, the anisotropy is dubious and would have little impact on estimation. For this reason, isotropic models have been estimated for both zones.

14.8 BLOCK MODEL

A 3D block model was created using Gemcom 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 relies on geostatistical analysis of the sample data and a detailed understanding of the geology to produce a robust estimate of the resource.

The model is rotated 045^o counter-clockwise from the UTM grid so that blocks are orthogonal to the drill sections (azimuth of 315^o) and mine grid established by Bond. The parameters for the block model are listed in Table 14.8.

A block model with 4 m x 4 m x 4 m blocks was created in Gemcom. The rock type element in the block model was coded with the Marc and AV solids using a 0% selection process. The rock model was then updated with the codes 101 and 201, respectively.

Table 14.8: Model Parameters for the Red Mountain Block Model

Coordinates				Origin Coordinates	Block Size (m)	Number of Blocks
Axis Direction	Actual Orientation	Axis	Axis Nomenclature			
Easting	045^o	X	Column	4800	4	125
Northing	315^o	Y	Row	1000	4	200
Elevation	Vertical	Z	Level	2000	4	125

Source: NAMC (2001)

Gold grades were interpolated within the individual zones (101 and 201). The coded rock model was only used for grade interpolation. During volumetric analysis, the rock type comes from the ore solid, not the block model. This technique will only report the volume of a block that falls within the solid. This percentage model replaces the older technique of creating an arithmetic model that contains the percentage of a block that falls within a solid, and weighting the volume of the block by the arithmetic model.

Several block models utilizing different interpolation techniques were generated for comparative purposes. The final run used ordinary kriging and a transposed data set (palinspastic recreation to the original deposit prior to fault offsets) combining the Marc and AV zones. This block model was titled “AVTRANSFORMEDOK.” Within the AVTRANSFORMEDOK, multiple block models were created, each containing a value for a specific block model component. The model components are itemised in Table 14.9.

Table 14.9: Block Model AVTRANSFORMEDOK Components

Model Component	Description
AVTRANS	Au grade interpolation by ordinary kriging
Rock Type	Block code to solid model type
Distance	Anistropic distance to the nearest sample point
KV	Kriging variance as calculated by Gemcom program
STD_DEV	√(KV)
POINTS	Number of points used in AVTRANS estimate
Ag_Trans	Ag grade interpolation by ordinary kriging

Source: NAMC (2001)

A bulk density of 3.0 was used for all blocks in the model grading >5 g/t Au. For blocks grading <5 g/t Au, a bulk density of 2.89 was used in the 2014 resource estimate. Three alternate block models were created to compare for the best-fit interpolation method, which would respect both the geology and the drill hole gold grade distribution. These were used to cross-validate interpolation results and are defined in Table14.10.

Table 14.10: Cross-Validation Block Models

Block Model Name	Description
ID^3 final	Inverse distance to the 3.0 power using identical parameters to the AVTRANSFORMEDOK model excepting an octant search was done.
NN	(Nearest neighbour) inverse distance to the 3.0 power using identical parameters to the AVTRANSFORMEDOK model excepting limiting the sample number to 1(nearest)
Standard	MIK model using first pass statistics provided by Snowden Mining Industry Consultants

Source: NAMC (2001)

14.9 INTERPOLATION

The AVTRANSFORMEDOK model was used for the resource estimate. Interpolation parameters for the MARC and AV zones were identical, except that the rotation about “Y” (ellipse dip) was changed from -75° to -60° to better conform to the geology. The JW also used the MARC and AV parameters, but the dip was changed to -45° for the same reason.

Composite statistics suggested that the most accurate interpolation would be performed by combining the Marc and AV zone composites for interpolation. This was performed by computing the dip slip displacement of the AV zone on the Rick Fault and then calculating a trigonometric transformation of the AV composite point data to its pre-faulted position adjacent to the Marc Zone. The Marc Zone was then interpolated and blocks were assigned their values.

The combined Marc and AV point data were then moved trigonometrically in reverse to the AV zone and the AV zone was interpolated.

The JW zone composite data were left in their original position and interpolated without transformation. The JW zone has good geological continuity but insufficient assay data to place it in a measured or inferred category. The JW zone was, for this reason, interpolated separately from the Marc and AV zones.

14.9.1 Interpolation Parameters & Statistics

Interpolation parameters and statistics are summarised in Tables 14.11 to 14.14 below.

Table 14.11: Anisotropy Angles

Anisotropy are angles defined by Rotation ZYZ	Marc	AV	JW
Rotation about Z from X towards Y	0.0	0.0	0.0
Rotation about Y from Z towards X	-75.0	-60.0	-45.0
Rotation about Z from X towards Y	0.0	0.0	0.0

Source: NAMC (2001)

Table 14.12: Search Parameters

First search radius (along anisotropy x axis)	60.0 m
Second search radius (along anisotropy Y axis)	60.0 m
Third search radius (along anisotropy Z axis)	15.0 m
Ellipsoidal search volume will be used	
Maximum samples per hole	3
High grade cutting/ trimming value – Au	44.0 g/t
High grade cutting/ trimming value – Ag	220.0 g/t
Values exceeding these values will be cut or trimmed to these values	

Source: NAMC (2001)

Table 14.13: Input Parameters for Variograms

Number	Type	Sill		Name
		Increment	Cumulative	
1	nugget	0.37	0.37	OK
	spherical	0.63	1.00	

Number	Type	Rotation	Angle 1	Angle 2	Angle 3	Range 1	Range 2	Range 3
1	type	ZYZ						
	spherical	ZYZ	0.00	-75 to -45	0.00	40.0	40.0	10.0

Source: NAMC (2001)

Table 14.14: Summary Interpolation Statistics for Marc & AV Zones Only

Number of data points 1739	Maximum X value......................... 5,049.59
Minimum X value 4,980.14	Maximum Y value........................ 1,466.52
Minimum Y value 1,080.15	Maximum Z value........................ 1,903.86
Minimum Z value 1,707.39	Maximum value 44.00
Minimum value............................. 0.01	Standard deviation 10.37518
Mean value 8.77706	Coefficient of variation.................. 1.18208
Variance............................. 107.64446	Samples ≤0 0
Log variance 1.39323	Log estimate of mean................... 9.57245
Geometric mean 4.76965	

Source: NAMC (2001)

14.10 CROSS-VALIDATION

14.10.1 Discussion

Initially, Snowden Mining Industry Consultants were asked to perform preliminary geostatistics on the deposit. Evaluation of Snowden's MIK parameters indicated that the interpolation was allowing high gold composite values to overweight areas that had had direct drill information of lower values (i.e., the interpolation was not honouring the drill data).

Dr. Alastair Sinclair, P.Eng. was asked to assist. Dr. Sinclair's MIK variography differed from Snowden's for the upper indicator bins. Subsequently, repeat trials were performed in an attempt to get a grade determination that honoured drill and geological interpretation. Satisfaction was not achieved and the MIK model was returned to Snowden's original parameters and used for comparative cross validation.

ID^4, ID^3 and ID^2 methods were evaluated by Dr. Sinclair through cross validation. He concluded was that a power approaching ID^3 was a best fit to the composite data. ID to the 3rd power was used as a comparative run to NN, MIK and OK methods. Evaluation of ID^3 on section and plan displayed that the method honoured drill composite data significantly tighter than MIK or OK methods and gave a sense of higher spatial predictability. The ID^3 model would result in less tonnes at a higher grade than OK or MIK using a 5 g/t Au cut-off. Considering the stockwork distribution of gold in the deposit, weighting spatial predictability to specific assay intervals did not seem to be reliable and a more conservative approach using OK was adopted.

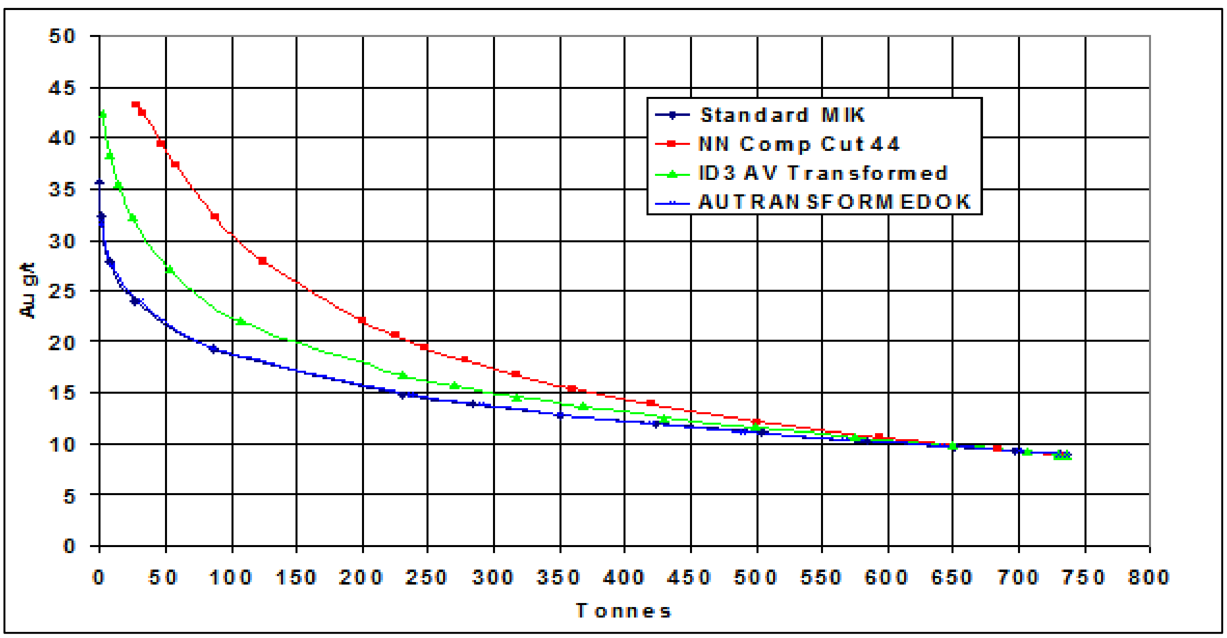
The drill assay validation illustrates the inherent problem of relying too heavily on specific assays being spatially representative. This spatial reliability was also evident in the creation of the stockwork model. Often, during relogging of core, a high-grade assay interval on the fringe of the stockwork would be derived from a 0.5 cm to 1.0 cm wide pyrite vein over a 1 m interval. Because these isolated veinlets could not be connected to a structure or other general mechanism, the intervals were not included in the resource stockwork outline. Similarly, due to the grade influence within the stockwork of a single discontinuous veinlet, spatial predictability of grade would have a greater confidence if a higher degree of smoothing takes place. It is for this reason that the ordinary kriging was chosen as a final method over ID^3.

The following cross validation discussion uses information from the NN, ID^3, Snowden MIK, and OK interpolations. "Nearest neighbour" is used as approaching polygonal resulting in estimating the highest grade associated with the minimum tonnes and the Snowden MIK represents the opposite end of the spectrum with the maximum amount of smoothing.

The Marc Zone, due to its high level of drill density, does not appear on the graphs to be affected by the MIK vs. OK method. On section and plan, however, regions that are on the tails of the deposit display MIK overestimating grade relative to drill data. This effect_is more evident in Figure 14-1. In the AV zone, where drill data spacing is more uniform at 25 m, the effects of the different interpolation methods are more pronounced (Figure 14-2). A summary of the cross-validation for all zones is given in Table 14.15.

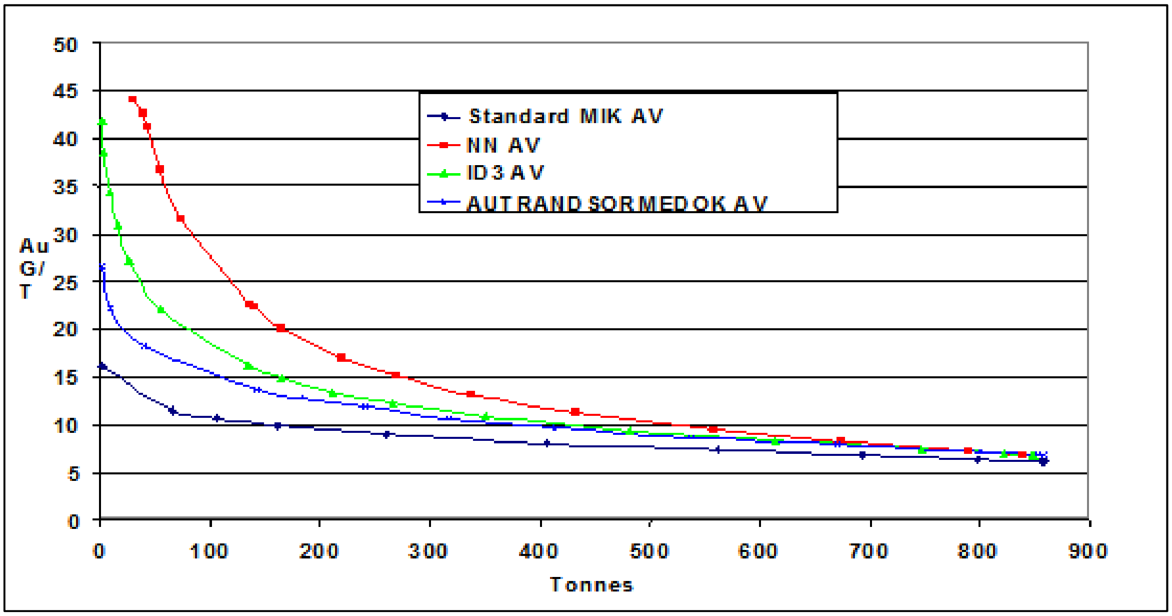
All four methods calculate the metal content within 1% of each other for the same volume in the Marc Zone and within 2% for the AV Zone, with the exception of the Standard MIK, which has an 11% difference to the AVTRANSFORMEDOK model in the AV zone.

Figure 14-1: Mark Zone Grade Tonnage



Source: NAMC (2001)

Figure 14-2: AV Zone Grade Tonnage



Source: NAMC (2001)

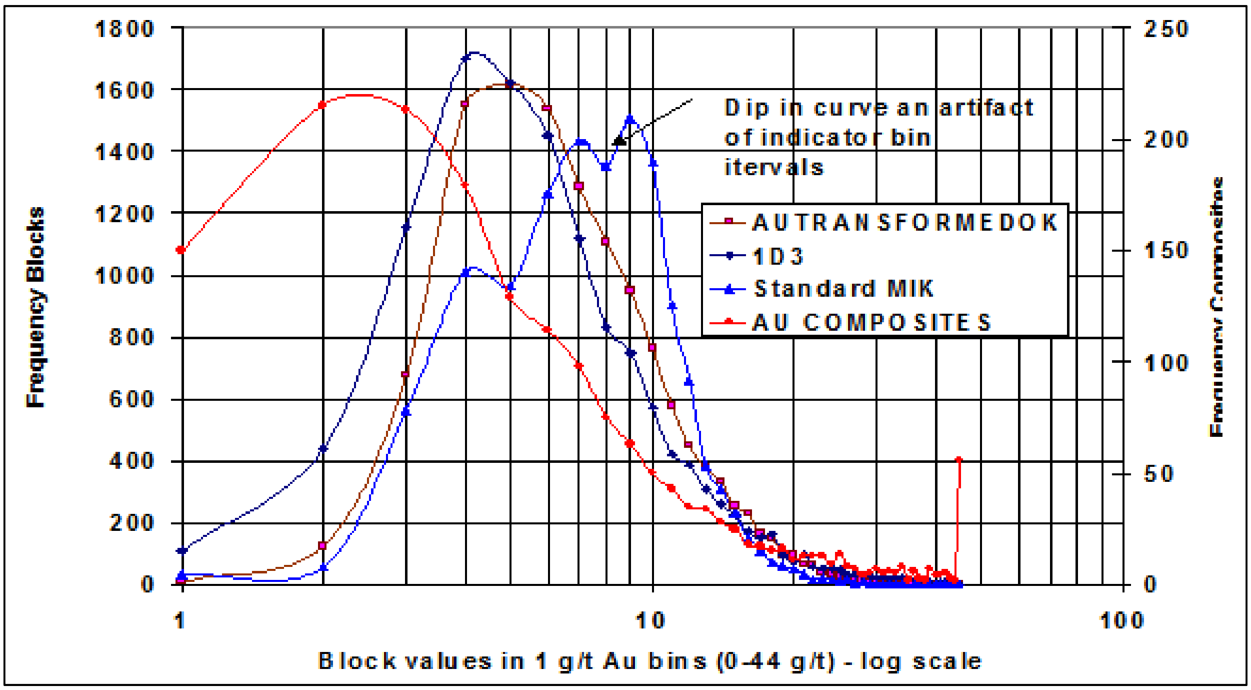
Table 14.15: Block Model Cross-Validation with MIK, ID3 & NN

Interpolation Method	Tonnes Total	Au grade g/t	Au grams Total	Contained Ounces Au
A. Marc Zone				
Standard	737,748	8.91	6,572,442	211,312
ID^3	736,778	8.93	6,580,746	211,579
NN	736,738	8.80	6,485,447	208,515
AVTRANSFORMED OK	736,722	8.89	6,552,836	210,682
Solid volume cross check	737,331			
B. AV Zone				
Interpolation Method	**Tonnes Total**	**Au grade g/t**	**Au grams Total**	**Contained Ounces Au**
Standard	859,632	6.12	5,260,349	169,127
ID^3 (Some blocks not filled)	849,944	6.85	5,820,216	187,127
NN (Some blocks not filled)	840,197	6.88	5,781,492	185,882
AVTRANSFORMED OK	859,437	6.87	5,905,065	189,855
Solid volume cross check	859,278			
C. JW Zone				
Interpolation Method	**Tonnes Total**	**Au grade g/t**	**Au grams Total**	**Contained Ounces Au**
AVTRANSFORMED OK	345,123	7.45	2,569,455	82,611
Solid volume cross check	345,243			

Source: NAMC (2001)

Figure 14-3 illustrates the effect of the different interpolation methods. The MIK model increases grade in the lower grade region and decreases grade in the higher grade region. Visual examination of composite data with the MIK model did not seem reasonable. As expected, the ID^3 and OK methods smooth composite data, with the OK model showing slightly to the left of the ID^3. The higher grade bin intervals are lowered, displaying a shift of metal content into the lower grade blocks.

Figure 14-3: Au g/t in Blocks



Source: NAMC (2001)

14.10.2 Conclusions

Variography for the MIK model supplied by Snowden had a rotating primary search direction from near-horizontal to near-vertical, changing incrementally with increasing indicator bins. NAMC staff contoured these bins by hand on section and plan and found no such trend. Contour grade shells were much better defined in the north-south direction with a dip component of which was best described by the Sinclair variography. The conclusion was reached by Sinclair and NAMC staff that OK interpolation using Sinclair variography best fit the geology and gold metal distribution.

ID^3 was an accurate method that compared well in terms of total metal content but implied too much accuracy in the spatial identification of metal location. In the opinion of the geologists, underground mining would experience a slightly lower grade and higher tonnage above a mining cut-off than indicated by the ID^3 model. It was for this reason that the OK model was chosen.

14.11 RESOURCE CLASSIFICATION

Resource classification was performed using relative kriging errors as described by Blackwell (1998), in which the relative kriging standard deviation (RKSD) is used as a means of determining measured, indicated and inferred resources. The RKSD is given by:

RKSD = σ_K

where σ_K is the $\sqrt{}$(kriging variance) for the estimate.

The RKSD is plotted against the number of sample points used to produce the RKSD classification criteria. RKSD values were not calculated for silver. Silver resource classification used the same block RKSD value as was used for gold.

Blackwell (1998) uses a much larger block size than currently used in the Red Mountain resource. As a trial comparison, The Marc Zone was kriged in both 4.0 m block sizes and then in a 12 m block size, with the kriging standard deviation recorded in the 4.0 m blocks contained within the 12 m blocks. The resource model 4.0 m blocks were then plotted against the 4.0 m within the 12 m blocks on a scatter diagram. Reasonable correlation gave the confidence to continue with this approach.

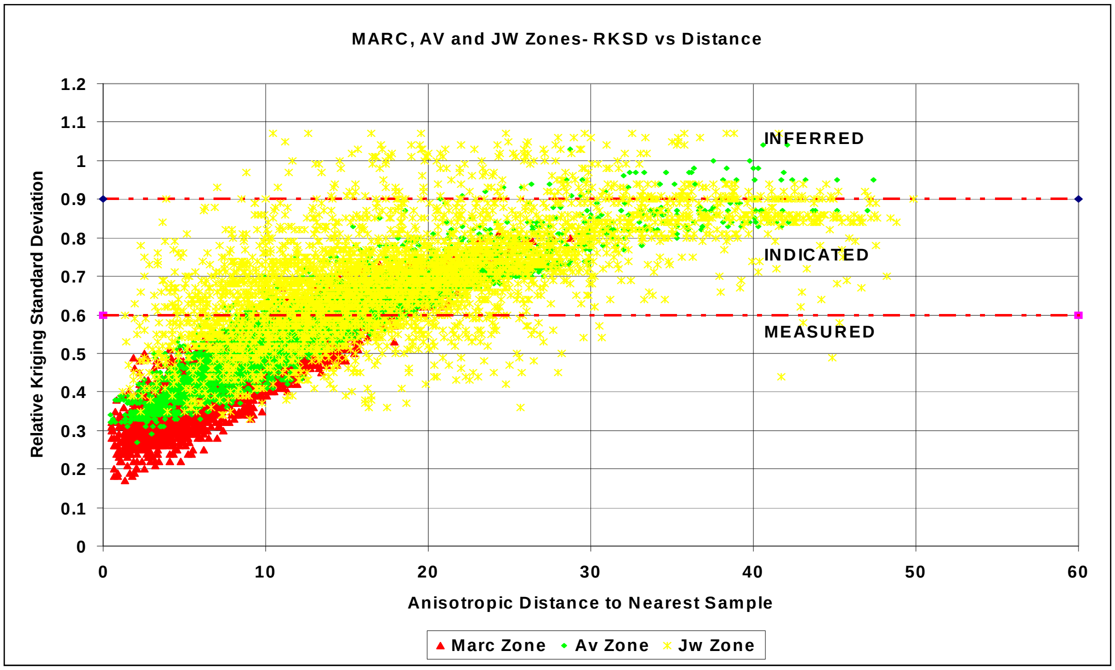
The above model was run using the Stanford University kriging package to test the Gemcom program for whether or not the kriging variance is relative or not. It was concluded that the kriging variance as produced by Gemcom is relative.

Using the 0.60 RKSD cut-off, no estimate is farther than ≈18 m from a sample composite for the measured category. This is slightly less than half the range used in the search ellipse. The indicated category uses a cut-off of 0.9 RKSD, which corresponds with the nearest sample being ≤40 m (equal to the range).

Cut-offs were determined firstly by evaluating what would normally be classified as measured by using distance as a criteria (i.e., Marc zone would equal 100% measured). Secondly, a geologist compared the wide range of assay variation surrounding the block with the block estimate. Thirdly, the block grade was compared to surrounding geological continuity. This reiterative process concluded that 0.60 and 0.90 RKSD values safely categorised the measured and indicated categories.

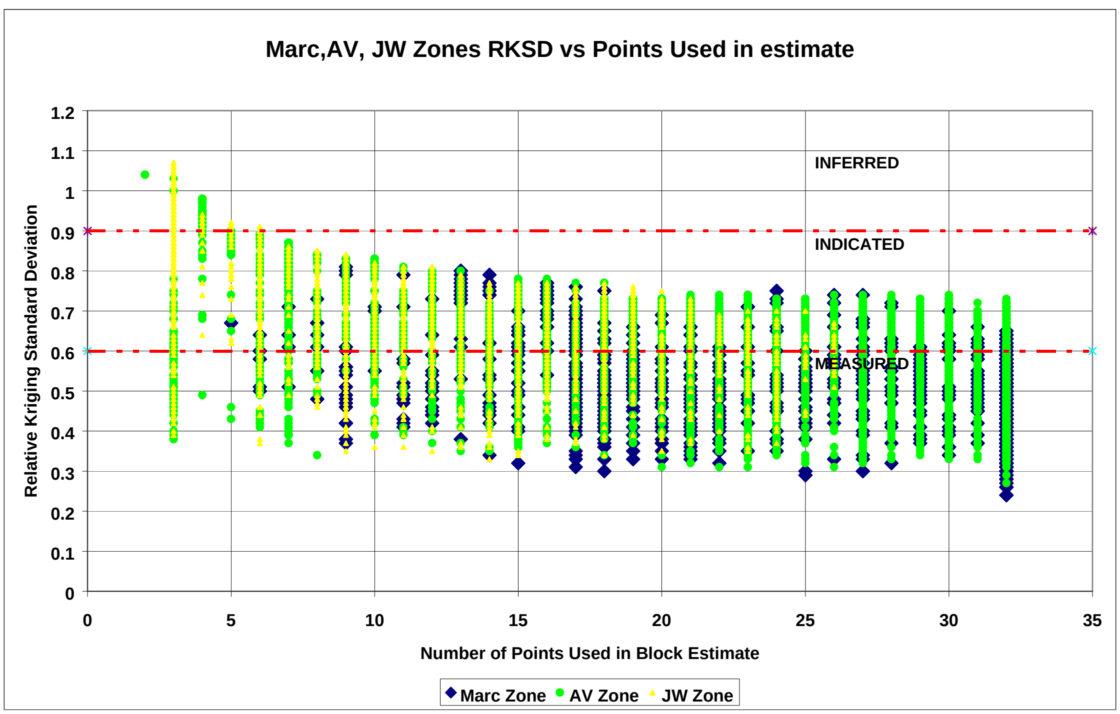
The advantage to this classification system is that even in areas of high drill density (<10 m x 10 m spacing), blocks that have a high uncertainty due to composite selection variance are downgraded to the indicated category. The geologist can assist in the classification, applying his confidence that the geological continuity is good or poor based on non-statistical information such as structure, vein density, rock type, etc. In this sense, it is a much more effective method than basing the classification on purely statistical means such as variography, variance or distance.

