MINERAL RESOURCE UPDATE for the RED MOUNTAIN GOLD PROJECT, NORTHWESTERN BC, CANADA

Prepared for:

IDM Mining Ltd.

Prepared by:
Dr. Gilles Arseneau, P. Geo.
Andrew Hamilton, P. Geo.

ARSENEAU Consulting Services Inc.

Effective Date: June 15, 2018
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Date & Signature Page

This report, entitled Mineral Resource Update for the Red *Mountain Gold Project, Northwestern BC, Canada* (effective as of June 15, 2018), was prepared and signed by the following authors:

Original document signed by:

"Gilles Arseneau"	*July 31, 2018*
Dr. Gilles Arseneau, P. Geo.	*Date Signed*

Original document signed by:

"Andrew Hamilton"	*July 31, 2018*
Andrew Hamilton, P. Geo.	*Date Signed*

Contents

Tables

Figures

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

Arseneau Consulting Services Inc. (ACS) was retained be IDM Mining Ltd. (IDM or IDM Mining) to prepare an update of the Mineral Resource Estimate for 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 incorporate new drilling information gathered by IDM and to update the mineral resources for the Red Mountain gold project to include the results of the 2017 drilling program.

On April 12, 2014, IDM optioned the property from Seabridge Gold Inc. (Seabridge) with the intent of initiating a Preliminary economic assessment (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, initiated project permitting under the British Columbia and Canadian Environmental Assessment review process, completed a total of 190 surface and underground drill holes, engaged JDS Energy & Mining (JDS) to prepare a PEA for the Red Mountain gold project that was published on September 3, 2014, an updated PEA also by JDS published on July 12, 2016, and a Feasibility Study (FS) also by JDS published on June 26th, 2017.

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).

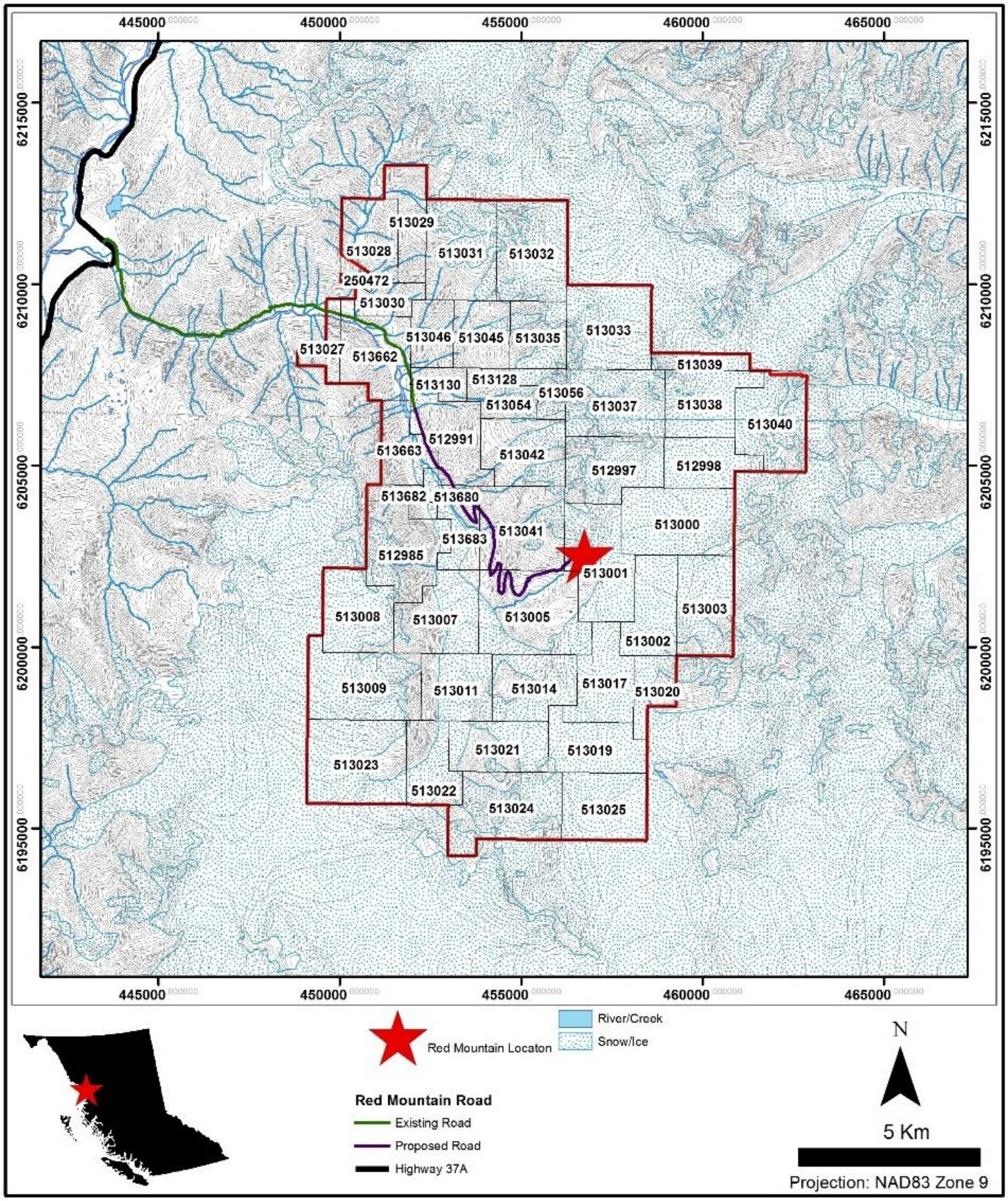
The area is characterised by rugged, steep terrain with weather conditions typical of the north coastal mountains including significant (+2 m) snow accumulation in the winter. Access to the site is presently by helicopter from Stewart with a flight time of 10 to 15 minutes. An existing road extends for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

The Red Mountain gold 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 (2017)

Figure 1.2: Red Mountain gold project Claim Map



Source: IDM (2014)

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

1.2.2 Ownership

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

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 were purchased by Bond's successor, Lac). The agreements each provide for a royalty in the form of a net smelter return ("NSR") and one of them, the Wotan agreement, has an area of influence.

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

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

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

Upon the commencement of commercial production, IDM will make an additional one-time $1.5 million cash payment to Seabridge and Seabridge will also retain a gold metal stream on the Red Mountain gold project to acquire 10% of the annual gold production from the Property at a cost of $1,000 per ounce up to a maximum of 500,000 ounces produced (50,000 ounces to Seabridge). Alternatively, Seabridge may elect to receive a one-time cash payment of $4 million at the commencement of production in exchange for the buy-back of the gold metal stream.

1.3 GEOLOGY & MINERALIZATION

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt. There are three primary stratigraphic elements in Stikinia and all are present in the Stewart area: Middle and Upper Triassic clastic rocks of the Stuhini Group, Lower and Middle Jurassic volcanic and clastic rocks of the Hazelton Group, and Upper Jurassic sedimentary rocks of the Bowser Lake Group. Many primary textures are preserved in rocks from all of these groups, and mineralogy suggests that the regional metamorphic grade is probably lower greenschist facies.

The current geologic model for Red Mountain is based on a favourable horizon that is defined by hydrothermal breccias. High grade mineralized zones are concordant with the breccias and hosted both within the breccias and their un-brecciated equivalents. A definitive origin for the host structure has not been identified. The favourable horizon was subject to folding during mid-Cretaceous transpressional tectonics, resulting in high amplitude, gently northwesterly plunging folds.

Mineralized zones consist of crudely tabular, northwesterly trending and variably dipping gold and silver bearing iron sulphide stockworks. Pyrite is the primary sulphide mineral; however, locally pyrrhotite is important. The stockworks zones are developed primarily within the Hillside porphyry, rafts of sedimentary and volcaniclastic rocks and brecciated contacts between the two lithologies.

The stockwork zones consist of pyrite micro-veins, coarse-grained pyrite veins, irregular coarse-grained pyrite masses and breccia matrix pyrite. In Hillside porphyry and breccia units, pale, strong sercite, k-spar, silica and carbonate alteration are associated with the gold/pyrite mineralization. Vein widths vary from 0.1 cm to approximately two metres 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 (Barnett, 1991). The stockwork zones are surrounded by more widespread zone of disseminated pyrite and pyrrhotite alteration, as well as common red sphalerite disseminations.

1.4 HISTORY, EXPLORATION & DRILLING

Placer mining commenced in Bitter Creek downstream from Red Mountain in the early 1900's. 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 1989, gold mineralization in the Marc and Brad zones were discovered by drilling. LAC Minerals Ltd. (LAC) acquired Bond in 1991. Surface drilling on the Marc, AV and JW zones continued in 1991, 1992, 1993 and 1994. Underground exploration of the Marc zone was conducted in 1993 and 1994. In 1995, LAC was acquired by Barrick who subsequently optioned the property to Royal Oak Mines Ltd. (Royal Oak) in 1996. North American Minerals Inc. (NAMC) purchased the property from the receivership sale of Royal Oak in 2000. NAMC subsequently sold the property to Seabridge in 2002 who optioned the property to Banks Island Gold Ltd. (Banks). Banks terminated the option in 2013 and the property reverted to Seabridge. Seabridge subsequently optioned the property to IDM in 2014.

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

Table 1.1: Red Mountain 1988-2017 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 mineralization by drilling in the Marc zone (3,623 m); airborne EM and magnetic survey.
1990	Exploration of Marc zone and adjacent area (11,615 m of drilling) by Bond.
1991	LAC acquired 100% of Bond. A 2,400 m drill program was completed on the Marc and AV Zones.
1992	Results of a 4,000 m drill program by LAC increased Red Mountain resources and indicated excellent potential for expansion.
1993	28,800 m of surface drilling defined the Marc, AV, and JW Zones and identification of the 141 Zone. An underground exploration adit allowed bulk sampling of the Marc zone. 8,600 m of underground drilling completed in the Marc zone.
1994	LAC completed a 350 m extension of the main decline, 30,000 m of underground drilling and 16,000 meters of surface drilling.
1995	Barrick Gold acquires Lac Minerals. No exploration work completed by Barrick. Royal Oak purchases the project from Barrick.
1996	Royal Oak undertakes exploration to explore for additional reserves. Extended underground drift by 304 m and completed 26,966 m of surface and underground drilling.
2000	North American Metals Corp. purchased the property and project assets from Price Waterhouse Coopers; completed detailed relogging of existing drill core and constructed a geological model for resource estimation purposes and exploration modelling.
2002-2012	Seabridge purchases property, completes two Preliminary Assessment Studies (PEA)
2012-2013	Banks Island Gold options property, three surface drill holes completed. Lidar survey and metallurgical work completed. Published PEA study.
2014	IDM enters into Option Agreement with Seabridge Gold. 12 core holes completed. Surface mapping and sampling completed, with emphasis on Cambria Zone.
2016	Underground workings dewatered and rehabilitated. IDM drilled 11 surface holes and 51 underground holes totalling 8,123.44 metres, completed surface rock and channel sampling, discovered gold mineralization at Lost Valley. Updated PEA was published. Metallurgical, geotechnical and hydrological work completed.
2017	IDM drilled 11 surface and 105 underground holes totalling 29,299.26 metres, completed surface rock and channel sampling, constructed a new geological model and published a Feasibility Study. Claim ownership transfers from Seabridge to IDM Mining. Environmental Application and Environmental Impact Statement submitted.

1.5 MINERAL PROCESSING & METALLURGICAL TESTING

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

Table 1.2: presents the estimated metallurgical recoveries based on the correlation between cyanide concentration and gold or silver recovery. The 141 zone recoveries are a weighted average of test work results from four composites. The overall projected recovery is a weighted average of recovery by zone and projected tonnages based on the mine plan.

Table 1.2: Estimated Metallurgical Recoveries

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

Source: JDS and Basemet Laboratories (2017)

1.6 MINERAL RESOURCE ESTIMATE

Numerous resource estimates were completed from 1989 to present. During 2000, NAMC conducted a detailed review of all data, re-logged all core within a 20 m envelope of the Marc, AV and JW mineralized zones and reviewed all exploration holes for potential inclusion into the resource. An extensive quality control and quality assurance (QA/QC) review was completed on all exploration work, and a comparative analysis was performed on drill hole data, underground bulk sampling, and geology. The 2000 NAMC resource was reviewed, cross checked and verified for accuracy in May 2014. In 2015, IDM drilled 12 core holes on the property in 2014, 62 holes in 2016 and 114 holes in 2017. The one hundred and ninety IDM drill holes combined with the past historical drilling on the property form the basis for the resource estimate in Table 1.3.

Table 1.3: Mineral Resource Statement at a 3 g/t Cut-off Grade, effective June 15, 2018*

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	715,100	10.65	41.46	244,800	953,300
Indicated	9,300	11.02	45.63	3,300	13,700
Inferred	0			0	0
AV Zone					
Measured	837,200	7.75	19.77	208,700	532,200

Indicated	116,500	8.47	20.81	31,700	78,000
Inferred	3,200	9.32	12.27	900	1,200
JW Zone					
Measured	275,600	7.96	20.07	70,500	177,800
Indicated	150,500	7.24	18.48	35,000	89,400
Inferred	4,900	8.83	16.88	1,400	2,600
141 Zone					
Indicated	234,700	4.86	7.04	36,700	53,100
Inferred	18,000	4.67	3.86	2,700	2,200
Smit					
Indicated	241,400	4.54	4.64	35,200	36,000
Inferred	48,100	5.28	2.26	8,200	3,500
Marc Footwall					
Indicated	28,600	5.76	10.79	5,300	9,900
Inferred	21,400	4.61	1.95	3,200	1,300
Marc Outlier Zone					
Indicated	12,100	5.24	28.64	2,000	11,100
Inferred	0			0	0
Marc NK Zone					
Indicated	37,500	7.4	8.26	8,900	9,900
Inferred	500	6.79	8.19	100	100
JW HW					
Indicated	39,900	5.66	32.28	7,300	41,400
Inferred	2,100	7.22	3.55	500	200
Bray					
Indicated	57,100	5.68	10.43	10,400	19,100
Inferred	73,800	4.66	7.49	11,100	17,800
Chicka					
Indicated	15,800	9.46	3.82	4,800	1,900
Inferred	600	5.3	1.57	100	0
JW FW					
Inferred	4,800	16.09	33.78	2,500	5,200
SF					
Inferred	54,600	6.88	17.55	12,100	30,800
Cambria					
Inferred	84,000	6.89	4.54	18,600	12,300

Total Measured & Indicated	**2,771,300**	**7.91**	**22.75**	**704,600**	**2,026,800**
Total Inferred	**316,000**	**6.04**	**7.6**	**61,400**	**72,000**

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

1.7 CONCLUSIONS & RECOMMENDATIONS

1.7.1 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 mineralized zone controls.

The Marc, AV and JW Zones forms the main portion of the mineralized deposit and require no further drilling.

The 141 and Smit Zones are drilled at nearly a 25 to 50 m grid spacing and shows reasonably good geological and grade continuity yielding a large portion of the deposits in the indicated category. The zones will require infill drilling to confirm the geological continuity prior to mining. The infill drilling carried out in 2017 confirmed the geological and grade continuity of the AV and JW and was successful in upgrading the inferred resources in these zones to indicated classification and upgrading some indicated blocks to measured in the Marc Zone.

The Cambria, SF and JW FW are currently classified as inferred due to wider spaced drill density. The existing drill holes display reasonable geological continuity and it is reasonable to assume that the majority of the inferred could be converted to indicated with additional drilling.

1.7.2 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.

1.7.3 Recommendations

Further work is recommended by ACS for the Red Mountain gold project. The total next phase work program is estimated at $6.3 million as follows:

- It is recommended that the 2017 feasibility study (FS) be updated to include the new indicated mineral resources defined as part of this report. The total estimated cost is expected to be in the order of $1.2 million for the feasibility update.

- It is recommended that a 10,000 metre drilling program of infill and step out drilling be carried out to upgrade and expand recently identified inferred resources. Total estimated cost for this program is $5.0 million.

- It is recommended that metallurgical test work be carried out on samples from the Smit Zone as there is potential to convert resources to reserves. Total estimated cost for this program is $100,000.

2.0 INTRODUCTION

2.1 BASIS OF TECHNICAL REPORT

Arseneau Consulting Services Inc. (ACS) was retained be IDM Mining Ltd. (IDM or IDM Mining) to prepare an update of the Mineral Resource Estimate for 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 incorporate new drilling information gathered by IDM and to update the mineral resources for the Red Mountain gold project to include the results of the 2017 drilling campaign.

Since acquiring the project, IDM has completed a comprehensive review and validation of the Red Mountain geological and environmental data, initiated project permitting under the British Columbia and Canadian Environmental Assessment review process, completed a total of 190 surface and underground drill holes, engaged JDS Energy & Mining (JDS) to prepare a PEA for the Red Mountain gold project that was published on September 3, 2014, an updated PEA also by JDS published on July 12, 2016, and a Feasibility Study (FS) also by JDS published on June 26th, 2017.

2.2 SCOPE OF WORK

ACS was requested to update the mineral resources for the Red Mountain gold project to include information from the recently completed IDM drilling program.

The ACS scope of work included:

- Compile the technical report including historical data and information provided by other consulting companies
- Carry out a site visit
- Validate and modify the existing wireframes
- Statistical review of the assay data
- Prepare block model for resource estimation
- Estimate mineral resource by Ordinary Kriging
- Validate block model resource and classify the mineral resource

2.3 QUALIFIED PERSON RESPONSIBILITIES & SITE INSPECTIONS

The Qualified Persons (QPs) preparing this technical report are specialists in the fields of geology, exploration and mineral resource estimation and classification.

As of the effective date of the report, 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 outlined in Table 2.1.

Table 2.1: Qualified Person Responsibilities

QP	Company	Report Section(s) of Responsibility
Andrew Hamilton, P. Geo		4, 5, 6, 7, 8, 9, 10, 11, 12 and 16
Gilles Arseneau, P.Geo.	ACS	1, 2, 3, 13, 14, 15, 17, 18, 19, 20 and 21

Source: JDS (2014)

2.4 SITE VISITS & INSPECTIONS

QP site visits were conducted as follows:

- Dr. Gilles Arseneau of ACS responsible for the preparation of the mineral resources visited the site on May 7, 2017 for one day.
- Andrew Hamilton last visited the site from October 3rd to 6th, 2017.

Both QP’s have visited the site on previous occasions.

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 an updated mineral resource estimate 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 ACS.

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

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

Engineering and geological information from historical documents was used in this report after determination by ACS that the work was performed 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 IDM in 2014, 2016 and 2017.

Metallurgical test work was conducted by LAC and NAMC staff from 1991 to 2001, and by JDS on behalf of IDM in 2017.

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. In the 1990's a road was pioneered from Highway 37A up the Bitter Creek valley to the base of Red Mountain. Two portions were washed out in 2011 and the road is mostly overgrown with vegetation.

The deposit is located under the summit of Red Mountain at elevations of between 1,600 and 2,000 m. The site is drained by Goldslide Creek, which flows southwest to the flank of the Bromley Glacier and by 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 al- 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 owns a 100% interest in 47 contiguous claims that comprise an area of 17,125.2 ha (Table 4.1 and Figure 4.2).

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

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

Figure 4.1: Red Mountain gold project Location Map

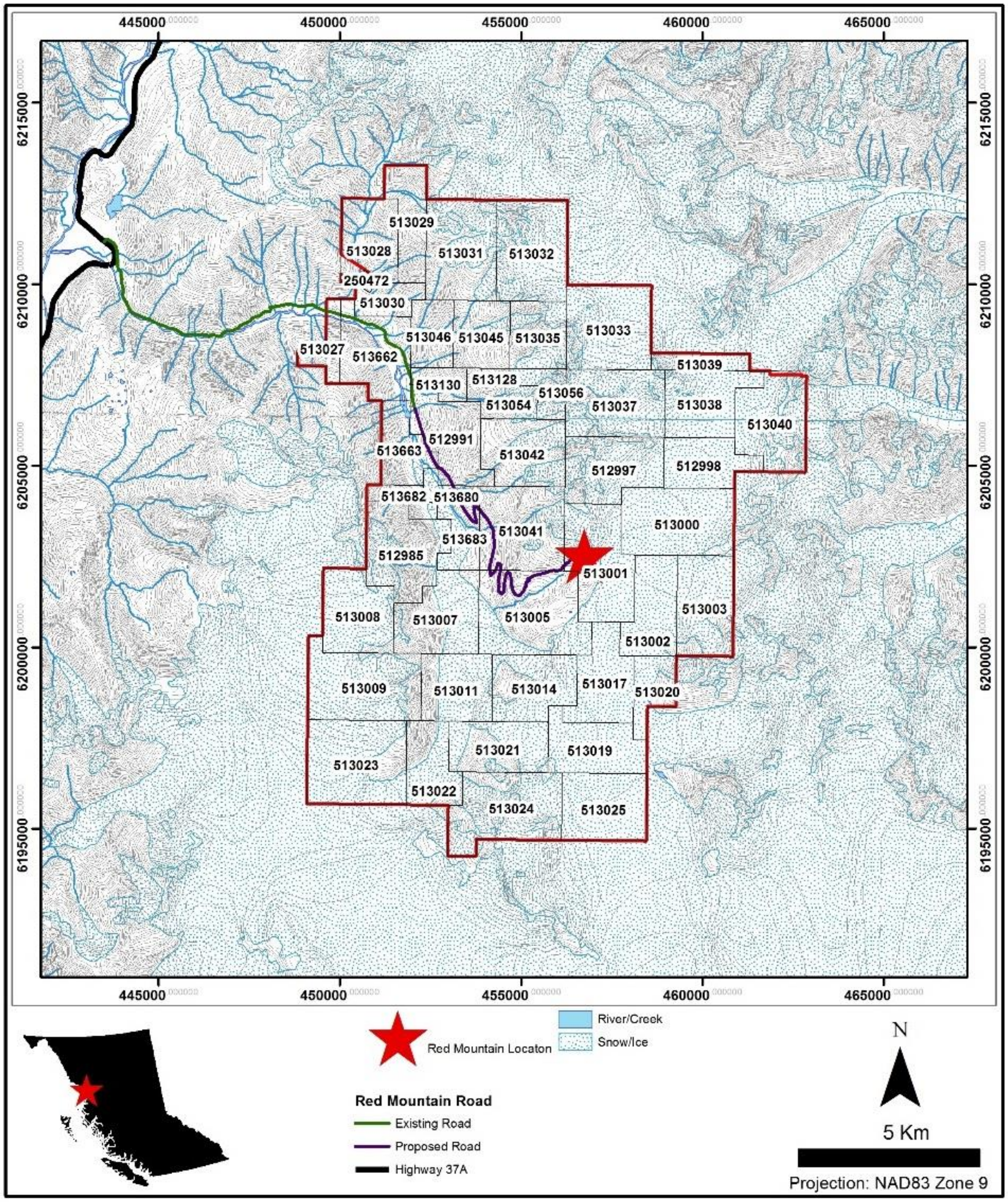
Source: IDM (2017)

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)

4.3 ROYALTIES, AGREEMENTS & ENCUMBRANCES

4.3.1 Royalties, Metal Stream and Other

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

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

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

The mineral resources in this report are also subject to a gold metal stream whereby Seabridge may acquire up to 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 time of commencement of production in exchange for the buy-back of the metal stream.