Figure 14-4: Marc, AV & JW Zones – RKSD vs. Distance



Source: NAMC (2001)

Figure 14-5: Marc, AV & JW Zones – RKSD vs. Points Used in Estimate



Source: NAMC (2001)

The JW zone has an erratic drill pattern compared to the Marc or AV zones. Although the geological continuity is well documented, the drill density does not satisfy the criteria of confidence to classify the zone above the inferred class. The RKSD values for the JW Zone indicate that infill drilling on this zone has a very high probability of raising the zone to the measured and indicated classification.

14.12 INTERPRETATIONS & CONCLUSIONS

A high degree of drilling and quality control work has been performed on the project by previous operators as well as by NAMC. Re-logging the core to create a geological model has placed a high degree of confidence in understanding ore controls and given the interpretation an unusually high degree of consistency and comprehension not often afforded mineral deposits. A great deal of credit belongs to previous management and staff for providing a system that was both well organised and had the concerted effort of dedicated professional scientists. This foundation was built upon in an orderly fashion by NAMC during the 2000 program and allows for a resource that is robust.

The author believes that the resource for Red Mountain has been calculated utilizing acceptable estimation methodology and the author is also of the opinion that the classification of measured, indicated and inferred resources in this report meet the definitions as stated by NI 43-101 and defined by the CIM Definition Standards For Mineral Resources and Mineral Reserves adopted by the CIM Council on November 27, 2010.

15.0 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE NON-COMPLIANCE

This Preliminary Economic Assessment does not support an estimate of Mineral Reserves, since a prefeasibility or feasibility study is required for reporting of Mineral Reserve estimates. This report is based on potentially mineable material ("mineable tonnes").

Mineable tonnages were derived from the resource model described in the previous section. Measured, indicated and inferred resources were used to establish mineable tonnes.

Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

15.2 MINEABLE TONNAGE ESTIMATION PROCESS

To determine the mineable tonnage at Red Mountain the following process was utilised:

1. Analyze geologic resource model for geometric properties, such as mineralised zone width, depth, length, and continuity.
2. Select the mining methods best suited for the deposit based on geometry, economics, and geotechnical restrictions.
3. Determine an economic cut-off grade based on expected operating cost, mining recovery, mining dilution, and commodity price assumptions.
4. Identify the blocks in the model that pass cut-off, and design production stope shapes around these blocks.
5. Query the production stope shapes for in-situ tonnage and grade data, apply mine dilution, and check the diluted stope grades against the cut-off grade, removing all stopes that fall below cut-off.
6. Develop a mine plan around the economically viable production stopes and run economic models on various production scenarios.
7. Repeat steps one through six, refining cut-off parameters and stope designs until an economically optimal mine plan is achieved.

15.3 RESOURCE MODEL SUB-BLOCKING

JDS used the resource block model discussed in the resource section 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 mineralised material blocks near the waste/mineralised material contact and to better estimate planned mine dilution.

Sub-blocking an existing block model effectively reduces only the blocks that are in contact with a resource boundary and removes those blocks, which extend into a waste zone. As such, there is generally a minor loss of tonnage during sub-blocking exercises, but if done properly, loss should be less than 1% by tonnage. Table 15.1 below summarises the change in block model resource at a 3.0 Au g/t cut-off before and after the sub-blocking exercise.

Table 15.1: Mineral Resource Before & After Sub-blocking

		JDS Sub-blocked Model			***ACS Model***		
Zone	**Class**	**Au (g/t)**	**Ag (g/t)**	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Tonnes**
Marc	Meas	9.26	40.02	649,000	9.26	40.06	652,000
Marc	Ind	9.53	30.13	12,000	9.71	30.33	11,000
AV	Meas	7.15	20.86	503,000	7.14	20.88	508,000
AV	Ind	7.34	20.94	286,000	7.32	21.03	284,000
AV	Inf	10.83	39.19	2,000	10.96	39.50	2,000
JW	Inf	7.66	12.59	330,000	7.67	12.57	331,000
Grand Total		**8.06**	**26.40**	**1,782,000**	**8.06**	**26.44**	**1,787,000**
Total Difference		**0.01%**	**-0.14%**	**-0.30%**			

Source: JDS (2014)

15.4 MINING METHOD SELECTION

The Red Mountain resource is made up of three major zones: Marc, AV, and JW. All three zones share a general north-south strike, with dips varying between 30° and 80°. The zones range from one to 40 m in width, 100 m to 200 m in strike length, and 60 m to 100 m in height.

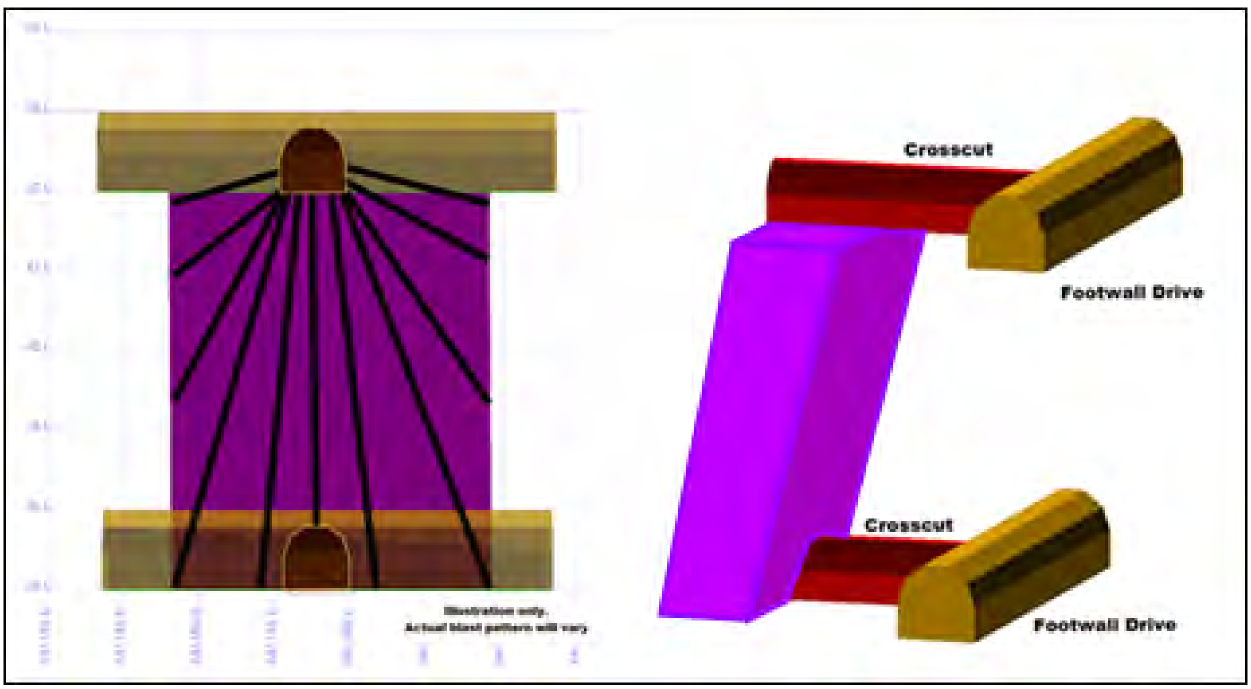
JDS selected sub-level longhole (LH) stoping with cemented rock fill (CRF) as the principal mining method at Red Mountain due to its high productivity, low cost, selectiveness, and successful history of application for deposits of this nature. Alternatively, drift and fill (D&F) mining would be used where conditions are not suitable for longhole stoping.

Longhole stoping is a semi-selective and productive underground mining method, and well suited for steeply dipping deposits of varying thickness. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralisation. In the planned longhole stopes at Red Mountain, a top and bottom drift delineate the stope and a dedicated longhole drilling machine drills blast holes between the two drifts. The drill holes are loaded with explosives and the stope is blasted, with broken material falling to the bottom drift for extraction. In longhole stopes, remote controlled load haul dump machines (LHD) are required to remove the blasted material from the stope once blasting commences.

One of the limitations with longhole stoping is that the dimensions of the stope should not exceed a longhole drilling machine's effective range, which, for top hammer drill rigs, is generally 30 m. Another limitation with longhole stoping is the stopes must remain open long enough to remove the mineralised material and then filled with an engineered backfill material (if pillars are not used). These limitations generally restrict level spacing to 30 m or less, and subject stope strike lengths to geotechnical review.

Transverse stoping would be the primary method at Red Mountain, whereby crosscuts would cut through the stope perpendicularly, and longhole fans would drill off the strike length of the stope. This method is beneficial for production rates as multiple stopes can be in operation at once on a level. The shortfall of transverse longhole mining is that a footwall drift is required outside the mineralised material zone for the entire strike length, and crosscuts must be driven long enough to maintain a safe distance between the footwall drift and the mineralised material zone, which depending on geotechnical constraints can vary from 15 to 50 m. This generally adds a significant amount of waste development to the operation. The method is shown in Figure 15-1.

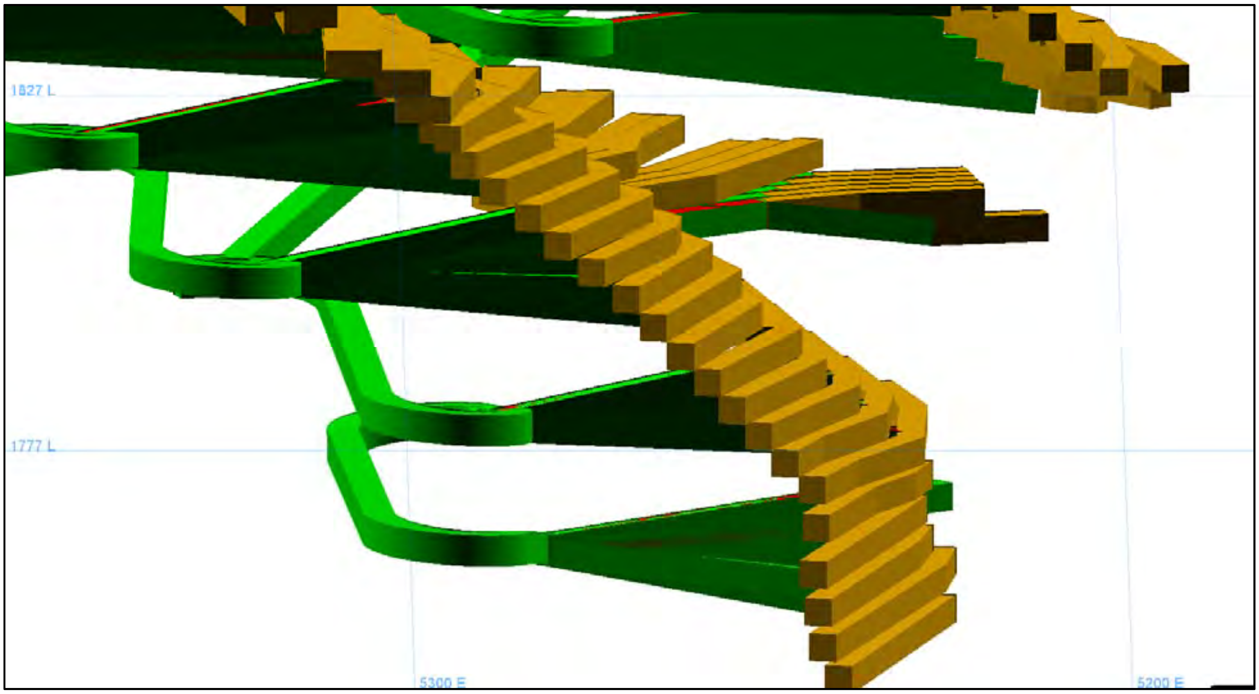
Figure 15-1: Transverse Longhole Stoping (Oblique View)



Source: JDS (2014)

Drift and fill mining would also be used at Red Mountain for areas of the deposit, which fall below an allowable dip for longhole stoping. The JW zone is for the majority a shallow dipping deposit and would be mined by drift and fill. Several small areas of the AV and Marc zone are too shallow for longhole stoping and would be mined by drift and fill. Figure 15-2 depicts the JW zone with green development drives accessing the shallow dipping zone (orange) in a fan pattern. Drift and fill mining would be fully mechanised, using a 4.0 x 4.0 m overhand drift profile.

Figure 15-2: Drift & Fill Stoping (Oblique View)



Source: JDS (2014)

15.5 GEOTECHNICAL PARAMETERS

The following information has been extracted from the 2008 PEA study prepared by SRK Consulting.

A geotechnical analysis of the Red Mountain deposit was conducted by Scott Broughton and Brennan Lang (Rock Group Consulting Engineers) and reviewed by Bharti Engineering in their 1994 Feasibility mine design. The summary of the geotechnical analysis is reported here.

The rock mass classification describes the ground in Marc zone as “good” according to the CSIR/RMR system. The gold mineralisation is the most competent rock type (RMR 80) and the hanging wall is the least competent (RMR 68).

Due to the shallow depth of mining, ground problems related to high stress are not expected. If mining induced stresses exceed the strength of a pillar, it would exhibit yielding, rather than rock burst type behavior.

The rock mass classification data was used as an input for the design of excavation spans. The procedure followed the empirical approach (stability graph) developed by Matthews and Potvin. The stability graph method indicates stable spans of 15 to 20 m width.

Given high RMR ratings and the large stable spans estimated by Rock Group Consulting Engineers, JDS selected longhole stopes of 20 m wide by 30 to 38 m tall.

15.6 STOPE DESIGN PARAMETERS

Stopes were designed based on mining method and geotechnical restrictions discussed above:

Table 15.2: Production Stope Design Criteria

Mine Method	Stope Width [a]	Stope Height [b]	Stope Length [c]	Dip (^{0})
Drift and Fill	4.0 m	4.0 m	N/A	0-59
Longhole Stoping	20.0 m	30.0 m	10.0 m	60-90

Source: JDS (2014)

15.7 MINE DILUTION & RECOVERY

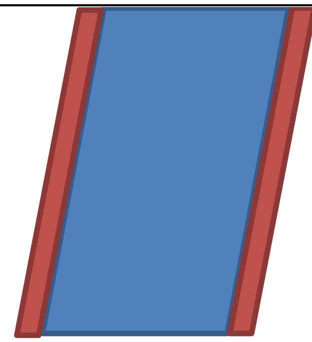
Dilution parameters were assigned to each stope to estimate overbreak experienced during mining operations. The ground at Red Mountain is sufficiently competent and wide enough that JDS feels overbreak would be minimal, at 0.5 m on each wall for longhole stopes, and 0.1 m for drift and fill stopes. Based on a generic stope dimension of 30 m x 20 m this equates to 5% and 10% dilution for longhole and drift and fill stopes respectively.

Table 15.3 outlines the dilution criteria used for all production stopes.

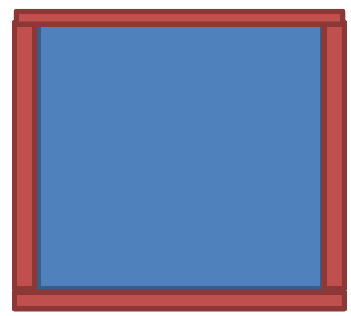
The geology of Red Mountain shows fairly strong and regular mineralisation/waste contacts, and it is estimated that mining recoveries would be high during production with appropriate longhole drilling and grade control. JDS has estimated a 95% mining recovery factor for mineralised material contained within the targeted production stopes.

Table 15.3: Mine Dilution Criteria

Longhole Stopes - Transverse		
Stope Width (a)	20	m
Stope Height (b)	30	m
Stope X section	600	sq.m
Footwall overbreak	0.5	m
Hangingwall overbreak	0.5	m
Footwall overbreak area	15	sq.m
Footwall overbreak area	15	sq.m
Dilution by Volume	5.0%	



Ore Drifts		
Drift Width (a)	4	m
Drift Height (b)	4	m
Stope X section	16	sq.m
Back and floor overbreak	0.1	m
Wall overbreak	0.1	m
Back/floor overbreak area	0.800	sq.m
Wall overbreak area	0.800	sq.m
Dilution by Volume	10.0%	



Source: JDS (2014)

15.8 CUT-OFF GRADE CRITERIA

A gold grade cut-off was established for Red Mountain, the parameters of which are stated below in Table 15.4.

The resource block model was queried at the production cut-off grades mentioned above for both drift and fill and longhole stoping zones to identify economic targets for stope design.

Figure 15-3 displays a long section view of the resource at a 3.0 g/t cut-off, the minimum selected for longhole stoping. The resource was first assessed for longhole stoping potential, as it is the lower cost, more productive method in comparison to drift and fill, and should be maximised.

The remaining resource was assessed for drift and fill potential and stopes were hand designed around areas that met the stope design criteria.

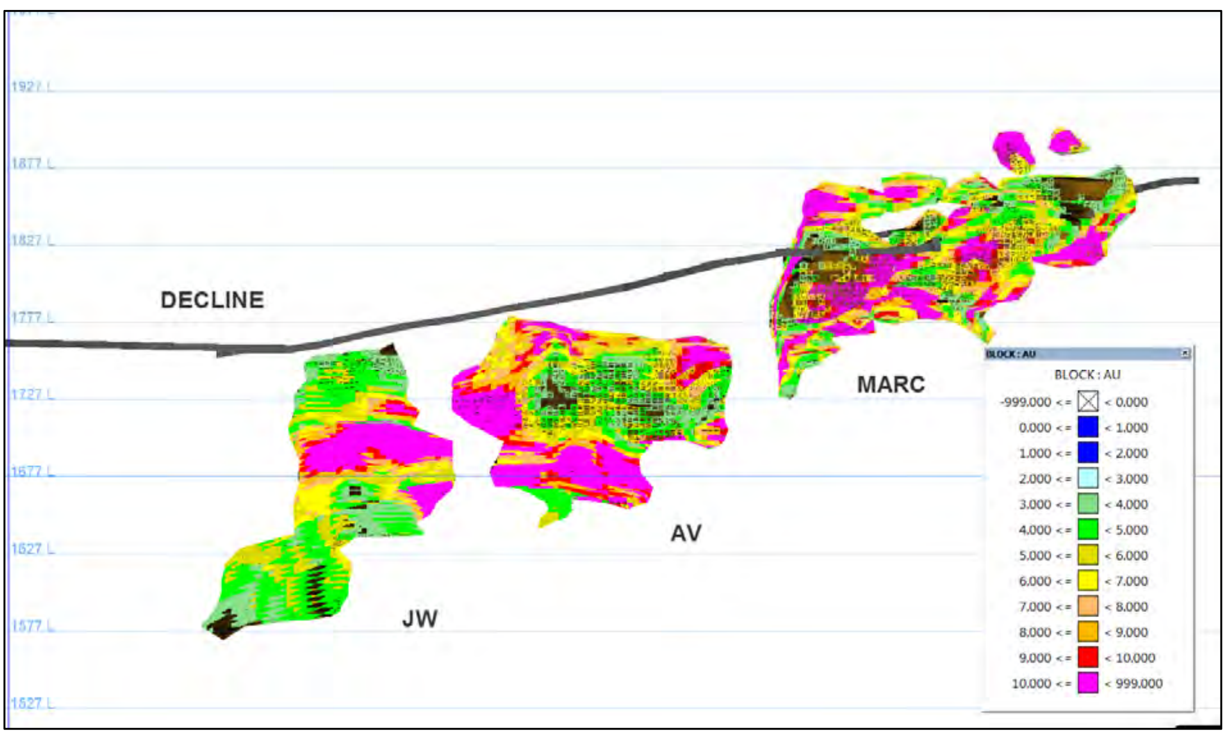
Figure 15-4 depicts a long section view of the designed production stopes passing economic cut-off.

Table 15.4: Cut-off Grade Parameters

Price of Gold	**1,250.00**	**CA$/oz Au**	**40.19**	**CA$/g Au**
Mill Recovery	91%	%	36.57	
Payable Metal from Refiner	99%	%	36.21	
Royalties	3.5%	$NSR	34.94	Net gold price
Mine Opex Calcs	**Longhole ($/t)**	**Drift & Fill ($/t)**		
Production	6.03	25.59		
Backfill	9.85	10.23		
Mine General	13.03	13.03		
Mine Labour	34.14	34.14		
TOTAL	**63.04**	**82.99**		
Opex	**Longhole ($/t)**	**Drift & Fill ($/t)**		
Mine	63.04	82.99		
Process	28.00	28.00		
G&A	9.00	9.00		
Total	**100.04**	**119.99**		
Calculated Cut-off (g/t Au)	**2.86**	**3.43**		
Chosen Cut-off (g/t Au)	**3.00**	**5.00**		

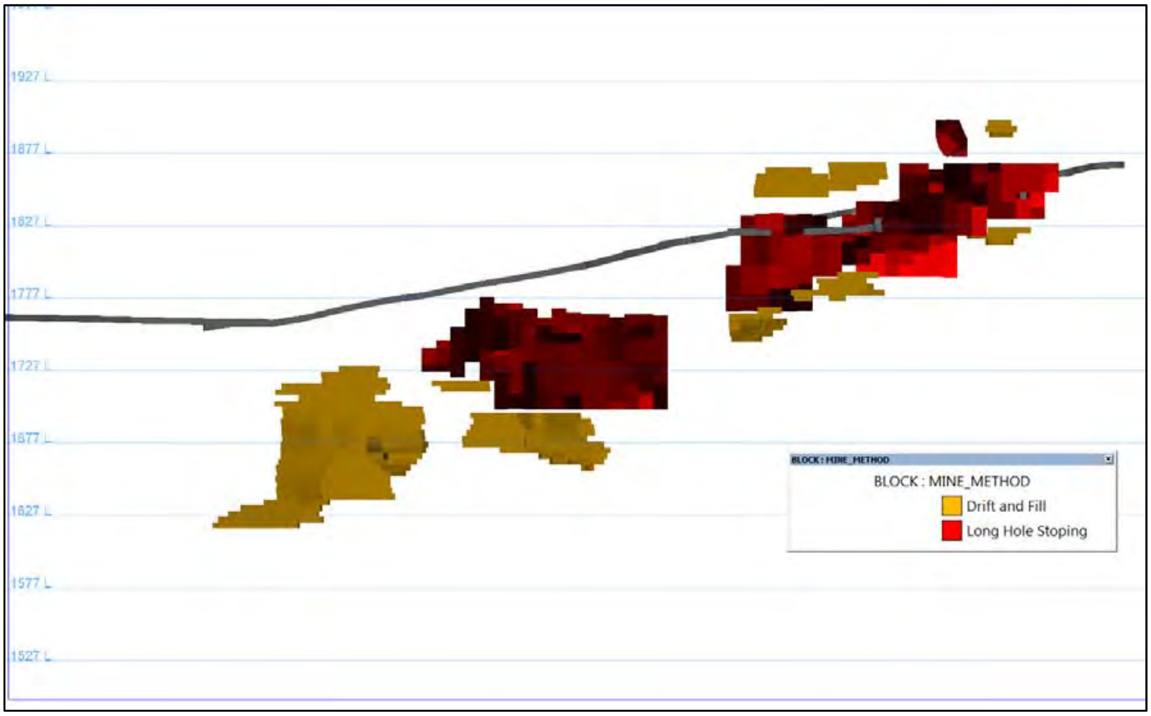
Source: JDS (2014)

Figure 15-3: Long Section View of Block Model at 3.0 Au g/t Cut-off (Looking East)



Source: JDS (2014)

Figure 15-4: Long Section View of Production Stopes (Looking East)



Source: JDS (2014)

15.9 MINEABLE TONNAGE

The mineable tonnage for Red Mountain is a product of the above steps taken, plus multiple iterations for optimizing operating parameters such as operating costs and stope designs during the mine planning process.

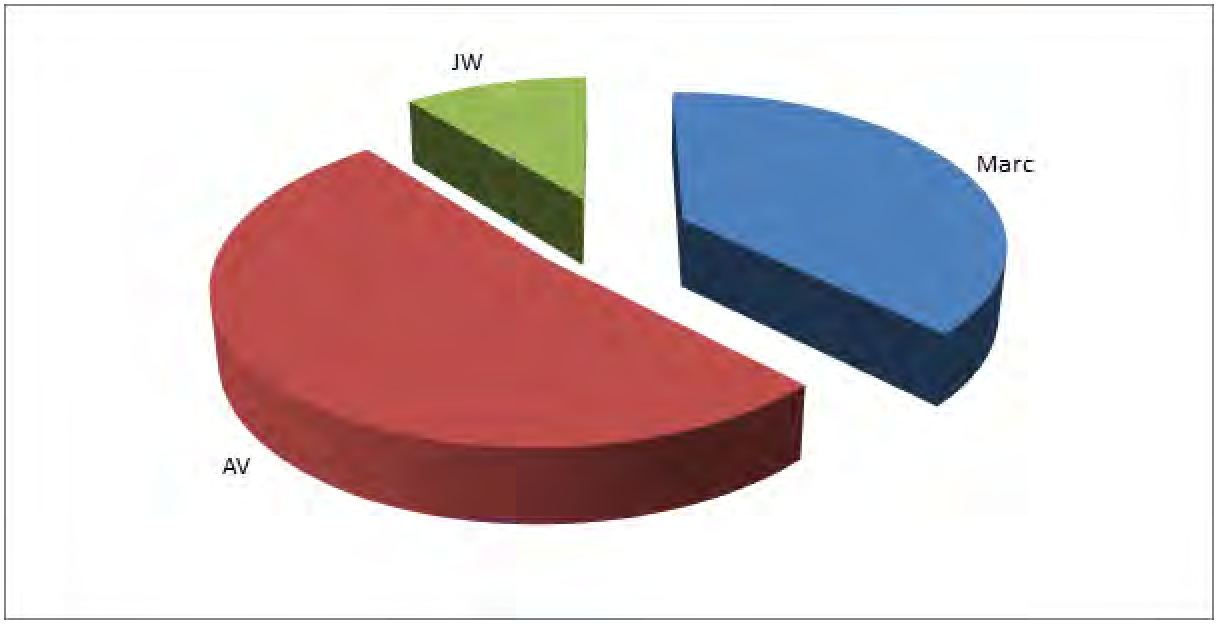
Table 15.5 and Figure 15-5 outline the diluted, recoverable, mineable tonnage used for mine planning purposes. For this level of study, no separation by resource class has been used, and as such, reported numbers contain inferred resources.