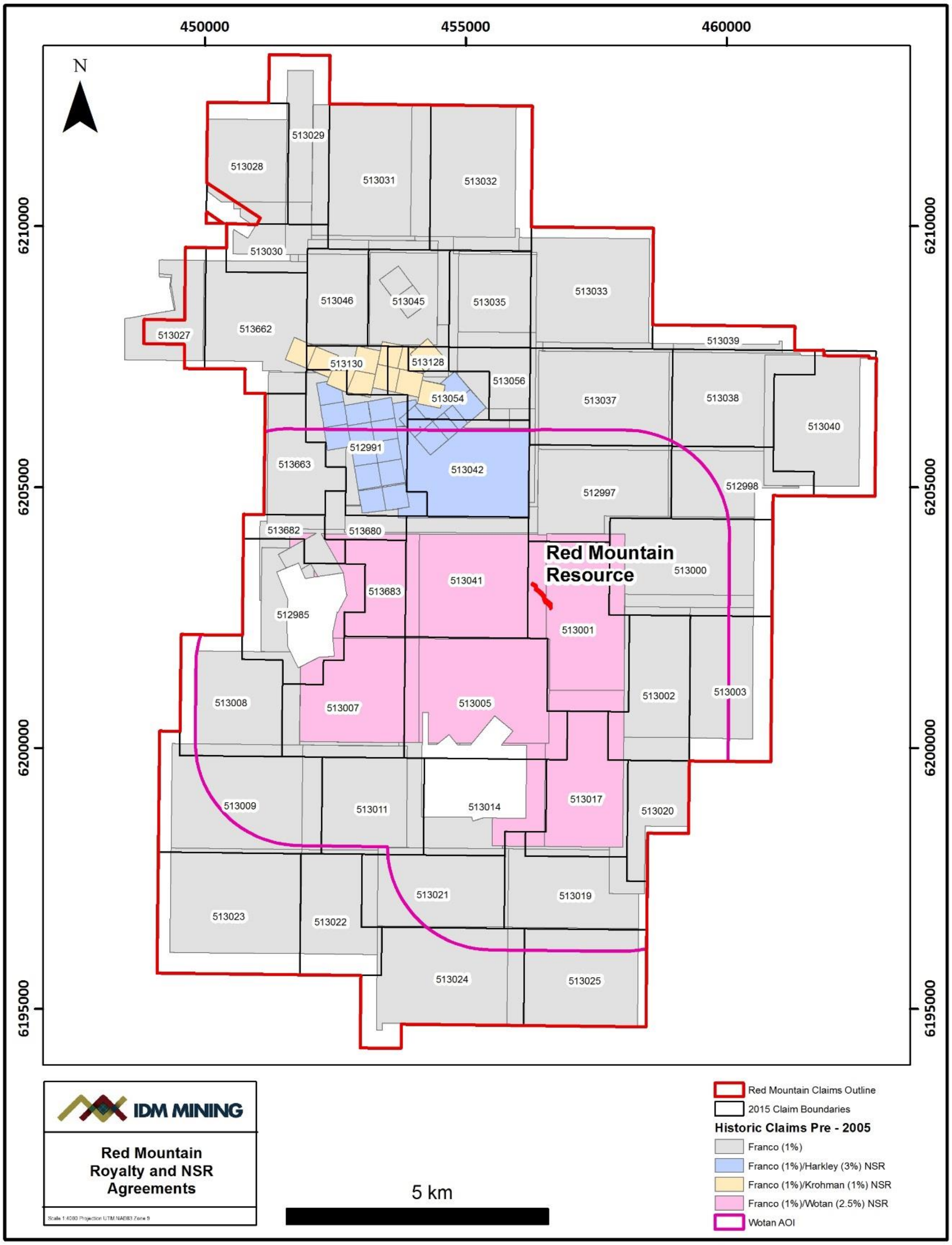
4.3.2 Underlying Agreements

The principal agreements governing the Red Mountain gold project are listed below, along with a summary of the more salient provisions and identified shading on Figure 4.3

1. Wotan Agreement: Agreement dated July 26, 1989 between Bond, Wotan and Dino Cremonese granting Bond an option to acquire seven mineral claims. IDM is obligated to pay Wotan an uncapped 2.5% NSR royalty on production from these claims, 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.

2. Krohman Sinitsin Agreement: IDM is obligated to pay Darcy Krohman and Greg Sinitsin a 1.0% NSR royalty on production from claims 513128 and 513130. IDM may buy out the royalty at any time for $500,000.
3. Harkley Fegan Scott Agreement: Option agreement dated September 26, 1989 between Bond, Harkley Silver Mines Ltd., Stephen Fegen and Wesley Scott, as amended by letter agreement dated September 30, 1992 between LAC and Harkley Silver. IDM is obligated to pay Harkley Silver an uncapped 3.0% NSR royalty on production from claims 513042 and 513054.
4. Franco Agreement: Separated Royalty Agreement dated May 25, 2017 between Franco and IDM granting an uncapped 1>0% NSR royalty on production from then existing claims sold by Barrick to Royal Oak. Franco is entitled to receive an additional $10.00 cash production payment per ounce on all ounces of gold produced from the property in excess of 1,850,000 ounces (“Production Payments”).
5. Seabridge Agreement: Option agreement dated April 15, 2014 between IDM and Seabridge Gold Inc., granted IDM the right to acquire 47 mineral claims. IDM is obligated to pay Seabridge a one-time $1.5 million upon the commencement of commercial production, and Seabridge also retained a gold metal stream on the Red Mountain gold project to acquire 10% of the annual gold production from the Property at a cost of US$1,000 per ounce up to a maximum of 500,000 ounces produced (50,000 ounces to Seabridge). Alternatively, Seabridge may elect to receive a one-time cash payment of $4 million at the commencement of production in exchange for the buy-back of the gold metal stream (the obligations described in this paragraph are referred to collectively as the “Secured Obligations”).
6. Mortgage Charge and Security Agreement: Security Agreement dated May 25, 2017 between IDM and Seabridge, granting Seabridge as general and continuing collateral security for the payment of the Secured Obligations a security interest in the mineral claims comprising the Red Mountain gold project (together with leases made in replacement thereof), and all records, rights and permits relating to or in connection with the mineral claims.

Figure 4.3: Red Mountain Royalty and NSR Agreements



4.4 ENVIRONMENTAL LIABILITIES & PERMITTING

4.4.1 Environmental Liabilities

A $1,000.000 cash reclamation bond has been posted with the provincial government against the property and can be recovered pending closure and remediation of certain environmental requirements, including the following:

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

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.

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

4.4.2 Required Permits & Status

Pursuant to section 3(1) of the Reviewable Projects Regulation pursuant to the Canadian Environmental Assessment Act (CEAA, 2012), 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 the issuance of an Environmental Assessment Certificate (EAC). The BC Environmental Assessment Office (EAO) issued a Section 10 Order to IDM on November 2, 2015 confirming the project will require an EAC. The EAO further issued a Section 11 Order on February 10, 2016 which outlined the requirements for the environmental assessment of the project under the BC EAA. The project will also require a review and decision pursuant to the Canadian Environmental Assessment Act (CEAA) 2012. The BC EAA and CEAA processes are coordinated and only one Application is required. The Application/EIS was accepted by the EAO and CEAA in November 2017 and is currently under review.

it is anticipated that the project will require approvals under the Mines Act (1996b), Environmental Management Act (2003) and Land Act (1996a). IDM will pursue synchronous permitting for provincial permits relating to the development and operation of the Project after receipt of the environmental assessment certificate. Using this approach, review timelines for provincial permit applications will be coordinated and agreed upon with the Major Mine Permitting Office. No decisions on commercial production related to provincial permits is possible until completion of the decision pursuant to the BC EAA.

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 Ltd. in 1994 to the Hartley Gulch-Otter Creek area. Currently this road is passable for only a few kilometres from the highway. The remainder is not passable, as sections have been subjected to washout or landslide activity.

5.2 CLIMATE

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

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

5.3 TOPOGRAPHY, ELEVATION & VEGETATION

A view showing the topography of the Red Mountain area is provided in Figure 5.1.

Figure 5.1: View of Red Mountain & Camp Looking Northeast (1,400 to 2,000 masl)

Source: IDM (2006)

From June 1993 to June 1994 and from 2014 to 2016, weather data were collected for the site. Several stations were monitored but the station most relevant to this study is the Upper Tram Station (Table 5.1). 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 were 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 were compared to the data collected by BC Ministry of Environment, Lands and Parks (MELP) from other stations in the area. Snow pack development at this site was very similar to snow pack development at the Bear Pass site until April when water equivalent peaked at Bear Pass. At Red Mountain, the peak was reached in early May. Snow densities are generally high in coastal British Columbia, reaching 50% by late winter.

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

The 1994 snow course data for the Red Mountain camp is shown in Table 5.3.

Table 5.3: 1994 Red Mountain Snow Course Data

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

Jun 1	142.7	740	52

Source: LAC (1994)

5.3.4 Seismic Activity

The National Building Code of Canada seismic source model (Horner 1994) places Stewart in Zone 2 for peak ground acceleration and Zone 4 for peak ground velocity, on a Risk Zone scale of 1 (low risk) to 6 (high risk). A site-specific seismic hazard assessment was carried out using the Cornell method incorporated in the McGuire program “RISKLL,” and ground motion attenuation relationships. Annual probabilities of exceeding a range of return periods are shown in Table 5.4 with the corresponding peak ground accelerations and velocities. This analysis indicates that the Red Mountain gold project area is in a region of moderate seismic risk. Seismic events occurring in the earthquake prone zone, which runs along the length of the Coast Mountains (Horner 1994), may cause ground motion at the Red Mountain gold project area.

Table 5.4: Probabilistic Seismic Ground Motion Analysis

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

Source: (LAC 2014).

5.3.5 Local Resources

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

5.3.6 Operating Conditions

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

5.3.7 Surface Rights

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

5.4 INFRASTRUCTURE

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

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

5.5 DEMOGRAPHICS

5.5.1 Population

Prior to 1914, the population of Stewart was in the order of 10,000 people. By 2001, the population declined to approximately 660 people, and then to 496 in 2006 (Government of Canada, 2006). The population of the District of Stewart was 401 in 2016 (Government of Canada, 2016).

At the time of the 2016 census by the Government of Canada, 41.1% 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 9.1%.

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 Nisg̱a'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 Nisg̱a'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 mineralization and visible gold at McAdam Point - rock sampling, geological mapping, hand trenching and diamond drilling (one 70 m AX hole). Rock sampling yielded an average of 0.475% MoS_2 over 137 m. One of the trenches yielded values of up to 64.45 g/t Au over 0.61 m.
- **1966-1973** Rehabilitation and extension of the underground workings at the Silver Tunnel vein on Roosevelt #1 claim; production of about 5,000 tonnes of unknown grade. The ore was processed at the Adam custom mill on lower Bitter Creek.
- **1976** Jack Claims (central and southern portions of Red Mountain) staked by J. Howard and optioned to Zenore Resources Ltd.
- **1977-78** Zenore Resources Ltd.: Logging and re-sampling of the 1967 drill core (samples assayed for molybdenum only); geological mapping, petrographic studies, rock geochemistry (assayed for copper, molybdenum, and gold).
- **1978-80** Falconbridge Nickel Mines Ltd: Reconnaissance program for porphyry copper-molybdenum targets in the Stewart area.
- **1987-88** Chuck Kowall, working with a 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 mineralization at the Marc and Brad zones was discovered. Lac Minerals Ltd. acquired Bond in 1991. Surface drilling on the Marc, AV and JW zones continued in 1991, 1992, 1993 and 1994. During 1993 and 1994, and underground decline with three cross-cuts through the Marc Zone for bulk sampling were completed, as well as providing access for underground drilling. (Smit and Sieb 1995). In 1995, Lac Minerals was acquired by Barrick Gold Corp, who subsequently sold the property to Royal Oak Mines in 1996. North American Metals Corp purchased the property from the receivership sale of Royal Oak in 2000. They subsequently sold the property in 2002 to Seabridge, who optioned the property to Banks Island Gold in 2012; minor surface work and 3 surface core holes were completed. Banks terminated the option in 2013 and the property reverted to Seabridge. Seabridge subsequently optioned the property to IDM in 2014. Details of the exploration program carried out by IDM are given in Section 9.7.

Table 6.1 is a recent chronological summary of exploration efforts on Red Mountain from 1988 to 2017:

Table 6.1: Red Mountain 1988-2017 Exploration Summary

1988-89	Staking of Red Mountain by Wotan Resources Inc.
1989	Red Mountain and Wotan properties optioned to Bond. Discovery of gold-silver mineralization by drilling in the Marc zone (3,623 m); airborne EM and magnetic survey.
1990	Exploration of Marc zone and adjacent area (11,615 m of drilling) by Bond.
1991	LAC acquired 100% of Bond. A 2,400 m drill program was completed on the Marc and AV Zones.
1992	Results of a 4,000 m drill program by LAC increased Red Mountain resources and indicated excellent potential for expansion.
1993	28,800 m of surface drilling defined the Marc, AV, and JW Zones and identification of the 141 Zone. An underground exploration adit allowed bulk sampling of the Marc zone. 8,600 m of underground drilling completed in the Marc zone.
1994	LAC completed a 350 m extension of the main decline, 30,000 m of underground drilling and 16,000 meters of surface drilling.
1995	Barrick Gold acquires Lac Minerals. No exploration work completed by Barrick. Royal Oak purchases the project from Barrick.
1996	Royal Oak undertakes exploration to explore for additional reserves. Extended underground drift by 304 m and completed 26,966 m of surface and underground drilling.
2000	North American Metals Corp. purchased the property and project assets from Price Waterhouse Coopers; completed detailed relogging of existing drill core and constructed a geological model for resource estimation purposes and exploration modelling.
2002-2012	Seabridge purchases property, completes two Preliminary Assessment Studies (PEA)
2012-2013	Banks Island Gold options property, three surface drill holes completed. Lidar survey and metallurgical work completed. Published PEA study.
2014	IDM enters into Option Agreement with Seabridge Gold. 12 core holes completed. Surface mapping and sampling completed, with emphasis on Cambria Zone.
2016	Underground workings dewatered and rehabilitated. IDM drilled 11 surface holes and 51 underground holes totalling 8,123.44 metres, completed surface rock and channel sampling, discovered gold mineralization at Lost Valley. Updated PEA was published. Metallurgical, geotechnical and hydrological work completed.
2017	IDM drilled 11 surface and 105 underground holes totalling 29,299.26 metres, completed surface rock and channel sampling, constructed a new geological model and published a Feasibility Study.

	Claim ownership transfers from Seabridge to IDM Mining. Environmental Application and Environmental Impact Statement submitted.

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. Numerous small, underground mines and small prospects were explored over the 20th century. The most significant historic producers were the Premier gold-silver mine (1918 intermittingly to 1996) and the Granduc Copper Mine (1965 to 1984).

In 1997, the Premier mine suspended operations. In 2017, Pretium's Brucejack gold-silver mine commenced commercial production.

6.3 HISTORIC MINERAL RESOURCE ESTIMATES

Several resource estimates were completed in the past for Red Mountain at a 3 g cut-off. Any mineral resource estimates prepared prior to 2001 do not follow the requirements of NI-43-101. Mineral resources stated in Table 6.2 are only stated for historical completeness and should not be relied upon as they are superseded by the mineral resources presented in Section 14 of this report.

Table 6.2: Historical Resource Estimates

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

2008	Seabridge[2]	Inferred	191,020	10.25	15.22	62,900	93,500
2013	Banks[3]	M&I	1,612,000	8.4	28.30	432,000	1,440,000
2013	Banks[3]	Inferred	807,000	5.4	10.20	140,000	260,000
2014	JDS[3]	M&I	1,454,300	8.15	29.57	380,900	1,382,800
2014	JDS[3]	Inferred	332,900	7.69	12.72	82,300	136,200
2016	ACS	M&I	1,641,600	8.36	26.00	441,500	1,379,800
2016	ACS	Inferred	548,100	6.10	9.00	107,500	153,700
2017	ACS	M&I	2,074,700	8.75	24.80	583,700	1,655,700
2017	ACS	Inferred	324,700	6.21	10.1	64,800	105,500

Source: JDS (2017) with modifications. (1) 0 g/t Au cut-off, (2) 6 g/t Au cut-off, (3) 3 g/t Au cut-off. The 2001 NAMC resource was the base for the 2014 JDS resource.

6.4 HISTORIC PRODUCTION

No historical production has taken place on the property.

7.0 GEOLOGICAL SETTING & MINERALIZATION

7.1 INTRODUCTION

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

7.2 REGIONAL GEOLOGY

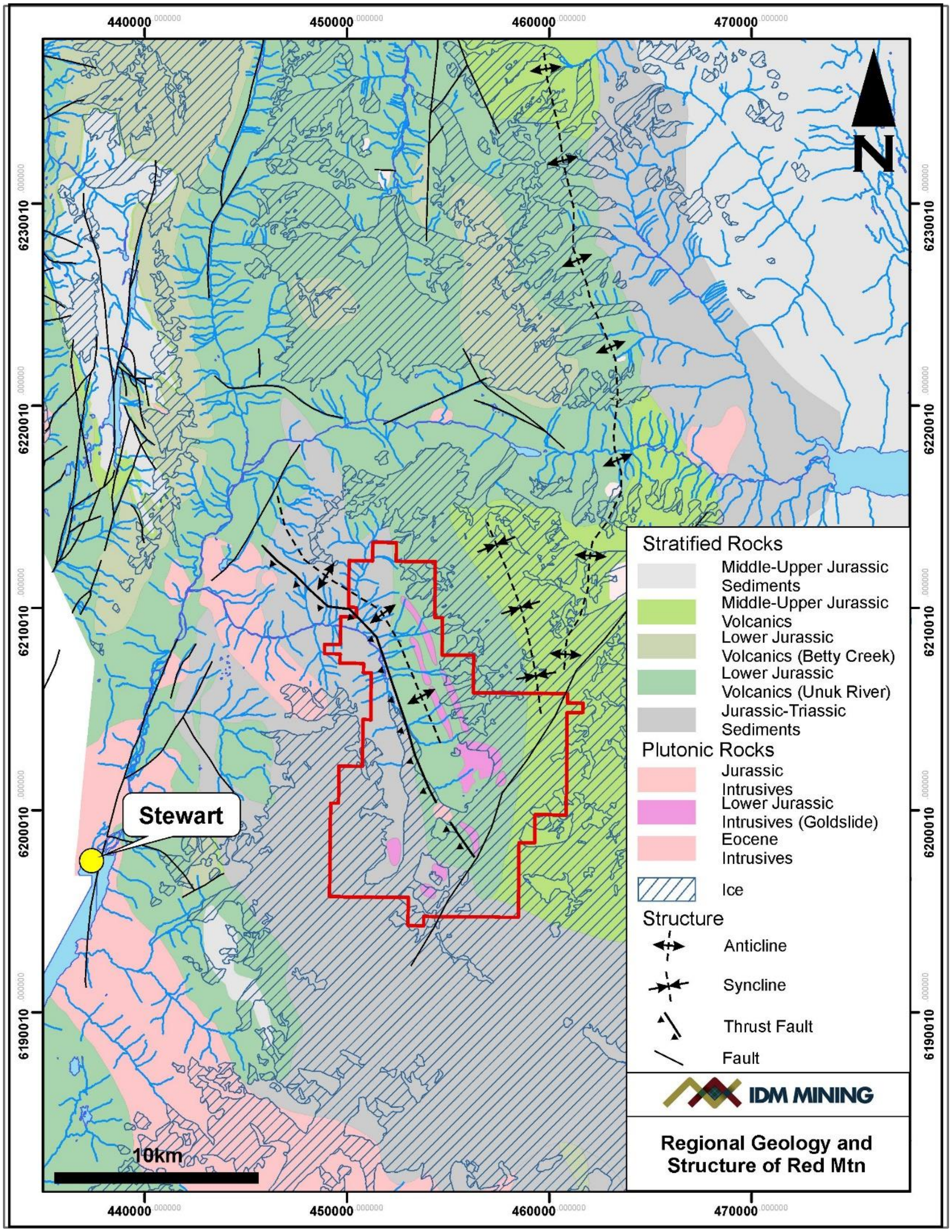
The regional geology of the Red Mountain area has been described by Greig et al (1994), Alldrick (1993) and Rhys et al (1995). The following description is drawn from these sources.

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt (Figure 7.1). There are three primary stratigraphic elements in Stikinia and all are present in the Stewart area: Middle and Upper Triassic clastic rocks of the Stuhini Group, Lower and Middle Jurassic volcanic and clastic rocks of the Hazelton Group, and Upper Jurassic sedimentary rocks of the Bowser Lake Group. Regional metamorphic grade is typically lower greenschist facies, locally to middle-greenschist. On the Red Mountain property, the Lisa Nunatak area exhibits moderate crenulation cleavage, suggesting a higher degree of regional metamorphism.

Intrusive rocks in the Red Mountain region range in age 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 mineralization in the Stewart area, including the Red Mountain resources (referred to as the Goldslide Suite). Intrusive rocks of this age are of variable composition (Rhys et al, 1995). Eocene intrusions of the Coast Plutonic Complex occur to the west and south of Red Mountain and are associated with high-grade silver-lead-zinc occurrences; gold-silver-bismuth+/-copper-lead-zinc mineralization recently identified in the Lost Valley area is likely Eocene age.

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

Figure 7.1: Regional Geology



Source: IDM (2016)

7.3 LOCAL GEOLOGY

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

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

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

145 Ma (Early Cretaceous) – The Alexandria terrain docks and formation of the Skeena fold belt starts. This event folded the rocks of the Stuhini, Hazelton and Bowser Lake groups.

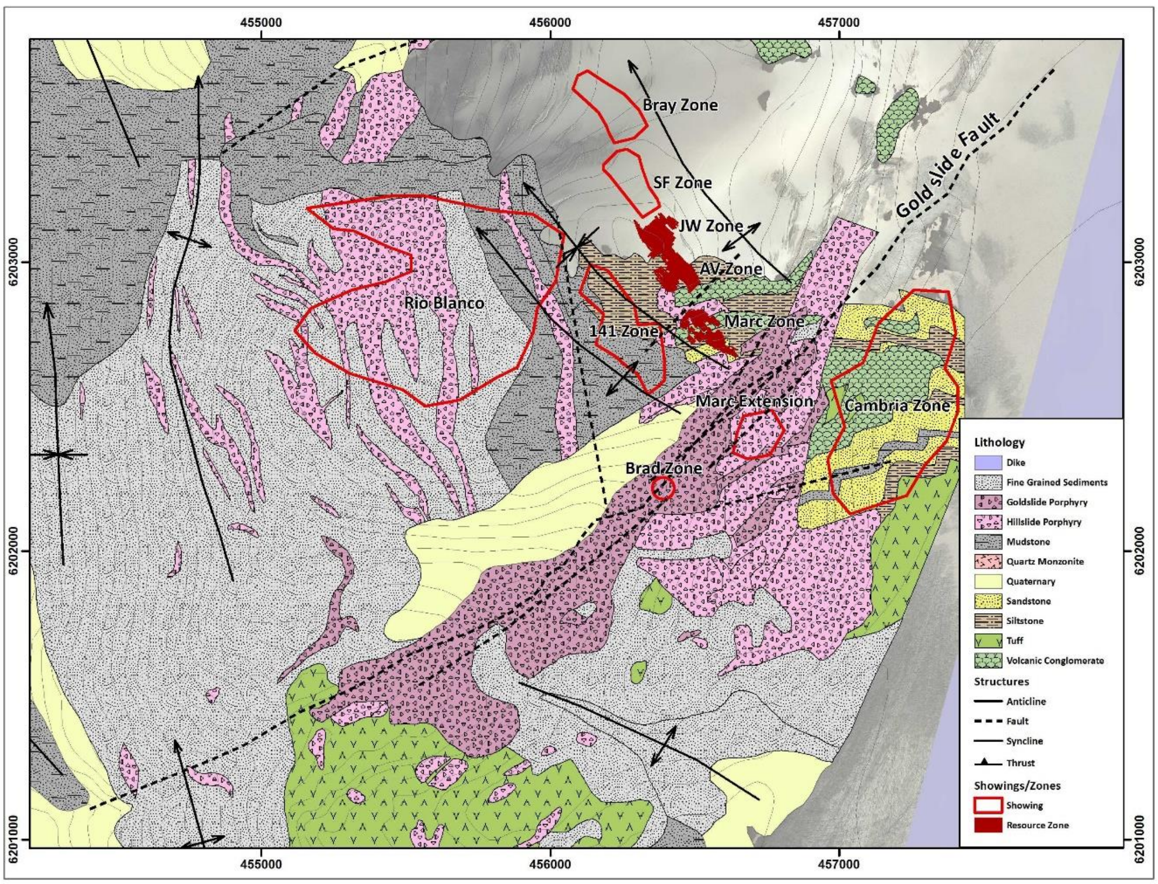
65 Ma (End of Cretaceous) – Deformation of Stikine terrain rocks is complete resulting in folded and doubly plunging structural culminations. The Red Mountain deposits have been rotated from a vertical (?) orientation to a westerly dipping, northerly plunging orientation in the eastern limb of the Bitter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

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

7.4 PROPERTY GEOLOGY

Property geology is shown on Figure 7.2. The oldest rocks, Middle to Upper Triassic mudstone, siltstone and chert of the Stuhini group outcrop over about two thirds of the mapped area. The Triassic rocks grade upward into Lower Jurassic Hazelton Group clastic and volcaniclastic rocks, which outcrop in the northeastern portion of the map area. The unconformity is not definitely identified on the property, however a pyroxene-phyric volcaniclastic breccia exposed west of the summit of Red Mountain has been suggested as the location of this unconformity. Rocks of both groups are folded about axes, which plunge towards 345° and dip steeply to the southwest. The Goldslide suite of intrusions, which have been identified from the Lisa Nunatak area of the south, through Red Mountain to the Hartley Gulch area to the north, are suggested to follow along an earliest Jurassic back-arc basin growth fault, with thick sequences of bimodal, andesite-dominated volcanic rocks to the east of this structure. The approximate contact between Triassic sedimentary rocks and Jurassic rocks runs parallel to the projected trace of the Bitter Creek antiform. This structure has been mapped by Greig et al (1994) to the northwest of the map area. Hazelton Group volcaniclastic rocks on the southwest limb of this structure were likely much thinner and may have been eroded away.