Table 15.5: Mineable Tonnage Used for Mine Planning

Zone	Tonnes	Au (g/t)	Ag (g/t)
Marc	516,000	8.62	35.88
AV	713,000	6.09	18.83
JW	149,000	8.06	11.71
Total	**1,378,000**	**7.25**	**24.44**

Source: JDS (2014)

Figure 15-5: Mineable Tonnage by Zone



Source: JDS (2014)

The mineable tonnage achieves a 77% resource-to-mineable-tonnage conversion ratio based on a resource cut-off of 3.0 g/t in situ. This is a moderately high conversion ratio in comparison to other projects, and can be attributed to the geometry of the mineralised zones at Red Mountain and continuous nature of the grade distribution.

Longhole stoping would contribute 82% of the mineable tonnage, and the remaining 18% would come from drift and fill mining.

16.0 MINING

16.1 MINE DESIGN CRITERIA

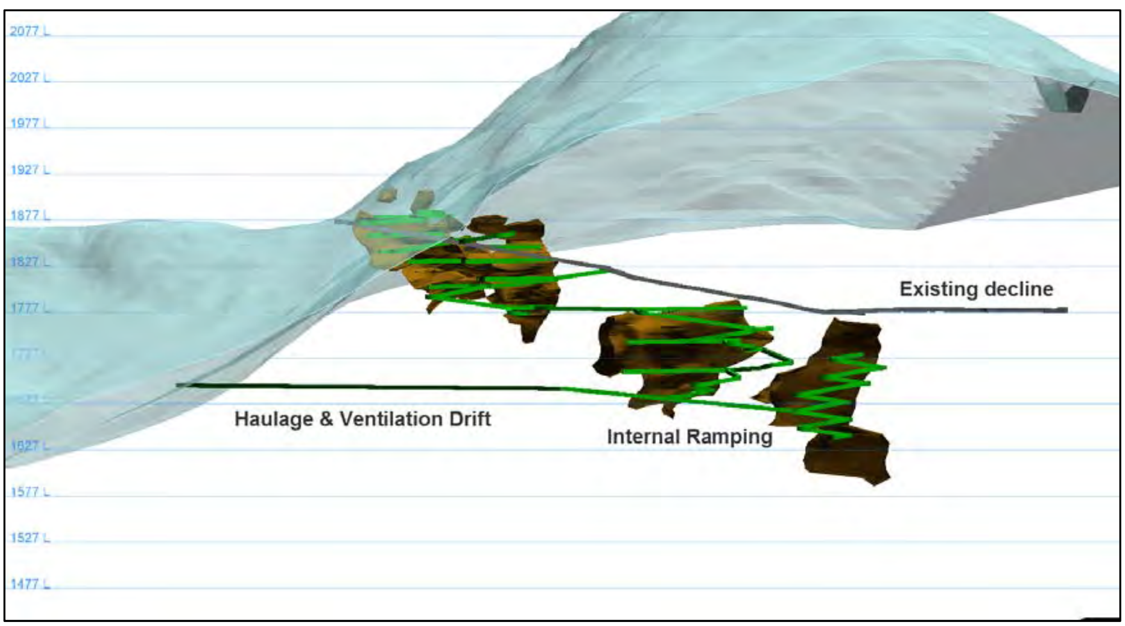
16.1.1 Mine Access

The Red Mountain deposit consists of a mineable resource extending 300 vertical metres. An exploration decline currently extends from a portal at 1,860 m elevation over the entire strike length of the known resource, well situated at the top of the mineralised zones. The existing decline is about 5 m wide by 4 m high and was driven at a grade of -17%.

The deposit is situated on a mountainside, and as such, there is an opportunity to establish a second portal and drive an incline towards the bottom of the mineable resource. This would provide the mine with a natural ventilation circuit, gravity drainage to prevent mine flooding, secondary egress, shorter haulage route to the process plant, and the potential to gravity feed broken muck to an extraction level via muck pass. The new portal was designed at 1,650 m elevation and is tied in to an existing tote road, which connects the future process plant area to the upper portal. The incline would be driven at a grade of +3.5% over 850 m.

An internal ramp system would be developed for each of the three 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. Mineralised zone development would be on a 4.0 m by 4.0 m profile. Figure 16-1 below depicts the mine access points.

Figure 16-1: Mine Access



Source: JDS (2014)

16.1.2 Production Rate Selection

The Red Mountain mine plan has been sized for a 1,000 tonne per day (t/d) operation. It is common for first pass evaluations of underground longhole operations to use a rule of thumb in which the mineable tonnes per vertical metre are applied to an annual vertical advance of 40 to 50 m. This exercise yielded a throughput in the order of 1,000 t/d, and was used for base case analysis. Detailed mine planning and cycle time evaluations were conducted to support this production rate selection. Production in the last year of mining was slightly increased to 1,085 t/d to finish mining in a full season.

Due to its altitude and high snowfall conditions, the mine would be operated on a seasonal basis during the months of March to November. From December to February, the mine would be on care and maintenance.

16.1.3 Production Sequencing

Production in longhole stoping zones would be mined in a primary → secondary fashion, whereby primary stopes would be mined first, backfilled with cemented rock fill, then secondary stopes would be mined and backfilled with either cemented rock fill, or uncemented run of mine waste where future mining does not take place below or next to the stope.

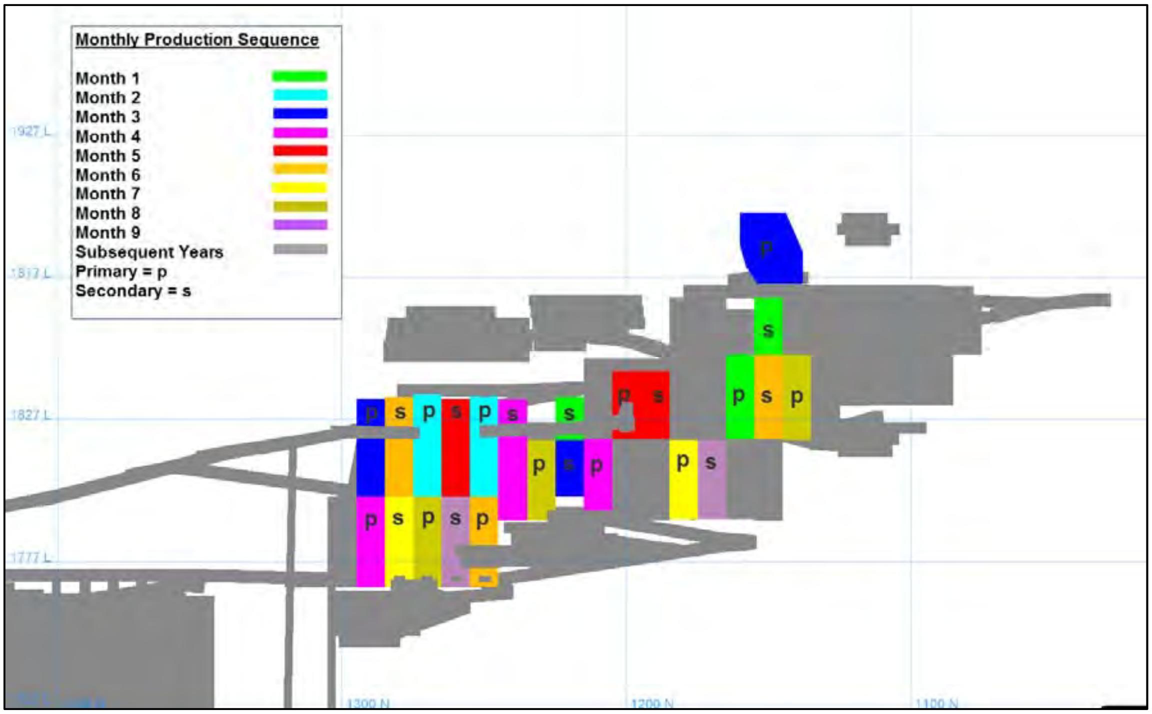
Drift and fill zones would be mined in a bottom-up fashion from a main access drift. From the main ramp, a drift would access the production area with a -15% crosscut. Once the production drift would be mined out on that level, it would be backfilled and the access crosscut would be slashed along the back and backfilled on the floor to allow access to the next level above, where the mining process would be repeated.

Figure 16-2 depicts the production sequencing on a monthly basis for Year 1 production.

Mineral zones were sequenced in order to prioritise highest grade, lowest mine operating costs, existing access development, and level of confidence in the resource. As such, Marc zone would be targeted in the first year of production, followed by AV in the second production year. Mining of JW zone would commence in Year 4.

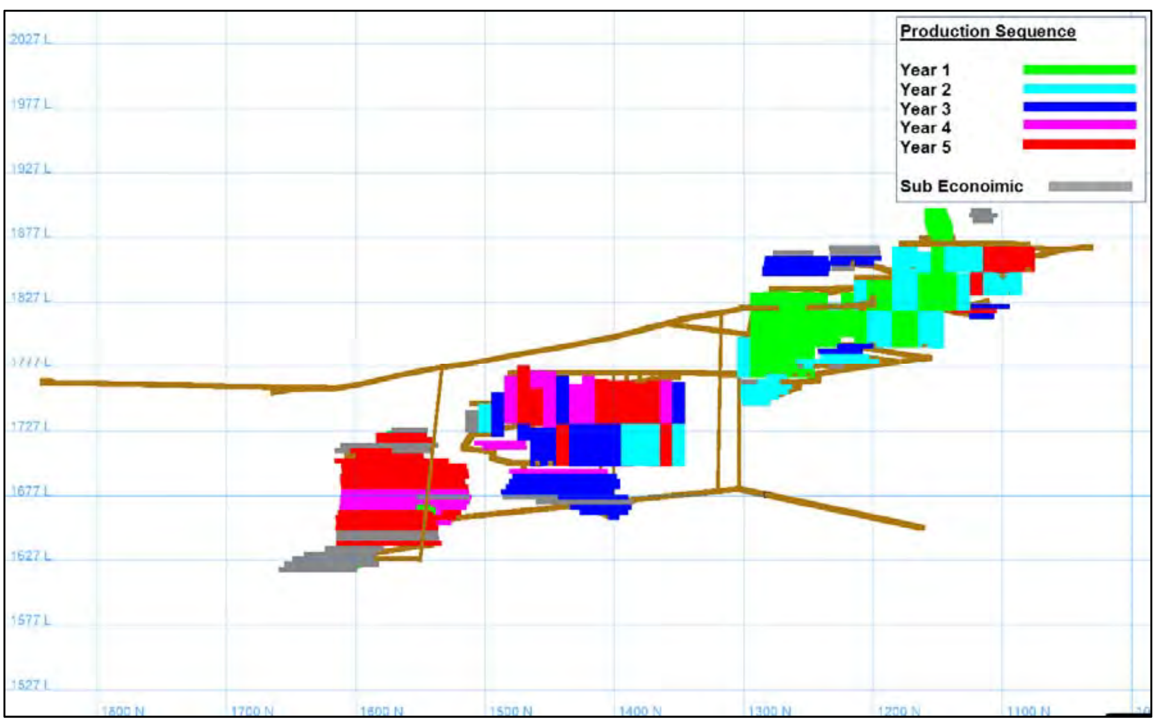
Figure 16-3 depicts the production sequencing on an annual basis.

Figure 16-2: Monthly Production Sequencing



Source: JDS (2014)

Figure 16-3: Annual Production Sequencing



Source: JDS (2014)

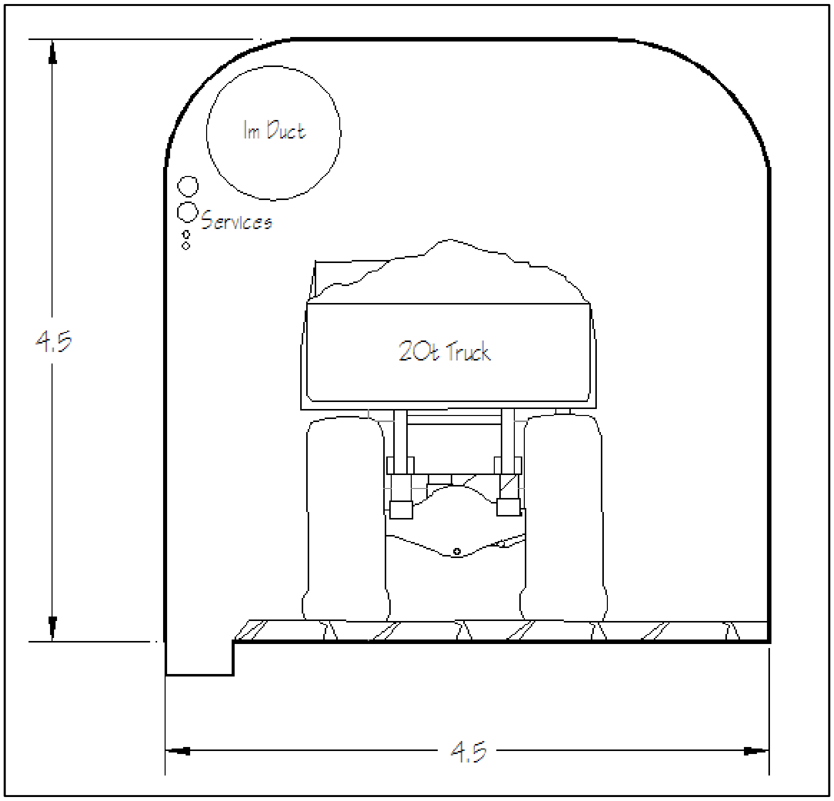
16.2 UNDERGROUND MINE DEVELOPMENT

16.2.1 Lateral Development

The ramp is designed at 4.5 m x 4.5 m arched to accommodate fully loaded 20 t or 30 t haul trucks and 42 inch round vent ducting. Footwall drifts would also be driven at 4.5 m x 4.5 m to allow haul truck access to the stope crosscuts. Crosscuts would be driven flat back style 4.0 m x 4.0 m to accommodate remote LHD entry.

Figure 16-4 depicts a typical ramp cross section.

Figure 16-4: Typical Ramp Cross-section



Source: JDS (2014)

16.2.2 Vertical Development

A muck pass at 3.0 m by 3.0 m profile is planned between the Marc and AV zones. All mineralised material from Marc and the majority of mineralised material from AV would be sent down the muck pass, where haul trucks would be loaded and would haul out of the lower portal to the mill site. JW zone is close enough to the lower portal incline that the haulage is short, so all mineralised material would be truck hauled from these drift and fill stopes. A grizzly would be installed at the top of the muck pass to remove oversize blasted material.

Ventilation raises at 3.0 m by 3.0 m profile would be established to provide fresh air for each of the mining zones. All raises would be driven with the use of Alimak raise climbers.

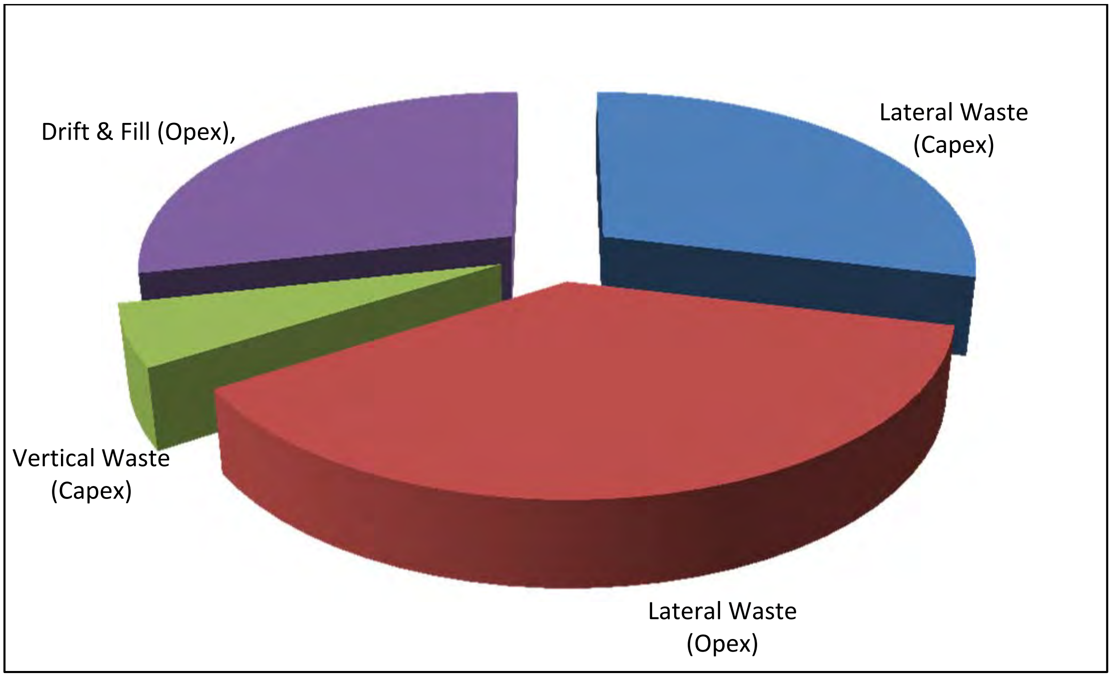
Total lateral and vertical development over the mine life is summarised in Table 16.1.

Table 16.1: Development Schedule

Development Type		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5
Lateral Waste (Capex)	m	**3,920**	1,420	1,640	860	-	-	-
Lateral Waste (Opex)	m	**5,080**	340	2,060	1,020	1,040	160	470
Lateral Drift & Fill (Opex)	m	**3,880**	-	-	1,130	730	1,520	500
Total Lateral	**m**	**12,880**	**1,770**	**3,700**	**2,990**	**1,770**	**1,680**	**970**
Total Vertical Waste (Capex)	**m**	**770**	**210**	**270**	**290**	**-**	**-**	**-**
Total Development Waste Rock	**t**	**531,100**	**154,290**	**198,300**	**101,390**	**48,140**	**7,390**	**21,600**

Source: JDS (2014)

Figure 16-5: Development Metres by Type



Source: JDS (2014)

16.3 UNDERGROUND MINE PRODUCTION

16.3.1 Mine Operations

Longhole stoping would be the main mining method at Red Mountain. Individual stope tonnages range from 2,300 t up to 34,000 t.

Longhole drilling of mainly downholes with 76 mm diameter is planned at sublevel spacing of 15 m to 35 m. Some stoping would include drilling of upholes. Slots would be developed by drop raising. ANFO would be used for longhole blasting.

The stopes would be mucked with remote controlled 4.6 m^3 (6-yard) LHDs. Mineralised material would be transported by the LHDs to level remucks. Muck would be loaded on the level into haul trucks and transported to the muck pass or directly to the process plant. Backfill would be dumped by LHD or truck into the stope.

Drift and fill mining would be used for narrow mineralised areas with a shallow dip. Overhand mining is planned in 4.0 m lifts. Backfill would be placed in each completed cut and compacted tight to the back with a rammer jammer attachment on a 3.1 m^3 LHD.

Backfill would consist of cemented rock fill for primary stopes and unconsolidated waste rock for secondary stopes. CRF consists of waste rock mixed with cement slurry. Waste rock would be sourced from the underground access development and the existing waste stockpile from the underground exploration program.

All of the development waste of 531,000 t shown in Table 16.1 would be used as mine backfill. The existing waste stockpile at the upper portal is estimated at 150,000 t and would also be consumed as backfill. No development waste would remain on surface at the end of the mine life.

Cement slurry would be produced underground by a portable batch plant consisting of a cement hopper and a mixing tank. The slurry would be piped to a spray bar, which discharges onto the waste rock in the truck box.

A cement binder content of 5% was assumed for longhole stoping and 3% for drift and fill mining. To provide sufficient wall stability, a higher cement content is required for longhole stoping. The quantities of cement required in the backfill mixes are estimates only, since no test work has been done to-date. Backfill testing is recommended for future studies to define the optimum mix recipes.

The following tables and figures outline the schedules for mine production and backfill placement.

Table 16.2: Mine Production Schedule

Year		Total	-2	-1	1	2	3	4	5
Working Days			**180**	**180**	**270**	**270**	**270**	**270**	**270**
Production	t/d				1,000	1,000	1,000	1,000	1,100
Marc									
Tonnes	t	**516,000**			270,000	190,000	29,000	0	27,000
Au	g/t	**8.62**			10.3	6.9	7.8	0.0	4.5
Ag	g/t	**35.88**			42.9	27.5	27.3	0.0	33.4
AV									
Tonnes		**713,000**			0	83,000	242,000	201,000	187,000
Au	g/t	**6.09**			0.0	7.2	6.4	5.8	5.5
Ag	g/t	**18.83**			0.0	21.4	16.4	20.0	19.6
JW									
Tonnes		**149,000**			0	0	0	70,000	79,000
Au	g/t	**8.06**			0.0	0.0	0.0	7.5	8.6
Ag	g/t	**11.71**			0.0	0.0	0.0	14.1	9.6
Total									
Tonnes		**1,378,000**			**270,000**	**272,000**	**271,000**	**271,000**	**293,000**
Au	**g/t**	**7.25**			**10.3**	**7.0**	**6.5**	**6.2**	**6.3**
Ag	**g/t**	**24.44**			**42.9**	**25.6**	**17.6**	**18.5**	**18.2**

Source: JDS (2014)

Table 16.3: Backfill Schedule

Year		Total	-2	-1	1	2	3	4	5
Opened Void Space									
Marc	m^3	**211,000**			113,000	78,000	9,000	-	11,000
AV	m^3	**238,000**			-	25,000	75,000	72,000	65,000
JW	m^3	**49,000**			-	-	-	23,000	27,000
Total	m^3	**498,000**			**113,000**	**103,000**	**85,000**	**95,000**	**103,000**
Backfill Required									
Marc	t	**423,000**			226,000	156,000	19,000	-	22,000
AV	t	**476,000**			-	50,000	150,000	144,000	131,000
JW	t	**98,000**			-	-	-	45,000	53,000
Total		**997,000**	**-**	**-**	**226,000**	**206,000**	**169,000**	**189,000**	**206,000**
Waste Stockpile	t		150,000	304,000	273,000	197,000	112,000		
Addl. Material Required	t	**164,000**						17,000	146,000

Source: JDS (2014)

16.3.2 Mine Services

16.3.2.1 Mine Ventilation

Airflow requirements were estimated based on diesel emissions of the underground equipment fleet, as shown in Table 16.4 below.

Table 16.4: Ventilation Requirements

Equipment	Max. Qty	Engine Power (kW)	Utilisation (%)	Total kW	Air Flow Required (m^3/sec)	Air Flow Required (cfm)
Dev. Jumbo - two-boom	2	82	10	16	1.0	
LHD - 4 yd^3	3	100	85	255	15.3	
20 Tonne Truck	3	240	100	720	43.2	
Fuel/Lube Truck	1	95	30	29	1.7	
Boom Truck	1	86	30	26	1.5	
Scissor Truck	3	120	30	108	6.5	
Longhole Drill	2	74	30	44	2.7	
Toyota	3	95	50	143	8.6	
Transmixer	1	155	50	78	4.7	
Mobile Equipment Subtotal	19				**85.1**	**180,000**
Air Losses (20%)					17.0	
Safety Factor (10%)					8.5	
Total Estimated Ventilation Requirements					**110.6**	**234,386**

Source: JDS (2014)

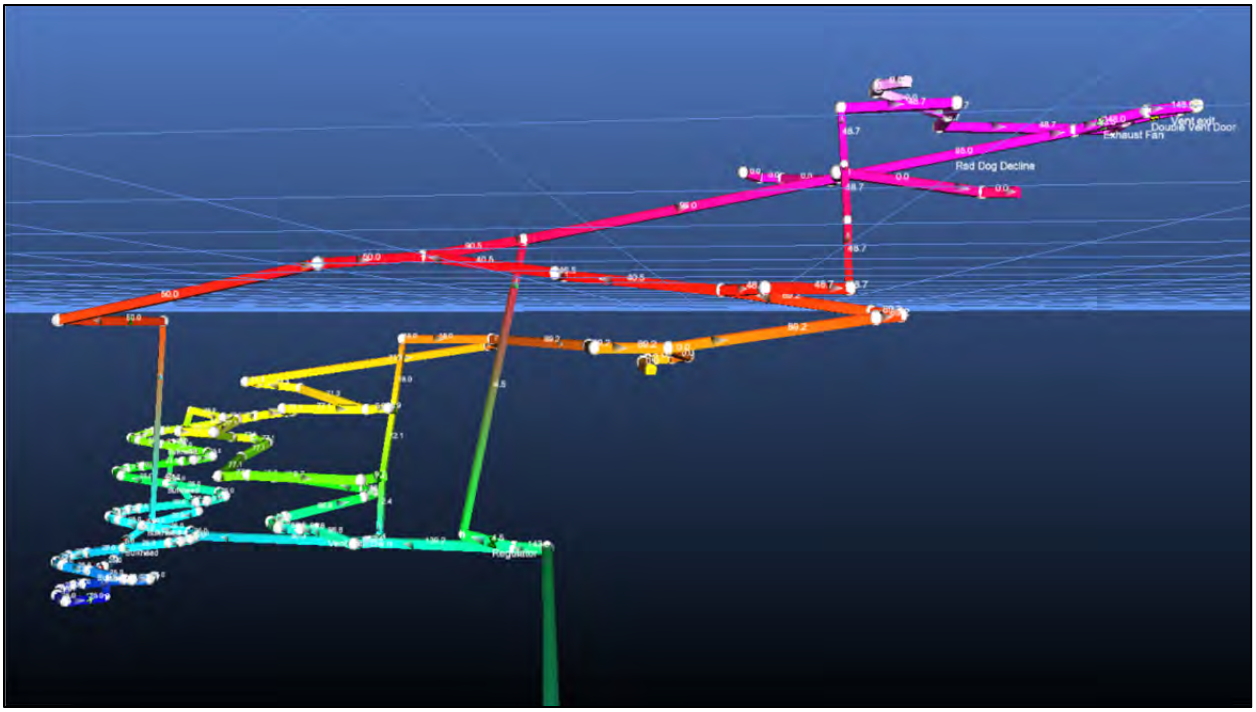
Ventsim® ventilation software was used to estimate power requirements for the ventilation network and a 400 HP exhaust fan was selected to supply the 234,000 cfm to the underground mine. This fan would be installed in a bulkhead at the upper portal and create a “pull” ventilation circuit.

The lower portal would serve as fresh air intake. Fresh air would flow through the incline and upcast through the mine.

Three ventilation raises of 3.0 m x 3.0 m would supply fresh air to production levels throughout the mine. Figure 16-6 displays the Ventsim network, arrows indicating the direction of airflow.

Auxiliary fans would circulate fresh air into active production areas, sized at 75 kW, 50 kW, and 30 kW, respectively.

Figure 16-6: Ventilation Simulation



Source: JDS (2014)

16.3.2.2 Mine Air Heating

Intake air would require heating during the winter months to prevent water from freezing underground and to provide acceptable working conditions while mining is ongoing. From December to February, when the mine is on care and maintenance, the airflow would be reduced to 30,000 cfm to minimise heating costs.

Mine air would be heated to + 2.0°C by a direct-fired propane heater located at the lower portal. A parallel heater drift would house the heater infrastructure. The air would be pulled into the heater drift by a 200 HP fan and blended with cold air entering through the main incline, allowing traffic to enter the mine without the use of double air lock doors.

Heating calculations were based on average monthly temperatures collected at the site weather station. It was estimated that 6.9 M ft^3 of propane would be required throughout the year.

16.3.2.3 Electrical Power

The majority of electrical power consumption at the mine would arise from:

- main and auxiliary ventilation fans
- mine air compressors

- batch plant
- drilling equipment
- dewatering pumps
- refuge stations.

High voltage cables would enter the mine via the two portals and would be distributed to electrical sub-stations near the mining zones. High voltage power would be delivered at 4160 V and reduced to 600 V at electrical sub-stations.

Total electrical power consumption for underground mining was estimated at 7,980,000 KWh per year.

16.3.2.4 Compressed Air

Compressed air would be required for stopers, jacklegs, face pumps and ANFO loading. An average air consumption of 0.5 m^3/s (1,000 cfm) was estimated. Compressed air would be provided by two stationary, electrically powered 100 kW compressors, with one operating and one on standby Reticulation of compressed air through the mine would be in steel pipes with 150 mm diameter.

16.3.2.5 Service Water Supply

Service water for drilling, dust control, washing and fire suppression would be sourced from a sump at the top of the underground workings and distributed in steel piping with 50 mm diameter.

16.3.2.6 Dewatering

Groundwater inflows into the mine will vary throughout the year. Increased flow rates can be expected during the snowmelt in spring. All mine water would be handled through the lower portal and directed into the tailings pond.

Old remuck bays would be converted to small sumps during access development. Small portable 15 kW submersible pumps would be installed in these sumps and would discharge into steel pipes with 150 mm diameter. Mine water from Marc and AV zones would be drained through drainholes into the main drainage piping system. A permanent sump with 100 kW submersible pumps would be installed at the bottom of JW zone since it is below the lower portal elevation.

16.3.2.7 Explosives Storage & Handling

Primary explosives storage magazines would be located on surface. Secondary magazines would be located underground to provide explosives for up to seven days. Bulk explosives and detonators would be stored in two separate facilities.

ANFO would be used as the major explosive for mine development and production. Packaged emulsion would be used as primer and for lifter holes.

Explosives handling, loading, and detonation would be carried out by trained and authorised personnel.

Typically, underground operations of this rock type require powder factors between 1.0 and 1.5 kg/t for successful blasting with good fragmentation. The following tables outline the expected consumption of explosives during development and production.

Table 16.5: Explosives Consumption for Development & Drift & Fill Headings

Explosive Consumption	Unit	per round
ANFO	kg	150
Geldyne 32 mm x 400 mm	kg	68
Xactex 19 mm x 600 mm	kg	14
Exel LP detonator 5 m	each	50
Electric detonator 3.5 m	each	2
B-Line	m	47
Total Explosive Consumption	kg	232
Powder Factor (based on 4 m round at 3.0 t/m^3)	kg/t	1.34

Source: JDS (2014)

Table 16.6: Explosive Consumption for Longhole Stope Production

Explosive Consumption	Unit	Units/stope
ANFO	kg	18,462
Geldyne 40 mm x 400 mm	kg	785
Pentex 90 g booster	each	169
Exel MS detonator 15 m	each	169
Exel MS detonator 18 m	each	169
Exel MS detonator 25 m	each	169
Electric detonator 3.5 m	each	10
B-Line	m	300
Total Explosive Consumption	kg	19,247
Powder Factor (based on 12,200 tonne stope)	kg/t	1.51

Source: JDS (2014)

16.3.2.8 Fuel Storage & Distribution

Haul trucks, LHDs and auxiliary mobile equipment would be refuelled on surface at a fuel station from a 15,000 litre Enviro-Tank located close to the lower portal. Drilling equipment would be refuelled underground with a fuel/lube truck.

16.3.3 Underground Transport of Personnel & Materials

During the initial year of underground development, access to the mine would be established through the upper portal and exploration decline. The existing tote road wouldbe upgraded to provide access to the upper portal. As soon as the lower portal and incline would be connected to the upper levels through the Marc / AV ramp system, all traffic would be directed through the lower portal.

A personnel carrier would be used to shuttle workers from surface to the underground workings during shift changes. Supervisors, mechanics, engineers, geologists and surveyors would use Toyota trucks as transportation underground. A boom truck and forklift would be used to transport supplies and consumables from surface to active underground workplaces.

16.3.4 Underground Mine Equipment

The required underground mobile equipment was determined based on the selected mining methods, mine production rate and geometry of the mine workings. Scheduled quantities of work in combination with cycle times, productivities, availabilities, and efficiencies formed the basis to determine the fleet size.

Table 16.7 summarises the underground mobile fleet.

Table 16.7: Mobile Mine Equipment Requirements

Description	Max Req.
Underground Equipment Quantities	
Haulage Truck (20 t)	3
LHD (3.0 m^3 - Rammer Jammer D&F)	1
LHD (4.6 m^3)	3
Jumbo (2 boom)	2
Longhole Drill (76mm)	2
Scissor Truck	2
Boom Truck	1
Fuel/Lube Truck	1
Mechanic Vehicle	1
Supervisors Vehicle	1
Man Carrier	1
Mine Rescue Vehicle	1
Cement Delivery Truck	1
Grader/Dozer	1
Forklift/Telehandler	1
Diamond Drill	1

Source: JDS (2014)

Haulage requirements for LHDs and trucks were estimated for mineralised material, waste and backfill.

Two development crews with dedicated jumbos, LHDs and scissor lifts would drive the critical path development at Marc and the incline during pre-production. Some development equipment would be used for drift and fill mining later in the mine life, when the main access development is completed.

Two-boom electro-hydraulic jumbos would be used for the mine development.

Bolting would be performed from scissor lifts using stopers and jacklegs.

Two top hammer longhole drills were specified to allow for redundancy and efficient production drilling in two zones. Drainholes and electrical cable holes would also be completed with the longhole drills.

Definition drilling would be performed with a small electric diamond drill.

16.3.5 Mine Equipment Maintenance

Mobile underground equipment would be maintained at the mine shop located at the mill facility site. Major rebuild work would be performed off site. Minor maintenance and repairs would be done underground to minimise tramming of equipment to surface. A mechanics truck would be used to perform underground repairs.

The mine shop would also include a mine dry and offices for mine supervision, engineering, geology, and maintenance planning. The mine shop complex would be constructed from ATCO skid-mounted trailers and wash cars. Sea cans and laydown areas would be used for warehousing and management of supplies.

16.3.6 Mine Personnel

The Red Mountain mine department would employ 39 people during mine development and ramp up. Once in full production there would be approximately 90 mine employees rotating on a two-week-on, two-week-off schedule.

Mine employees would live in Stewart during their two-week rotations and travel to and from the mine site through a bus service that would run twice daily. During the one-hour trip to site, all employees would receive half-time pay. The mine would operate on two 12-hour shifts per day, seven days per week

Table 16.8 below outlines the anticipated mine labour force quantities, and rotation schedules.

Table 16.8: Mine Labour Force

Position	Roster	Rotation	On Site	LOM/ Max. Quantity	Year -1	Year 1
Mining Operations						
Mine Super/Mine GF	Staff	2&2	1	2	1	2
Mine Supervisor/Shift Boss	Hourly	2&2	2	4	3	4
Safety / Mine Rescue / Training Officer	Hourly	2&2	1	2	1	2
LH Drillers/Blasters	Hourly	2&2	4	8	0	8
Ground Support, Hanging Services	Hourly	2&2	4	8	3	8
Fuel/Lube/Boom/Grader/Telehandler	Hourly	2&2	2	4	1	4
Jumbo & LHD Operator	Hourly	2&2	10	20	11	20
Truck Operator	Hourly	2&2	6	12	5	8
Mining Operations - Total			**30**	**56**	**25**	**56**
Backfill						
Backfill Crew - Bulkheads, Piping, Monitor	Hourly	2&2	2	4	0	4
CRF Plant Operators	Hourly	2&2	1	2	0	2
Backfill - Total			**3**	**6**	**0**	**6**
Mining Maintenance						
Mine Maintenance Supervisor/Lead Hand	Hourly	2&2	2	4	3	4
Maintenance Planner	Staff	2&2	1	2	1	2
HD Mechanic	Hourly	2&2	4	8	3	8
Welder	Hourly	2&2	1	2	1	2
Electrician	Hourly	2&2	1	2	1	2
Mine Maintenance - Total			**9**	**18**	**7**	**18**
Mining Technical Services						
Chief Mine Engineer/Sr. Mine Engineer	Staff	2&2	1	2	1	2
Mine Planning Engineer	Staff	2&2	1	1	1	1
Surveyor	Staff	2&2	1	2	1	2
Mine Technologist	Staff	2&2	1	2	1	2
Senior Geologist	Staff	2&2	1	1	1	1
Grade Control Geologist	Staff	2&2	1	2	1	2
Technical Services - Total			**5**	**10**	**7**	**10**

Source: JDS (2014)

17.0 RECOVERY METHODS

17.1 PROCESSING METHODS

Gold and silver will be extracted by cyanidation from mine feed delivered to the mill complex. The mine feed will be stage crushed by a jaw and cone crusher and stored in a feed hopper bin. At a nominal rate of 1000 t/d, the crushed feed will be processed in a conventional rod and ball mill grinding circuit followed by thickening prior to leaching. Target grind size will be 95% passing 38 µm.

Gold and silver extraction will be accomplished by leaching in five sequential leach tanks. Following leaching, precious metals will be extracted from the pregnant solution with carbon in pulp (CIP) adsorption occurring in 6 reaction tanks. Precious metals will be removed from the carbon by carbon elution followed by electrowinning and refining to produce doré. A carbon regeneration circuit will re-activate the carbon for reuse in the CIP circuit. Tailings will be treated with SO_2 and air to destroy cyanide prior to discharge to the tailing impoundment area.

All equipment will be enclosed in a weatherproof building with the exception of the thickener, five leach tanks and the process water tank. These items will be located outdoors adjacent to the mill facility.

The mill will operate with 24 hours per day, 7 days per week for 8 months during the warmer months. Mill availability while operating was assumed to be 92% to allow for regular maintenance, unplanned maintenance and power outages.

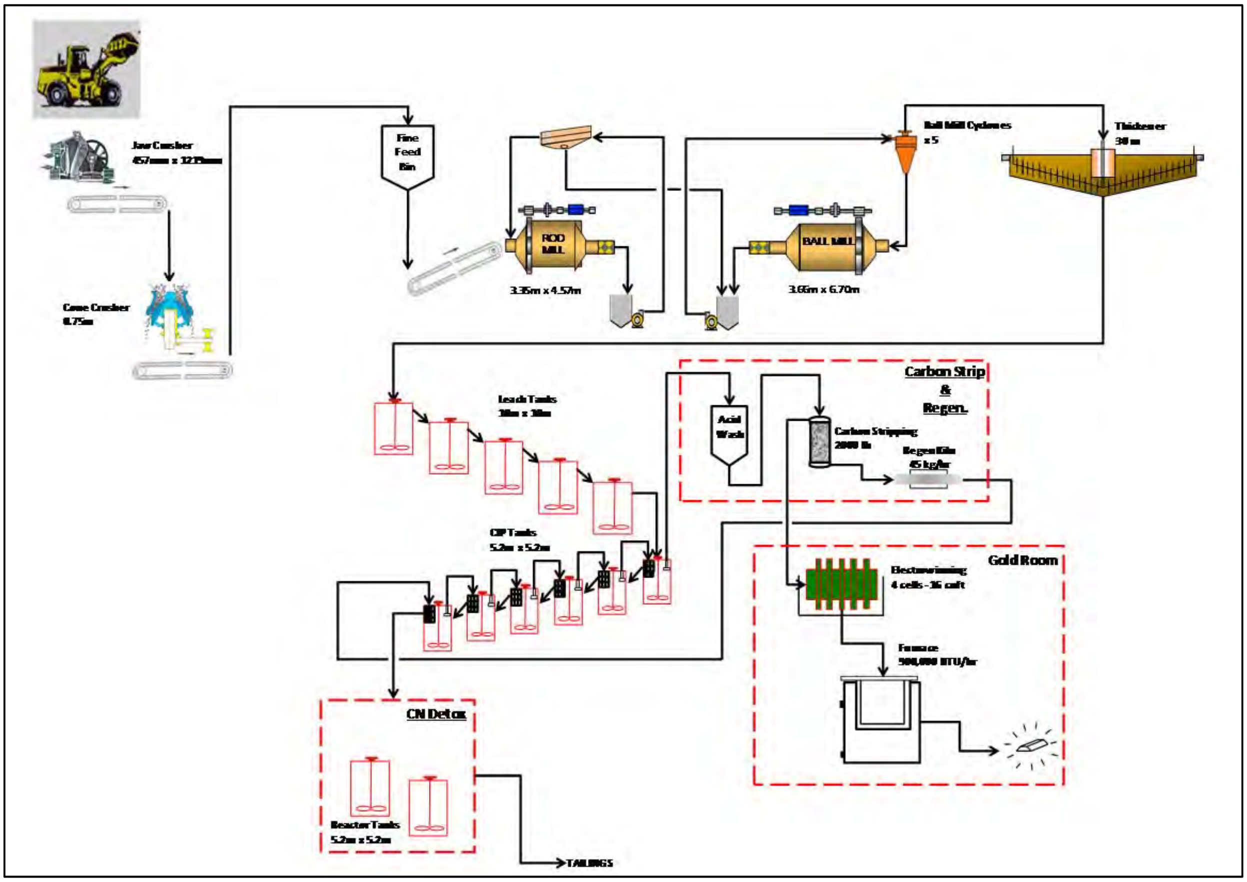
Selected slurry streams will be sampled, prepared and assayed on site for metallurgical accounting purposes. A small assay lab and metallurgical preparation facilities will supplied in containerized modules located near the mill facility.

Due to the limited site space, only a limited supply of reagents and mill supplies will be stored at the mill. A large supply of items will be maintained in Stewart at a larger warehouse facility and will be transported to the mill as required.

17.2 FLOW SHEET

A simplified schematic of the process flow sheet is displayed in Figure 17-1.

Figure 17-1: Simplified Process Flow Sheet



17.3 PLANT DESIGN & EQUIPMENT CHARACTERISTIC

Run of mine will feed a 457 mm x 1,219 mm jaw crusher then a 0.75 m short head cone crusher both operated in open circuit. The crushed feed will have a target size of minus 20 mm and will be transferred to a fine feed surge bin by conveyor.

The feed bin will deliver material to the grinding circuit and will be first ground in a 3.35 m x 4.57 m rod mill (11 ft x 15 ft) powered by a 750 kW motor. The rod mill will be operated in closed circuit with a 1.22 x 2.44 m vibrating screen with 5 mm slot openings. The oversize from the screen will be returned to the rod mill and the undersize will pass to the ball mill. The 3.66 x 6.70 m ball mill will be powered by a 1,119 kW motor and will be operated in closed circuit with five cyclones having a diameter of 200 mm.

The cyclone overflow from the grinding circuit will flow by gravity to the 30 m diameter thickener where the pulp density will be increased to approximately 45% solids. Thickener overflow will be returned to the process water tank along with reclaimed water from the tailings impoundment area. The process water will be reused in the grinding circuit.

The thickened slurry will be pumped to a series of five leach tanks. The tanks will be 10 x 10 m and will provide nominal 48 hours of leaching time. Air will be added to the leach tanks as required to maintain elevated oxygen levels. The leach circuit will have both cyanide and pH levels monitored and maintained. Based on current test results, the leach circuit will be operated at a cyanide concentration of 600 to 1000 ppm and the pH of the circuit will be maintained between 10.5 and 12 pH with lime.

After leaching, the slurry will flow by gravity to six 5.2 x 5.2 m CIP tanks. The last stage of CIP will overflow onto a 20 mesh safety screen to recover carbon fragments generated in the CIP tanks. The undersize from the safety screen will be pumped to the INCO SO_2/air cyanide destruction process. Once treated, the tailings will flow to the tailings impoundment area for final disposal.

Fresh carbon will be conditioned then added to the last tank of the CIP circuit. Carbon will be transferred counter current from the last CIP tank to the first tank using recessed impeller pumps to minimize attrition and wear on the carbon. Carbon from the first CIP will be dewatered on a vibrating screen and be processed by acid washing, stripping and regeneration. A nominal 1,000 kg of carbon will be processed in a single cycle. Two strip cycles will be required per day.

Carbon stripping will be done using a caustic cyanide solution at elevated temperature and pressure. The solution containing the gold and silver will then be passed to the electrowinning cells for recovery of the precious metals. The electrowinning sludge will be recovered and smelted to doré in a furnace.

Carbon regeneration will be performed in a horizontal rotary kiln. After regeneration, the carbon will be quenched and be available for reuse in the CIP circuit.

17.4 REAGENTS HANDLING & STORAGE

Reagent mixing and storage will be in the plant for NaCN, Lime, flocculant. Smaller scale reagent mixing and handling for NaOH and HCl related to the carbon stripping will be contained within the carbon stripping system. Similarly, the reagents required for cyanide destruction, copper sulphate and sulphur dioxide, will be mixed and stored in the cyanide destruction area.

The reagent with the largest consumption will be lime and a lime silo and slaking unit will be placed adjacent to the mill facility. The lime slaking unit capacity was 454 kg/h, which will meet the peak demand of 10 kg/t required by some of the more difficult to treat zones.

The cyanide mixing system will be performed in a two-tank system to ensure constant supply. Cyanide will be delivered to various locations in the process by metering reagent pumps. The reagent system capacity was assumed to provide a maximum of 2.5 kg/t of cyanide.

A flocculant mixing unit designed to deliver a maximum of 50 g/t to the thickener has been assumed. Other minor reagents required for the production of doré are borax, $NaNO_3$ and silica. Minor amounts sulphuric and nitric acid will be used in the assay laboratory.

17.5 ENERGY, WATER & PROCESS MATERIALS

A summary of the energy demand for major mill equipment is displayed in Table 17.1.

Table 17.1: Summary of Mill Energy Demand

Installed HP	5,283
Installed kW	3,939
Usage factor	0.75
Operating kW	2,956
kWh/a	17,026,560

Power will be delivered to the mill MCC at 4160 V.

There will be three water systems at the mill facility: a fresh water system to supply mill potable water requirements, a process water system to supply mineral processing requirements; and a separate fire suppression water system.

The process water system will deliver an average of 2,000 m^3/d of water to the grinding and leach circuit. Reclaim water from the thickener overflow and the tailings impoundment area will be recycled to the process water tank. The required amount of make-up water was assumed to be 25%. Sources of make-up water, (approximately 500 m^3/d) will be source from mine dewatering. A 9.1 x 6.5 m process water tank will contain about 420 m^3 of water. This tank will be located near the mill and has two process water pumps to supply water to the mill.

18.0 PROJECT INFRASTRUCTURE

18.1 SUMMARY

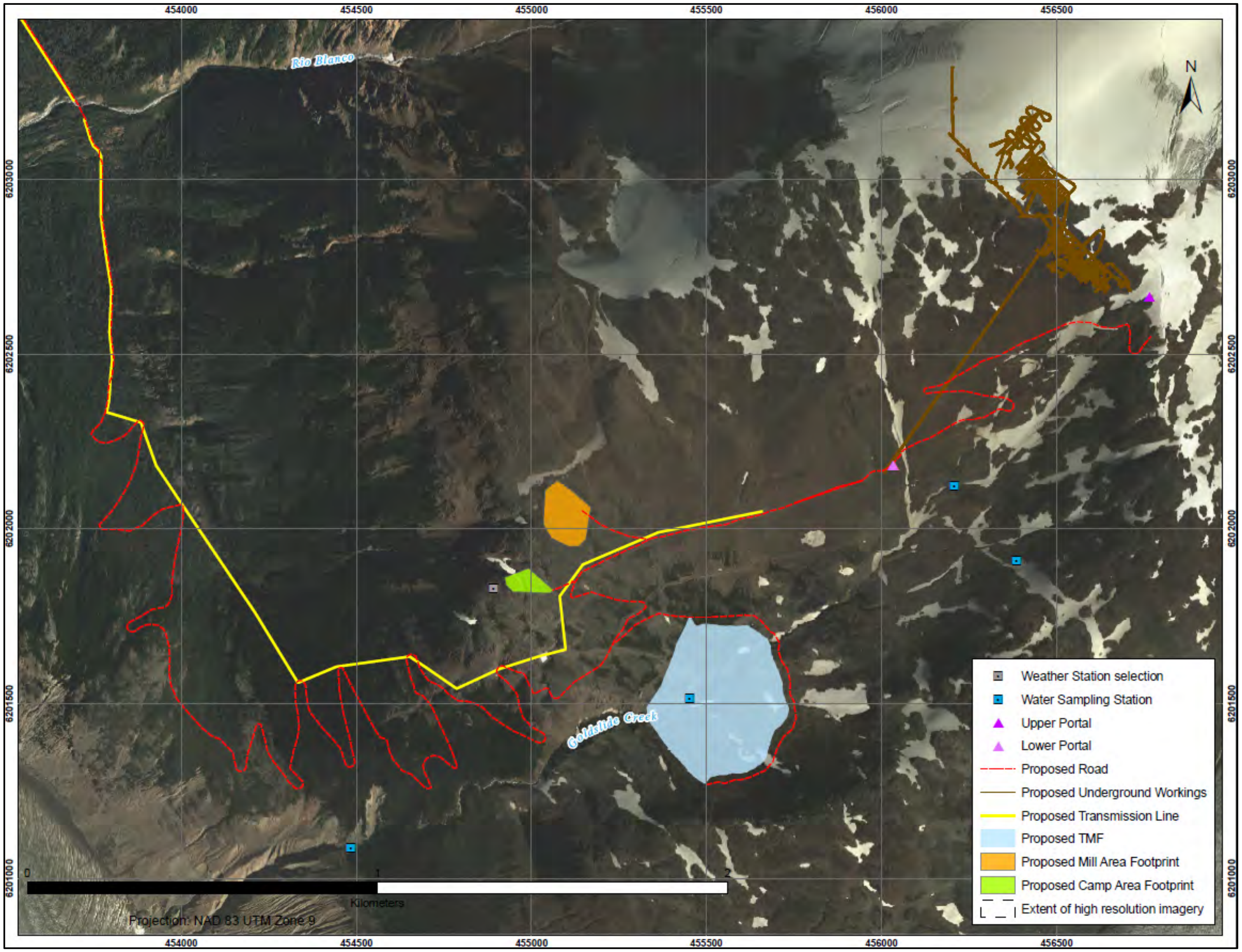
The project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 25 km seasonal access road from Highway 37A to the project site.
- 5 km of on-site service roads to access the mine portals, tailings management facility and other working areas.
- Tailings management facility and impoundment.
- Temporary development waste storage areas (note that waste rock generated by development and mining is rehandled into the underground workings as backfill).
- Crushing and grinding circuits and gold extraction plant.
- Administration office, mine dry, maintenance shop, warehouse and emergency camp.
- Electrical connection to BC Hydro, transmission line adjacent to the seasonal access road and on-site substation and distribution network.
- Process and fire water storage and distribution.
- Sewage collection and treatment system.

18.2 GENERAL ARRANGEMENT

The site general arrangement is shown in Figure 18-1. The facilities have been located where the terrain is less steep and minimal earthworks are required to provide suitable footprint for the surface structures and equipment. The tailings management facility would be located in the Cirque valley adjacent to the mill complex. The proposed location minimises the dam footprint, construction volume and catchment area, while maximizing the storage capacity of the impoundment.

Figure 18-1: Site General Arrangement



Source: IDM (2014)

18.3 SITE ACCESS

18.3.1 Site Access Road

Approximately 25 km of seasonal access road is required to connect the Highway 37A to the project site. Where possible, the road would follow the existing right-of-way developed by Lac Minerals in 1994. The road is planned to be a radio controlled, single-lane, gravel mine access road with inter-visible turnouts. The road would be approximately 6 m wide and include a ditch at one side and adequate space cleared to accommodate an overhead power line. The maximum designed grade of the road is 14%.

Personnel would reside in Stewart and commute to and from the project site on a daily basis. Consumables would be consolidated at a warehouse in Stewart and delivered to site as required.

18.3.2 On-Site Roads

Where necessary, the existing tote road network would be upgraded to provide light vehicle access to the different working areas and facilities, including: the upper mine portal, mill complex, tailings management facility and environmental monitoring points.

The road to the upper portal serves as an alternative route for underground trucks to deliver mineralised material to the mill. This road is planned to be 8.4 m wide including a 1.5 m tall shoulder berm for single-lane truck traffic. The maximum design grade of the road is 15%.

18.3.3 Avalanche Control

Prior to mill start-up each year, a helicopter supported avalanche control campaign would be conducted to reduce the avalanche hazard potential. During operations, avalanche survey and control flights would be taken after key environmental events (snow, freeze thaw cycles, etc.).