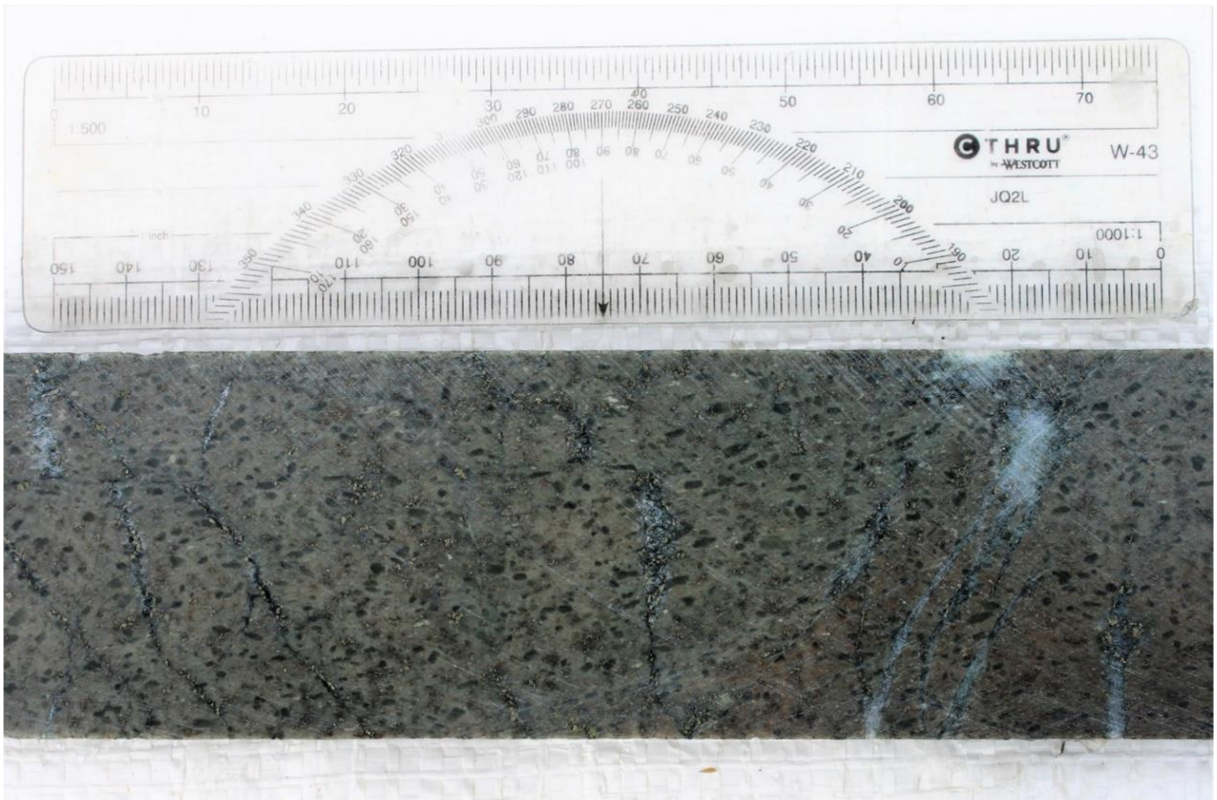
Figure 7.2: Red Mountain Property Geology



Source: IDM (2017)

At least four phases of the Early Jurassic Goldslide suite of intrusions have been identified in the map area. The Hillside porphyry (Figure 7.3), a fine to medium-grained hornblende and plagioclase porphyry, occurs near the summit of Red Mountain and along the ridge to the southeast of the summit. The medium to coarse-grained hornblende +/- quartz phyric Goldslide porphyry (Figure 7.4), is distinguishable from the Hillside porphyry by mineralogy and phenocryst size. It is exposed along the Goldslide Creek valley, extending from the surface expression of the Marc Zone to the southwest for two kilometres. Rhys et al (1995) report a Pb/U date of 197.1 +/-1.9 Ma for a sample of Goldslide. Finally, sills of the Biotite porphyry intrude Upper Triassic sedimentary rocks on the west side of Red Mountain. 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).

Figure 7.3: Hillside Porphyry



Source: IDM (2017)

Figure 7.4: Goldslide Porphyry with chlorite-epidote alteration

Source: IDM (2017)

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

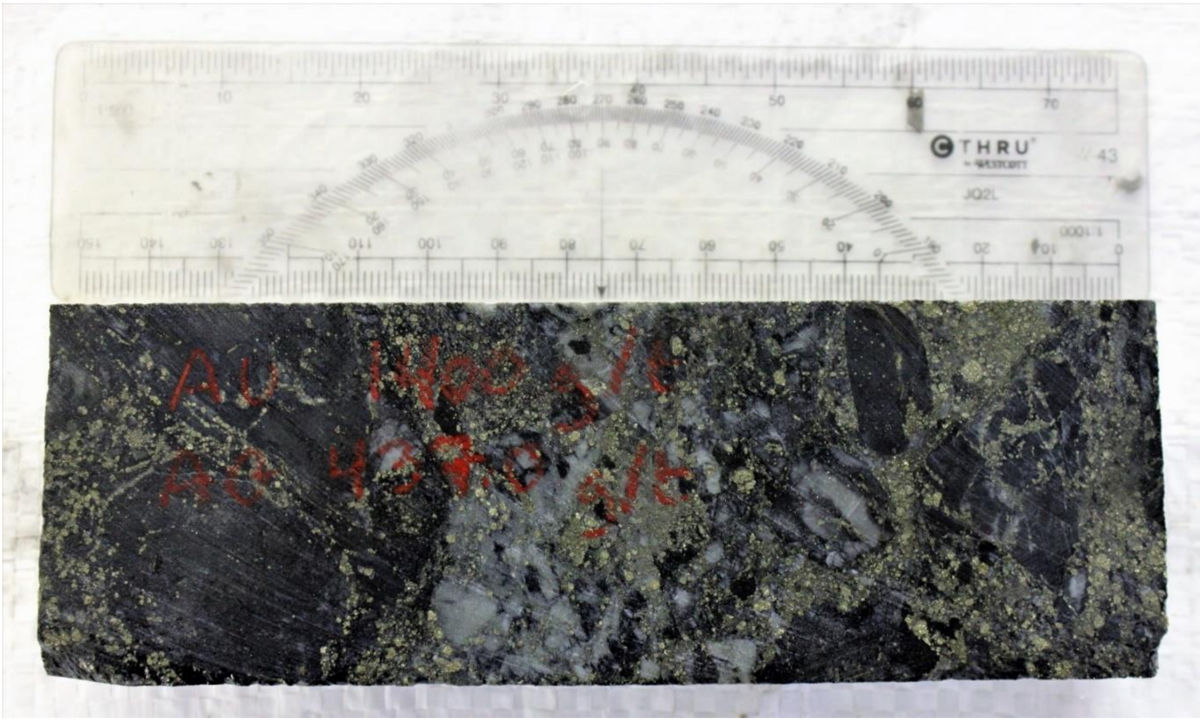
Bedded rocks occurring proximal to the mineralized zones vary in composition, ranging from siliciclastic-dominant siltstones to sandstones, in areas there is minor graphite, to bedded ash tuffs, with local mixed to transition zones (see Figure 7.5). Volcanic units often host lapilli to agglomeritic clasts.

Figure 7.5: Example of bedded tuff with fine muddy laminae

Source: IDM (2017)

Multiple types of intrusive and hydrothermal breccias have been identified throughout the property and strongly disrupted bedding are common along the contacts of these intrusions, particularly in association with the Hillside porphyry. Mineral resources at Red Mountain generally follow early, syn-intrusive breccias of multiple types. Breccia types include contact breccias, crackle and mosaic breccias, and intrusive breccias. These are best exposed in underground workings. The Hillside porphyry contains large rafts of the sedimentary rocks ranging in size from one or two metres to several tens of metres. An example of a strongly mineralized breccia, containing mudstone and Hillside porphyry clasts is shown in Figure 7.6

Figure 7.6: Strongly mineralized mudstone - Hillside porphyry breccia



Source: IDM (2017)

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

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

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

Post-mineralization faulting has affected all rock units at Red Mountain. Rhys et al (1995) 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 mineralized zones, such as the Rick Fault. The latter group are noted by Rhys et al (1995) to have contain more gouge and have broader alteration envelopes than the former.

7.5 MINERALIZED ZONES

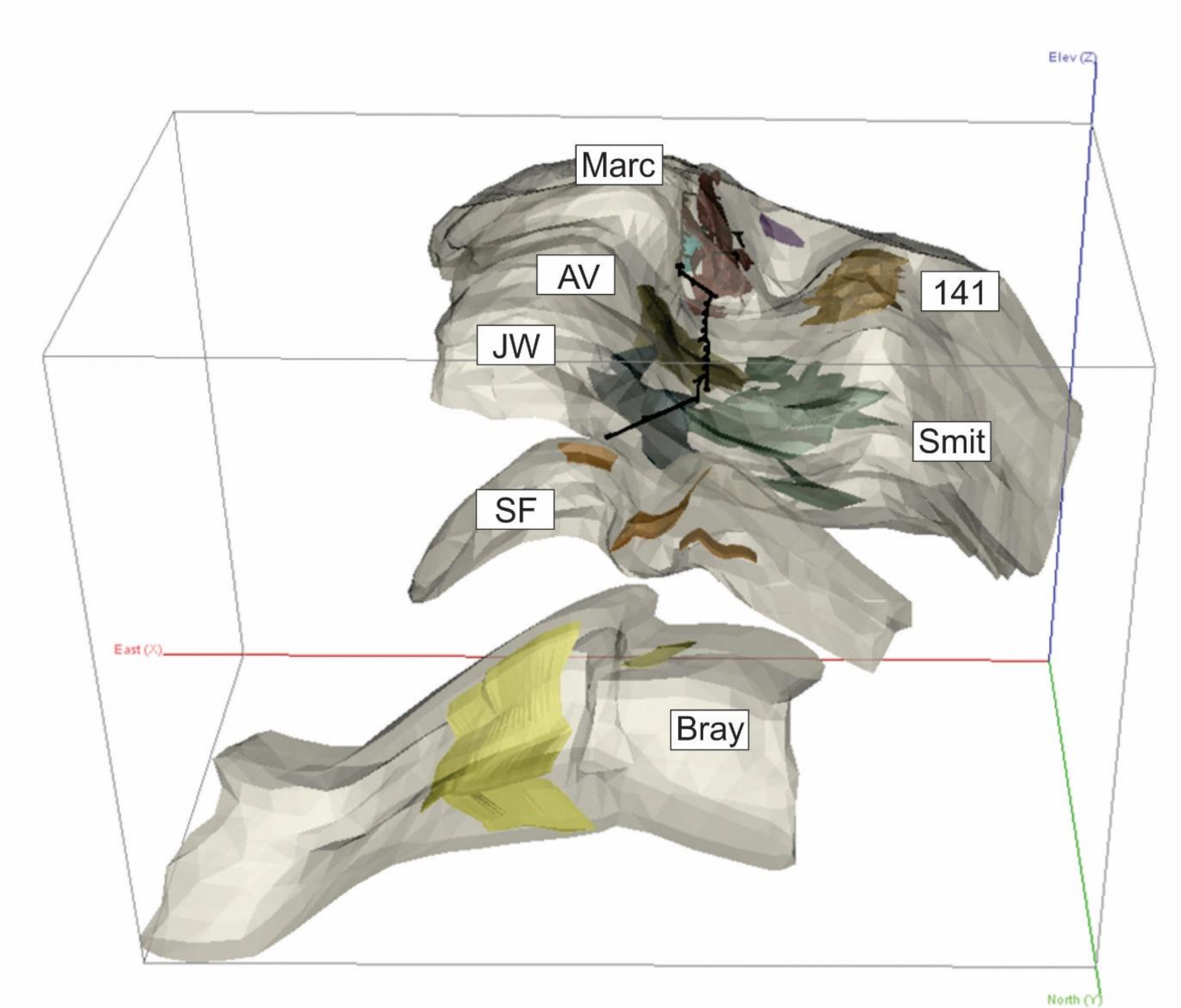
7.5.1 Updated Geologic Model

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

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

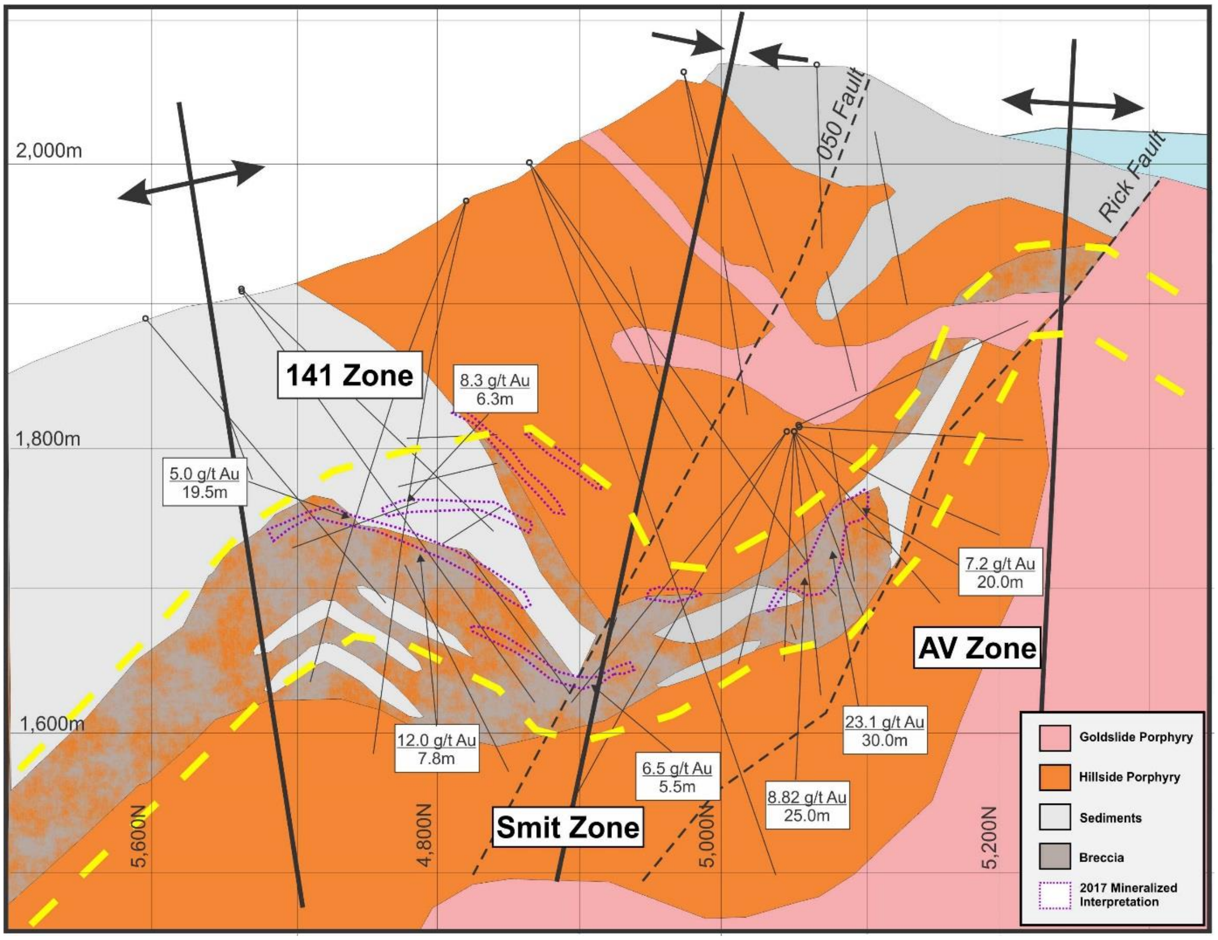
The favourable breccia horizon was clearly folded during mid-Cretaceous transpressional tectonics, resulting in relatively high amplitude (~200m) gently northwesterly plunging folds. An isometric view of the interpreted, folded horizon is shown in Figure 7.7, looking southeast. The folds have been cut by northeasterly trending normal faults, downdropping the horizon to the northwest (note that diagram is in mine grid where north is 315°). Figure 7.8 shows Cross Section 1350N, illustrating the geometry of the favorable breccia horizon and the location of several of the mineralized zones.

Figure 7.7: Current Geological Model of Folded favorable Breccia horizon and Mineralized Zones



Source: IDM (2017)

Figure 7.8: Cross Section 1350N

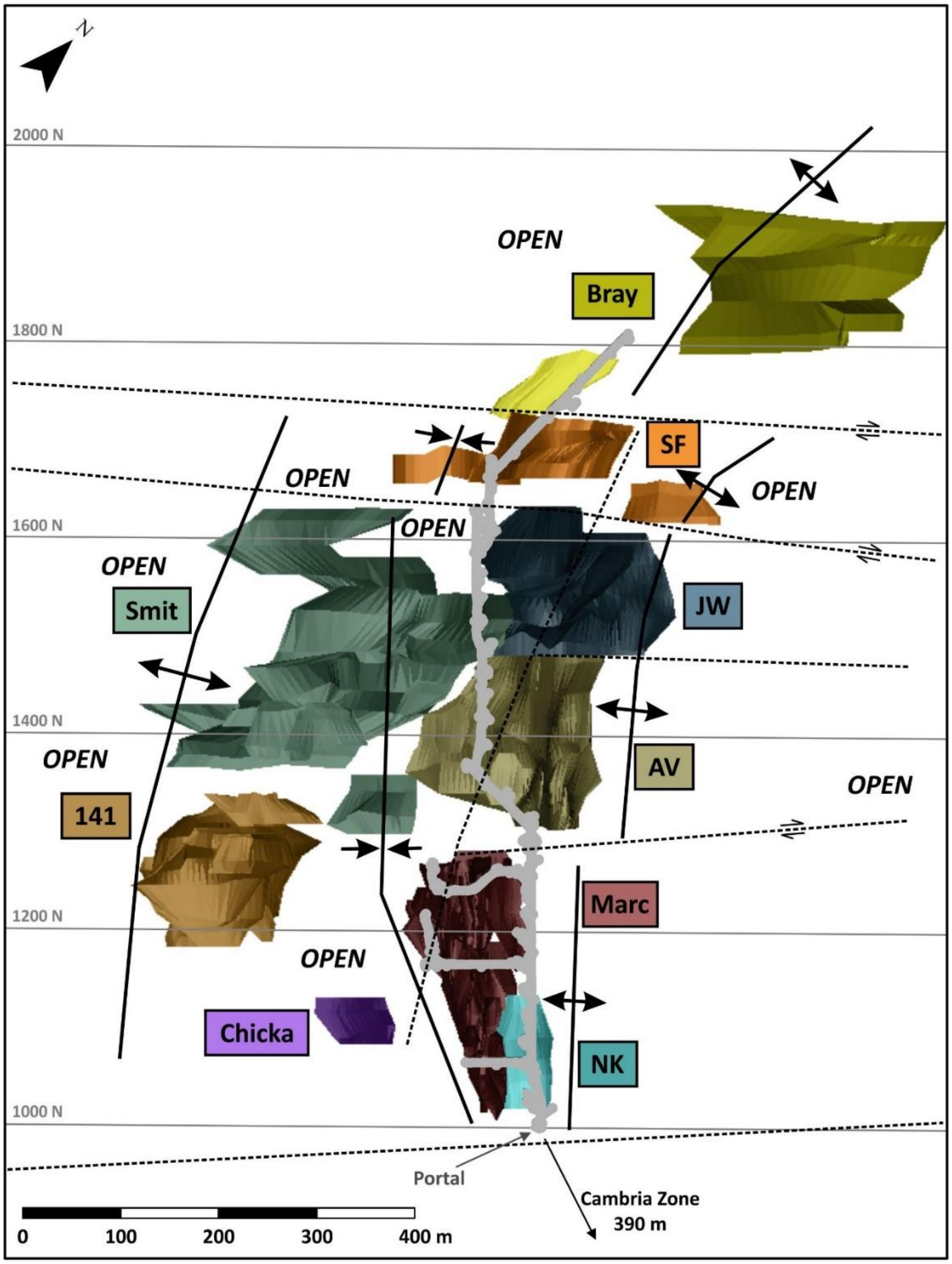


Source: IDM (2017)

7.5.2 Mineralized Zones

The mineralized zones consist of crudely tabular, northwesterly trending and variably dipping gold bearing iron sulphide stockworks. A plan of the zones with names is given in Figure 7.9. Pyrite is the dominant sulphide, however locally pyrrhotite is important. The stockworks zones are developed primarily within the Hillside porphyry and brecciated Hillside, with volumetrically less in the rafts of sedimentary and tuffaceous rocks and sediment-dominant breccias. Although locally anomalous gold values are present within the Goldslide porphyry, significant gold-bearing sulphide stockwork zones have not been located in this rock unit, although it often occurs in close proximity to mineralized zones.

Figure 7.9: Plan View of Mineralized Stockwork Zones



Source: IDM (2017)

The stockwork zones consist of pyrite disseminations, irregular coarse-grained pyrite veins and stockworks with a pale, strongly sericite altered rocks (Figure 7.10). Potassium feldspar, silica, calcite and iron-carbonate alteration are common, particularly in the Hillside porphyry host. Pyrite vein widths vary from 0.1 cm to approximately 200 cm, with 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 zones 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.

Figure 7.10: Typical coarse-grained pyrite veins

Source: IDM (2017)

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

8.0 DEPOSIT TYPE

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

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

1. Approximately 200 Ma, the intrusion of the Hillside porphyry into Stuhini and unconsolidated lower-most Hazelton Group strata. Large rafts of sedimentary rocks are encapsulated the intrusion; contact brecciation between porphyry and sedimentary rocks.
2. The Hillside porphyry cools and contracts. The contraction causes microfracturing of the porphyry and breccia zones. Early pyrite was deposited into these fractures.
3. Ongoing cooling, and alteration of hosts rocks by hydrothermal fluids, with fracturing and brecciation of coarse-grained pyrite veins. Additional coarse-grained pyrite is deposited. The early gold mineralization including petzite is deposited as small inclusions in pyrite grains.
4. Intrusion of the Goldslide porphyry including quartz-phyric phase. The intrusion drives a pulse of hydrothermal fluids primarily containing native gold with local tellurides and sulphosalts into fractures and rims of the in the coarse-grained pyrite veins.
5. Final infilling of remaining fractures in the coarse-grained pyrite veins with gold minerals, fibrous quartz, calcite, feldspar and sericite.
6. Intrusion of biotite-phyric phase of Goldslide Suite.
7. Mid-Jurassic extensional tectonism.
8. Cretaceous transpressional tectonics; recumbent folding of mineralization and favourable breccia horizon.
9. Intrusion of multiple phases of 57.3 Ma McAdam Point stock; intrusive related/porphyry gold-molybdenum quartz stockworks and disseminations.