18.4 POWER SUPPLY

Total average annual power consumption is estimated to be approximately 25,000,000 kWh/a (4.3 MW). The project envisions connecting the Red Mountain mine and mill complex to the BC Hydro electrical transmission system near Stewart, BC. It is expected that two electrical substations would be required. The first sub-station will step down power for transmission to site at 38 kV, and the second; to step down for on-site distribution at 4160 and 600 V. It is planned that the transmission line would run adjacent to the seasonal access road.

Further studies in coordination with BC Hydro are required.

18.5 FUEL STORAGE & DISTRIBUTION

Diesel would be trucked to the project site from Stewart on an as-needed basis and stored in a 15,000 litre 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 & lube truck.

18.6 WATER MANAGEMENT

18.6.1 Surface & Underground Water

A network of surface ditches would be used to collect and divert runoff to the environment before it reaches the mill and tailings complex. Ditches below the temporary development waste storage areas would direct water to the tailings management facility.

Water pumped from the underground mine would be used as make-up water in the mill process or diverted to the tailings management facility.

18.6.2 Fire Water

The firewater tank would be located on the hillside above the mill complex and ancillary facilities in order to provide sufficient hydraulic head for firefighting. The tank would be sized to provide volume for firefighting at any one of the surface structures for a period of two hours.

18.6.3 Potable Water

Potable water for site personnel, emergency showers, etc. would be trucked in and stored in reusable containers located around the site, including: the administration offices, first aid, emergency camp, maintenance facility, mill and in several locations underground.

18.6.4 Sewage Collection & Treatment

Sewage would be treated in a septic tank and field. Solid waste, stored in the septic tank, would be pumped out and trucked to Stewart for disposal.

18.7 WASTE MANAGEMENT

18.7.1 Temporary Development Waste Storage Areas

Waste rock generated from the development of the underground mine would be stored in two temporary surface storage areas located adjacent to the Upper and Lower portals. Both areas would contain approximately 150,000 t of potentially acid generating waste rock (~300,000 t total). The schedule of total waste rock stored on surface is summarised in Table 18.1.

Table 18.1: Temporary Development Waste Storage Areas

Year	Units	-2	-1	1	2	3	4	5
Waste Stockpile	t	150,000	304,000	273,000	197,000	112,000	-	-

Source: JDS (2014)

All waste is planned to be rehandled into underground stopes as backfill over the project life.

A network of surface ditches would be used to divert runoff below the waste storage areas to the tailings management facility.

18.7.2 Tailings Management Facility

The conceptual design for the tailings management facility expands on technical data available as part of a draft report prepared for Lac Minerals by Klohn-Crippen (1994). The tailings management plan envisions storage of approximately 1.4 Mt of processed material at a long-term settled density of 1.2 t/m^3. The potentially acid generating tailings would be stored subaqueously in a lined impoundment behind a dam located in the Cirque valley adjacent to the mill complex, as shown in Figure 18-1. As recommended by Klohn, subaqueous disposal is proposed in order to control the acid generating potential of the tailings. The dam would be designed to maintain saturated conditions throughout the tailings mass. The location of the dam minimises the footprint, construction volume and catchment area, while maximizing the storage capacity of the impoundment. The dam would be located downstream of the mill complex and this minimises the power required for tailings pumping.

The tailings dam is designed to be a rock-filled structure with granular filter zones on the upstream side. The impoundment and upstream face of the dam would be covered with a geosynthetic liner to minimise seepage of tailings and water into the surrounding area. The filter zones would provide a bed for the liner and ensure that fine tailings are not piped through the rock-fill dam in the event of liner damage. The upstream and downstream slope angles would be 26.6° (2H:1V slopes).

The dam would be constructed using the downstream method with locally borrowed talus and colluvium materials. The talus and colluvium would be screened on site to produce material for the granular filter zones.

An initial starter dam would be constructed to contain the first two years of tailings production in order to minimise upfront capital expenditure. The dam would be raised several times over the mine life to increase the storage capacity and maintain a minimum of 5 m freeboard at all times. The proposed schedule of dam construction is summarised in Table 18.2.

Table 18.2: Tailings Management Facility Raise Schedule

Year	Units	-2	-1	1	2	3	4	5
Dam Crest	masl	-	1,443	1,461	1,469	1,469	1,472	1,472
Raise Height	m	-	18	18	8	-	3	-
Dam Construction Volume[1]	Mm^3	-	0.3	0.3	0.5	-	0.2	-
Impoundment Capacity[2,3]	Mt	-	0.2	0.5	1.1	1.1	1.4	1.4
Cumulative Total Milled	Mt	-	-	0.3	0.5	0.8	1.1	1.4

Source: JDS (2014). Notes: 1. Dam construction volume includes rockfill, sand and gravel, and riprap placement. 2. Impoundment capacity is solids only and excludes volume available for freeboard. 3. Impoundment capacity based on long-term, settled density for tailings of 1.2 t/m^3.

The dam would have sufficient freeboard to contain runoff collected during the winter shutdown period. Water from the impoundment would be recycled to the mill as makeup water and a siphon and backup spillway would be used to discharge excess water from the impoundment once water quality guidelines are met. The tailings management plan assumes discharging approximately 3.0 Mm^3 of water each year. Alternatively, ditches would be used to divert the majority of inflows around the impoundment and limit the need to discharge water.

At closure, the tailings facility would be covered by a geosynthetic liner and 1 m thickness of material to prevent infiltration. The cover would be graded to create natural drainage to prevent erosion and a permanent spillway constructed to carry excess water away from the dam toe. A care and maintenance program would be established to monitor the performance of the dam.

The capital cost for the tailings management facility is estimated from first principles based on volumetric material movement, material placement and additional costs for the HDPE liner and mechanical components.

Further field and engineering work is required for higher levels of study.

18.7.3 Industrial Waste

All methods of reducing and recycling would be employed to minimise the amount of industrial waste generated by the project. Solid waste (scrap steel, wood, etc.) would be collected in bins; empty chemical totes, lubricant drums, etc. would be collected and compacted. All industrial waste would be back-hauled off-site for disposal or recycling in an appropriate manner.

18.8 MILL COMPLEX & ANCILLARY FACILITIES

The mill equipment would be enclosed within pre-engineered steel buildings.

Skid-mounted trailers are planned for the administration offices, mine dry and first aid room. Once project construction and commissioning are complete, the construction offices would be re-purposed to provide emergency shelter. The emergency camp will include propane heat, dorm accommodation, microwaves and a supply of frozen meals. The surface maintenance shop will be a single-bay, fabric covered structure suitable for preventative maintenance (PM) services, basic repairs and component replacement. More extensive repair work would be conducted off-site. Equipment would be washed underground.

The primary warehouse would be located in Stewart. Sea-containers would be used to store spare parts and consumables at site.

18.9 EXPLOSIVES STORAGE

Explosives would be stored at a secured and monitored site located approximately 1 km from populated, high traffic areas. The final location of the explosives storage site would be determined as part of future prefeasibility or feasibility studies. Boosters and detonators would be stored in locked and barricaded sea-containers and separated according to Natural Resource Canada guidelines.

18.10 INFORMATION TECHNOLOGY & COMMUNICATIONS

The administration offices, mill complex, maintenance facility and first aid offices would include a wireless computer network and satellite phone system. Hand-held radios would be used to provide voice-communication between personnel on surface.

19.0 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

19.1 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 Piesold 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 utilised this data. The historic environmental database was utilised 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, socioeconomics, 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 the need for the additional studies summarised in Table 19.1 to update the baseline to current environmental conditions, to address refinement of the project design and current regulatory requirements. The gap analysis results indicated additional information is required for the project area's atmosphere/climate, surface hydrology, aquatics, water and sediment, terrestrial wildlife and fish habitat. IDM will also complete comprehensive baseline studies of rock geochemistry, archaeology and heritage resources, land use, cultural, and socioeconomic baseline studies will be completed to characterise the regional human environment. Where available, traditional ecological knowledge will inform the assessment of the effects of the project. Further, IDM Mining will work with the Nisga'a with respect to the collection of Traditional Knowledge and Use Information to further identify potential project effects. Mitigation measures will be developed in consultation with the Nisga'a through the Environmental Assessment process.

Table 19.1: Anticipated Priority Studies for 2014 & 2015

Baseline Component	Additional Information
Terrain and Physiography	New mapping is required to reflect changes as the glaciers of the Cambria Icefield have retreated. Updates to a natural hazards assessments in the Bitter Creek Valley proximal to the proposed mine and access route.
Water Quality	Update water quality data to address gaps and project design refinements.
Climate	Install meteorological station to extend historical data.
Hydrology	Extend monitoring of stream flow to support project design, fisheries and water quality assessments.
Hydrogeology	Extend monitoring of groundwater to support project design, fisheries and water quality assessments.
Wildlife and Vegetation	Establish baseline for wildlife for the Bitter Creek Valley to support the assessment of project effects.
Fisheries	Establish baseline for fisheries for the Bitter Creek and Bear River to support the assessment of project effects.
ARD/ML	Further 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 in the vicinity of project facilities.
Socioeconomics	Establish a baseline for the socio-community and economic characteristics of the area to support an assessment of project effects
Culture and Heritage	It is anticipated that an archaeological overview assessment will be required pursuant to the *BC Heritage Act*, and to address the interests and concerns of First Nations.
First Nations Interests	It is anticipated that consultation with First Nations may require additional studies to address First Nations interests.

19.2 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 Premier Gold Mine, Dolly Varden Silver Mine, Eskay Creek, and Snip.

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.

19.2.1 Vegetation

The majority of the Red Mountain project site lies within the alpine area above the local treeline, which occurs at approximately 1300m 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 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 characterised 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 characterised by interspersed sedge and mountain heather shrubs (BC Ministry of Forest Bro51, 1997).

Some clearing of timber will be required for the construction of infrastructure. This is roughly estimated to be 25 hectares of timber clearing over the entire project.

19.2.2 Wildlife

The wildlife species present within the proposed Red Mountain 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 unforested area surrounding the immediate project 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 (Royal Oak, 1996).

19.2.3 Fisheries & Aquatic Resources

The Red Mountain 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, pink salmon, chum salmon, sockeye salmon, and trout.

Fish resource and fish habitat studies conducted in 1993 indicated that there is limited usage and available fish habitat in Bitter Creek. Dolly Varden was the main fish species to use Bitter Creek and the lower reaches of Roosevelt Creek (Royal Oak, 1996).

19.3 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 100 people during the two-year construction phase. During the six-year mine operational phase, seasonal employment of up 394 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.

19.3.1 Social Community

The workforce for construction and operations is expected to live in Stewart, which has sufficient facilities and infrastructure to accommodate the potential increase in residents both during construction and 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 District of Stewart was 494 in 2011.

19.3.2 First Nation Land Use

The Red Mountain property falls within the Nass Wildlife Area as set out in the Nisga'a Final Agreement (NFA). Pursuant to the NFA, the Nisga'a Nation has rights to the management and harvesting of fish and wildlife within the Nass Wildlife Area. The Nisga'a Nation holds the estate in fee simple to a parcel of land known as Scamakounst. A former Indian Reserve, Scamakounst is defined as Category A Land in the NFA, and is situated on the east bank of the Bear River approximately 500 m across from Stewart, BC. Waters draining from Red Mountain flow into Bitter Creek, which in turn flows into the Bear River upstream of Scamakounst.

Nisga'a Nation citizens are closely tied to the land and practice traditional and cultural activities, including seasonal resource harvesting of terrestrial and marine plants, hunting and trapping wildlife, and fishing. The website of the Nisga'a Lisims Government outlines the importance of the land to the Nisga'a Nation's culture, governance and survival. The Nisga'a Lisims Government stresses the importance of the Nisga'a's traditional system of land ownership which "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).

The Nisga'a Lisims Government also writes about the importance of the land as the staging ground of the Nisga'a's traditional history stories (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 with Nisga’a 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
- tourism.

In addition to the traditional cultural importance of the land, the Nisga’a Nation is active in modern economic resource development including, forestry tenures, commercial recreation, angling licences and traplines.

19.3.3 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 elections are held every two years and the most recent election was in late 1999. At that time, there was a change of Mayors but the attitude of the local government remains favourable to commercial development in the area.

The Red Mountain 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-Stikene, which provides government services to the area.

19.4 ENVIRONMENTAL APPROVALS

The Red Mountain project will require a review by the British Columbia Environmental Assessment Office (EAO) pursuant to the *British Columbia Environmental Assessment Act* (BCEAA) 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 various former Property operators.

Upon project approval, a number of permits from various government agencies will be required. No technical difficulties are anticipated for obtaining these permits. A reclamation bond must be deposited with the government on the issuance of permits. It is anticipated that the cost of the bond will increase from the $1,000,000 that is already withheld.

Royal Oak Mines Inc., a previous Property owner, entered into the environmental review process in 1996. However, the project was subsequently withdrawn from the process due to financial difficulty. In 2000, a Review of Baseline Studies at Red Mountain (SRK, 2000a) which presented the results of a review of the baseline studies that were completed, identified gaps in the information, and recommended further data and analysis that would be required to support a project application.

No federal decision pursuant to the *Canadian Environmental Assessment Act* 2012 (CEAA, 2012) is required.

19.5 ANTICIPATED PROVINCIAL PERMITS & AUTHORISATIONS

Provincial permitting and licensing (statutory permit processes) is expected to proceed concurrently with the environmental review pursuant to the *BC EAA*. No permits for the project will be issued before an Environmental Assessment Certificate (EAC) is obtained. Consequently, IDM will apply for concurrent permitting within the environmental review process for all permits. Concurrent permitting will expedite the permitting process following issuance of the EAC and reduce the time to start of construction.

Table 19.2 presents a list of provincial authorisations, licences, and permits required to develop the project. The list includes only the major permits and is not intended to be comprehensive.

Table 19.2: List of Anticipated Provincial Permits & Authorisations

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*
s.14 Investigative Permit	Ministry of Forests, Lands and Natural Resource Operations	*Heritage Conservation Act*
s.14 Inspection Permit	Ministry of Forests, Lands and Natural Resource Operations	*Heritage Conservation Act*
S.12 Site Alteration Permit	Ministry of Forests, Lands and Natural Resource Operations	*Heritage Conservation Act*
Burning Reference Number	Ministry of Forests, Lands and Natural Resource Operations	*Wildfire Act*
s.9 Approval or authorisation 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		

Source: JDS (2014)

19.6 ANTICIPATED FEDERAL PERMITS & AUTHORISATIONS

Federal authorisations and permits that may be required are listed in Table 19.3.

Table 19.3: List of Anticipated Federal Permits & Authorisations

Permits	Agency	Legislation
Explosives Permit	NRCan	*Explosives Act*
Authorisations under the *Fisheries Act*	DFO	*Fisheries Act*

Source: JDS (2014)

19.7 MINE CLOSURE

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 in the form of cemented paste backfill. Following closure, the underground mine will be allowed to flood and the mine portals and ventilation raises will be collapsed or blocked. To seal the lower portal, a permanent hydrostatic concrete plug will be installed to prevent outflow of mine water. The upper portal will be sealed to protect against surface water entering the mine. Any potentially acid generating mine tailings left on surface at the Red Mountain Cirque will be stored in an engineered lined tailings management facility, sloped, and capped to prevent potential acid generation problems.

The structures on the Red Mountain Gold Property 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 21.0. It is assumed that the salvage values from mill, mobile and stationary equipment will be adequate to pay for the cost of closure and reclamation.

19.8 SITE MANAGEMENT & MONITORING

IDM will design, construct, operate, and decommission the Red Mountain 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)
- *Wildlife Act* (BC).

IDM will develop and implement an Environmental Management System that will define the processes by which compliance will be met and demonstrated, and 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 will develop a water management plan that applies to all mining activities undertaken during all phases of the Red Mountain project. The goals of this management plan will be to:

- 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 and guidelines.

19.9 WASTE ROCK & TAILINGS DISPOSAL

The main waste management issue for the Red Mountain project is the prevention and control of ARD from the tailings and any potentially acid generating rock that is produced during mine development or operation.

The Red Mountain project will create both 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 Red Mountain Portal. Project data indicates that 90,000 tonnes are currently stored there; specifically, there are 5,000 tonnes adjacent to the portal and 85,000 tonnes 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 pHs and moderate sulphate levels. Paste pHs 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 recontouring 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).

19.10 SITE MONITORING

As a condition of the Mineral and Coal Exploration Activities & Reclamation Permit No. MX-1-422 (BC MEM, 2002), Seabridge is required to complete annual monitoring activities to document conditions at the Red Mountain site, including:

- Collection and analysis of seep and crib drainage samples.
- Monitoring of dump weathering.
- Documenting general site conditions.

Through the IDM Option Agreement with Seabridge (reference: Section 4.3.1 this report), responsibility of the annual monitoring work is transferred to during proposed operations, an environmental monitoring program will be implemented. Details of the monitoring program will be determined in further studies.

19.11 WATER MANAGEMENT

Potential water sources consist of underground mine development drainage, decant from the tailings storage facility and basin drainage from Goldslide Creek.

Water stored in the tailings facility will be naturally decanted and drawn down for mill processing water through a reclaim barge in the pond. The tailings management plan assumes discharging approximately 3.4 Mm^3 of water each year once water quality guidelines have been met. Alternatively, ditches would be used to divert the majority of inflows around the impoundment and limit or eliminate the need to discharge water.

Cyanide destruction of tailings will occur at the processing plant prior to discharge into the tailings management facility. As currently conceived, there will not be a need for a water treatment plant at the site. However, should further investigations determine that some level of treatment is required, it has been determined that a water treatment plant can be utilised to treat mine drainage water and the runoff from temporary stockpiles of waste rock if required prior to placing waste rock underground as stope backfill.

20.0 MARKET STUDIES & CONTRACTS

20.1 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.

20.2 CONTRACTS

No contractual arrangements for concentrate trucking, port usage, shipping, smelting or refining exist at this time. Furthermore, no contractual arrangements have been made for the sale of gold doré at this time.

20.3 ROYALTIES

The project was evaluated utilizing the following royalties:

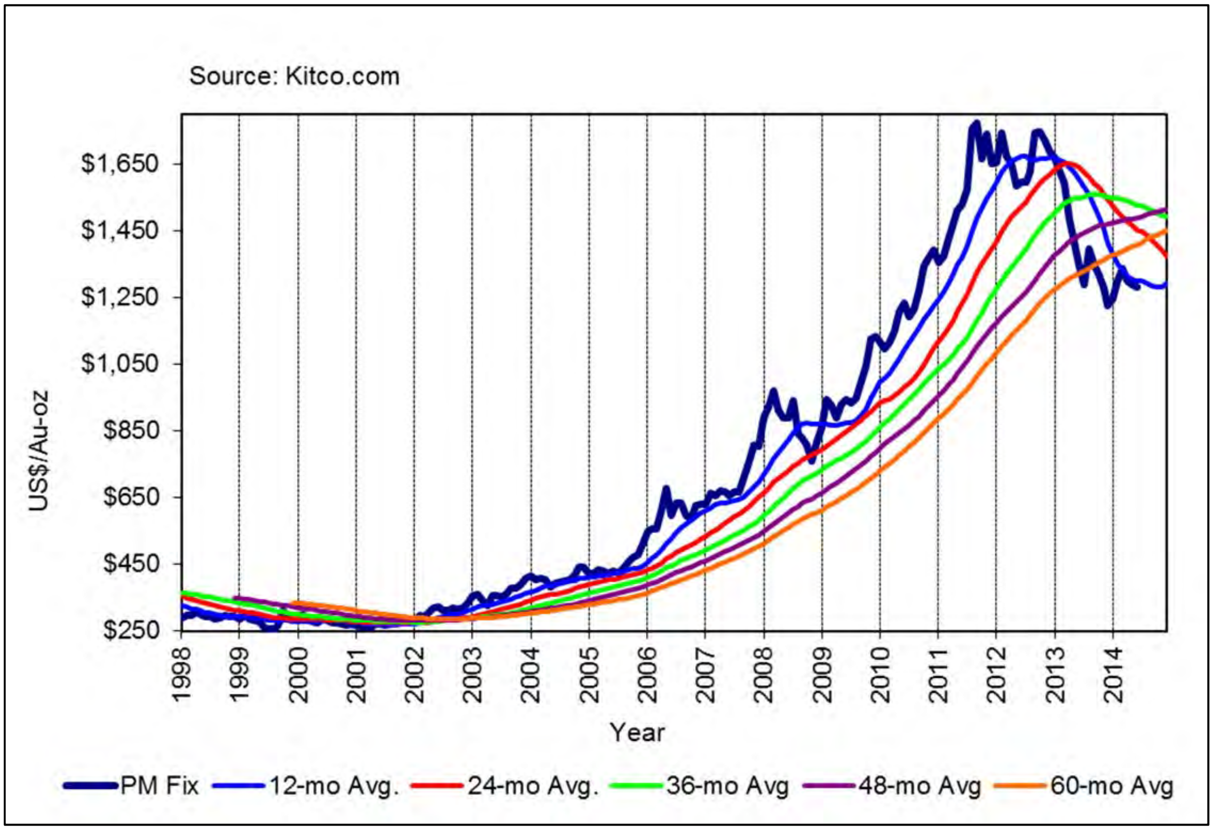
- 1.0% NSR royalty to Barrick
- 2.5% NSR royalty to Wotan
- 10% of payable gold is sold to Seabridge at a discount price of US$1,000.
- British Columbia mineral royalties, which comprise:
  - 2% net current proceeds tax
  - 13% net revenue tax.

20.4 METAL PRICES

The base and precious metal markets benefit from terminal markets around the world (London, New York, Tokyo, Hong Kong) and fluctuate on an almost continuous basis. Historical metal price for gold is shown in Figure 20-1 and demonstrates the change in metal price from 1998 through to June 2014.

Four metal price scenarios were evaluated as part of the economic analysis. Base Case pricing for is based on approximately 5% less than the lower of spot and three-year trailing average prices as at June 30, 2014. The Base Case pricing was used in the parameters established for mine planning. The foreign exchange rate used in all three scenarios is between the three-year trailing average and spot exchange rate as at June 30, 2014.

Figure 20-1: Average Gold Cash Price as at June 2014



Source: Kitco.com (2014)

Table 20.1 summarises the metal prices and exchange rates used to run various scenarios in the economic analysis.

Table 20.1: Metal Price & Foreign Exchange Rates used in Economic Analysis Scenarios

Parameter	Unit	Base Case	Base Case +10%	Base Case - 10%	Lower of Three-Year Trailing & Spot as at June 30, 2014
Gold Price	US$/oz	1,250	1,375	1,125	1,315
Silver Price	US$/oz	20.00	22.00	18.00	21.00
Exchange Rate	USD:CAD	0.95	0.95	0.95	0.95

Source: JDS (2014)

21.0 CAPITAL & OPERATING COSTS

21.1 CAPITAL COSTS SUMMARY

The capital cost estimate (CAPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a preliminary study.

The CAPEX estimate includes the costs required to develop, sustain, and close the operation for the planned five-year mine life. The construction schedule is based on an approximate two-year build period.

All costs are presented in Canadian dollars (CAD) unless otherwise noted.

The CAPEX estimate summary is shown in Table 21.1.

Table 21.1: LOM Capital Costs

Capital Cost	Pre-Production $M	Sustaining/ Closure $M	LOM Total $M
Crushing & Milling	23.8	0.0	23.8
Tailings Pond	3.7	11.6	15.3
Power	10.2	0.0	10.2
Mine Development	10.5	4.8	15.3
Infrastructure	2.5	0.6	3.1
Surface Equipment	1.1	0.0	1.1
Site Access Roads	5.9	0.0	5.9
Owner, Indirects, EPCM	8.6	0.0	8.6
Closure (Net of Salvage Value)	0.0	1.4	1.4
Subtotal Pre-Contingency	**66.2**	**18.4**	**84.7**
Contingency	9.9	2.8	12.7
Total Capital Incl. Contingency	**76.1**	**21.2**	**97.4**

Source: JDS (2014)

21.1.1 Mining

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 outlines the stationary equipment capital required to support the Red Mountain mine plan presented in this report. All costs are in Canadian dollars.

Table 21.2: Stationary Equipment Capital Costs

Stationary Equipment Type	UNITS Req'd	Price Per Unit	LOM Cost
Site Access & General Earthworks	1	100,000	**100,000**
Main Ventilation Fan	1	542,000	**542,000**
UG Heating System	1	400,000	**400,000**
Compressors	2	100,000	**200,000**
Batch Plant	1	150,000	**150,000**
Portable welder	1	12,000	**12,000**
UG Aux Fans 75 kW	2	50,000	**100,000**
UG Aux Fans 50 kW	8	12,500	**100,000**
UG Aux Fans 30 kW	4	25,000	**100,000**
Pumping Stations	2	50,000	**100,000**
Face Pumps	4	12,500	**50,000**
Portable Refuge Stn	2	25,000	**50,000**
Mine Rescue Equip	1	100,000	**100,000**
Cap Lamps & Chargers	1	17,000	**17,000**
Stench Gas System	1	13,000	**13,000**
Mine Eng Equip & Software	1	100,000	**100,000**
Portal (Incline)	1	100,000	**100,000**
Plant,Office & Maint Site Earthworks	1	100,000	**100,000**
Fuel Storage at Site	1	75,000	**75,000**
Water Supply & Distribution	1	20,000	**20,000**
Water Storage (Potable, Process & Fire)	1	100,000	**100,000**
Sewage Storage Tank	1	50,000	**50,000**
Powder Mag (2 Containers)	1	40,000	**40,000**
Mine Dry/Office (45 persons/shift)	1	250,000	**250,000**
Maintenance Shop Surface	1	150,000	**150,000**
Warehouse, Storage Area (Mine Site)	1	20,000	**20,000**
Communications	1	20,000	**20,000**
Subtotal	**C$**		**3,059,000**

Source: JDS (2014)

Mine development capital for lateral and vertical development, consisting of pre-production capital and sustaining capital, is summarised in Table 21.3.

Table 21.3: Capital Development Cost Schedule

		Year	-2	-1	1	2	3	4	5
	Total								
Pre-Prod. Capex ($M)	10.5			10.5					
Sustaining Capex ($M)	4.8				2.6	2.3			

Source: JDS (2014)

21.1.2 Sustaining Capital Costs

The majority of the sustaining capital costs are related to mine development, tailings pond, and infrastructure such as underground heating systems, mine engineering equipment and software, and ventilation equipment. All sustaining costs are expected to occur between Year 1 and Year 4 of the mine life.