10. Remobilization of gold and sulphides at Lost Valley during subsequent thrusting.

11. North-south faults with minor offset; pyrrhotite dominant gold-silver-base metal veins.

12. Intrusion of andesite and lamprophyre dykes.

9.0 EXPLORATION

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

9.1 PROPERTY GRIDS

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

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

9.2 GEOLOGICAL MAPPING

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

9.3 GEOCHEMICAL SAMPLING

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

9.4 GEOPHYSICS

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

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

9.5 PETROLOGY, MINERALOGY, AND RESEARCH STUDIES

A significant number of research studies have been completed on the Red Mountain gold project. These include:

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

9.6 IDM EXPLORATION PROGRAMS

After acquiring an option on Red Mountain in 2014 IDM commenced exploration on the property, including soil sampling (546 samples), rock sampling (440 samples), channel sampling (241 samples) and 12 diamond drill holes totalling 2223.0 m (McLeod, 2014). Additionally, historic core was re-logged, and 68 infill samples taken in areas of strong alteration and mineralization.

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

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

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

9.7 EXPLORATION POTENTIAL

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

9.8 COMMENT ON SECTION 9

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

10.0 DRILLING

10.1 INTRODUCTION

A total of 659 surface and underground diamond drill holes (170,407.43 m) have tested a variety of targets on the Red Mountain property. Of these, 406 holes totalling 100,298 m were drilled by Bond and LAC between 1989 and 1994, and 60 holes totalling 29,671 m were drilled by Royal Oak in 1996. No drilling was carried out by NAMC. During 2012, Banks Island completed 3 drill holes for 681 m in the Marc zone.

The majority of the historical drilling tested the Marc, AV and JW zones. A total of 368 drill holes from the Bond and LAC programs, including 207 surface drill holes and 161 underground drill holes, tested these areas.

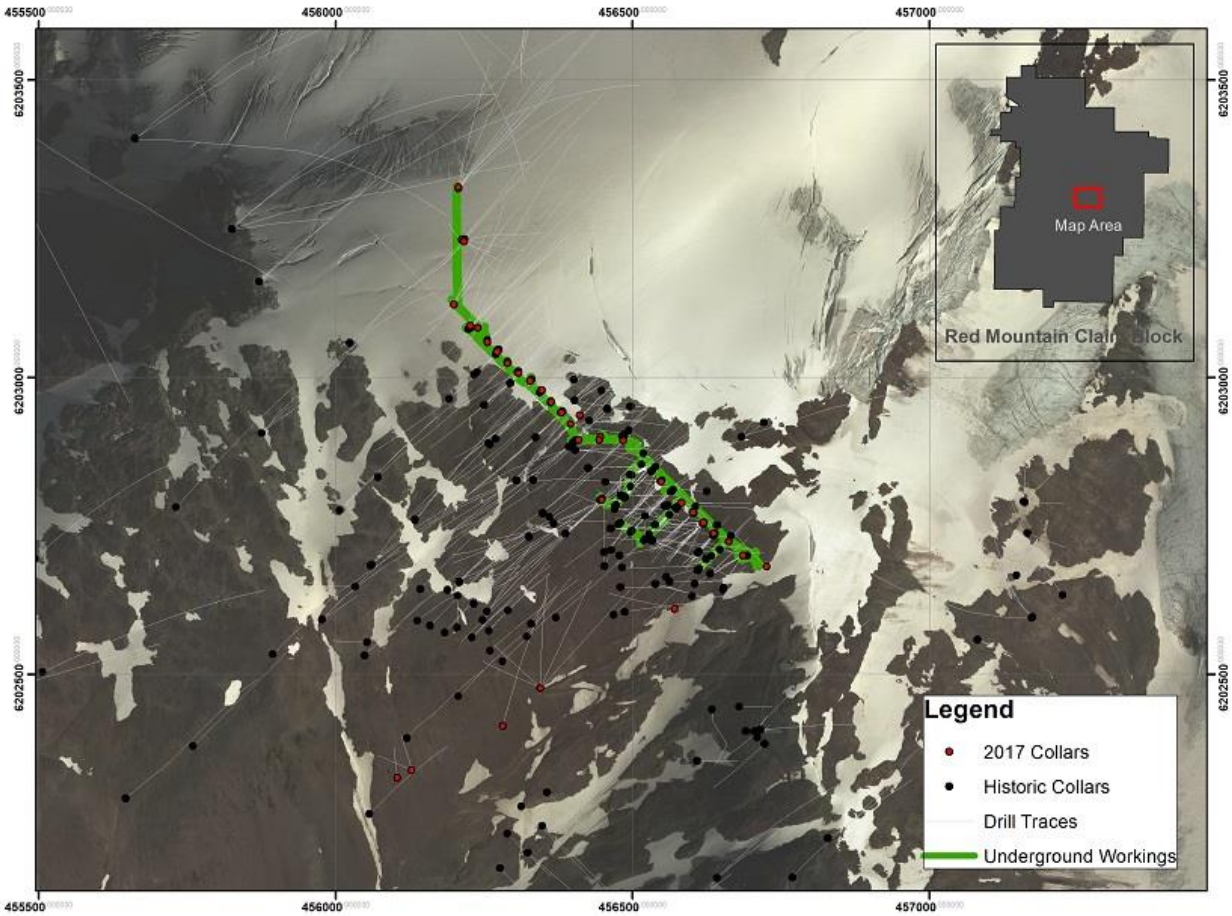
The location of a majority of drill holes on the property are shown on Figure 10.1, which is centred on the resource areas and main prospects.

In 2014 IDM Mining completed 12 surface drill holes totalling 2223 m, including 2 in the AV zone, 3 in the 141 Zone, 2 in the Marc Extension zone and 5 on exploration targets in the Cambria zone.

In 2016 IDM completed 62 holes totalling 8123 metres, including 51 underground drill holes in the Marc, AV and JW Zones, and 11 surface drill holes including 5 in Lost Valley, 1 in the Brad zone and 5 in the 141 Zone.

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

Figure 10.1: Red Mountian Drill Plan Resource Areas & Main Prospects



Source: IDM (2017)

10.2 SURFACE DRILLING CONTRACTORS

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

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

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

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

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

10.3 UNDERGROUND DRILLING CONTRACTORS

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.

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

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

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

10.4 FIELD PROCEDURES

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

10.5 CORE LOGGING

10.5.1 Bond and LAC Logging

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

- Geotech log: Recovery, RQD fracture count, hardness and fracture filling. Carried out by a geological technician.

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

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

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

10.5.2 **Royal Oak Logging**

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

10.5.3 **NAMC Logging**

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

10.5.4 **Banks Island Logging**

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

10.5.5 **IDM Logging**

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

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

10.6 RECOVERY & RQD

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

10.7 DRILL COLLAR AND DOWN HOLE SURVEYS

10.7.1 Drill Collar Surveys

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

For most Bond and LAC surface drill holes from 1990 to 1993, the collar orientations appear to be ideal set up orientations as shown in Table 10.1. For 1994 surface drill holes the first down hole survey orientation was used for collar orientation.

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

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

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

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

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

10.7.2 Down Hole Surveys

With the exception of the 1989 drill holes and a few of the 1990 drill holes, which had acid dip tests, most holes drilled on the property until 1996 have Sperry Sun surveys, the predominant down hole survey technique at the time. Banks Island used a Reflex Easy Shot instrument and collected the surveys after the hole was completed. For their 2014 program IDM used a Ranger multi-shot survey instrument, but no surveys were obtained for six of the twelve holes. The IDM 2016 drilling program used a combination of a Reflex multi-shot surveys and Reflex Gyro surveys. Only Reflex multi-shot surveys were used in 2017. Details of the down hole surveys and collar surveys for all programs given in Table 10.1.

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

Table 10.1: Details of collar and downhole surveys

Year	Company	Surface or UG	Collar Location	Collar Orientation	Survey Type	Comments
1989	Bond	S	Y	Y	Acid	Acid dip tests only
1990	Bond	S	Y	N	Sperry	~90 m spacing, ideal collar coords
1991	Lac	S	Y	N	Sperry	~90 m spacing, ideal collar coords
1992	Lac	S	Y	N	Sperry	~90 m spacing, ideal collar coords
1993	Lac	S	Y	N	Sperry	~60 m spacing, ideal collar coords
1993	Lac	UG	Y	Y for most	Sperry	Some holes <80 m in length have no surveys. Holes >100 metres have surveys every 60 m or at the bottom of the hole.
1994	Lac	S	Y	N	Sperry	First at ~15 m then every 60 m, data from first test used for collar.
1994	Lac	UG	Y	Y for most	Sperry	First at ~15 m depth then every 30 m
1996	Royal Oak	S	Y	Y for ~25% of holes	Sperry	Variable spacing, 50 to 100 m or more
1996	Royal Oak	UG	Y	N	Sperry	Variable spacing, 50 to 100 m or more
2013	Banks Island	S	?	N	Reflex	Every 31 metres

2014	IDM	S	Y	Y & N	Ranger MS	Readings taken every 6 m. If surveyed there are collar coords otherwise ideal cords were entered.
2016	IDM	S	N	N	Reflex	Reflex every 6 m
2016	IDM	UG	Y	Y & N	Reflex, Gyro	Reflex or Gyro every 3 m
2017	IDM	S	Y	Y & N	Reflex	Every 3 m
2017	IDM	UG	Y	Y & N	Reflex	Every 3 m

10.8 DRILL HOLE ADJUSTMENTS

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

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

10.9 SAMPLE LENGTH/TRUE THICKNESS

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

10.10 DRILL SPACING

Drill spacing on the Red Mountain gold project is variable depending on the stage of exploration or development of a particular zone.

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

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

10.11 DRILL INTERCEPTS

Table 10.2 shows a selection of intersections through the main resource zones to illustrate typical grades and widths the deposit.

Table 10.2: Typical Drill Intersections

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

10.12 COMMENTS ON SECTION 10

In the opinion of the responsible QP, the quantity and quality of the geological, geotechnical, collar and down-hole survey data collected by the past and present operators on the Red Mountain gold project are sufficient to support mineral resource estimation as follows:

- Drilling procedures and core logging meets industry standards:
- Recovery data from drill core data are acceptable;
- Collar surveys have been performed using industry-standard instrumentation;
- Down-hole surveys were collected at the time of the programs using industry-standard instrumentation.
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the resource areas;
- Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths;
- Drill spacing has been adequate to first outline, then infill and define mineralized zones. Drill hole spacing does vary with the stage of exploration and development;
- Drill hole intercepts as summarized in Table 10-2 appropriately reflect the nature of the gold mineralization, and include areas of higher-grade intervals in low-grade drill intercepts;
- No factors were identified with the data collection from the drill programs that could materially affect resource estimation accuracy or reliability.

11.0 SAMPLE PREPARATION, ANALYSES & SECURITY

11.1 SAMPLING METHODS

11.1.1 Soil Sampling

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

11.1.2 Rock and Channel Sampling

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

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

11.1.3 Drill Sampling

Bond and LAC 1989 -1992

Drill core samples from 1989 to 1992 were collected over 1.50 m intervals regardless of geology. After geological (and some geotechnical) logging of the core was completed, BQTK-sized core was manually split in half. One-half was submitted to for sample preparation and analysis and the other half was kept for future reference at the core storage facility in Stewart, British Columbia.

LAC 1993 - 1994

Drill core samples from the 1993 and 1994 programs were typically collected over 1.0 m intervals and occasionally over 1.50 m intervals. In some cases, effort was made to break sample intervals at lithological or mineralogical boundaries, resulting in sample intervals shorter than 1.0 m. After detailed geotechnical and geological logging was completed, the core was sawn in half. As in previous programs, half of the core was submitted to the lab for sample preparation and analysis. The second half of the core was stored at the core storage facility in Stewart, British Columbia.

During these large programs, up to four diamond blade rock saws were running to cut core. A foreman was hired to oversee core sawing, sample tags and standard insertion to ensure that this process worked efficiently and to ensure good quality control. A sample sheet, with sample numbers and from-to distances filled in by the logging geologist, was used to assure as best as possible that sample numbers corresponded with the right intervals when samples were collected.

Royal Oak 1996

Royal Oak typically collected samples over 1.0 m (underground and surface) and 1.5 m (surface intervals and these lengths comprise over 75% of their samples. Minimum and maximum sample lengths are 0.3 m to 6.0 m respectively. Sampling was carried out at the camp in Goldslide Creek where sample intervals were sawn. Multi-part sample tag portions were inserted into the core boxes between each sample interval, with the other part was placed in the sample bag.

Banks Island 2013

Banks Island sampled over 0.25 to 1.5 metre intervals that honored geological boundaries. It is known that the core was sawn, however no other sampling procedures, or the location where sampling was carried out, were documented.

IDM 2014-2016

Samples from the 2014 IDM drilling program were collected over 1.0 m intervals for a majority of sampling and never less than 0.5 m in length and seldom crossed lithological boundaries. Sampling took place at the camp in Goldslide Creek. The core was sawn and the upper half was placed in a sample bag and sent for assay. Sample tags were placed in the bag and under the second half of the core in the boxes. The core is stored on pallets at the camp on Red Mountain.

Sampling protocols were the same in 2016 with the exception that in longer sections of suspected barren to low grade low rock, particularly in some of the surface drill holes, 1.5 metre samples were taken. Additionally, for 20 HQ diameter underground holes drilled for metallurgical samples, a full half was sent for the test work, ¼ was sent for regular assay and ¼ was retained for future reference.

11.1.4 **Whole Rock Samples**

During the Bond and LAC drilling programs, samples were collected from drill core for whole rock analysis. Samples were collected every 20 to 30 m or with major lithological changes. Proximal to or within the mineralized zones samples were taken every 10 metres. Samples were half core and a minimum of 0.5 metres long. For samples already selected for conventional assay a portion of samples pulp was submitted for whole rock analysis.

11.1.5 **1993-1994 LAC Underground Chip Samples**

During the 1993 and 1994 programs the ramp and crosscut faces were sampled after every round. Chip samples were collected from fresh faces using a grid with 1.5 x 1.5 m panels, with each face being three panels wide by two panels high. Chips were collected evenly from within the panels.

11.1.6 **1993-1994 LAC Bulk Samples**

A muck sample was collected from every underground round, either from the main decline or from the cross cuts designed to assess the Marc Zone mineralization. From crosscut rounds within potential ore, and for several rounds on either side, the muck was stockpiled on surface. A grid was overlain on the stockpile and 20 samples were taken from each round. If the average grade of the resulting assays was less than 2.0 g/t Au the muck was put onto the waste pile. If the average grade was over 2.0 g/t Au, the stockpiled muck was taken through the bulk sampling process. Twenty-three rounds from the underground were treated in this manner.

11.2 ANALYTICAL LABORATORIES

Several primary laboratories have been used for Red Mountain samples over the history of the project as shown in Table 11.1. For a majority of drill hole samples, Eco-Tech Labs was the primary laboratory.

Table 11.1: Laboratory Summary Table

Operator	Laboratory	Time Period	Sample Type Analysed
Bond Gold/Lac	Min-En Labs, North Vancouver	1989-1991	Surface drill hole samples
Bond Gold/Lac	Bondar-Clegg, North Vancouver	1989-1992	Check assays on drill pulps
Lac	Acme Labs, North Vancouver	1989 -1991	Whole rock samples
Lac	Acme Labs, North Vancouver	1992	Surface drill hole samples
Lac	Eco Tech Labs, Stewart, B.C.	1993-1994	Surface and underground drill hole samples
Lac	Chemex, North Vancouver, B.C.	1993	Overflow drill samples
Lac	X-RAL, Don Mills, Ontario	1993	Whole rock samples
Lac	Chemex, North Vancouver, B.C.	1994	Whole rock samples
Lac	Chemex, North Vancouver, B.C.	1993-1994	Check assays on drill rejects and pulps
Royal Oak	Eco Tech Labs, Kamloops, B.C.	1996	Surface and underground drill hole samples
Royal Oak	Bondar-Clegg, North Vancouver	1996	Check Assays on drill pulps
NAMC	Chemex, North Vancouver, B.C.	2000	Check assays on drill rejects and pulps
Banks Island	AGAT, Mississauga, ONT	2013	Surface drill hole samples
IDM	Acme (BV), Vancouver, B.C.	2014	Surface drill hole samples
IDM	ALS Global, North Vancouver, B.C.	2016-2017	Surface & UG drill samples, rock samples
IDM	ActLabs, Kamloops, B.C.	2016	Check assay on drill pulps
IDM	MS Analytical, Langley, B.C.	2017	Check assay on drill pulps

The ISO accreditations of all labs from 2000 and prior is not known. AGAT Labs, Acme (Bureau Veritas), ALS Global, ActLabs and MS Analytical are all ISO 9001:2008 accredited laboratories. All laboratories are also ISO/IEC 17025:2005 accredited for some specific tests including fire assays with AA and gravimetric finishes.

11.3 SAMPLE PREPARATION & ANALYSIS

11.3.1 Sample Preparation

Sample preparation for drill samples of drying as required, crushing, and selection of a sub-split which is then pulverized to produce a pulp sample sufficient for analytical purposes. Table 11.2

summarizes the sample preparation procedures used by the primary and, where applicable, by the check assay laboratories. Note that crushing and grinding practices for Acme (Bureau Veritas) have changed between work carried out in 1992 and 2014.

Table 11.2: Sample Preparation Procedures

Laboratory	Procedure
Min-En	Dry, 2 stage crushing to -1/8", 500 g split pulverised to 95% passing -120 mesh.
Bondar-Clegg	Dry, crush and pulverize to -150 mesh (onn rejects only for checks).
Eco-Tech	Dry, crush to -10 mesh, 250-400 g split pulverized to 85% passing -140 mesh.
Acme Labs	Dry, crush to -10 mesh, 250 g split pulverized to 85% passing -150.
Chemex	Dry, crush to -10 mesh, 200-300 g split pulverized to 90% passing -150 mesh.
AGAT	Dry, crush to 75% passing -10 mesh, 250g split pulverized to 85% passing -200 mesh.
Acme (BV)	Dry, crush to 70% passing -10 mesh, 250 gram split pulverized to 85% passing -200 mesh
ALS Global	Dry, crush to 70% passing -10 mesh, 1000 gram split pulverized to 85% passing -200 mesh

For the 1993, 1994 and 1996 programs all sample preparation by Eco-Tech was carried out at their facility in Stewart, B.C. For the 2013 Banks Island program samples were prepared at the AGAT facility in Terrace, BC. For the 2014 IDM program samples were prepared at the Acme facility in Smithers, B.C. before being forwarded to Vancouver, B.C., for analysis. The 2016 samples were prepared at ALS Global in Terrace B.C.

11.3.2 Sample Analysis

The analytical methods used on drill core and check assays from Red Mountain are summarized in Table 11.3.

Table 11.3: Analytical Methods

Laboratory	Procedure
Min-En	Fire assay for gold a 30 g sample with an AA finish. Results over 0.5 oz/T Au (~17 g/t) re-assayed with a gravimetric finish. Multi element ICP package.
Bondar-Clegg	Fire assay for gold and silver on a 30 g sample with an AA finish. Results over 0.2 oz/T Au (~7 g/t) re-assayed with a gravimetric finish.
Acme	Fire Assay for gold on a 30 g sample, Mutli element ICP on a 0.5 g sample. Whole rock by lithium borate fusion with an ICP finish.
Eco-Tech	Fire assay for gold on a 30 g sample with an AA finish. Results >10 g/t Au re-assayed with a gravimetric finish and if >30 g/t Au a metallic assay was performed. Ag assayed using an aqua regia digestion and an AA finish on a 2 g sample. 31 element ICP package.
XRAL	Whole rock analyses by XRF.
Chemex	Fire assay for gold on a 30 g sample with an AA finish. Results >10 g/t Au re-assayed with a gravimetric finish and if >30 g/t Au a metallic assay was performed. Ag assayed using an aqua regia digestion and an AA finish. Also multi element ICP on 1993 over flow samples. Whole rock analyses by XRF.
AGAT	Fire assay for gold on a 30g sample with an ICP-OES finish, results >10 g/t re-assayed using a gravimetric finish. 45 element ICP-OES package with aqua regia digestion.

Acme (BV)	Fire assay for gold on a 30 g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. 36 element ICP-ES on a 0.25 g sample.
ALS Global	Fire assay for gold on a 30g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. Ag by Acid digestions with AA finish, repeated if >100 g/t Ag, 48 element 4 acid, ICP-MS package.
MS Analytical	Fire assay for gold on a 30g sample with AA finish. Results >10 g/t re-assayed using a gravimetric finish. Ag over-limits from ICP by fire assay with gravimetric finish, multi-element 4 acid ICP package

For the 1993, 1994 and 1996 programs most gold and silver analyses were performed at Eco-Tech's Stewart facility, while the ICP analyses were carried out at Eco-Tech's Kamloops facility. The exception for this is for late 1994, starting in November when the Eco-Tech's Stewart analytical facility closed and both fire assay and ICP work was done at the Kamloops facility. For the 1996 Royal Oak samples, all analytical work was carried out at Kamloops.

11.4 QUALITY ASSURANCE AND QUALITY CONTROL

The Quality Assurance and Quality Control (QAQC) for the Red Mountain drilling programs has previously been presented by Anderson (2000) and reported in Craig (2001) and Smee (1993). All historic QAQC data was recompiled and assessed in early 2016.

11.4.1 Bond and LAC QAQC 1989-1992

There is little, if any information regarding the insertion of QAQC materials (standards, blanks, duplicates) into the sample stream by Bond or LAC prior to 1993.

A significant amount of check assaying was carried out on samples from the 1989 to 1992 drill holes with 1,243 (1121 pulps and 122 rejects) of 13,256 samples (9.48%) submitted to Bondar Clegg.

The compiled data show small to modest high biases for the Bondar Clegg check assay analyses. For gold, Bondar Clegg results were 2.8% and 4.73% higher than the original Min-En results for pulps and rejects respectively. For silver, Bondar Clegg results were 1.02% and 2.3% higher than Min-En for pulps and rejects respectively. Four samples, two pulp and two reject, were removed from the analysis due to outlier results in the Bondar Clegg dataset.

The results indicate good assay accuracy between the two labs. The higher bias in the rejects results may be due to the preparation of a second pulp from a second split.

11.4.2 Lac QAQC 1993-1994

Standards

LAC initiated the use of standards in 1993 but the number was very limited at only 53 in total. The standards used were Canmet standards as shown in Table 11.4. Note that in 2000, +/-2 standard deviations were used as failure limits for all standards. Current industry standards are to use +/-2 standard deviations as a warning limit and +/-3 standard deviations as failure limits, and this has been followed here.

Table 11.4: Red Mountain Canmet Standards

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

When drilling recommenced in April 1994, a more stringent standard insertion program was instituted with an insertion approximately every 20 samples. While some of the 1993 Canmet standards were used, for this program four site specific standards were created by CDN Resource Laboratories of Delta, BC, using material from the Marc Zone bulk samples (Sanderson, 1994). 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. Standard values and +/-3SD failure limits, based on the round-robin results and analysis, are presented in Table 11.5.

Table 11.5: Red Mountain LAC Site Specific Standards

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

The results of the standard insertions from the 1993 and 194 Lac drilling programs are summarized in Table 11.6.