21.1.3 Closure Costs & Salvage Value

A $4.5M closure cost was estimated based on reclamation of TMF, process plant, roads, and general site. A $3.1M salvage value was assumed based on varying factors (5% - 15%) of capital cost components. The closure cost is assumed to occur in Year 6, one year following the end of commercial production.

21.1.4 Contingency

An overall contingency of 15% was applied to the LOM capital costs of the project. Total project contingency amounts to $12.7 M.

21.2 OPERATING COST SUMMARY

The OPEX estimate is based on a combination of experience, reference project, budgetary quotes and factors as appropriate with a preliminary study.

The operating cost estimate in this study includes the costs to mine, process the mineralised material to doré, and general and administrative expenses (G&A). These items total the mine operating costs and are summarised in Table 21.4.

Table 21.4: LOM Operating Costs

Operating Cost	$/t processed	LOM $M
Mining	66.54	91.7
Milling	27.67	38.1
G&A	10.91	15.0
Total	**105.13**	**144.9**

Source: JDS (2014)

21.2.1 Mining

Mine operating costs are summarised below in Table 21.5.

Table 21.5: Underground Mine Operating Costs

Mine Opex	Cost per tonne (CAD)
Production Costs	8.50
Development Costs	3.73
Backfill	7.15
Equipment & Fuel	10.13
Electrical Power & Mine Air Heating	2.90
Labour	34.14
Total	**66.54**

Source: JDS (2014)

Costs are averaged over the life of the mine, and range from $73 per tonne in Year 1, to $59 per tonne in Year 5. The fluctuation in costs is mostly due to a decrease in mine development costs over the mine life. By Year 5 most waste development apart from stope crosscuts and incremental footwall drives have been established, and as such less manpower, equipment, ventilation, and power is required to maintain a steady production rate of 1,000 t/d.

Mine operating costs have been built up using a combination of first principle engineering and equivalent project scaling.

21.2.1.1 Production & Development Costs

The following table outlines the unit costs for mineralised material development, ramp development, raise development, and production mining. Note that labour is not included in these tables and is accounted for in the labour component of mine operating costs.

Table 21.6: Mineralised Material Development Unit Costs

Cost Summary	$/m	$/t
Equipment	58.40	1.18
Drill Steel and Bits	140.61	2.85
Explosives	361.30	7.31
Ground Support	95.91	1.94
Piping	86.32	1.75
Electrical	42.46	0.86
Ventilation	34.30	0.69
Load/Haul	444.53	9.00
TOTAL	**1,263.82**	**25.59**

Source: JDS (2014)

Table 21.7: Ramp Development Unit Costs

Cost Summary	$/m	$/t
Equipment	59.91	0.99
Drill Steel and Bits	142.72	2.37
Explosives	340.68	5.66
Ground Support	108.90	1.81
Piping	181.77	3.02
Electrical	50.16	0.83
Ventilation	34.30	0.57
Load/Haul	541.98	9.00
TOTAL	**1,460.42**	**24.25**

Source: JDS (2014)

Table 21.8: Alimak Raise Unit Cost

Item	Units	Unit Cost	Units Required	Cost
Excavate 4x4 Nest	metre	3,713.46	15	55,702
Equip Nest	each	36,438.22	1	36,438
Install Muck Wall	each	3,708.72	1	3,709
Drive 3.0 x 3 Raise - 8ft round	metre	2,390.23	150	358,535
Install Laser	each	7,465.71	1	7,466
Screen Raise	metre	622.45	150	93,368
Strip out Alimak Nest	each	11,915.60	1	11,916
Total Cost				**567,132**
Cost per Metre				**3,781**

Source: JDS (2014)

Alimak raise costs are based on contractor quote.

Table 21.9: Longhole Unit Costs

Cost Summary	$/t
Drill/Blast Supplies	$5.35
Load/Haul	$0.67
Total	**$6.03**

Source: JDS (2014)

21.2.1.2 Backfill Costs

Red Mountain would utilise CFR backfill for all primary longhole stopes, and all drift and fill stopes. Primary longhole stopes would require 5% cement, while drift and fill backfill only would require 3% cement content. For this study it has been assumed dry cement would be delivered to site for $350 per tonne. Secondary longhole stopes would be backfilled with waste rock, and would only require the cost of backhauling waste material.

Table 21.10: Backfill Unit Costs

Backfill Type	Units	Unit Cost
Primary Longhole Stopes (5% cement content)	$/t	15.80
Secondary Longhole Stopes (0% cement content)	$/t	3.68
Drift and Fill Stopes (3% cement content)	$/t	10.23

Source: JDS (2014)

21.2.1.3 Equipment & Fuel

Diesel fuel is estimated at $1.10 per litre, delivered to site. Mobile equipment fuel consumption is an average 60,000 litres per month, which accounts for approximately $2.43 per tonne of mineralised material mined. Equipment maintenance costs are listed below in Table 21.11.

Table 21.11: Mobile Equipment Maintenance Costs

	Maintenance / Parts	Lubes / Filters	Tires	No Fuel
Equipment	**$/h**	**$/h**	**$/h**	**$/h**
Jumbo (2 boom)	13.03	4.28	0.21	17.52
Longhole Drill	5.93	3.43	1.25	10.61
LHD	59.35	5.86	2.36	67.57
Haulage Truck	44.80	4.18	6.94	55.92
Grader	5.08	1.48	0.50	7.06
ANFO Loader	16.89	3.70	0.42	21.01
Boom Truck	8.20	1.93	0.42	10.55
Mechanics Truck	8.20	1.93	0.42	10.55
Fuel-Lube Truck	8.20	1.93	0.42	10.55
Supervisor/Service Vehicle	1.00	0.50	0.42	1.92
Cement Truck	1.00	0.50	0.42	1.92
Scissor Truck	8.20	1.93	0.42	10.55
Jackleg/Stoper	0.29	0.85	-	1.14
Forklift/Telehandler	1.67	0.33	0.66	2.66

Source: JDS (2014)

Mobile equipment would be leased to the mine at an average cost of $7.78 per tonne of mineralised material mined, which includes a residual buyout at the end of the mine life. Details on leasing can be found below in Table 21.12.

It should be noted that no cost has been accounted for any major overhauls, rebuilds, or replacements of the equipment, as it is assumed the lease agreements would account for these items. The mine life is less than five years running seasonally, thus it is not expected any major equipment would need replacing during this time frame.

Table 21.12: Mobile Equipment Lease Costs

Underground Equipment Type	Units Req'd	Unit Price	Upfront Fee	Annual Lease Payments /Unit	Residual Buyout	LOM Total Lease
Haulage Truck (20 t)	3	265,400	2,654	49,041	67,745	**779,818**
LHD (3.0 m^3 - Rammer Jammer)	1	224,800	2,248	41,539	57,382	**267,322**
LHD (4.6 m^3)	3	850,000	8,500	157,063	216,968	**3,032,347**
Jumbo (2 boom)	2	1,156,000	11,560	213,605	295,076	**2,749,328**
Longhole Drill (76mm)	2	944,000	9,440	174,432	240,962	**2,245,126**
Scissor Truck	2	150,000	1,500	27,717	38,288	**356,747**
Boom Truck	1	225,000	2,250	41,575	57,433	**267,560**
Fuel/Lube Truck	1	58,000	580	10,717	14,805	**68,971**
Mechanic Vehicle	1	60,000	600	11,087	15,315	**71,349**
Supervisors Vehicle	1	60,000	600	11,087	15,315	**71,349**
Man Carrier	1	60,000	600	11,087	15,315	**71,349**
Mine Rescue Vehicle	1	115,000	1,150	21,250	29,354	**136,753**
Cement Delivery Truck	1	245,000	2,450	45,271	62,538	**291,343**
Grader/Dozer	1	149,500	1,495	27,625	38,161	**177,779**
Forklift/Telehandler	1	80,500	805	14,875	20,548	**95,727**
Stopers	1	6,700	67	1,238	1,710	**7,967**
Jacklegs	5	6,400	64	1,183	1,634	**38,053**
Life-of-Mine Total						**10,729,000**

Source: JDS (2014)

21.2.1.4 Electrical Power & Mine Air Heating

Electrical power for the Red Mountain operation would be provided by a BC Hydro overhead transmission line running adjacent to the seasonal access road, with onsite substation and distribution networks. As such, Red Mountain would not suffer from high power costs associated with running diesel generators. Based on BC Hydro's rate of $0.04 per kWh and an annual consumption of 7.98 M kWh, power operating costs have been estimated at $319,000 per year or $1.18 per tonne of mineralised material mined.

Mine air would be heated through propane burners at a consumption rate of 792 ft^3/h. At a propane cost of $0.75 per litre, it is estimated the operating cost of the mine air heater would be $556,000 per year, or $2.07 per tonne of mineralised material mined.

21.2.1.5 Labour

Labour requirements are discussed in Section 16 of this report. Mine operating labour costs are based on hourly and salaried employees working seasonally at Red Mountain. Salaries and wages are built with a base pay followed by a loading for CPP, EI, WCB, stat holidays, vacation, RSP contributions, and halftime pay during travel to/from the mine site.

Table 21.13 on the following page outlines the base and fully loaded salaries and wages.

21.2.2 Processing & Infrastructure Capital Costs

The following items are included in each capital cost category:

- **Crushing & Milling** – All equipment (including ancillary equipment), equipment installation, piping & electrical material and labour, process building, and plant services.
- **Tailings Pond** – Earthworks and construction related to the construction of the tailings management facility.
- **Power** – Costs related to the switchyard, powerline, and substation as well as site distribution for power required at site.
- **Infrastructure** – Site access and general earthworks, ventilation fans, portable refuge station, portal, offices, warehouse and storage areas, maintenance shop, communications, and batch plant.
- **Surface Equipment** – Construction equipment such as loaders and graders, lifting and support equipment, busses, vans, trucks, and emergency vehicles.
- **Site Access Roads** – Road costs to connect the plant site, portal, and TMF.
- **Owner, Indirects, & EPCM** – Costs related to field construction expenses, general EPCM, and G&A costs to be incurred in the pre-production period.

Table 21.13: Mine Labour Salaries

Position		Base Pay	Loaded Pay Rate
Mining Operations			
Mine Super/Mine GF	Salary	$200,000	$247,636
Mine Supervisor/Shift Boss	Hourly	$32.00	$77.47
Safety / Mine Rescue / Training Officer	Hourly	$25.50	$58.55
LH Drillers/Blasters	Hourly	$25.50	$58.55
Ground Support, Hanging Services	Hourly	$28.70	$67.94
Fuel/Lube/Boom/Grader/Telehandler	Hourly	$22.35	$44.02
Jumbo & LHD Operator	Hourly	$31.00	$71.10
Truck Operator	Hourly	$28.70	$67.94
Backfill			
Backfill Crew - Bulkheads, Piping, Monitor	Hourly	$22.35	$44.02
CRF Plant Operators	Hourly	$25.50	$38.55
Mining Maintenance			
Mine Maintenance Supervisor/Lead Hand	Hourly	$32.00	$77.47
Maintenance Planner	Salary	$110,000	$141,436
HD Mechanic	Hourly	$45.00	$89.90
Welder	Hourly	$28.70	$67.94
Electrician	Hourly	$45.00	$89.90
Mining Technical Services			
Chief Mine Engineer/Sr. Mine Engineer	Salary	$170,000	$212,236
Mine Planning Engineer	Salary	$120,000	$153,236
Surveyor	Salary	$90,000	$117,836
Mine Technologist	Salary	$90,000	$117,836
Senior Geologist	Salary	$110,000	$141,436
Grade Control Geologist	Salary	$80,000	$106,036

Source: JDS (2014)

21.2.3 Process 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 average annualized mineralised treatment rate 276 kt. The crushing plant is designed at 1,000 t/h. Labour rates and benefit loadings are based on information supplied by JDS. All reagent costs estimates are detailed in Section 21.2.3.1. The process operating costs are summarised in Table 21.14.

Table 21.14: Process Operating Costs

Processing Operating Costs	$/t processed	$M/a	LOM $M
Consumables	15.69	4.3	21.6
Labour	11.98	3.3	16.5
Total Processing OPEX	**27.67**	**7.6**	**38.1**

Source: JDS (2014)

21.2.4 Processing OPEX Consumables

Table 21.15 demonstrates the processing consumable costs.

Table 21.15: Processing Consumables

Consumables	$/t Processed	$M/a	LOM $M
Power	2.84	0.8	3.9
Natural Gas	0.39	0.1	0.5
Rod Mill Grinding Media Liners	0.70	0.2	1.0
Ball Mill Grinding Media Liners	2.00	0.6	2.8
Maintenance Labour & Supplied	5.77	1.6	8.0
Reagents	3.99	1.1	5.5
Total	**15.69**	**4.3**	**21.6**

Source: JDS (2014)

21.2.4.1 Reagent Costs

Reagent costs were based on recent quotes. Table 21.16 summarised the reagent and chemical requirements.

Table 21.16: Reagent Requirements & Costs

Reagent	kg/t Required	$/kg	$/t
Lime	6.1	0.30	1.84
NaCN	0.8	2.00	1.60
Carbon	0.1	1.00	0.05
Other Reagents	0.1	5.00	0.50
Total			**3.99**

Source: JDS (2014)

21.2.4.2 Processing Labour Requirements

Table 21.17 summarises all processing labour requirements.

Table 21.17: Processing Labour Requirements & Costs

Process Labour	Salary/Hourly	Base Pay	Loaded Pay	Quantity
Milling Operations				
Mill General Foreman	Salary	$150,000	$188,636	2
Mill Shift Supervisor	Hourly	$40	$58	4
Mill Operators	Hourly	$32.00	$47.47	12
				0
Milling Operations - Total				**18**
Met Lab & Quality Control				
Plant Metallurgist	Salary	$130,000	$165,036	2
Assay Technicians	Hourly	$40.00	$58.45	2
Metallurgical Technician	Staff	$90,000	$117,836	2
				0
Met Lab & Quality Control - Total				**6**
Plant Maintenance				
Mill Maintenance Shift Foreman	Salary	$150,000	$188,636	2
Maintenance Planner	Hourly	$45.00	$64.90	2
Millwrights	Hourly	$45.00	$64.90	2
Instrument Technicians	Hourly	$45.00	$64.90	2
Plant Maintenance - Total				**8**
Total Process Labour				**32**

Source: JDS (2014)

21.2.5 G&A Costs

General and Administration costs include for all off-site and on-site activities including road avalanche control, site services equipment cost, Stewart office operating costs and associated labour. The summary of costs is shown in Table 21.18, averaged over the life of mine.

Table 21.18: Breakdown of G&A Costs

G&A Cost	$/t Milled	Avg $M/a	LOM $M
G&A Expenses	1.21	0.3	1.7
G&A Labour	9.70	2.7	13.4
Total	**10.91**	**3.0**	**15.0**

Source: JDS (2014)

21.2.6 G&A Labour Requirements

Table 21.19 lists the site supervision and support personnel requirements and costs.

Table 21.19: G&A Labour Requirements & Costs

G&A Labour	Salary/Hourly	Base Pay	Loaded Pay	Quantity
General Manager	Salary	$230,000	$283,036	1
General Manager - Cross Shift	Salary	$184,000	$228,756	1
Administration	Salary	$70,000	$93,882	2
Purchaser	Salary	$100,000	$129,636	2
Environment Manager	Salary	$120,000	$153,236	2
Environment Technician	Salary	$80,000	$106,036	2
Clerk - Shared	Salary	$50,000	$68,768	2
Site Services	Hourly	$45.00	$74.90	8
Warehouse men	Salary	$50,000	$68,768	2
Bus Drivers	Hourly	$28.70	$43	2
Total G&A Labour				**24**

Source: JDS (2014)

22.0 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, head grades, operating costs, capital costs, and discount rates 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 realised 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 summarised in Section 21 of this report (presented in 2014 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.1 ASSUMPTIONS

Four metal price cases were evaluated to better understand the value drivers in each scenario.

All costs, metal prices and economic results are reported in Canadian dollars (CAD or C$) unless stated otherwise. All cases have identical LOM plan tonnage and grade estimates demonstrated in Table 22.1. On-site and off-site costs and production parameters were also held constant for each scenario evaluated.

Table 22.1: LOM Plan Summary

Mine Life	Years	5.0
Resource Mined	Mt	1.4
Waste Mined	Mt	0.0
Total Mined	**Mt**	**1.4**
Strip Ratio	w:o	0.0
Throughput Rate	t/d	1,022
Average Au Head Grade	g/t	7.25
Average Ag Head Grade	g/t	24.44
Au Payable	koz	277.0
	koz/a	55.5
Ag Payable	koz	852.0
	koz/a	170.6

Source: JDS (2014)

Other economic factors common to all four cases include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario)
- Closure cost of $4.5 M and salvage value of $3.1 M were considered
- Nominal 2014 dollars
- Revenues, costs, 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, G&A) in Year 1 (based on a nine-month operation)
- Results are presented on 100% ownership
- No management fees or financing costs (equity fund-raising was assumed).

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 metal prices and USD:CAD exchange rate assumptions used in economic analysis. Base Case pricing for is based on approximately 5% less than the lower of spot and three-year trailing average prices as at June 30, 2014. The Base Case pricing was used in the parameters established for mine planning. The foreign exchange rate used in all three scenarios is between the three-year trailing average and spot exchange rate as at June 30, 2014.

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 realised 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: Metal Price & Foreign Exchange Rates used in Economic Analysis Scenarios

Parameter	Unit	Base Case	Base Case + 10%	Base Case - 10%	Lower of Three-Year Trailing & Spot as at June 30, 2014
Gold Price	US$/oz	1,250	1,375	1,125	1,315
Silver Price	US$/oz	20.00	22.00	18.00	21.00
Exchange Rate	USD:CAD	0.95	0.95	0.95	0.95

Source: JDS (2014)

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 20) of this report.

Table 22.3 indicates the NSR parameters that were used in the economic analysis.

Table 22.3: NSR Parameters Used in Economic Analysis

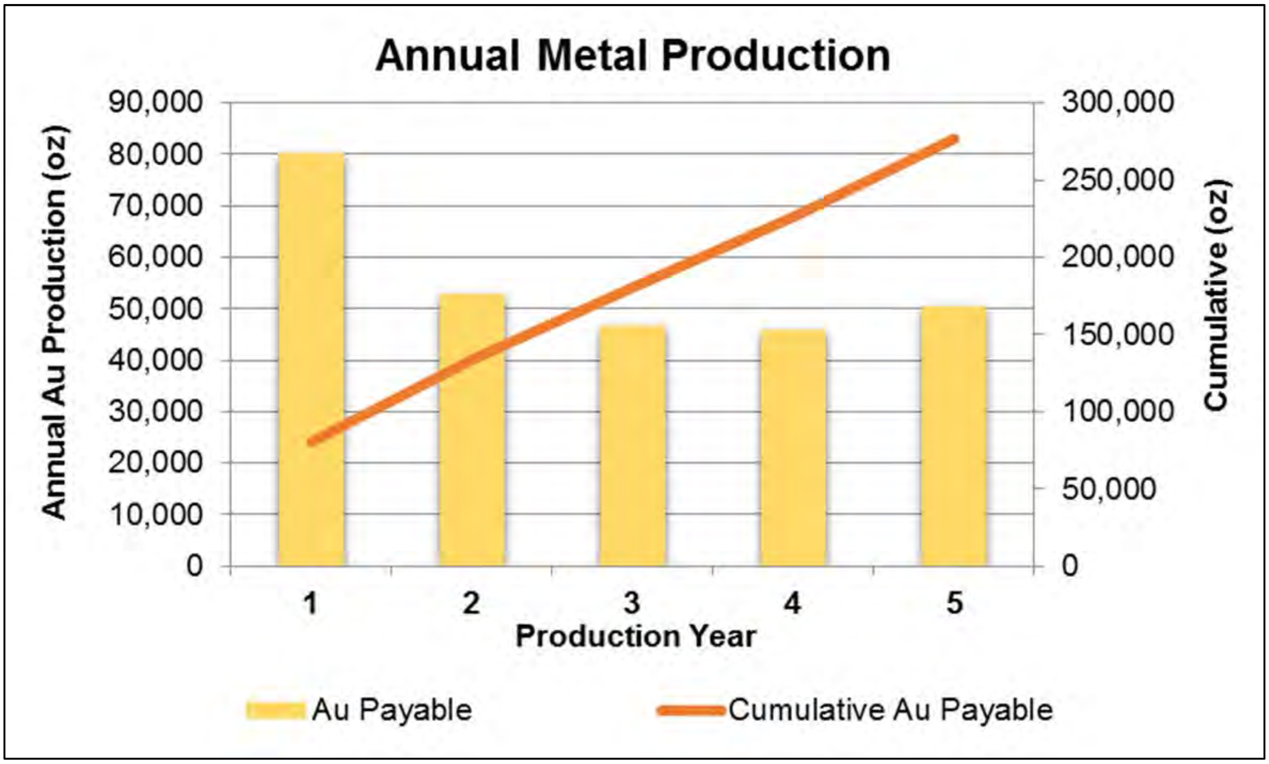
Assumptions & Inputs	Unit	Value
Operating Days	days/a	270
Barrick Gold Royalty	% NSR	1.00
Wotan First Nations Royalty	% NSR	2.50
Recoveries		
Marc Zone		
Au	%	90.5
Ag	%	85.0
AV Zone		
Au	%	82.0
Ag	%	71.0
JW Zone		
Au	%	93.0
Ag	%	86.0
NSR Parameters		
Au Payable	%	99
Au Refining Charge	US$/pay oz	5.00
Ag Payable	%	99
Ag Refining Charge	US$/pay oz	0.50

Source: JDS (2014)

Figure 22-1 and 22-2 show breakdowns of the amount of gold and silver recovered during the mine life. A total of 277 koz of gold and 852 koz of silver is projected to be produced during the mine life.

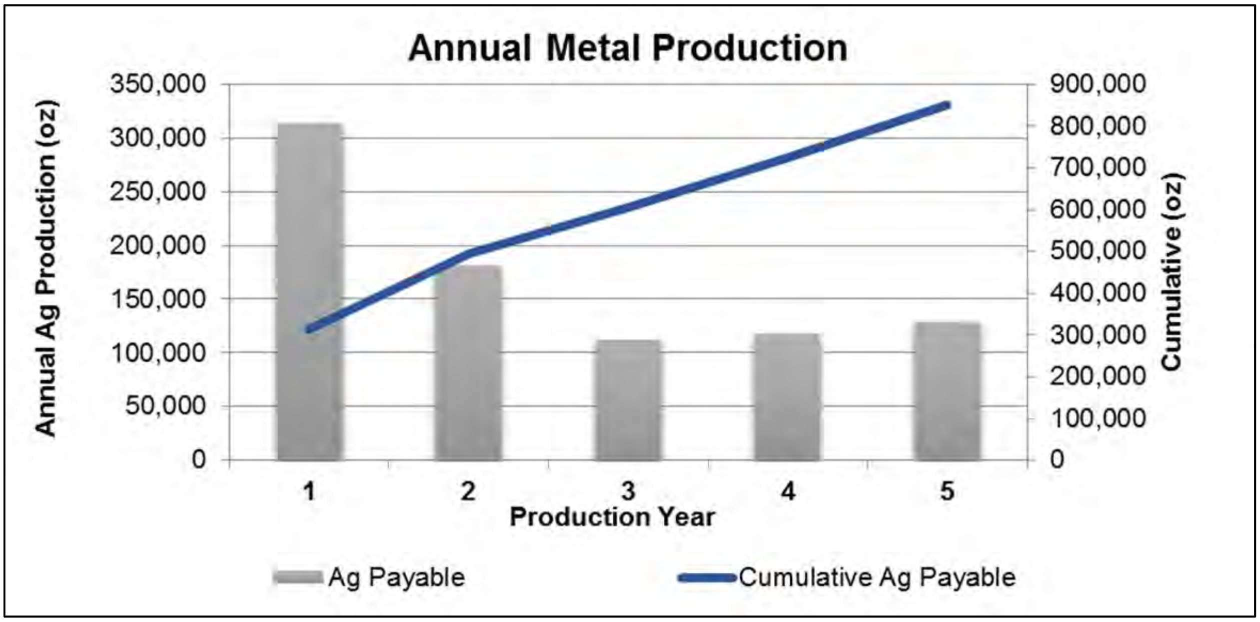
Figure 22-1 and Figure 22-2 also illustrate the amount of payable metal for the project. Figure 22-3 demonstrates the breakdown of LOM net smelter return for the base case, which amounted to $361.6 M.

Figure 22-1: Payable Gold Doré Production by Year



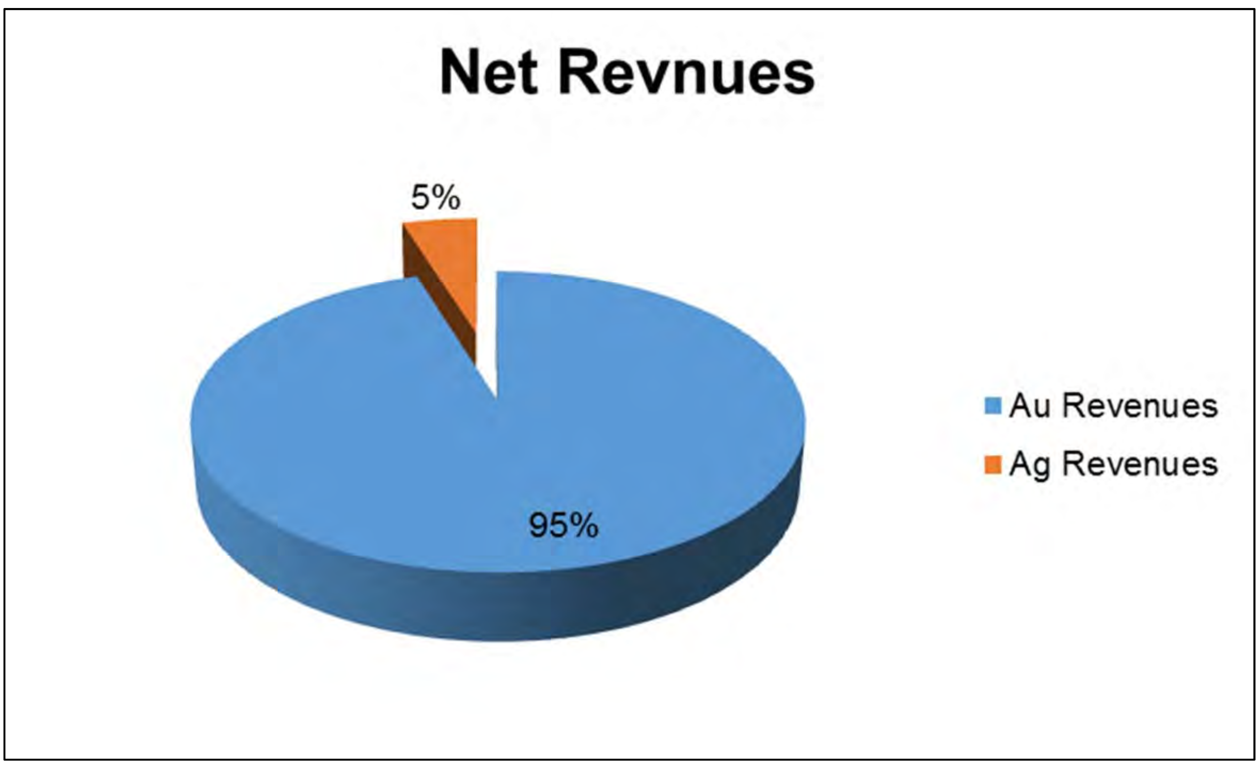
Source: JDS (2014)

Figure 22-2: Payable Silver Doré Production by Year



Source: JDS (2014)

Figure 22-3: LOM Project Net Revenue Breakdown



Source: JDS (2014)

22.3 SUMMARY OF CAPITAL COST ESTIMATE

During the two-year pre-production period, the initial capital costs amount to $76.1M. This includes costs for site development, processing plant, on-site infrastructure, pre-production operating costs, etc. This also assumes the leasing of the mining fleet starting in 2016. Leasing of the mining fleet was assumed to determine the project value for all scenarios. The total equipment cost of the leased portion of the mining fleet amounts to $9.2M.

A 15% contingency is included in the initial capital costs. A breakdown of the initial capital costs is shown in Table 22.4 and Figure 22-4. Details on capital costs can be found in Section 21.0.

Sustaining and closure capital cost estimates amount to $21.2M and were assumed to occur from 2017 to 2020 with a majority of these costs for the tailings pond. A 15% contingency is included in the sustaining and closure capital expenditures. Mine equipment that is included in the sustaining capital costs account for the ancillary, spares and other miscellaneous mine equipment that is assumed to not be leased. A breakdown of the sustaining and capital costs is shown in Table 22.4 and Figure 22-5.

Closure costs of $4.5M net of salvage value of $3.1M plus contingency were assumed to occur in Year 6.

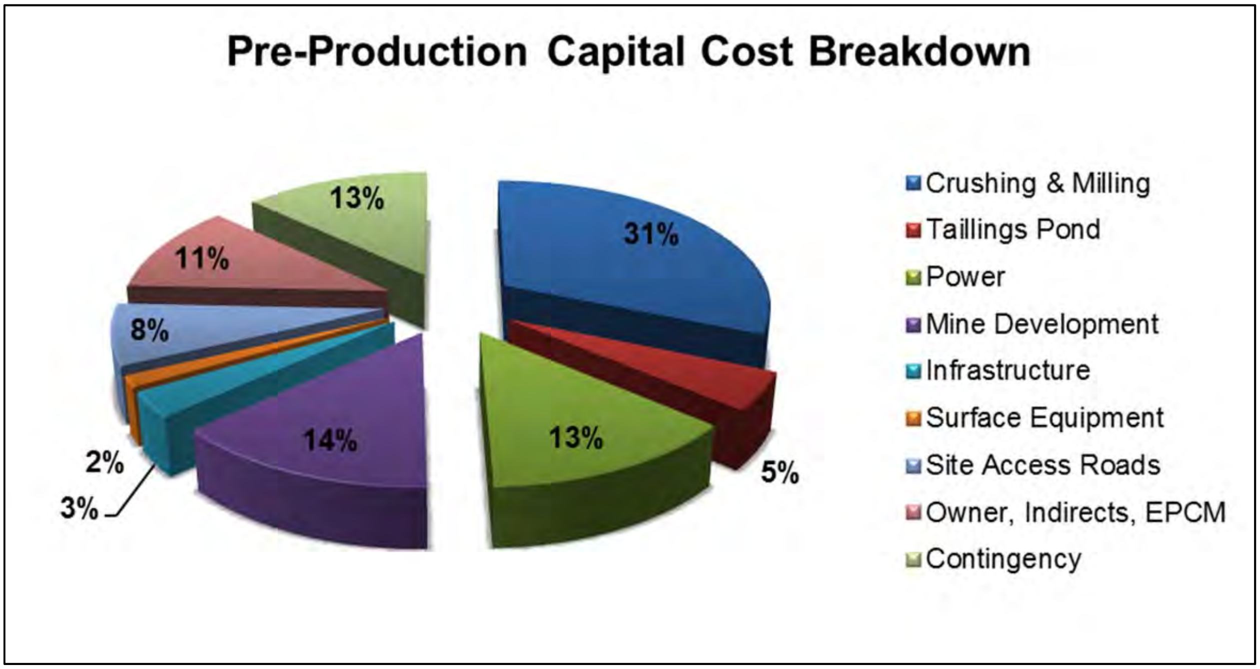
Details on the capital costs can be found in Section 21 of this report.

Table 22.4: Summary of LOM Capital Costs

Capital Cost	Pre-Production ($M)	Sustaining/ Closure ($M)	Total ($M)
Crushing & Milling	23.8	0.0	23.8
Tailings Pond	3.7	11.6	15.3
Power	10.2	0.0	10.2
Mine Development	10.5	4.8	15.3
Infrastructure	2.5	0.6	3.1
Surface Equipment	1.1	0.0	1.1
Site Access Roads	5.9	0.0	5.9
Owner, Indirects, EPCM	8.6	0.0	8.6
Closure (Net of Salvage Value)	0.0	1.4	1.4
Subtotal Pre-Contingency	**66.2**	**18.4**	**84.7**
Contingency (15%)	9.9	2.8	12.7
Total Capital Incl. Contingency	**76.1**	**21.2**	**97.4**

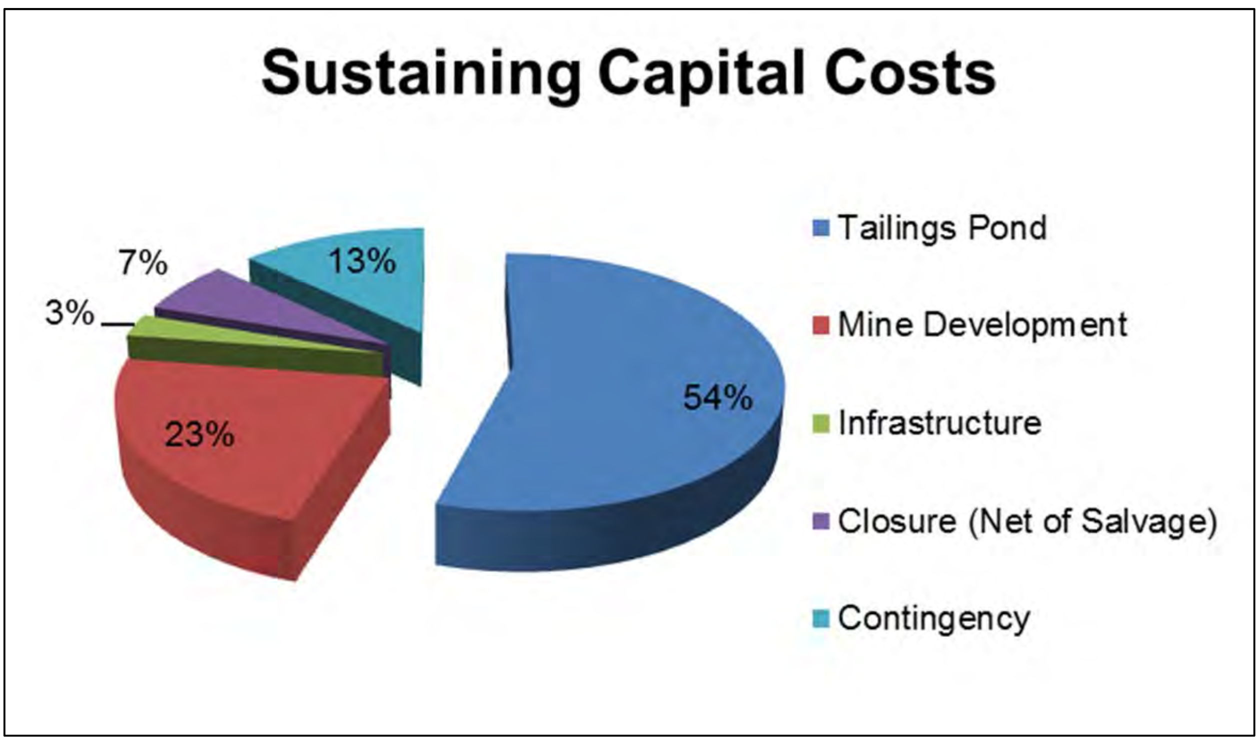
Source: JDS (2014)

Figure 22-4: Breakdown of Pre-Production Capital Costs



Source: JDS (2014)

Figure 22-5: Breakdown of Capital Expenditures during Production



Source: JDS (2014)

22.4 SUMMARY OF OPERATING COST ESTIMATES

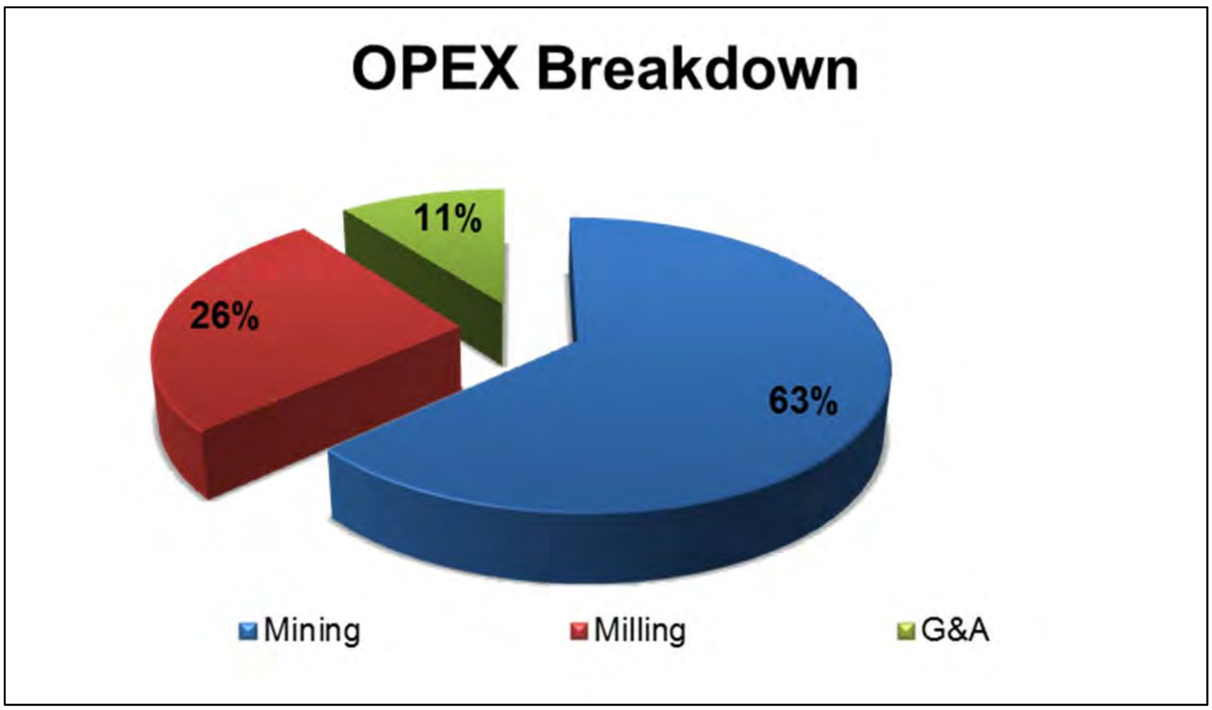
Total LOM operating costs amount to $144.9M. The total LOM operating costs translate to an average cost of $105.13/tonne milled. A breakdown of these costs is outlined in Table 22.5 and Figure 22-6.

Table 22.5: Operating Costs

Operating Cost	$/t processed	LOM $M
Mining	66.54	91.7
Milling	27.67	38.1
G&A	10.91	15.0
Total	**105.13**	**144.9**

Source: JDS (2014)

Figure 22-6: Breakdown of Operating Costs



Source: JDS (2014)

22.5 LEASING

The economic analysis assumes that primary mine equipment would be leased. The total value of the mine equipment to be leased is $9.2M. The terms used to calculate the lease payments are in line with the market conditions and include the following:

- Up-front fee of 1%
- 20% residual value; 80% lease
- 5% lease interest rate
- 5-year lease term.

Lease payments (including up-front payments) for the life of mine total $10.7 M.

22.6 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. A tax model was prepared by a specialised mining tax accountant with applicable British Columbia mineral tax regime. The tax model contains the following assumptions:

- 15% federal income tax rate
- 11% British Columbia provincial tax rate
- British Columbia mineral taxes which include:
  - 2% net current proceeds tax
  - 13% net revenue tax.

Total taxes for the project amount to $40.1 M.

22.7 ROYALTIES

The economic analysis for the project accounts for the following royalties:

- 1.0% NSR royalty to Barrick
- 2.5% NSR royalty to the Wotan First Nations
- 10% of payable gold is sold to Seabridge at a discount price of US$1,000.
- BC mineral royalties as discussed in Section 22.6.

22.8 ECONOMIC RESULTS

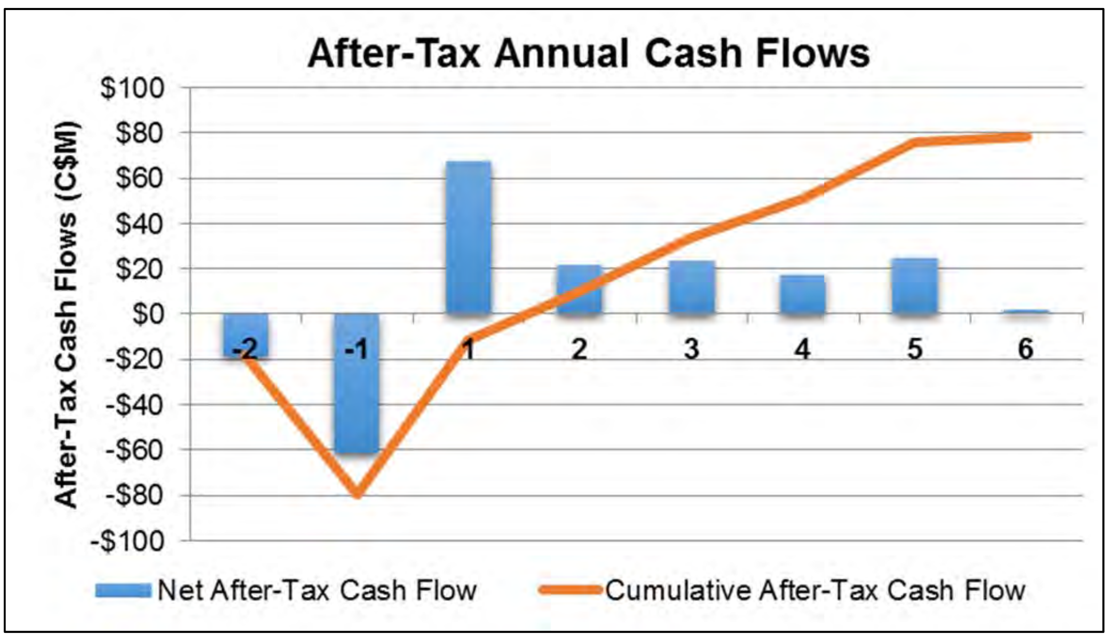
At this preliminary stage, the project is economically viable with an after-tax internal rate of return (IRR) of 32.9% and a net present value using a 5% discount rate (NPV5%) of $57.6M using the Base Case metal prices. In addition, three additional scenarios were evaluated using Base Case price +10%, Base Case pricing -10%, and the lower of spot and three-year trailing average metal prices as at June 30, 2014. Table 22.6 through Table 22.9 on the following pages summarise the economic results of each scenario evaluated.

The scenario utilizing three-year trailing average metal prices resulted in the highest performance and project value due to the highest metal prices of all three scenarios. The metal prices in the "Base Case" yielded the lowest economic value of the project, however, still showed strong economic results.

The break-even gold price for the project's Base Case is approximately US$849/oz, based on the LOM plan presented herein and a silver price of US$20/oz.

Figure 22-7 through Figure 22-10 show the projected cash flows for the project used in the different scenarios of the economic analysis.

Figure 22-7: Annual After-Tax Cash Flows for Base Case Scenario



Source: JDS (2014)

Table 22.6: Summary of Results for Base Case Scenario

Summary of Results	**Unit**	**Value**
Au Price	US$/oz	1,250
Ag Price	US$/oz	20.00
F/X Rate	US$: C$	0.95
Mine Life	Years	5.0
Resource Mined	Mt	1.4
Waste Mined	Mt	0.0
Total Mined	Mt	6.4
Strip Ratio	w:o	0
Throughput Rate	t/d	1,022
Average Head Grades		
Average Au Head Grade	g/t	7.25
Average Ag Head Grade	g/t	24.44
Au Payable	koz	277.0
	koz/a	55.5
Ag Payable	koz	852.0
	koz/a	170.6
NSR (Net of Royalties)	LOM C$M	361.6
	$/t mined	262.37
Operating Costs	LOM C$M	144.9
	$/t mined	105.13
Au Cash Cost	US$/oz	516.23
Au Cash Cost (Net of By-Product)	US$/oz	454.73
Capital Costs		
Pre-Production Capital	C$M	66.2
Pre-Production Contingency	C$M	9.9
Total Pre-Production Capital	**C$M**	**76.1**
	$/t mined	**55.24**
Sustaining & Closure Capital	C$M	18.4
Sustaining & Closure Contingency	C$M	2.8
Total Sustaining & Closure Capital	**C$M**	**21.2**
	$/t mined	**15.4**
Total Capital Costs Incl. Contingency	C$M	97.4
	$/t mined	70.64
Working Capital	C$M	3.4
Pre-Tax Cash Flow	LOM C$M	$119.4
	C$M/a	$23.9
Taxes	LOM C$M	$40.1
After-Tax Cash Flow	LOM C$M	$79.2
	C$M/a	$15.9
Economic Results		
Pre-Tax NPV$_{5\%}$	**C$M**	**$90.1**
Pre-Tax IRR	**%**	**43.3%**
Pre-Tax Payback	**Years**	**1.3**
After-Tax NPV$_{5\%}$	**C$M**	**$57.6**
After-Tax IRR	**%**	**32.9%**
After-Tax Payback	**Years**	**1.5**

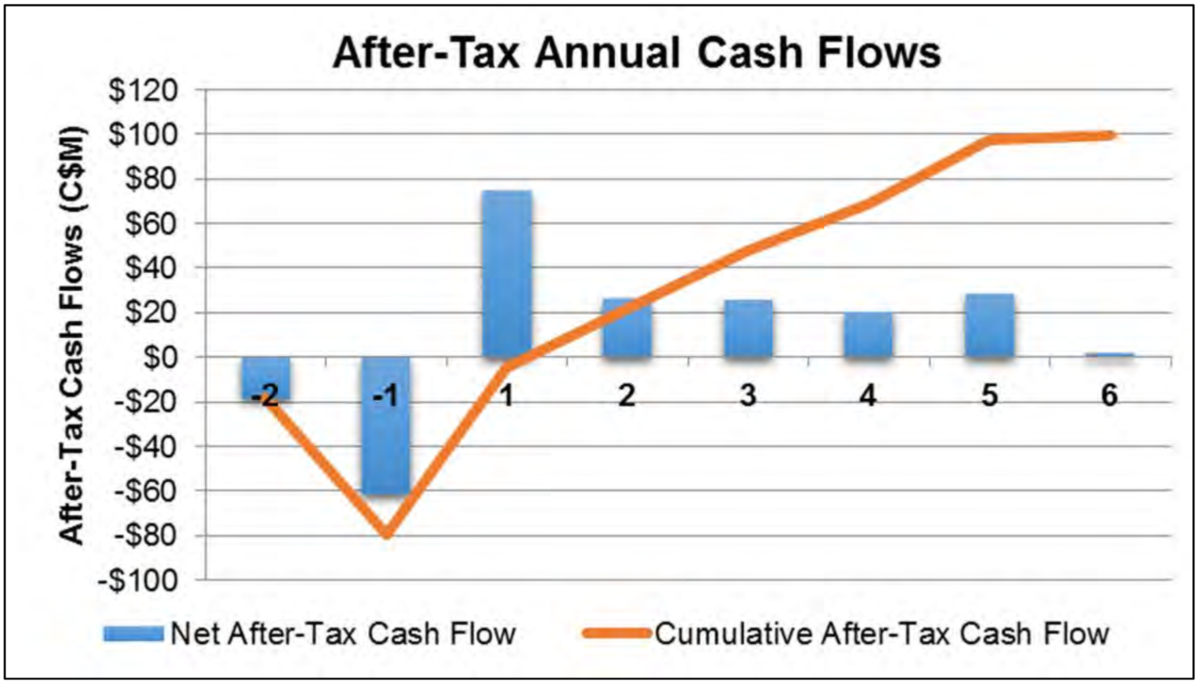
Source: JDS (2014)

Table 22.7: Summary of Results for Base Case +10%

Summary of Results	***Unit***	***Value***
Au Price	US$/oz	1,375
Ag Price	US$/oz	22.00
F/X Rate	USD:CAD	0.95
Mine Life	Years	5.0
Resource Mined	Mt	1.4
Waste Mined	Mt	0.0
Total Mined	Mt	6.4
Strip Ratio	w:o	0
Throughput Rate	t/d	1,022
Average Head Grades		
Average Au Head Grade	g/t	7.25
Average Ag Head Grade	g/t	24.44
Au Payable	koz	277.0
	koz/a	55.5
Ag Payable	koz	852.0
	koz/a	170.6
NSR (Net of Royalties)	LOM C$M	395.0
	$/t mined	286.60
Operating Costs	LOM C$M	144.9
	$/t mined	105.13
Au Cash Cost	US$/oz	517.42
Au Cash Cost (Net of By-Product)	US$/oz	449.76
Capital Costs		
Pre-Production Capital	C$M	66.2
Pre-Production Contingency	C$M	9.9
Total Pre-Production Capital	**C$M**	**76.1**
	$/t mined	**55.24**
Sustaining & Closure Capital	C$M	18.4
Sustaining & Closure Contingency	C$M	2.8
Total Sustaining & Closure Capital	**C$M**	**21.2**
	$/t mined	**15.4**
Total Capital Costs Incl. Contingency	C$M	97.4
	$/t mined	70.64
Working Capital	C$M	3.4
Pre-Tax Cash Flow	LOM C$M	$152.8
	C$M/a	$30.6
Taxes	LOM C$M	$52.1
After-Tax Cash Flow	LOM C$M	$100.7
	C$M/a	$20.2
Economic Results		
Pre-Tax $NPV_{5\%}$	**C$M**	**$118.0**
Pre-Tax IRR	**%**	**53.4%**
Pre-Tax Payback	**Years**	**1.0**
After-Tax $NPV_{5\%}$	**C$M**	**$75.6**
After-Tax IRR	**%**	**40.6%**
After-Tax Payback	**Years**	**1.2**

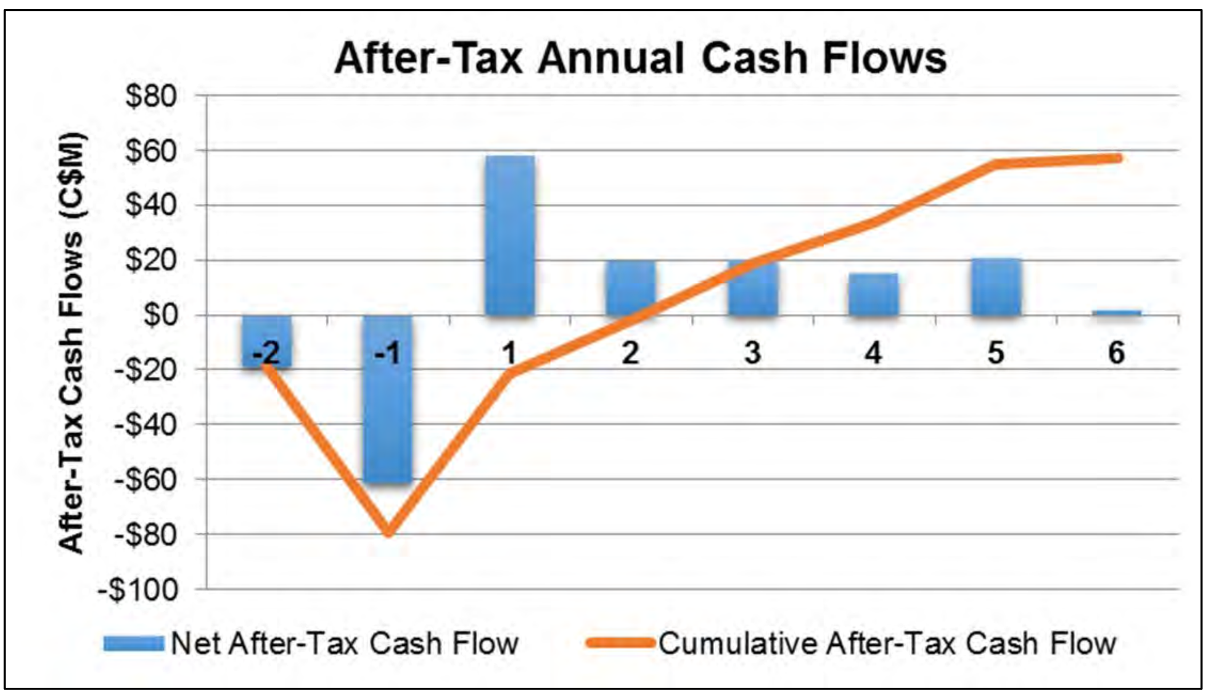
Source: JDS (2014)