Table 11.6: Summary of Standard Insertions

Standard	Number of Analyses	Mean of Analyses	Expected Value	Percentage difference	No. High Fails	No. Low Fails
MA-1b	22	17.1	17.00	+0.6	5	3
MA-2b	39	2.11	2.39	-11.7	1	29
MA-3	37	7.26	7.49	-3.1	2	13
LAC 1	235	1.91	1.90	+0.5	6	1
LAC 2	242	3.18	3.19	-0.3	3	2
LAC 3	281	6.57	6.35	+3.5	2	2
LAC 4	124	14.50	14.15	+2.4	0	0

In general, the Canmet standards did not perform well relative to their +/-3 standard deviation failure limits. Many failures may be attributable to quite tight failure limits relative to standards of similar grades from other commercial suppliers, as the ranges for a majority of results for each standard appear to indicate reasonable accuracy. The majority of failure were low relative to the expected values, suggesting that assay data may underestimate gold values.

The LAC standards performed well indicating good assay accuracy. Standards LAC 1 and LAC 2 show no biases relative to the expected values. Standards LAC 3 and LAC 4 do show small positive biases but most values still fall within the +/-3 standard deviation failure limits. Examples timeline plots are shown in Figure 11.1 (LAC 2) and Figure 11.2 (LAC 3). Note that a tightening of results relative to the expected values is evident in both plots at approximately samples 185 and 210 respectively, corresponding to the moving of all analytical work from the Stewart Eco-Tech facility to the Kamloops facility.

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

Figure 11.1: Timeline Plot for Standard LAC 2

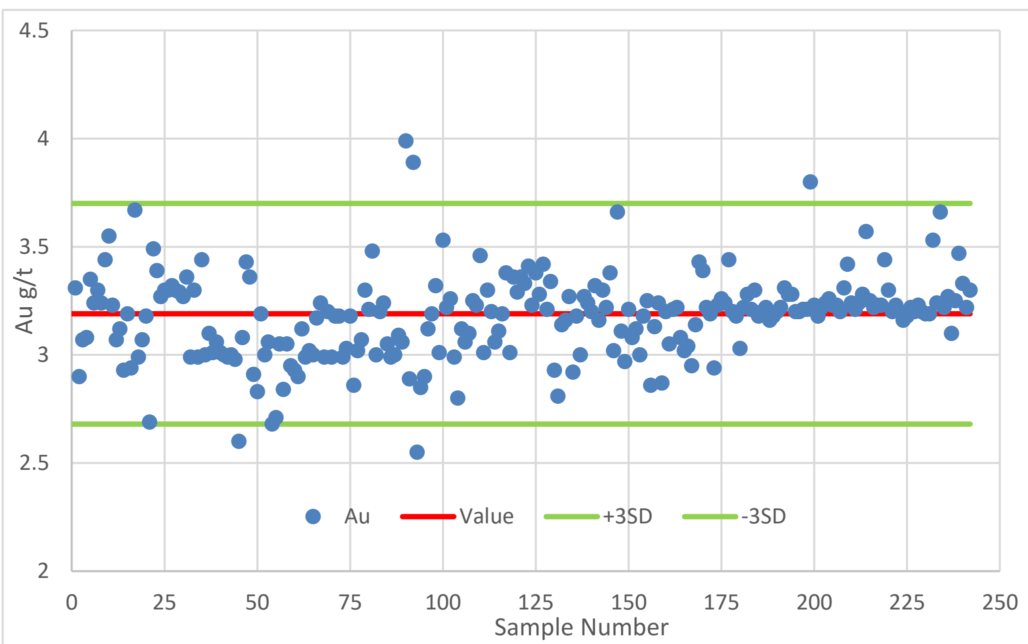
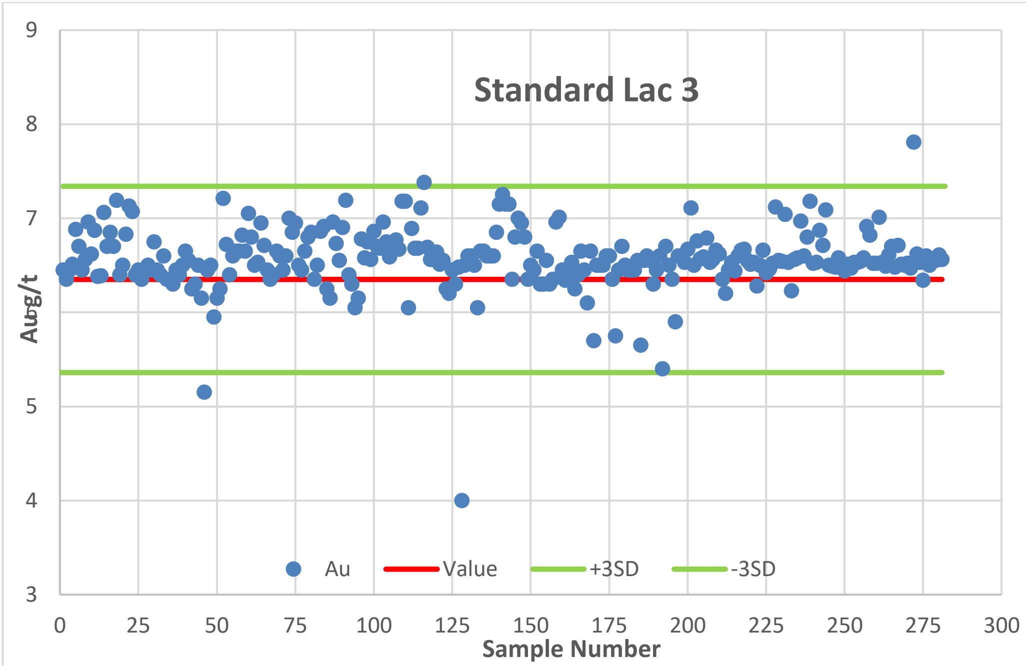


Figure 11.2: Timeline Plot for Standard LAC 3



Check Assays

A rigorous check assay program was implemented by LAC in 1993 with a protocol whereby 1 in every 10 pulps and 1 in every 20 duplicates half were to be submitted to Chemex for check assay. This protocol was not used in 1994 and instead two cross sections, both in the AV zone, were chosen for check assays. From Section 1400N rejects were sent to Chemex and from Section 1500N pulps were sent to Chemex.

In total, check assays were submitted from 168 of 301 surface and underground LAC drill holes totalling 3060 checks from 31,064 original samples (9.9%). The samples actually submitted did not end up in the proportions suggested by the protocols with 371 pulp submittals and 2689 reject submittals. Nine hundred and twenty-five check assays in the historic Access database were not included in the compilation as the material, pulp or reject, could not be determined. Results are summarized in Table 11.7.

Table 11.7: Summary of 1993-1994 Check Assays

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

The pulp check assays results show a modest to strong high bias by Chemex for gold and a very small high bias for silver. The high bias for gold occurs in samples with values of over 3.0 g/t. There is a consistent low bias by Chemex at all grade levels compared to Eco-Tech with the reject checks. This bias has not been resolved, although it is possible that fine gold could have settled during transport of the rejects resulting in lower values. The influence of a different level of sample support (original pulp versus new pulp from a second split of rejects) is also not known.

No standards or blanks were included with check assay shipments to Chemex.

Duplicates

Anderson (2000) reported a LAC 1993-94 duplicate database consisting of 369 samples. From twenty-one 1994 underground drill holes, a high and low-grade sample was collected within the mineralized zones for each hole. The first half of core was assigned a sample number and the resulting pulps were analyzed twice. The second half was assigned a new sample number and also analyzed twice. If needed gravimetric and metallic assays were carried out. Additionally, four holes (U94-1155, (94-1156, U94-1157 and U94-1158) were drilled in the Marc Zone, on Section 1275N, in a 1.0 metre box spacing to test variance. The first three of these and hole U94-1160 were sampled from top to bottom and original and duplicate halves were analyzed (no extra pulp splits).

The comparison of results from the first pulp from both original and duplicate halves of the core (n=369) for the global dataset show extremely good assay precision with the originals having a mean of 8.02 g/t Au and the duplicates having a mean of 8.05 g/t Au. On an individual assay basis, there is some modest variability, probably reflecting differing proportions of the sulphide veins in opposing halves of core.

Duplicate holes

During 1994, four short drill holes were drilled on section 1275 N from collar points 1 m apart in a square pattern. As well as to serve as individual assay duplicates the purpose was to evaluate the variance within the stockwork zone over full intersection distances. Table 11.8 summarises the weighted assay averages for the higher-grade intervals in the four drill holes from 13 to 29 m.

Table 11.8: Weighted Assay Averages

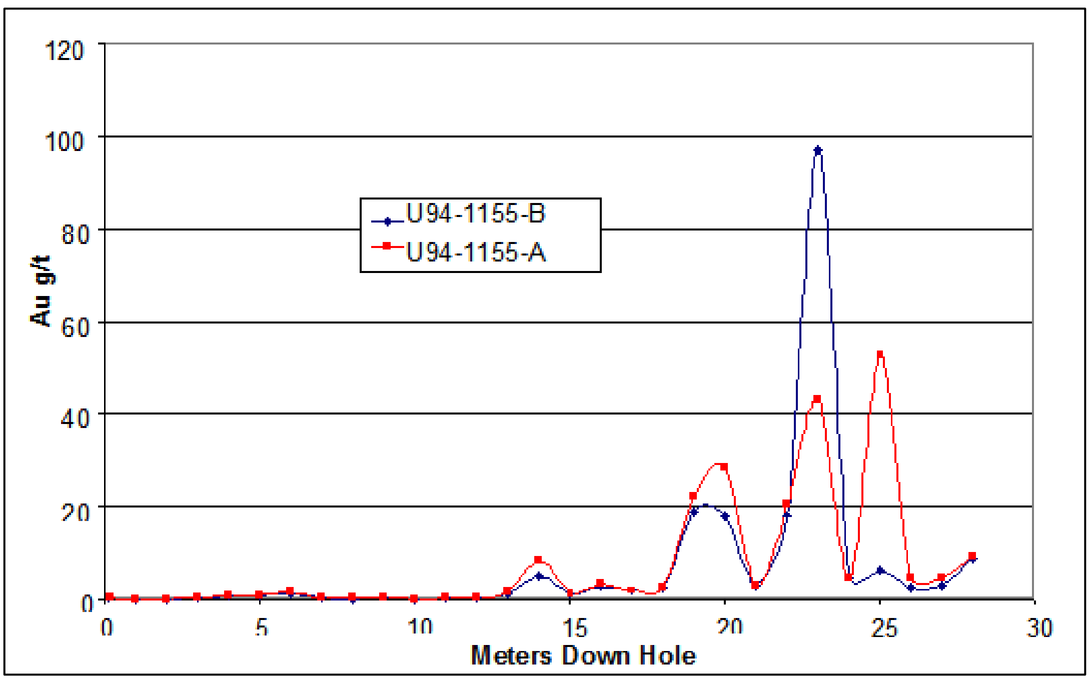
Drill Hole	From 13 to 29 m (Au g/t)
U94-1155	18.21
U94-1155, second half	12.11
U94-1156	16.43
U94-1156, second half	17.48
U94-1157	19.96
U94-1157, second half	18.32
U94-1158	16.31

Figure 11.3 to Figure 11.5 display the down hole assay comparisons for each half of the core for holes U94-1155 to U94-1157. Figure 11.6 displays the variance of holes U94-1155 to U94-1158 for the first ½ split of core in each hole.

Variance on an assay by assay in the two half-split comparison is relatively normal for a gold deposit and affects almost all ranges of assays. This would be expected in the Red Mountain style of stockwork. Stereonet analysis of the stockwork veining show that only 20% of the veins have a consistent trend within the stockwork envelope (Barclay, 2000) with the balance being relatively random. This randomness and rapid thickening and thinning over sub-metre and sub-centimetre distances was observed in both core and cross cuts and is an explanation for variance in grade as gold grade is associated with the percentage of coarse pyrite in a given interval.

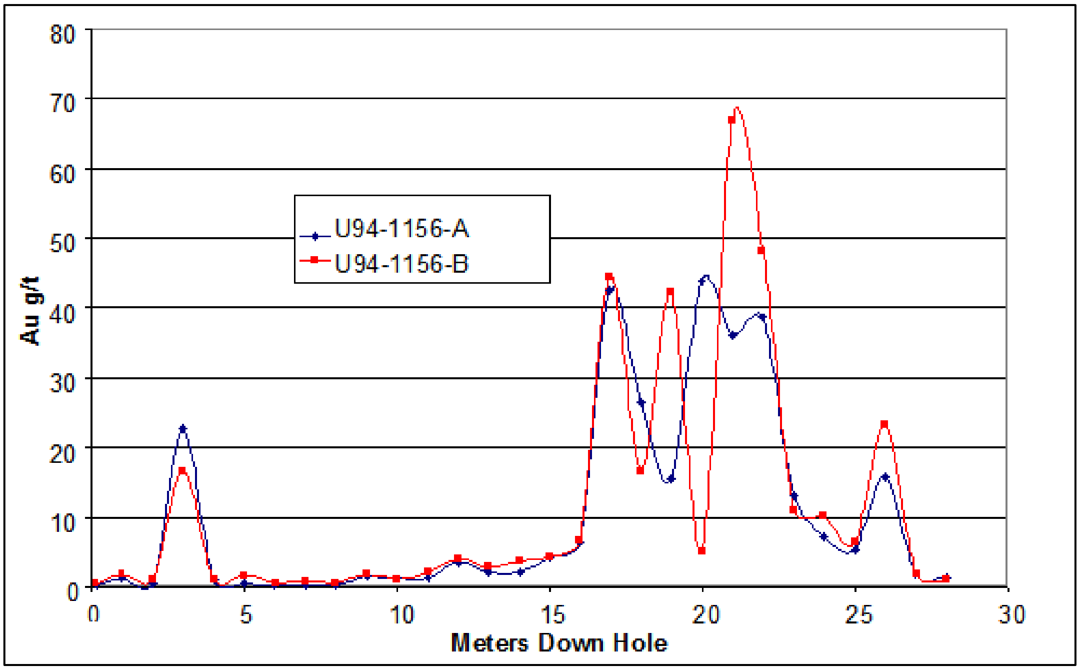
This variance is evident in the four individual drill holes (Figure 11.3 to Figure 11.5). When these plots considered in conjunction with the mean results for the 369 duplicates presented above, which suggest extremely good global precision, it is evident that variability on an individual sample basis can vary considerably, particularly at higher grades, as can be seen in Figure 11.7.

Figure 11.3: U94-1155 Gold Assay on Both Halves of Core



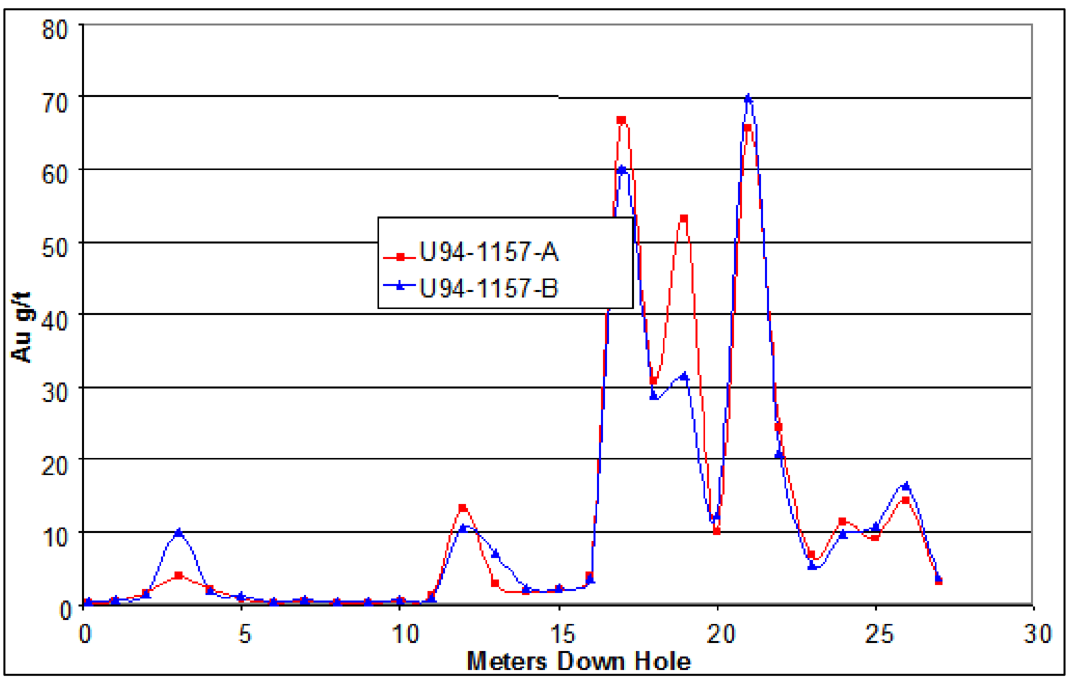
Source: NAMC (2001)

Figure 11.4: U94-1156 Gold Assays on Both Halves of Core



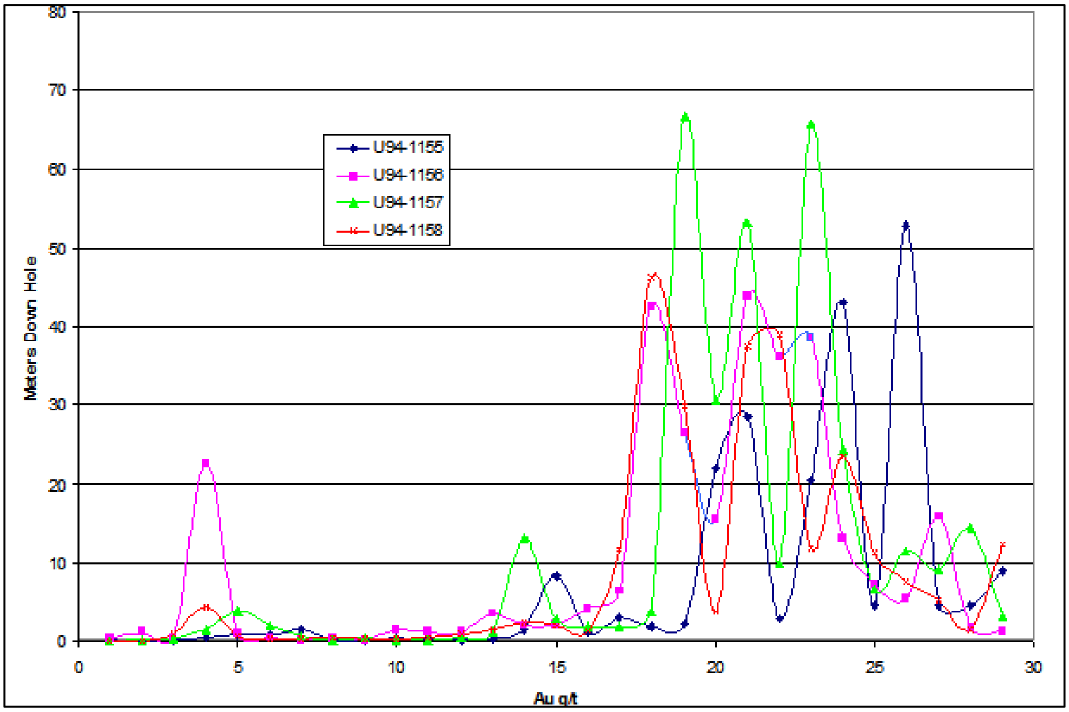
Source: NAMC (2001)

Figure 11.5: U94-1157 Gold Assays on Both Halves of Core



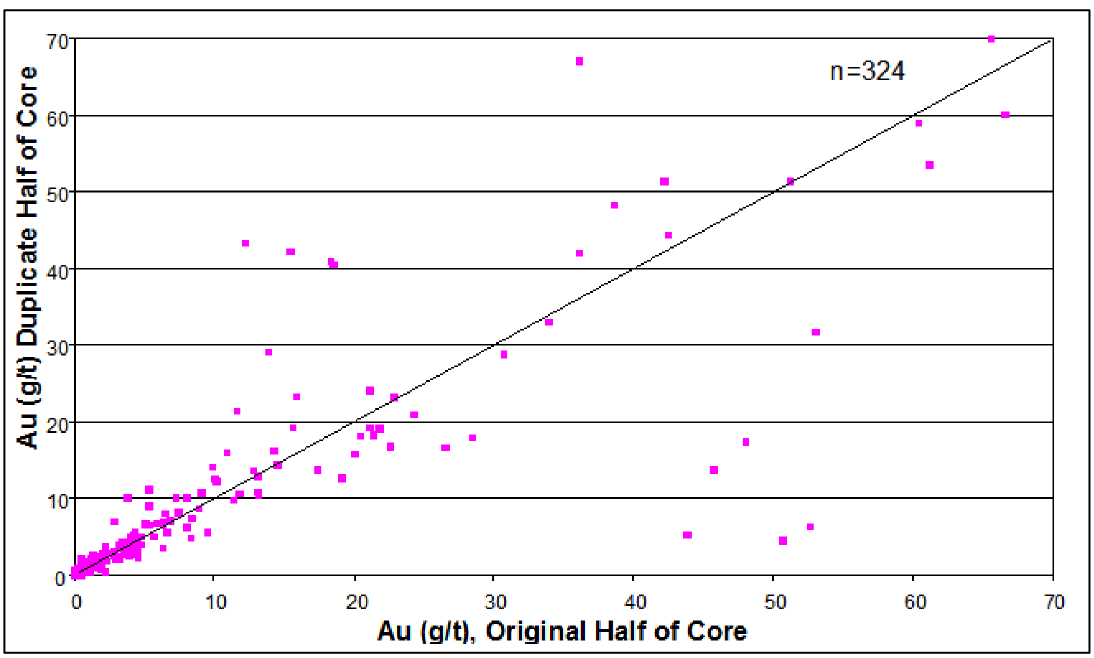
Source: NAMC (2001)

Figure 11.6: Gold Assay Comparison for DDH U94-1155, -1156, -1157 and -1158



Source: NAMC (2001)

Figure 11.7: Comparison of Original Gold Assays vs. Duplicate Halves of Core



Source: NAMC (2001)

Lab Audits/Visits

An important part of LAC's QAQC program were routine visits to the Eco-Tech laboratory facilities in Stewart. This was done on a regular basis during the 1993 and 1994 programs by a LAC geologist.

Early in 1993 Eco-Tech had a small facility in Stewart which could not cope with the large volume of samples and the quality of some results were suspect. In order to resolve this Eco Tech built a separate sample preparation facility in July 1993 which was inspected by a sampling consultant form Vancouver who considered the updated facilities adequate.

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

Extra Sample Splits

In 1994 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. When a duplicate assay was carried out by Eco-Tech on the same pulp, the average was given on the analytical certificate for the sample result, with the two individual results given at the end of the certificate with other QA/QC data. With samples with a second pulp (re-split), the assay from the original pulp was given as the sample result with the re-split result at the end of the certificate. As noted by Smit (2000) the individual assays were never compiled but would be useful if done, as an additional assessment of sample variance.

11.4.3 Royal Oak 1996

Royal Oak did not include QAQC materials in their drill hole sample shipments, but they did submit 221 pulps to Bondar Clegg for check assay. For both gold and silver Bondar Clegg results exhibited small low biases relative to the original Eco-Tech results. None of the Royal Oak holes are currently within resource areas.

11.4.4 NAMC QAQC 2000

NAMC submitted 197 samples, 167 of pulp and 30 of reject from mainly 1993 and 1994 drill holes in the Marc and AV mineralized zones for check assay. The results for this modest program indicated that Chemex was biased low relative to the original results by ~4.5% for gold, for both pulps and rejects.

Results for 9 LAC standards (3 different) included with these check samples indicate good assay accuracy.

11.4.5 Banks Island 2013

Banks Island inserted standards, coarse field blanks and pulp duplicates in their sample stream, at a rate of one for every 20 samples. In addition, they randomly inserted a few pulp blanks.

Details of the standards and pulp blank purchased form WCM Minerals of Burnaby, BC, are given in Table 11.9. The coarse field blank used came from a local quarry along the highway near the mouth of Bear Creek (coordinates 55°57'26"N, 129°58'52"W). The rock was from a barren Bitter Creek pluton of quartz monzonite composition.

Table 11.9 Banks Island Standard Reference Material

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

A total of 6 standard insertions were made, with all returning values within the +/-3SD limits, but both having average values 7% to 8% below the expected values. Two of nine coarse blanks failed, one after a 13.8 g/t Au sample suggesting contamination, the other unexplained. Visual inspection of the pulp duplicate results for gold indicate good assay precision.

11.4.6 IDM QAQC 2014

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

The standards used were from CDN Labs in Vancouver with values and limits as shown in Table 11.10. Timeline plots show good accuracy for gold. For silver Acme is biased high relative to the expected value by about 6 percent but most results still fall within failure limits. This bias may be related to the relatively high grade of the standard for an ICP analysis.

Table 11.10: 2014 CDN Labs Standards

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

The field blank used came from the same local quarry as used by Banks Island. All results were within the failure limits of 3 times detection limit for gold (DL was 5 ppb so failure limit is 15 ppb).

11.4.7 IDM QAQC 2016-2017

IDM started using a stronger QAQC program in 2016 consisting of a QC material once every 10 samples rotating between standards, blanks and field duplicates.

Four standards, two from CDN Labs and two OREAS standards have been used. Expected values and limits are shown in Table 11.11. Timeline plots shows good assay accuracy for both gold and silver. Figures 11.8 and 11.9 show the results for Standard CDN-GS-5Q for gold and silver respectively.

Table 11.11: 2016 Red Mountain Standards

Standard	Au Value	+3SD	-3SD	Ag Value	+3SD	-3SD
CDN-GS-1Q	1.24	1.36	1.12	40.7	44	37.4
CDN-GS-5Q	5.59	6.12	5.06	60.3	66.2	54.4
Oreas 60C	2.47	2.22	2.71	4.87	4.2	5.54
Oreas 62E	9.13	10.36	7.9	9.86	10.88	8.83

Figure 11.8: StandardsCDN-GS-5Q Gold Values

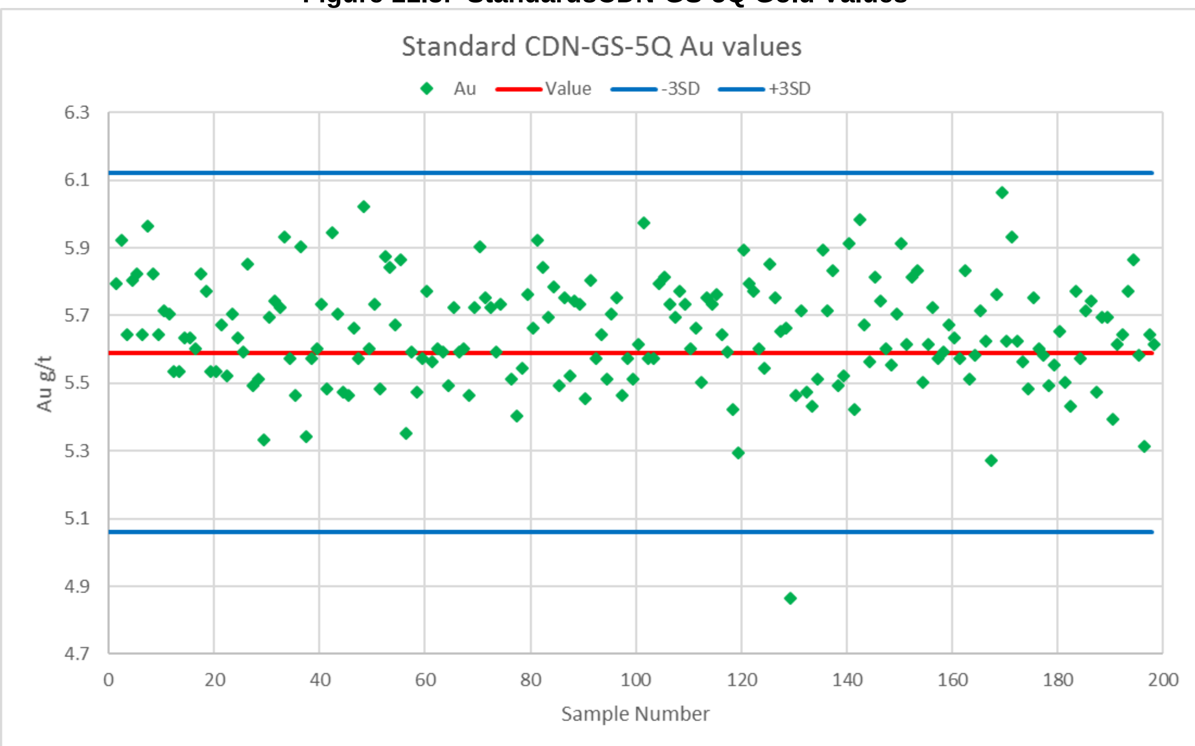
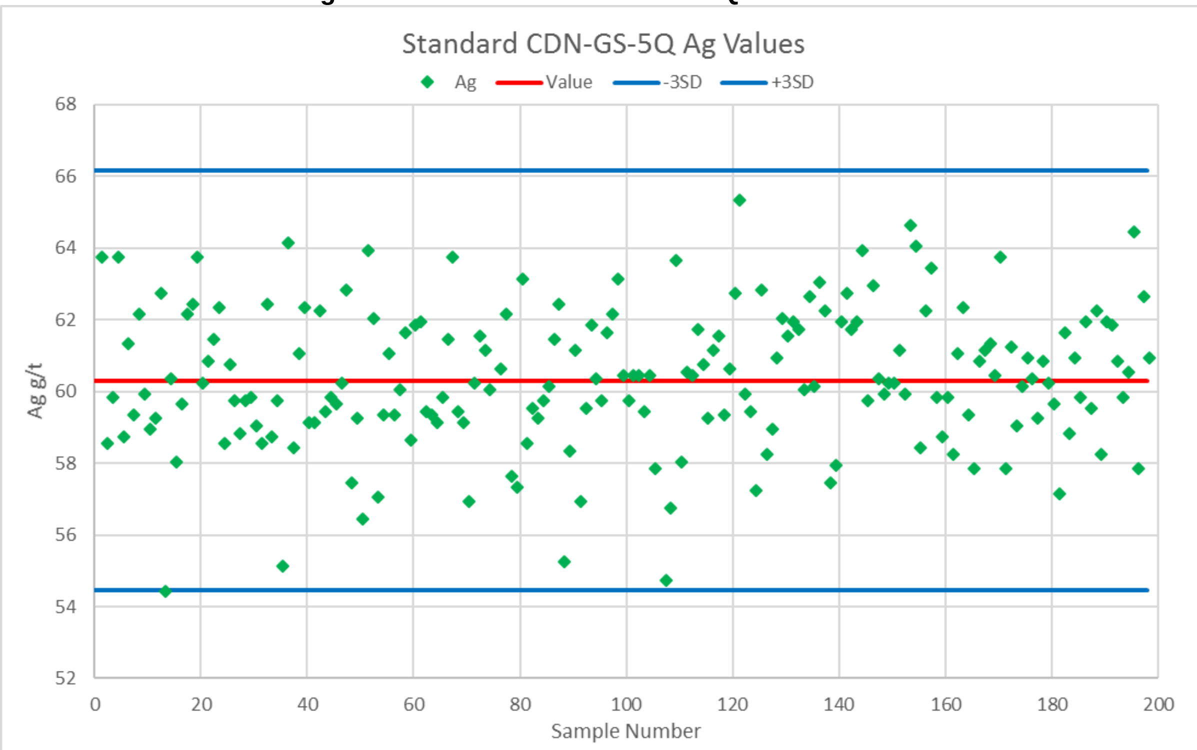


Figure 11.8: StandardsCDN-GS-5Q Sivler Values



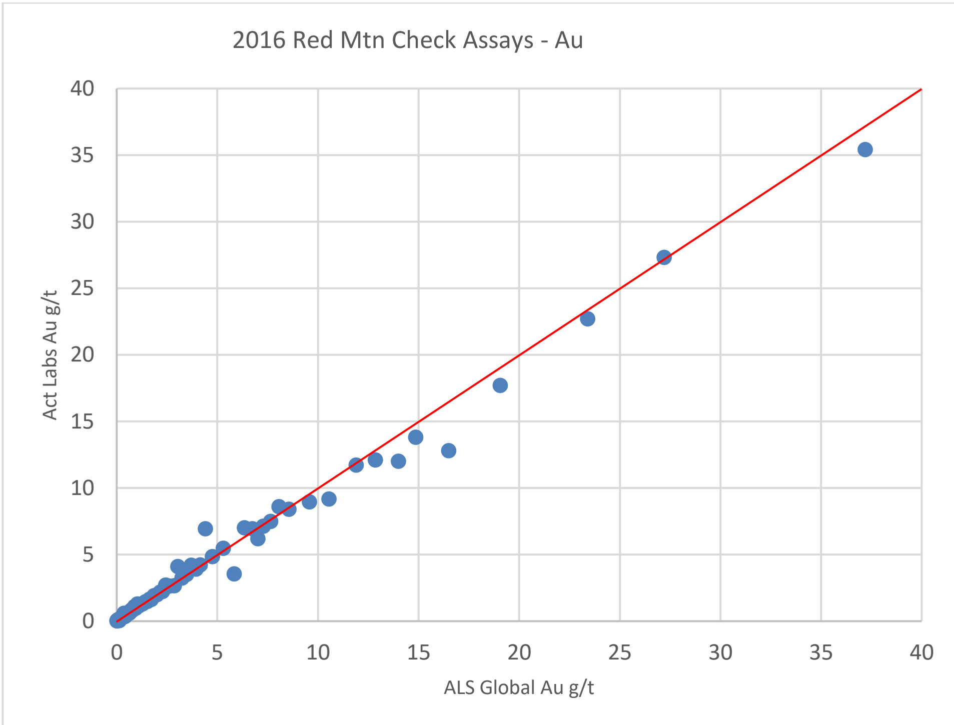
The same field blank as used 2014 has been used in the 2016 and 2017 programs. For gold, there were three failures in the 65 - 110 ppb range a few milder failures in the 20 – 30 ppb range. The stronger failures were found to follow high grade gold samples (after 63.8 and 1400 g/t values) and represent cases of mild contamination. A silver failure of 2.3 g/t Ag could not be explained. The number and tenor of both gold and silver failures are not considered serious and will have negligible effect on resource estimation.

For field duplicates, the full second half of NQ core was submitted for the duplicate sample, and in the case of HQ core the last ¼ core was submitted as the duplicate to match the ¼ core submitted as the original. Both gold and silver show moderate variability at all grade ranges reflecting the variable distribution of coarse stockwork pyrite in original and duplicate pairs.

IDM submitted 98 pulps from 2016 drill holes for check assay to ActLabs in Kamloops. The samples were selected mainly from within mineralized intersections but also included a few samples selected from low grade to un-mineralized sample intervals. Correlations for both gold and silver are good indicating good accuracy between laboratories. A plot for gold is shown in Figure 11.10.

In 2017 ninety-five pulps from 2017 drill holes were submitted for check assay to MS analytical in Langley B.C. As with the check assays from 2016 correlations for both gold and silver are good indicating good accuracy between laboratories.

Figure 11.10: 2016 Check Assay Comparison



11.5 COMMENTS ON QAQC

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

Standard results indicate no issues with assay accuracy as do the check assays that compare pulps to pulps as a measure of inter-lab accuracy. Similarly, true duplicate comparisons indicate good assay precision, although the dataset is quite small.

Historic comparisons for some sets of data between original pulp results, and the results of rejects sent as checks or comparisons between differing analyses on the same pulps (e.g. AA vs gravimetric), or a combination of both, are problematic as the sample support and analytical ranges of the different methodologies, respectively, are not the same.

Current QAQC protocols follow standard industry practices and are deemed adequate for inclusion of the assay data in resource estimation.

11.6 DATABASES

Information in Bray (2000) indicates that in 1993 and 1994 all Bond and LAC data were in a series of FoxPro databases. In 2000, these were combined into a smaller number of “master” FoxPro databases and then into a single master Microsoft (MS) Access database. This MS Access database contained much of the project data and was used in 2000 to populate a Gemcom Red Mountain drill database that formed the basis for the current mineral resource estimate.

11.7 SECURITY

11.7.1 Security

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

Bond security measures were not recorded at the time and normal security processes for the period are assumed.

LAC followed a diligent process of flying the core directly to the core storage facility in Stewart where logging and sampling was carried out under LAC supervision. Samples were delivered directly to the Eco-Tech laboratory located in Stewart accompanied by sample submittal forms.

Royal Oak samples were collected in the Goldslide Creek camp and subsequently delivered from the project area to the Eco-Tech sample preparation facility in Stewart.

NAMC samples were collected by a staff professional geologist and delivered to the Chemex laboratory under the direct supervision of the geologist.

In 2014, samples were shipped in rice bags and delivered from the project to a commercial trucking company based in Stewart. The samples were then delivered to Acme lab’s sample preparation facility in Smithers, B.C. The same procedure was used in 2016 and 2017 except that sample shipments were delivered to the ALS Global sample preparation facility in Terrace, B.C.

11.7.2 Storage

All drill core from 1989 to 1996 (Bond, Lac and Royal Oak) is stored in a fenced compound immediately next to the Stewart airstrip. The bulk samples and rejects are also stored in this location but have deteriorated to a point whereby they are no longer usable.

The Banks Island core was initially stored in the Banks Island warehouse in Smithers, BC. The authors are unaware of the current location of the Banks Island core or if it still exists.

Core from the 2014, 2016 and 2017 IDM drilling programs is stacked on pallets at the Goldslide Creek camp. Sample rejects have not been maintained. Pulps are currently stored at the ALS Global facility in Terrace but storage in Stewart is being arranged.

11.8 COMMENTS ON SECTION 11

In the opinion of the QP the quality of the analytical data is sufficiently reliable to support mineral resource estimation. Sample collection, preparation, analysis, and security were generally performed in accordance with exploration best practices and industry standards as follows:

- Sample collection and preparation for samples that support mineral resource estimation has been in line with industry-standard methods for the pyritic, stockwork hosted gold mineralization that occurs at Red Mountain;
- Drill core samples were analysed by independent laboratories using industry-standard methods for gold and silver analyses;
- Drill programs have included the insertion of an adequate number of QAQC materials;
- The QAQC program results do not indicate any problems with the analytical programs, and demonstrate that the results are accurate and precise;
- Sample security has relied upon the fact that the samples were always attended to by drill crews or company staff while at the project site or logging facilities, and delivered to the lab either directly by project staff or commercial trucking companies;
- The data that was collected was entered in databases and validated through visual checks prior to being imported into the master drill database(s).
- Current sample storage procedures and storage areas are consistent with industry standards.

12.0 DATA VERIFICATION

12.1 LAC DATABASE VERIFICATION

Data verification has been carried out by previous operators of the project including Bond, LAC and NAMC. In 2000, NAMC cross-referenced and catalogued all data from previous operators.

For all but the 2014 IDM program data have been transferred from paper format to electronic format. Data were entered into the computer by data entry personnel. All 1993 LAC data were checked in January and February of 1994. In 1994, LAC instituted a system where all drill hole data were entered and checked by different people as soon as possible after logging. The geologist who logged a hole was responsible to ensure all data was entered, checked and that data printouts were with completed logs in the files. Merging new data into the master drill databases was done by the system manager.

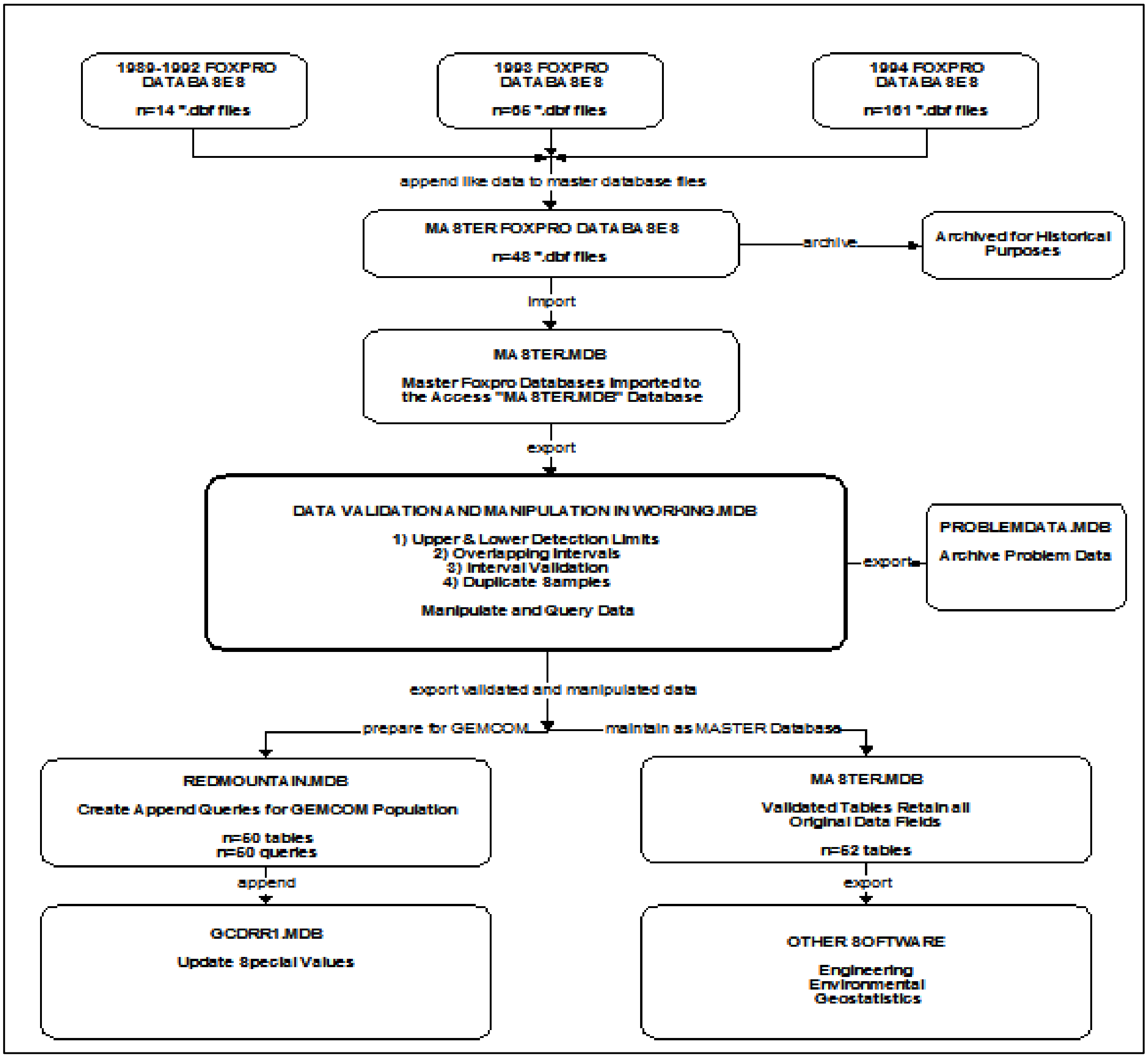
12.2 ELECTRONIC DATA VERIFICATION

LAC collected and organised over one gigabyte of electronic information during their work on the Red Mountain property during 1993 and 1994. As the project was under fast track conditions by LAC management, the programs were never compiled into a cohesive database that was accessible by a single program. NAMC, upon receiving the project data, undertook to create and validate a Microsoft Access database that held all of the site exploration and environmental work.

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

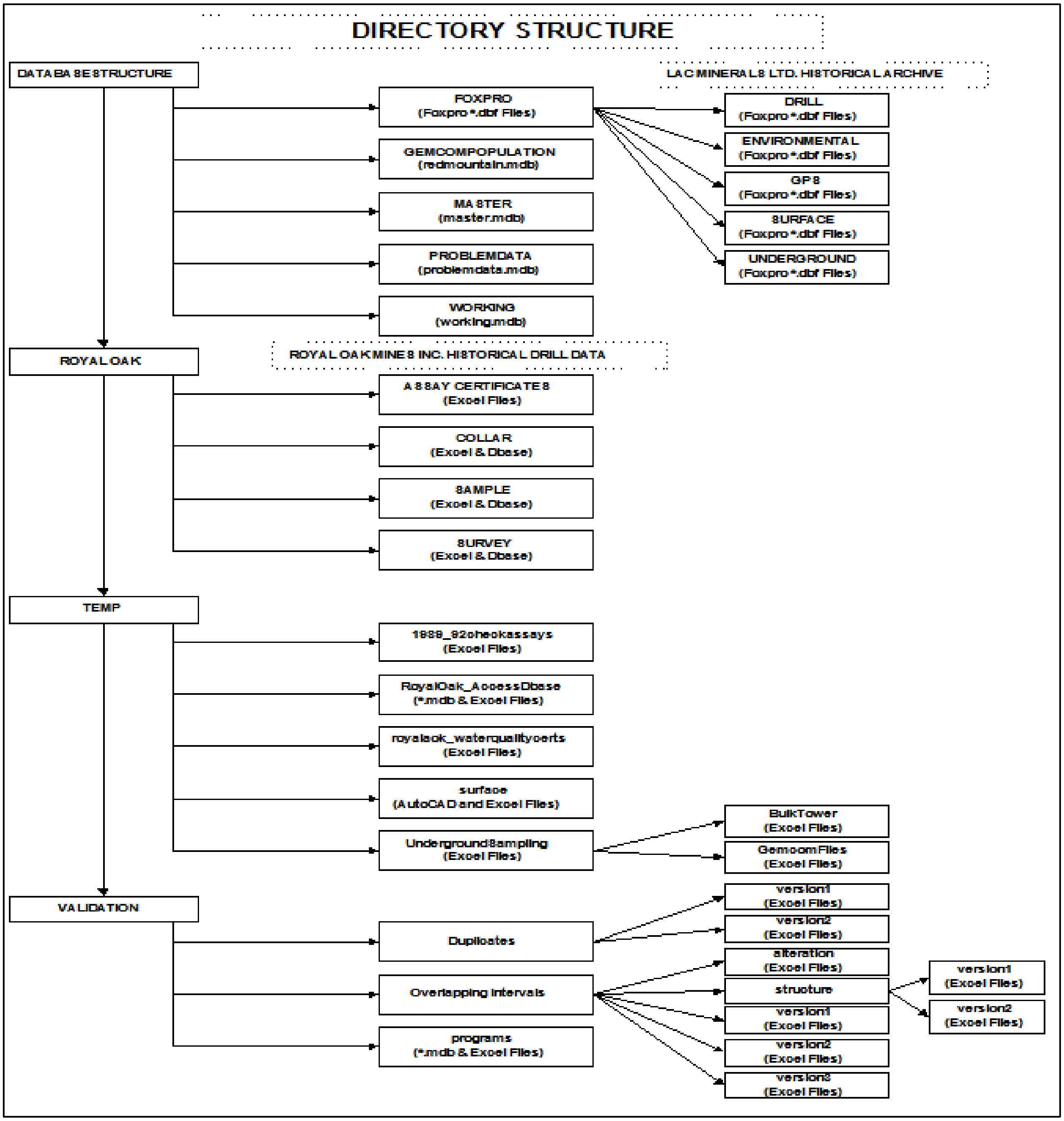
Flow sheets illustrating the database compilation procedures and resulting directory structure as shown in Figure 12.1 and Figure 12.2 respectively.

Figure 12.1: Data Validation Flowchart



Source: NAMC (2001)

Figure 12.2: Directory Structure



Source: NAMC (2001)

12.3 NAMC METALLURGICAL COMPOSITES

NAMC compiled five metallurgical composite suites from drill core. Samples were taken from intervals in the Marc and AV zones and were selected to give an average gold grade and distribution similar to the estimated milled head grade of 5-15 g/t Au. These composites were taken from the remaining half of drill core in the boxes, sawn to a ¼ sample and individually bagged in the original sample interval length. The samples were sent to Process Research Associates Ltd. where they were dried, weighed and 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 device.