Figure 22-8: Annual After-Tax Cash Flows for Base Case +10% Scenario



Source: JDS (2014)

Figure 22-9: Annual After-Tax Cash Flows for Base Case -10% Scenario



Source: JDS (2014)

Table 22.8: Summary of Results for Base Case -10%

Summary of Results	***Unit***	***Value***
Au Price	US$/oz	1,125
Ag Price	US$/oz	18.00
F/X Rate	USD:CAD	0.95
Mine Life	Years	5.0
Resource Mined	Mt	1.4
Waste Mined	Mt	0.0
Total Mined	Mt	6.4
Strip Ratio	w:o	0
Throughput Rate	t/d	1,022
Average Head Grades		
Average Au Head Grade	g/t	7.25
Average Ag Head Grade	g/t	24.44
Au Payable	koz	277.0
	koz/a	55.5
Ag Payable	koz	852.0
	koz/a	170.6
NSR (Net of Royalties)	LOM C$M	328.2
	$/t mined	238.15
Operating Costs	LOM C$M	144.9
	$/t mined	105.13
Au Cash Cost	US$/oz	515.05
Au Cash Cost (Net of By-Product)	US$/oz	459.69
Capital Costs		
Pre-Production Capital	C$M	66.2
Pre-Production Contingency	C$M	9.9
Total Pre-Production Capital	**C$M**	**76.1**
	$/t mined	**55.24**
Sustaining & Closure Capital	C$M	18.4
Sustaining & Closure Contingency	C$M	2.8
Total Sustaining & Closure Capital	**C$M**	**21.2**
	$/t mined	**15.4**
Total Capital Costs Incl. Contingency	C$M	97.4
	$/t mined	70.64
Working Capital	C$M	3.4
Pre-Tax Cash Flow	LOM C$M	$86.0
	C$M/a	$17.2
Taxes	LOM C$M	$28.1
After-Tax Cash Flow	LOM C$M	$57.8
	C$M/a	$11.6
Economic Results		
Pre-Tax NPV$_{5\%}$	**C$M**	**$62.2**
Pre-Tax IRR	**%**	**32.5%**
Pre-Tax Payback	**Years**	**1.8**
After-Tax NPV$_{5\%}$	**C$M**	**$39.5**
After-Tax IRR	**%**	**24.5%**
After-Tax Payback	**Years**	**2.1**

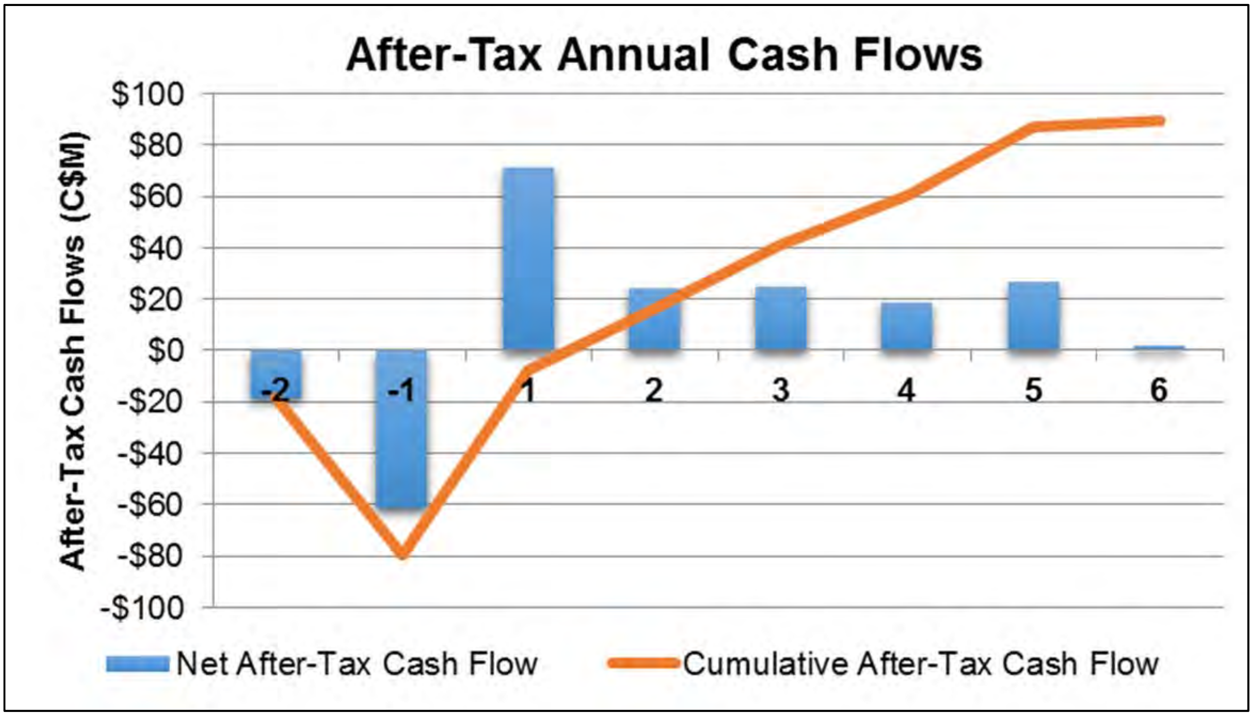
Source: JDS (2014)

Table 22.9: Summary of Results for Lower of Three-Year Trailing & Spot Metal Prices Scenario

Summary of Results	***Unit***	***Value***
Au Price	US$/oz	1,315
Ag Price	US$/oz	21.00
F/X Rate	USD:CAD	0.95
Mine Life	Years	5.0
Resource Mined	Mt	1.4
Waste Mined	Mt	0.0
Total Mined	Mt	6.4
Strip Ratio	w:o	0
Throughput Rate	t/d	1,022
Average Head Grades		
Average Au Head Grade	g/t	7.25
Average Ag Head Grade	g/t	24.44
Au Payable	koz	277.0
	koz/a	55.5
Ag Payable	koz	852.0
	koz/a	170.6
NSR (Net of Royalties)	LOM C$M	378.9
	$/t mined	274.95
Operating Costs	LOM C$M	144.9
	$/t mined	105.13
Au Cash Cost	US$/oz	516.85
Au Cash Cost (Net of By-Product)	US$/oz	452.27
Capital Costs		
Pre-Production Capital	C$M	66.2
Pre-Production Contingency	C$M	9.9
Total Pre-Production Capital	**C$M**	**76.1**
	$/t mined	**55.24**
Sustaining & Closure Capital	C$M	18.4
Sustaining & Closure Contingency	C$M	2.8
Total Sustaining & Closure Capital	**C$M**	**21.2**
	$/t mined	**15.4**
Total Capital Costs Incl. Contingency	C$M	97.4
	$/t mined	70.64
Working Capital	C$M	3.4
Pre-Tax Cash Flow	LOM C$M	$136.7
	C$M/a	$27.4
Taxes	LOM C$M	$46.3
After-Tax Cash Flow	LOM C$M	$90.4
	C$M/a	$18.1
Economic Results		
Pre-Tax $NPV_{5\%}$	**C$M**	**$104.6**
Pre-Tax IRR	**%**	**48.6%**
Pre-Tax Payback	**Years**	**1.1**
After-Tax $NPV_{5\%}$	**C$M**	**$67.0**
After-Tax IRR	**%**	**36.9%**
After-Tax Payback	**Years**	**1.3**

Source: JDS (2014)

Figure 22-10: Annual After-Tax Cash Flows for Lower of Three-Year Trailing & Spot Metal Prices Scenario



Source: JDS (2014)

22.9 SENSITIVITIES

A sensitivity analysis was performed on the Base Case metal pricing scenarios to determine which factors most affected the project economics. The analysis revealed that the project is most sensitive to metal prices, followed by head grades and operating costs. The project showed the least sensitivity to capital costs. Table 22.10 along with Figure 22-11 outline the results of the sensitivity tests performed on after-tax $NPV_{5\%}$ for the Base Case evaluated.

In addition, various scenarios were evaluated showing the project's sensitivity to gold and silver price.

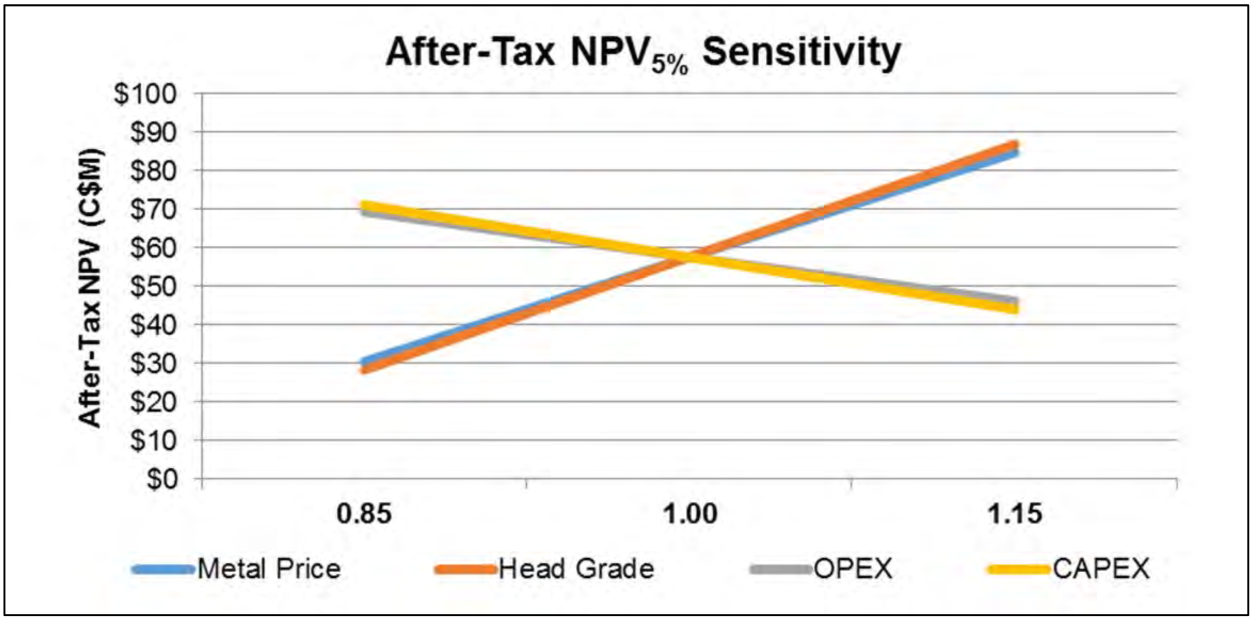
The project was also tested under various discount rates. The results of these tests for the Base Case are demonstrated in Table 22.11.

Table 22.10: Sensitivity Results for Base Case Scenario
(Au @ US$1,250/oz, Ag @ US$20/oz, F/X@ US$0.95:C$1.00)

After-Tax NPV$_{5\%}$ ($M)			
Variable	**-15%**	**100%**	**+15%**
Head Grade	28.3	57.6	86.7
Operating Costs	69.2	57.6	46.0
Capital Costs	71.3	57.6	43.9

Source: JDS (2014)

Figure 22-11: Sensitivity Results for Base Case Scenario (Au @ $/1,250 oz)



Source: JDS (2014)

Table 22.11: Base Case Scenario Discount Rate Sensitivity

Discount Rate	Pre-Tax NPV $M	After-Tax NPV $M
%	119.4	79.2
5%	90.1	57.6
7%	80.4	50.5
8%	75.9	47.2
10%	67.6	41.0
12%	60.1	35.5

Source: JDS (2014)

23.0 ADJACENT PROPERTIES

There are no adjacent properties relevant to the scope of this report.

24.0 OTHER RELEVANT DATA & INFORMATION

There is no other relevant data or information relative to the scope of this report.

25.0 INTERPRETATIONS & CONCLUSIONS

25.1 RESOURCE

A high degree of drilling and quality control work has been performed on the project by previous operators. Re-logging the core to create a geological model has created confidence in the understanding of mineralised zone controls.

The Marc Zone main portion of the mineralised deposit requires no further test work.

The AV Zone is drilled at nearly a 25 x 25 m grid spacing and shows good geological and grade continuity, yielding a large portion of the deposit in the measured category. It does require infill drilling for final mine stope planning on Section 1425. Both geology and gold assays align themselves well on sections either side of 1425 indicating there is a high probability that infill drilling will mimic flanking sections.

The JW zone is currently classified as inferred due to a lack of drill density. The existing drill holes display good geological continuity. The stockwork mechanism is consistent in virtually every hole and matches well with the reconstruction of the other two zones. Further infill drilling to provide assay information for mine planning and an upgrade in resource classification is recommended.

The AV Tails and 141 Zone have not been seriously evaluated for potential. Both, having scant drill data and thin horizons, require a re-examination for mineralogical controls.

25.2 METALLURGY

Further metallurgical test work would be beneficial to design and optimise mill feed processes including but not limited to deposit recovery variability studies, kinetic rate studies, carbon loading, pulp viscosity, comminution and thickener studies.

Samples from the deposit should be subjected to mill feed material sorting testing to determine the amenability of the mineralisation to mill feed sorting. Mill feed material sorting can provide a significant reduction to operating costs by supplying fill for underground, effectively increasing mill tonnage and reducing the generated tailings per tonne mined.

25.3 INFRASTRUCTURE & TAILINGS MANAGEMENT FACILITY

There are engineering options for grid power supply that require further investigation. There is potential to reduce grid power initial capital costs.

The tailings management facility may have optimisation potential through further design engineering in conjunction with detailed scheduling and tailings dam volume requirements.

25.4 RISKS

It is the conclusion of the QPs that the PEA summarised in this technical report contains adequate detail and information to support the potentially positive economic result. The PEA proposes the use of industry standard equipment and operating practices. 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 dilution, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

25.5 OPPORTUNITIES

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.

Pre-sorting mineralised material has had success in other mines and greatly enhanced both mining and processing efficiencies. If future test work on the Red Mountain deposit indicates pre-sorting viability, the benefits have the potential to be substantial.

26.0 RECOMMENDATIONS

Further work is recommended in two phases: completion of an exploration program and subsequent commencement of a prefeasibility study (PFS). Prior to initiating a prefeasibility study, current inferred mineralised material that is considered potentially economic in the PEA study should be drilled to an indicated level for inclusion into future resource estimations in preparation for a PFS level study. Paralleling long lead-time test work and engineering with exploration resource definition drilling is recommended. Cost estimates for the recommended phases of work are included below in Table 26.1 and Table 26.2.

Table 26.1: Phase 1 Exploration & Pre-PFS Engineering Cost Estimate

Item	Cost
Underground Resource Definition Drilling	
Assay	$40,000
Labour	$280,000
Underground	$530,000
Drilling	$864,000
Camp	$456,000
Helicopter	$330,000
Subtotal	$2,500,000
Pre-PFS Engineering	
Pre-sorting Mineralised Material Test Work & Bench Test Work	$50,000
BC Hydro Impact and Facility Study	$50,000
Tailings Management Facility Design	$200,000
Access Road Detailed Design	$50,000
Total Phase 1 Estimate	**$2,850,000**

Source: JDS (2014)

Table 26.2: Phase 2 Prefeasibility Study Cost Estimate

Item	Cost
Revised Resource Estimation	$50,000
Mine Planning & Reserve Estimation	$225,000
Metallurgical Test Work	$200,000
Processing Design	$300,000
Project Infrastructure	$175,000
Tailings Management Facility	$100,000
Report Compilation	$50,000
Total Phase 2 Estimate	**$1,100,000**

Source: JDS (2014)

27.0 UNITS OF MEASURE, ABBREVIATIONS AND ACRONYMS

27.1 27.1 UNITS OF MEASURE

Table 27-1: Units of Measure

'	Foot
"	Inch
µm	Micron (micrometre)
AT	Assay ton
C$	Canadian dollars
CAD	Canadian dollars
cfm	Cubic feet per minute
cm	Centimetre
dpa	Days per annum
dmt	Dry metric tonne
ft	Foot
ft^3	Cubic foot
g	Gram
hr	Hour
ha	Hectare
hp	Horsepower
kg	Kilogram
km	Kilometre
km^2	Square kilometre
kt	Kilotonnes
kW	Kilowatt
KWh	Kilowatt-hour
L	Litre
lb or lbs	Pound(s)
m	Metre
M	Million
m^2	Square metre
m^3	Cubic metre
min	Minute
mm	Millimetre
Mtpa	Million tonnes per annum
Mt	Million tonnes
MW	Mega watt
ºC	Degree Celsius
oz	Troy ounce
ppb	Parts per billion
ppm	Parts per million
s	Second
t	Metric tonne

t/a	Tonnes per annum
tpd	Tonnes per day
tph	Tonnes per hour
US$	US dollars
V	Volt
W	Watt
wmt	Wet metric tonne

27.2 ABBREVIATIONS AND ACRONYMS

Table 27-2: Abbreviations & Acronyms

% or pct	Percent
AA	Atomic absorbtion
AAS	Atomic absorption spectrometer
ABA	Acid base accounting
Ag	Silver
ARD/ML	Acid rock drainage/metal leaching
Au	Gold
ANFO	Ammonium Nitrate/Fuel Oil
ARD	Acid rock drainage
Banks	Banks Island Gold Inc.
Barrick	Barrick Gold Corporation
BC	British Columbia
BC EAA	British Columbia Environmental Assessment Act
Bond	Bond Gold Canada Inc.
CAPEX	Capital costs
CEAA	Canadian Environmental Assessment Act
CIM	Canadian Institute of Mining
CIP	Carbon-in-Pulp
CRF	Cemented rock fill
COG	Cut-off grade
Cu	Copper
D&F	Drift and Fill
EAC	Environmental Assessment Certificate
EAO	British Columbia Environmental Assessment Office
Eco-Tech	Eco-Tech Laboratories located in Stewart, BC
G&A	General & Administrative
Ha	Hectare
HDPE	High density polyethylene
ICP	Inductively coupled plasma
ID^2	Inverse distance squared
IDM	IDM Mining Ltd.
IRA	Inter-ramp angles

IRR	Internal rate of return
JDS	JDS Energy & Mining Inc.
LAC	Lac Minerals Ltd.
LH	Longhole stoping
LHD	Load haul dump machines
LOM	Life of mine
masl	Metres above sea level
MELP	BC Ministry of Environment, Lands and Parks
MIK	Multi-indicator kriging
Min-En	Mineral Environments Laboratories of North Vancouver, BC
N,S,E,W	North, South, East, West
NAD	North American Datum
NAMC	North American Metals Corp.
NFA	Nisga'a Final Agreement
NI 43-101	National Instrument 43-101
NN	Nearest neighbour
NPV	Net present value
NSR	Net Smelter Return
NTS	National Topographic System
OPEX	Operating costs
PEA	Preliminary economic assessment
PFS	Preliminary feasibility study
PH	potential of hydrogen; a measure of acidity or alkalinity of a solution
PM	Preventative maintenance
PPM	Parts per million
Project	Red Mountain Project
QA/QC	Quality assurance/quality control
QP	Qualified Person
RKSD	relative kriging standard deviation
ROM	Royal Oak Mines Inc.
Seabridge	Seabridge Gold Inc.
SG	Specific gravity
TMF	Tailings management facility
UTM	Universal Transverse Mercator
Wotan	Wotan Resources Corp.
X,Y,Z	Cartesian coordinates, also Easting, Northing and Elevation

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APPENDIX A – Qualified Person Certificates

CERTIFICATE OF AUTHOR

I, Dunham L. Craig, P.Geo., do hereby certify that:

1. I am currently employed as Project Manager by JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C 2T6;

2. This certificate applies to the technical report titled "Preliminary Economic Assessment NI 43-101 Technical Report, Red Mountain Gold Project, North Western BC, Canada (the "Technical Report") with an Effective Date of July 23, 2014 prepared for IDM Mining Ltd.

3. I am a Registered Professional Geoscientist (P.Geo.#20406) registered with the Association of Professional Engineers, Geologists of British Columbia;

   I am a graduate of the University of British Columbia with a B.Sc. degree in Geology in 1988. I have practiced my profession continuously since 1988;

   I have held senior mine production and mine technical positions in mining operations in Canada and Central America during which I was responsible for scoping, pre-feasibility, and bankable feasibility studies. From 1995 to 2000 I was responsible for discovery and integral to production of a mining operation in British Columbia and as Project & Construction Manager for a mine in Central America. I have conducted resource estimation, reserve estimates, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person from 1995 to 2014.

   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 the NI 43-101;

4. I visited the Red Mountain project site on May 29, 2014 and intermittently between April 2000 until October 2000;

5. I am responsible for Sections 1,2,3,4,5,6,7,8,9,10,11,12,14,19,20,21,22,23,24,25,26,27 and 28 of the Technical Report;

6. I have prior involvement with the property that is the subject of this Technical Report. I intermittently worked at the Red Mountain project site for a total of 25 days for the purposes of determining quality control, geological interpretation, resource estimating criteria and metallurgical test work. I directly supervised consultants and staff performing evaluation work for the purpose of resource calculations and metallurgical test work. During this period I was employed as Vice President of Exploration and Corporate Development at North American Metals Corp, the previous owners of the Red Mountain Project until 2001. I was the author of the NI 43-101 Red Mountain Project, British Columbia Canada Technical Report prepared for Seabridge Resources Inc. dated March 4, 2002;

7. 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;

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#860 – 625 Howe Street Vancouver, BC V6C 2T6

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t 250.763.6369 f 250.763.6302
#200 – 532 Leon Avenue, Kelowna, BC V1Y 6J6

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: July 23, 2014
Signing Date: September 3, 2014

Dunham Craig, P.Geo.

CERTIFICATE OF AUTHOR

I, Gordon Doerksen do hereby certify that:

1. I am currently employed as V.P. Technical Services with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver, BC, V6C 2T6;

2. This certificate applies to the technical report titled "Preliminary Economic Assessment NI 43-101 Technical Report, Red Mountain Gold Project, North Western BC, Canada (the "Technical Report") with an Effective Date of July 23, 2014 prepared for IDM Mining Ltd.

3. I am a Professional Mining Engineer (P.Eng. #32273) registered with the Association of Professional Engineers, Geologists of British Columbia. 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.

   I am a graduate of Montana Tech with a B.Sc. in Mining Engineering (1990). I have been invlolved 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.

   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.

4. I visited the Red Mountain project site on May 29, 2014;

   I am responsible for Sections 15 and 16 of the Technical Report;

5. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the National Instrument 43-101;

6. I have not had prior involvement with the property that is the subject of the Preliminary Economic Assessment of the Red Mountain Project;

7. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

8. 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: July 23, 2014
Signing Date: September 2, 2014

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA G. E. DOERKSEN # 32273

SEPT 2, 2014

Gordon Doerksen, P.Eng.

PARTNERS IN
ACHIEVING
MAXIMUM
RESOURCE
DEVELOPMENT
VALUE

JDS Energy & Mining Inc.

CERTIFICATE OF AUTHOR

I, Scot G. Klingmann, P.Eng., do hereby certify that:

1. I am currently employed as Senior Engineer by JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C 2T6;

2. This certificate applies to the technical report titled "Preliminary Economic Assessment NI 43-101 Technical Report, Red Mountain Gold Project, North Western BC, Canada (the "Technical Report") with an Effective Date of July 23, 2014 prepared for IDM Mining Ltd.;

3. I am a Mining Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia (P.Eng. #32339);

4. I am a graduate of Queen's University with an M.Sc. in Mining Engineering, 1999. I have practiced my profession continuously since 1999;

5. I have worked in technical and management positions at mines in Canada. I have been an independent consultant for two years and have performed mine design, planning, cost estimation, technical due diligence reviews and report writing for mining projects around North America;

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 the NI 43-101;

7. I visited the Red Mountain project site on May 29, 2014;

8. I am responsible for Section 18 of the Technical Report;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. 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;

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: July 23, 2014
Signing Date: August 30, 2014

Aug. 30, 2014.

PROFESSIONAL
PROVINCE OF BRITISH COLUMBIA
S. G. KLINGMANN
32339
ENGINEER

Scot Klingmann, P.Eng.

CERTIFICATE OF AUTHOR

I, Tom W. Shouldice, P.Eng., do hereby certify that:

1. I am currently employed as a Consulting Metallurgist for TS Technical Services Ltd. with an office at address; 1215 Canyon Ridge Place, Kamloop BC, V2H 0A1

2. This certificate applies to the technical report titled "Preliminary Economic Assessment NI 43-101 Technical Report, Red Mountain Gold Project, North Western BC, Canada (the "Technical Report") with an Effective Date of July 23, 2014 prepared for IDM Mining Ltd.

3. I am a Registered Professional Metallurgical Engineer (P.Eng. #27489 registered with the Association of Professional Engineers, Geologists of British Columbia;

   I am a graduate of Queens University with a B.Sc. in Metallurgical Engineering (1993). I have been involved in mineral processing since 1989 and have practiced my profession continuously since 1993. I have held senior mill production and mill technical positions in mining operations in Canada and the US. I have worked as VP of Technical Services and President of a large metallurgical testing facility located in Canada. For the past year, I have been a consultant and have performed metallurgical test work planning and auditing, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person, for several engineering studies and technical reports

   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 the NI 43-101;

4. I am responsible for Section 13 and 17 of the Technical Report;

5. I have had no prior involvement with the property that is the subject of this Technical Report;

6. 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;

7. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: July 23, 2014
Signing Date: August 19, 2014

Thomas Shouldice, P.Eng.

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA T.W. SHOULDICE 27489

Aug 19, 2014