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)

12.4 2016 AND 2017 DATA VERIFICATION

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

12.4.1 Collar Table

Drill collar locations were audited through examination in three dimensional GEMs software to ensure that collars were properly located in underground drill stations and in the case of surface holes coincident, within reasonable limits, with the topographic surface. No anomalies were noted.

12.4.2 Survey Table

The down hole survey table form the GEMs database was checked by examining the changes from one survey to the next in all holes for both azimuth and dip. A total of six holes from the 1993 and 1994 surface drilling programs have anomalous azimuth or dip deviations that should be checked through a combination of re-examining the Sperry Sun photos and looking at the mineralization data for the presence of pyrrhotite. One of these holes, M93157 pierces the 141 Zone solid, while the rest do not intersect resource solids.

12.4.3 Assay Table

Most original historical assay certificates are available in the Red Mountain files. A check was made between the gold and silver values in the GEMs data base and values on the historical assay certificates for assays from within the resource solids. A selection of drill holes from all resource zones was made that tried to cover different years of drilling and assayers, as well as being spatially representative. The number of historical assays checked for each zone is given in Table 12.2.

Table 12.2: Assay Validation Summary

Zone	No. Holes Checked	No. of Assays in Solid	No. of Assays Checked	% Checked
Marc	10	1978	202	10.2
Marc Footwall	3	53	11	20.7
AV	5	442	116	26.2
AV Lower	2	21	5	23.8
JW	3	104	20	19.2
JW Lower	1	36	6	16.7
132 Zone	2	95	11	11.6
141 Zone	7	328	76	23.2
Totals	**33**	**3057**	**447**	**14.6**

Overall the database was found to be very clean. Two instances of errors in the second decimal place for gold were found and are most likely data entry errors. A third discrepancy was found whereby a gold value of 4.33 was entered instead of the 3.98 listed on the certificate. No discrepancies were noted in silver values.

The 2016 and 2017 assays were validated by comparing the data base values to certificates obtained directly from ALS Global. Assays from certificates representing between 10% and 15% of from each year were evaluated. No discrepancies were found.

12.4.4 ACS Site Visit and Check Samples

ACS carried out a site visit to the Red Mountain gold project on May 7, 2017 for one day. During the site visit, ACS verified the property access, logistics and surface geology.

During a similar site visit in 2016 the underground workings were examined, and seven check samples were collected for validation. Three samples were collected from the Marc Zone from the underground cross cuts and four samples were collected from drill core stored in Stewart. Table 12.3 summarizes the results of the re-sampling program carried out by Arseneau Consulting Services Inc. (ACS).

Overall the ACS sample results agree well with the previous results. The sampling program was not intended to be a robust validation program, instead the samples were only collected to verify that the Red Mountain gold project did host gold and silver mineralization in the range of grades that have been reported for the Project in the past.

Table 12.3: Results of 2016 re-sampling program

Sample Number	Sample Location	Original Au Value (g/t)	Re-assay Au Value (g/t)	Original Ag Value (g/t)	Re-assay Ag Value (g/t)
I95066	1100 cross cut	4.95	7.43	26	16
I95067	1200 cross cut	1.3	0.1	1.7	<5
I95068	1295 cross cut	6.5	1.25	48	<5
I95069	DH941148	1.26	1.69	0.8	<5
I95070	DHM93154	3.95	6.62	3.8	<5
I90571	DHM9054	4.78	7.03	38	42
I95072	DH941122	5.7	2.35	0.05	<5

12.5 COMMENTS ON SECTION 12

The QP has reviewed the appropriate reports and data and is of the opinion that the data verification programs undertaken on the data collected adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation.

13.0 MINERAL PROCESSING & METALLURGICAL TESTING

The following Section is taken from JDS (2017) with minor modifications. ACS takes responsibility for this section prepared by Tom Shouldice, P. Eng. on behalf of JDS.

13.1 INTRODUCTION

Historical metallurgical testing was performed on Red Mountain samples by Lakefield Research (1991), Brenda Process Technology (1994), and International Metallurgical and Environmental (1997), a derivative of Brenda Process Technology. The majority of the test work conducted between 1991 and 1997 focused on cyanide leaching as the primary process for extracting gold and silver from the deposit.

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

In 2015, test work was completed by Gekko Systems (Gekko). Gravity, flotation and comminution test work was completed to test the amenability of the Red Mountain deposit to Gekko's Python modular plant. The results from that study are also applicable for generic flotation plants.

Results from 1991 to 2015 are documented in the 2016 PEA.

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

The 2016-2017 test work program was completed on variability and composite samples for Marc, AV, JW, and 141 zones. Initially, the test work focused on the PEA flowsheet, which included rougher flotation followed by concentrate leach. Pyrrhotite levels varied significantly in the deposit; and due to the increased reactivity and oxidation of the material, were found to dramatically affect flotation performance. As a result, WOL became the focus of the program. Optimization continued primarily on the Marc zone master composite and was confirmed with the AV, JW, and 141 samples. The final flowsheet included a primary grind to 80% passing (P_{80}) 25 microns (µm) followed by CIL recovery of gold and silver.

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

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

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

13.2 BASE MET LABS 2016 TO 2017 METALLURGICAL TESTING

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

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

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

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

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

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

- Direct cyanide leaching, often referred to as whole ore leaching (WOL); and
- Flotation of gold-bearing sulphide material to produce a rougher concentrate followed by regrinding and cyanide leach (FRL).

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

13.2.1.1 Variability Composite Sample Selection

A total of 36 variability samples were constructed using samples from 2016 drill holes for three zones: 18 from Marc zone, designated as MV, and nine from both AV and JW zones. The composites were selected to test a range of feed grades and geological lithologies in each zone, including various sulphur levels. Composites were constructed from contiguous half core intervals from the same drill hole to maintain spatial representation. In some instances, the gold content was quite variable between adjacent samples.

The chemical contents of key elements for the variability samples are displayed in Table 13.1.

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

Table 13.1: Head Assay Data for BL0084 Variability Composites

Composite ID	Zone	Chemical Content				
		Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
MV1	Marc Zone	5.79	42	7.5	6.66	0.03
MV2	Marc Zone	9.06	46	9.0	9.85	0.04
MV3	Marc Zone	3.52	7.4	6.7	4.96	0.03
MV4	Marc Zone	3.96	41	6.1	5.50	0.02
MV5	Marc Zone	12.3	68	11.2	13.2	0.02
MV6	Marc Zone	5.10	24	9.8	10.4	0.03
MV7	Marc Zone	3.73	5.9	8.0	5.39	0.03
MV8	Marc Zone	5.81	7.4	7.0	4.68	0.01
MV9	Marc Zone	8.53	42	7.5	5.38	0.11
MV10	Marc Zone	7.14	18	7.7	7.09	0.19
MV11	Marc Zone	0.87	0.9	7.4	5.37	0.05
MV12	Marc Zone	22.4	12	11.2	11.7	0.04
MV13	Marc Zone	5.33	50	8.5	9.83	0.02
MV14	Marc Zone	3.94	2.4	9.3	5.98	0.01
MV15	Marc Zone	14.7	16	10.8	12.2	0.02
MV16	Marc Zone	4.84	2.7	7.7	4.81	0.02
MV17	Marc Zone	16.2	71	13.0	17.4	0.02
MV18	Marc Zone	2.35	38	8.4	8.94	0.01
JW1	JW Zone	0.86	0.7	6.4	3.80	0.02
JW2	JW Zone	5.70	26	10.4	13.0	0.02
JW3	JW Zone	6.77	130	9.2	10.2	0.02
JW4	JW Zone	7.31	11	8.9	10.6	0.02
JW5	JW Zone	5.35	3.8	15.1	11.4	0.04
JW6	JW Zone	2.82	15	12.2	14.4	0.02
JW7	JW Zone	2.75	1.1	9.8	6.33	0.02
JW8	JW Zone	6.10	6.2	9.25	6.00	0.01

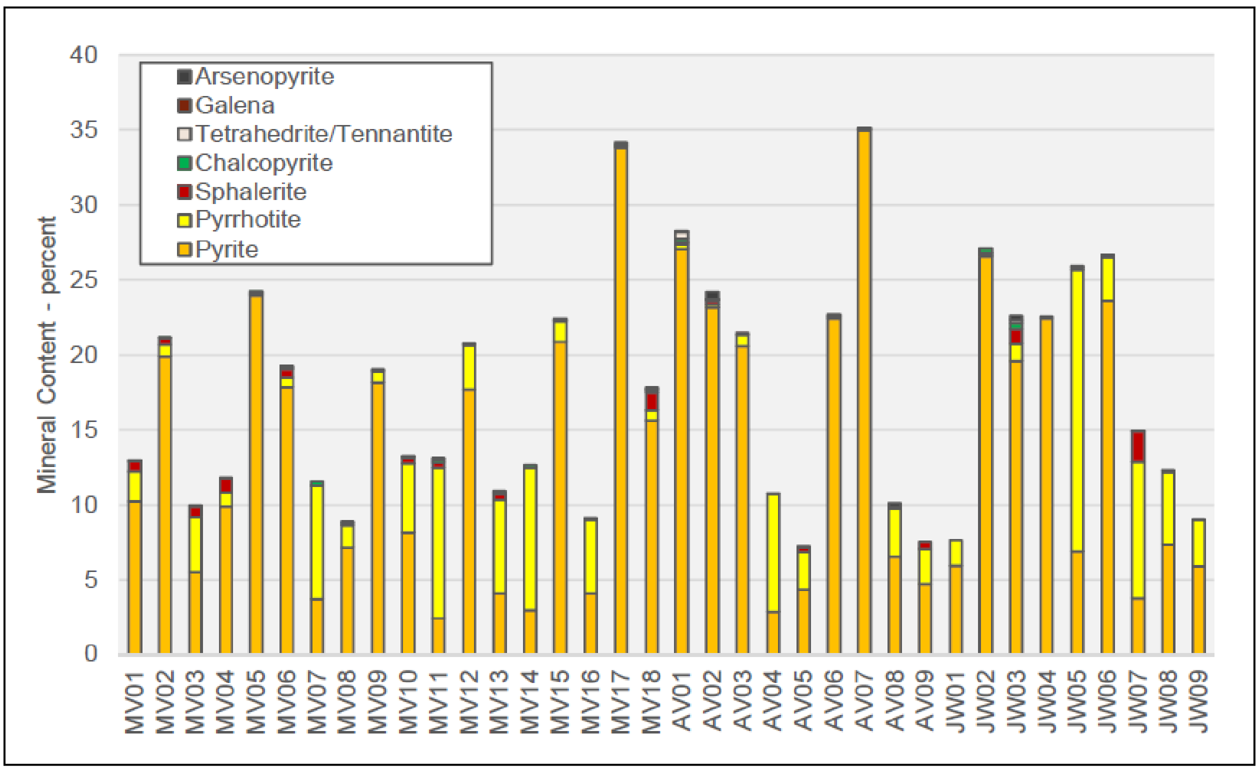
Composite ID	Zone	Chemical Content				
		Au (g/t)	Ag (g/t)	Fe (%)	S (%)	TOC (%)
JW9	JW Zone	1.83	1.5	7.2	4.37	0.01
AV1	AV Zone	6.96	22	11.9	14.1	0.03
AV2	AV Zone	3.90	11	9.9	11.7	0.02
AV3	AV Zone	4.76	15	9.7	10.9	0.05
AV4	AV Zone	6.41	1.4	7.4	4.65	0.01
AV5	AV Zone	0.38	1.7	7.6	4.13	0.02
AV6	AV Zone	5.15	28	10.0	11.5	0.02
AV7	AV Zone	31.0	49	14.1	18.8	0.02
AV8	AV Zone	29.5	42	11.2	11.9	0.01
AV9	AV Zone	0.21	3.0	5.8	3.42	0.01

Source: Base Met (2017a)

13.2.1.2 Mineralogical Characterization of Variability Composites

The mineral composition of each variability sample was determined by QEMSCAN - Bulk Mineral Analysis (BMA) determinations. The main non-sulphide minerals included muscovite, quartz, feldspars, and chlorite. The sulphide minerals, which are of particular interest, are shown in detail in Figure 13.1:. Pyrite and pyrrhotite represented the majority of the sulphide minerals in the samples, at levels up to about 35% in some samples. Pyrrhotite is a highly reactive mineral and susceptible to oxidation, which could negatively affect flotation performance.

Figure 13.1: Sulphide Mineral Content for BL0084 Variability Composites

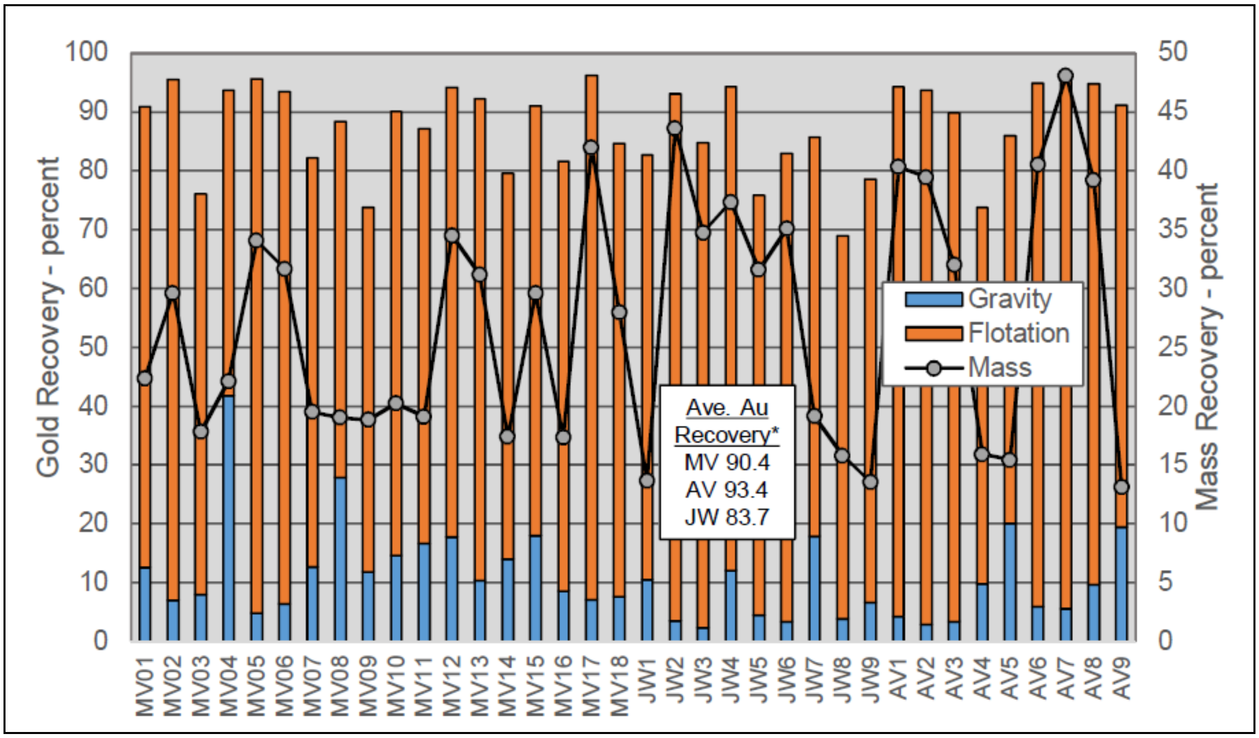


Source: Base Met (2017a)

13.2.1.3 Gravity and Rougher Flotation Testing on Variability Composites

Gravity concentration followed by bulk sulphide rougher flotation tests were conducted on all variability composites at a primary P_{80} grind size of 150 µm. The flotation tests were conducted at natural pH and used Potassium Amyl Xanthate (PAX) as the sulphide mineral and gold collector. A graphical summary of the gravity - flotation tests is displayed in Figure 13.2.

Figure 13.2: Gravity and Rougher Flotation Results for BL0084 Variability Composites



Source: Base Met (2017a)

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

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

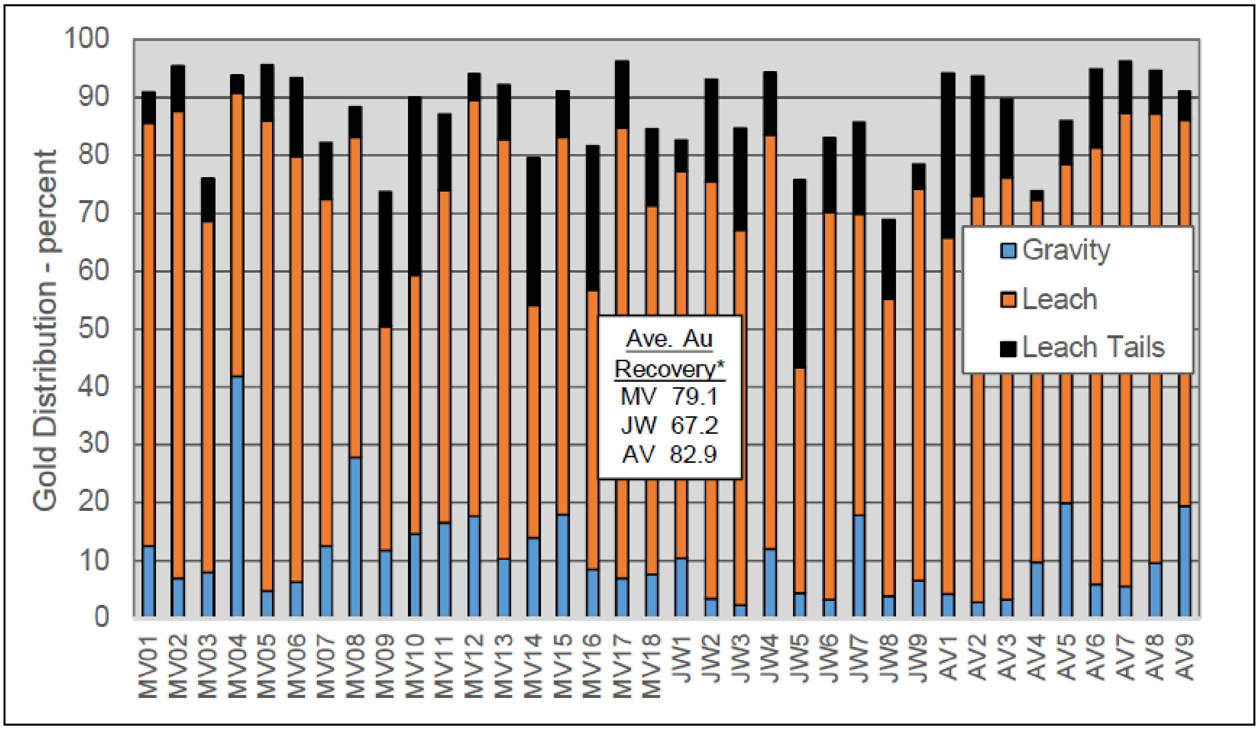
13.2.1.4 Rougher Concentrate Cyanidation Tests on Variability Composites

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

A summary of gold and silver distributions, including in the cyanide leach tailings, are shown in Figure 13.3 and Figure 13.4. The cumulative sums of the blue and orange bars are considered the final recovery position of the combined gravity, flotation, and flotation concentrate leach process. The average final recovery for each zone is shown in the inset table. The black portion of the bar indicates the metal lost to the leach tailings.

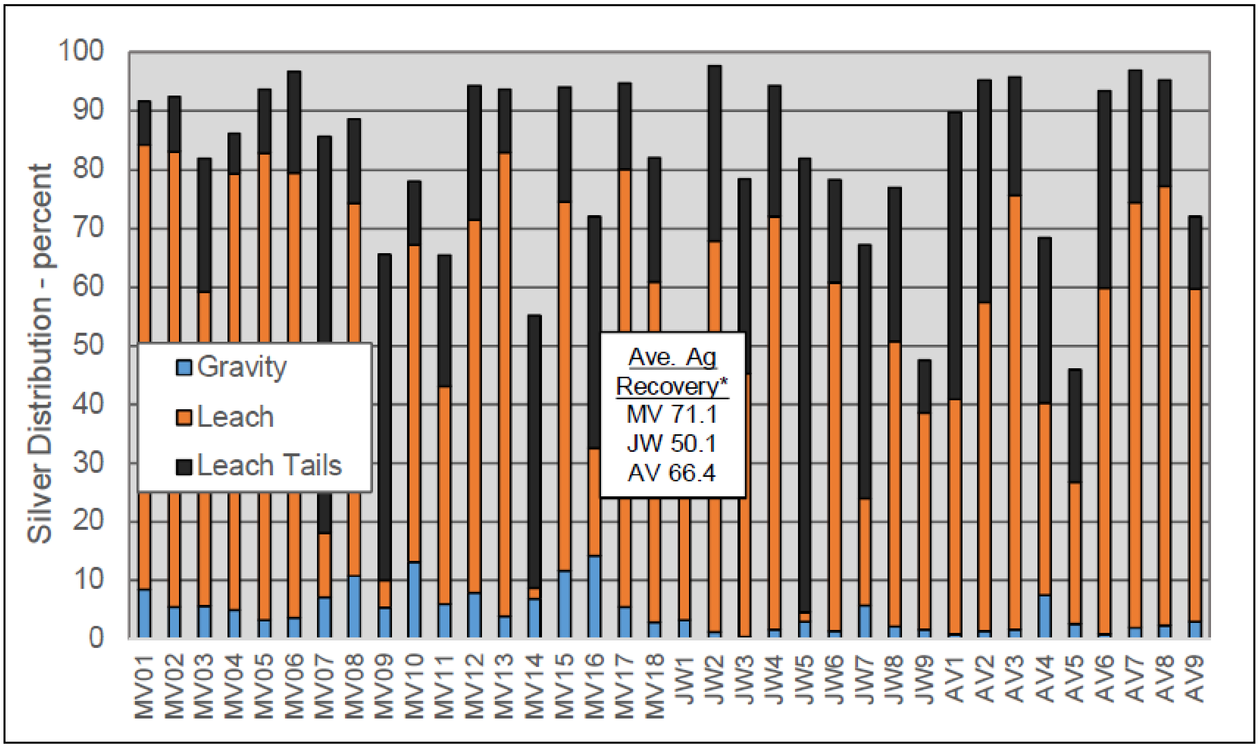
As shown, many of the samples had significant gold and silver losses to the leach tailings. Overall recovery, which includes gravity concentrate and leach extraction, ranged from 43 to 91% gold and 5 to 84% silver.

Figure 13.3: Overall Gravity/Flotation/Leach Au Recoveries for BL0084 Variability Composites



Source: Base Met (2017a)

Figure 13.4: Overall Gravity/Flotation/Leach Ag Recoveries for BL0084 Variability Composites

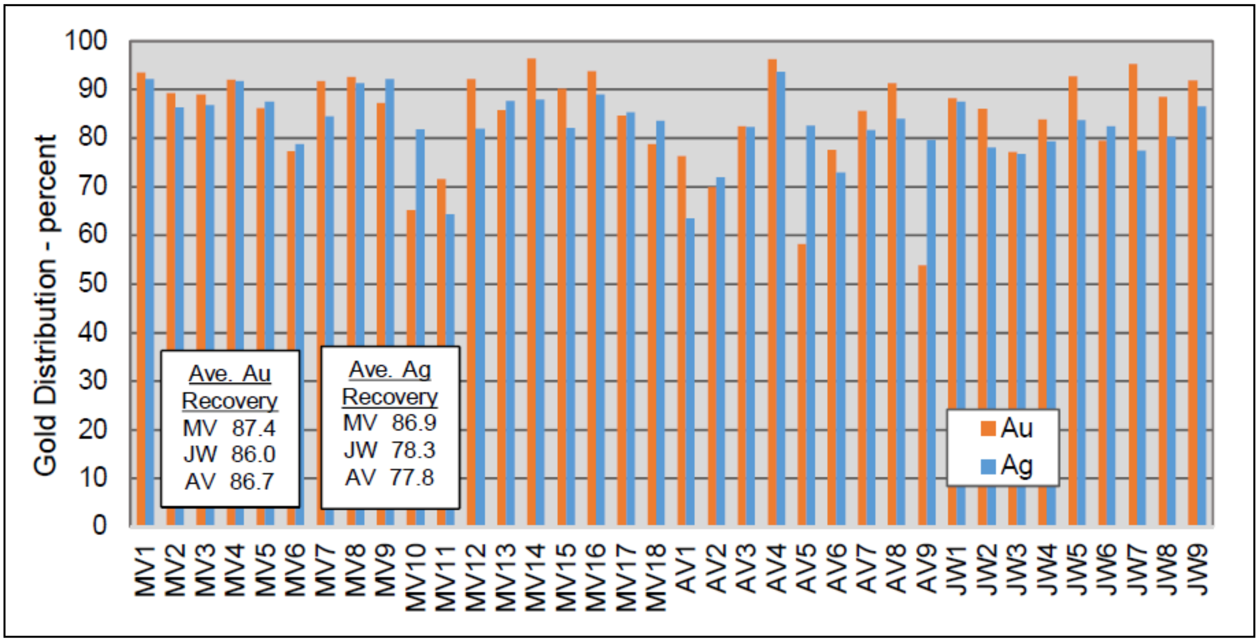


Source: Base Met (2017a)

13.2.1.5 Whole Ore Leach Tests on Variability Composites

Whole ore carbon in leach tests were conducted on all variability samples at primary grind sizes targeting a P_{80} of 40 µm. Tests were conducted at a pH of 11.0, and a NaCN concentration of 1,000 ppm. Lead nitrate was added at 250 g/t, and carbon was added at 50 g/L. Oxygen was sparged at 2, 6, 24, and 48-hour time intervals. The gold and silver recoveries are shown in Figure 13.5. The inset tables display weighted average recoveries by feed grade.

Figure 13.5: Whole Ore Leach Results for BL0084 Variability Composites



Source: Base Met (2017a)

13.2.1.6 Master Composite Sample Selection

Master Composites for the three zones were constructed using the variability composite ratios depicted in Table 13.2.

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

Table 13.2: Composition of BL0084 Master Composites

Marc Master Composite		AV Master Composite #1		JW Master Composite #1	
Composite ID	**Contribution (%)**	**Composite ID**	**Contribution (%)**	**Composite ID**	**Contribution (%)**
MV1	7	AV1	20	JW1	11
MV2	11	AV2	22	JW2	26
MV3	0	AV3	8	JW3	0
MV4	10	AV4	10	JW4	34
MV5	5	AV5	13	JW5	9
MV6	6	AV6	12	JW6	20
MV7	0	AV7	6	JW7	0
MV8	7	AV8	0	JW8	0
MV9	3	AV9	9	HW9	0
MV10	8				
MV11	8				
MV12	5				
MV13	10				
MV14	0				
MV15	2				

Marc Master Composite		AV Master Composite #1		JW Master Composite #1	
Composite ID	**Contribution (%)**	**Composite ID**	**Contribution (%)**	**Composite ID**	**Contribution (%)**
MV16	2				
MV17	6				
MV18	10				
Total	**100**	**Total**	**100**	**Total**	**100**

Source: Base Met (2017a)

Head assays for the Master Composites are shown in Table 13.3. The JW zone had limited material for composite construction and the overall composite was relatively low in weight. Therefore, subsequent optimization testing in this program focused on the Marc and AV master composites. Final conditions derived from the Marc and AV zone were then applied to the JW master composite.

Table 13.3: Head Assay Data for BL0084 Master Composites

Composite ID	Chemical Content										
	Au (g/t)	**Ag (g/t)**	**S (%)**	**Fe (%)**	**TOC (%)**	**Cu (g/t)**	**Pb (g/t)**	**Zn (g/t)**	**As (g/t)**	**Sb (g/t)**	**Te (g/t)**
Marc Master Composite - Head 1	9.1	33	8.98	8.5	0.05	355	162	2,290	471	75	52
Marc Master Composite - Head 2	11.0	32	8.77	8.7	-	341	181	2,530	502	76	53
Average	**10.1**	**33**	**8.88**	**8.6**	**0.05**	**348**	**173**	**2,410**	**487**	**76**	**53**
AV Master Composite #1 - Head 1	6.45	16	10.9	9.2	0.01	657	58	1,130	481	313	35
AV Master Composite #1 - Head 2	6.16	16	10.9	9.4	-	737	57	1,210	497	352	30
Average	**6.31**	**16**	**10.9**	**9.3**	**0.01**	**697**	**58**	**1,170**	**489**	**333**	**33**
JW Master Composite #1 - Head 1	5.63	13	12.3	10.2	0.02	467	36	606	307	94	29
JW Master Composite #1 - Head 2	5.55	16	12.2	10.2	-	479	39	518	327	102	32
Average	**5.59**	**14**	**12.3**	**10.2**	**0.02**	**473**	**38**	**562**	**317**	**98**	**31**

Source: Base Met (2017a)

13.2.1.7 Gold Mineralogical Assessment on Master Composites

All three master composites were submitted for gold trace mineral searches to investigate gold occurrences in the samples. Due to the limited number of gold occurrences found, the results should be taken as indicative only. A summary of the mineral associations is shown in Table 13.4.

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

Table 13.4: Master Composite Gold Deportment Percents by Mineral Species

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

Source: Base Met (2017a)

13.2.1.8 Master Composite Metallurgical Testing

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

A series of leach tests were conducted on the whole feed of the Marc and AV Master Composites. A summary of test conditions and results is summarized in Table 13.5.

Table 13.5: Master Composite Whole Ore Leach Test Results

Test	Grind K_{80} (µm)	Gravity	pH	$Pb(NO_3)_2$ (g/t)	NaCN (ppm)	Carbon (g/L)	O_2 Sparged	Pre-Ox 24hr	Extraction	
									Au (%)	Ag (%)
Marc Master Composite										
43	68	No	11.0	0	1,000	0	Yes	None	**80**	**76**
44	68	Yes	11.0	0	1,000	0	Yes	None	**82**	**84**
45	50	No	11.0	0	1,000	0	Yes	None	**85**	**82**
46	37	No	11.0	0	1,000	0	Yes	None	**86**	**86**
47	68	No	11.0	0	500	0	Yes	None	**81**	**79**
48	68	No	11.0	0	2,000	0	Yes	None	**81**	**77**
49	68	No	11.0	0	3,000	0	Yes	None	**81**	**81**
50	68	No	10.0	0	1,000	0	Yes	None	**81**	**78**
51	68	No	11.5	0	1,000	0	Yes	None	**81**	**79**
52	68	No	12.0	0	1,000	0	Yes	None	**79**	**79**
53	68	No	11.0	250	1,000	0	Yes	None	**81**	**74**
54	68	No	11.0	500	1,000	0	Yes	None	**80**	**72**
55	68	No	11.0	0	1,000	0	No	Air	**82**	**75**
56	68	No	11.0	0	1,000	0	Yes	O2	**82**	**76**
77	37	No	11.0	250	1,000	50	Yes	None	**89**	**90**
78	68	No	11.0	250	1,000	50	Yes	None	**86**	**86**
117	37	No	11.0	250	1,000	50	Yes	None	**90**	**89**
118	21	No	11.0	250	1,000	50	Yes	None	**92**	**92**
119	17	No	11.0	250	1,000	50	Yes	None	**93**	**93**
AV Master Composite #1										
57	70	No	11.0	0	1,000	0	Yes	None	**79**	**70**
58	70	Yes	11.0	0	1,000	0	Yes	None	**76**	**74**

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

Source: Base Met (2017a)

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

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

Flotation tests were conducted on the Marc and AV Master Composites to determine metallurgical response using a bulk sulphide flotation flowsheet. Initial testing on variability samples indicated gold losses in the flotation stage outweighed additional processing costs of WOL. Finer primary grinding prior to flotation, as well as the addition of copper sulphate were tested on the master composites to assess if an increase in gold recovery was achievable with either of these methods. The results are summarized in Table 13.6.

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

Table 13.6: Master Composite Flotation Test Results

Test	Composite	Primary Grind K_{80} (µm)	Reagents		Flotation Recovery			
			PAX (g/t)	$CuSO_4$ (g/t)	Mass (%)	Au (%)	Ag (%)	S (%)
71	Marc Master	150	35	0	24	91	87	91
72	Marc Master	68	50	0	22	92	85	89
73	Marc Master	150	35	150	27	93	87	95
74	AV Master #1	150	35	0	30	92	91	93
75	AV Master #1	70	50	0	27	92	90	93
76	AV Master #1	150	35	150	30	91	92	94

Source: Base Met (2017a)

13.2.1.9 Analysis

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

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

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

- **Recovery** - The optimized flotation recovery was achieved with a primary P_{80} grind size of 150 µm, 8-minute laboratory float time, PAX/$CuSO_4$/MIBC reagent addition, and a 30% mass pull. The average gold and silver recovery to the rougher concentrate was 92.6% and 87.3%, respectively. Using WOL results, subsequent leach recoveries of 90.4% Au and 94.7% Ag were predicted at a P_{80} regrind size of 20 µm. Overall recoveries of 83.7% Au and 82.6% Ag were used for the analysis. Based on Tests 118 and 121 in Table 3.5, WOL recovery was projected to be 90.8% Au and 88.6% Ag. This was achieved with a primary P_{80} grind size of 25 µm, 250 g/t $Pb(NO_3)_2$ addition and a CIL configuration.

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

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

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

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

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

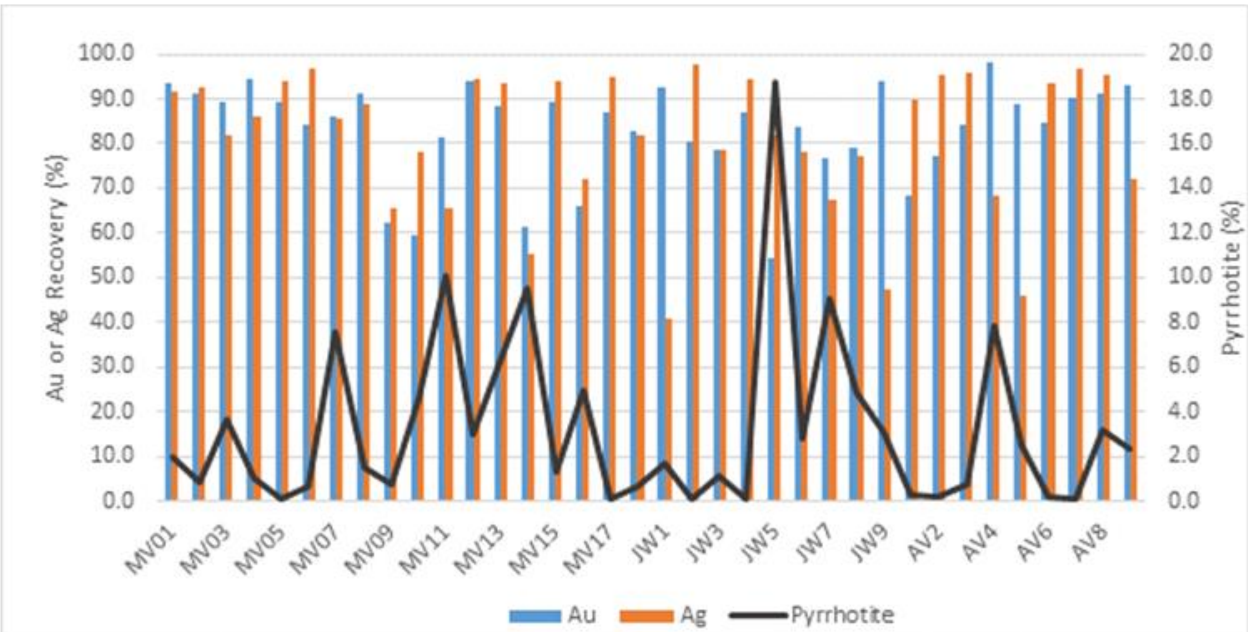
- **Flotation Variability** - There were several composites with very low flotation performance for gold. Further analysis of the variability data indicated that only TOC levels and gold feed grade had a small influence on results, but much of the variance in the recovery data remained. The overall performance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

  Pyrrhotite levels varied significantly in the deposit, for some samples representing most of the sulphide mineral mass. Pyrrhotite is very reactive and oxidizes rapidly, degrading flotation performance. Stockpiling ores containing pyrrhotite would also likely have a negative impact on flotation. The effect of pyrrhotite on flotation performance is shown in Figure 13.6.

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

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

Figure 13.6: Pyrrhotite Content vs. Flotation Recovery



Source: Base Met (2017)

13.2.2 **Comminution Testing**

SAG Mill Comminution (SMC) tests and Bond ball mill work index tests were completed on each variability composite. A summary of results is shown in Table 13.7. The SMC test results were on the hard end of the spectrum, mostly measuring between 80th and 95th hardness percentiles in the JK database.

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

Table 13.7: Comminution Results for BL0084 Variability Composites

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

Source: Base Met (2017a)

Two sets of master composites were generated for Marc, AV, and JW testing. One set of master composites was constructed exclusively for comminution testing to determine the crushing and abrasion indices for each zone. The results are summarized in Table 13.8.

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

Table 13.8: Crushing Work Index and Abrasion Index by Zone

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

Source: Base Met (2017a)

13.2.3 Cyanide Destruction Testing

Cyanide destruction testing was conducted on the three master composites. Optimization testing was conducted on leach tailings from Marc Master at a P_{80} of 44 µm. These conditions were applied to AV and JW leach tailings. Tests were also then conducted on the three master composites at a P_{80} of 25 µm. A summary of conditions and results are shown in Table 13.9.

Table 13.9: Master Composite Cyanide Destruction Test Results

Test	Particle Size K_{80} (µm)	Retention Time (mins)	Reagents Used		No of Displacements	Solution Concentration (ppm)			
			SO_2 (g/g CN_{WAD})	Cu (mg/L)		CN_{WAD}	Cu	Fe	Calc' CN_{TOT}
D1 - Marc	44	57	6	0	3.2	135.0	49.1	104.6	427.2
D2 - Marc	44	54	6	600	3.4	1.8	1.71	0.0	1.9
D3 - Marc	44	54	5	810	2.2	3.0	6.67	0.0	3.0
D4 - Marc	44	55	5	300	2.4	1.2	2.61	0.0	1.2
D5 - Marc	44	84	6	300	3.2	1.0	1.40	0.0	1.0
D6 - Marc	44	86	5	300	3.2	1.7	2.97	14.8	43.1
D7 - Marc	44	78	4	300	3.5	0.9	2.19	0.5	2.4
D8 - AV	41	69	4	300	4.6	1.4	3.09	0.0	1.4
D9 - JW	40	86	4	300	3.7	1.6	4.69	0.0	1.6
D10 - Marc	21	86	4	300	2.9	48.2	90.4	30.1	132.2
D11 - AV	23	86	4	300	2.6	61.3	98.6	6.2	78.6
D12 - JW	25	86	4	311	1.6	55.3	97.8	38.7	163.5
D13 - JW	25	86	4	515	1.6	78.7	129.8	8.6	102.8

Source: Base Met (2017a)

Initial tests on Marc Master leach tailings, at a P_{80} of 44 µm, indicated 300 mg/L copper, SO_2 to CN_{WAD} ratios of four and six and about 80 minutes' retention time were required to reduce CN_{WAD}

to about 1 ppm. When these conditions were applied to JW and AV leach tailings at 40 µm, the final CN_{WAD} concentration was about 1.5 ppm. When conditions were applied to leach tailings at 25 µm, the CN_{WAD} concentrations in the tailings were quite high, at 48 to 79 ppm. Additional optimization testing was carried out in BL0184 to reduce these levels.

13.2.4 Solid-Liquid Separation Testing

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

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

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

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

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

13.2.5 Optimized Test Work – BL0184

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

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

13.2.5.1 Sample Selection

Marc zone test work was carried out using master composite samples generated during the BL0084 test program. Additional AV and JW composites were created for this test program to enhance the variability datasets within each deposit. Four variability composites were also created for the 141 zone, a deposit projected to encompass approximately 4% of the total tonnage mined. The chemical composition of each sample is shown in Table 13.10.

Table 13.10: Head Assay Data for BL0184 Variability Composites

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

Source: Base Met (2017b)

AV and JW master composites were also generated from the variability composites. The composition of each composite was developed with direction from the Project geologist in order to ensure that the anticipated lithologies within each deposit were well represented in the masters. Based on the FS mine plan, a few variability samples from BL0084 were also included to improve spatial representation. The composition of each master composite is summarized in Table 13.11 and the resulting head grades are show in Table 13.12.

Table 13.11: Composition of BL0184 Master Composites

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

Source: Base Met (2017b)

Table 13.12: Head Assay Data for BL0184 Master Composites

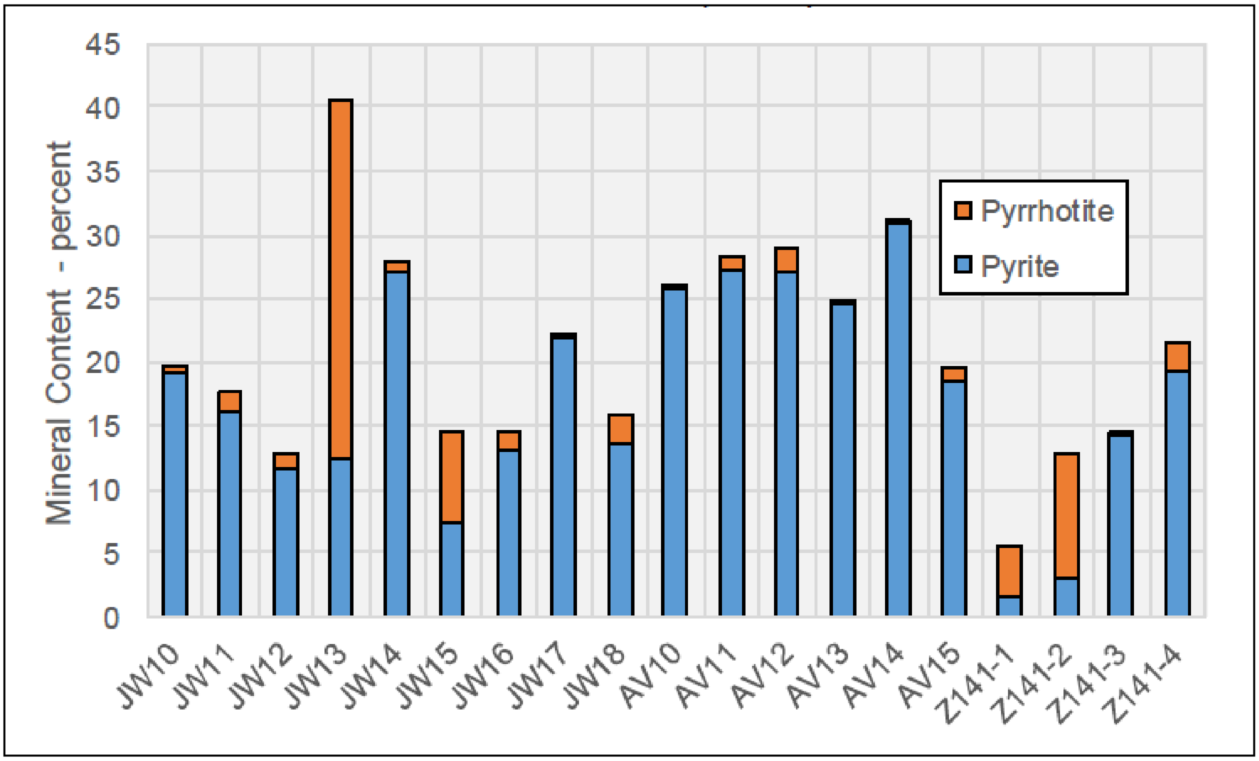
Composite ID	Chemical Content										
	Au (g/t)	**Ag (g/t)**	**S (%)**	**Fe (%)**	**TOC (%)**	**Cu (g/t)**	**Pb (g/t)**	**Zn (g/t)**	**As (g/t)**	**Sb (g/t)**	**Te (g/t)**
Marc Master Composite - Head 3	7.63	33	8.6	8.37	0.07	303	181	2,510	452	83	46
Marc Master Composite - Head 4	8.34	40	8.9	8.60	0.05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

Source: Base Met (2017b)

13.2.5.2 Mineralogical Characterization

The mineralogy of each variability sample was determined by QEMSCAN using the BMA protocol. The analysis was focused to determine sulphide mineral species present. Figure 13.7 shows the iron sulphide content in each sample. Iron sulphides represent the majority of the sulphur in the sample, averaging about 99%. Most samples consist primarily of pyrite; however, there are a few samples that have high levels of pyrrhotite, namely JW13, JW15, Z141-1, and Z141-2.

Figure 13.7: Iron Sulphide Content in BL0184 Variability Composites



Source: Base Met (2017b)

13.2.5.3 Comminution

Bond ball mill work index testing at a closing screen of 106 µm was carried out on AV and JW variability composites where sufficient sample was available. The results are summarized in Table 13.13. The new AV samples were found to be moderately hard while the JW samples were found to be hard to very hard.

Table 13.13: Bond Ball Mill Work Index Results for BL0184 Variability Composites

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

Source: Base Met (2017b)

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

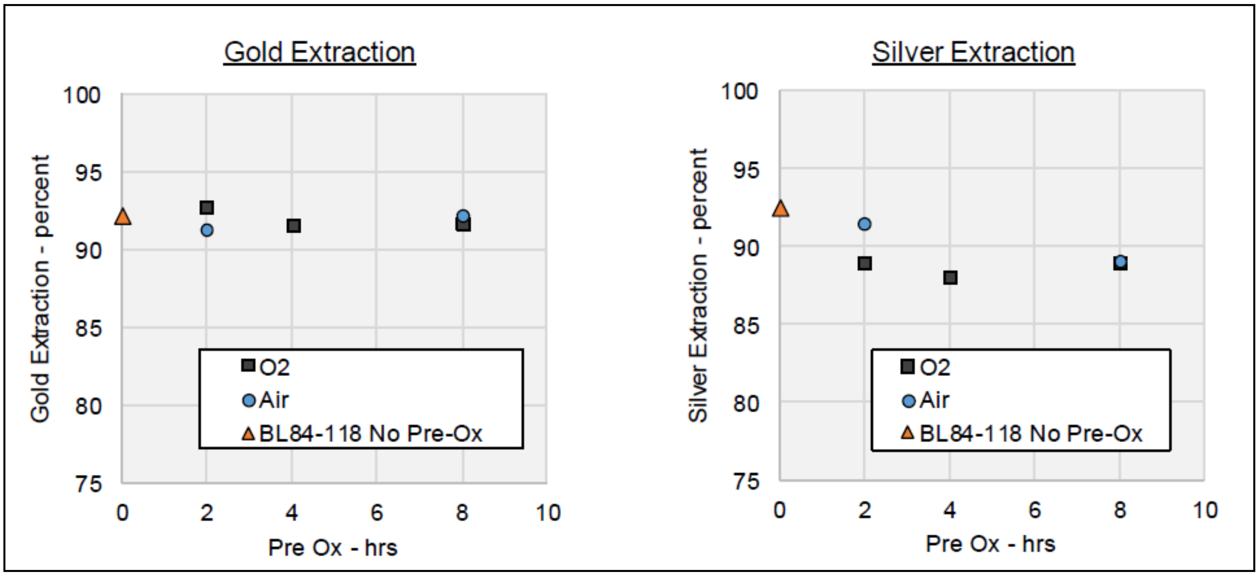
13.2.5.4 Pre-Oxidation Test Work

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

Using Marc Master Composite, pre-oxidation times of eight, four, and two hours were examined. Oxygen was used for the sparging gas. Air was also examined as the oxidation gas for eight and two hours. A graphical presentation of the related tests is displayed in Figure 13.8 and Figure 13.9, along with a test from the previous program with no pre-oxidation.

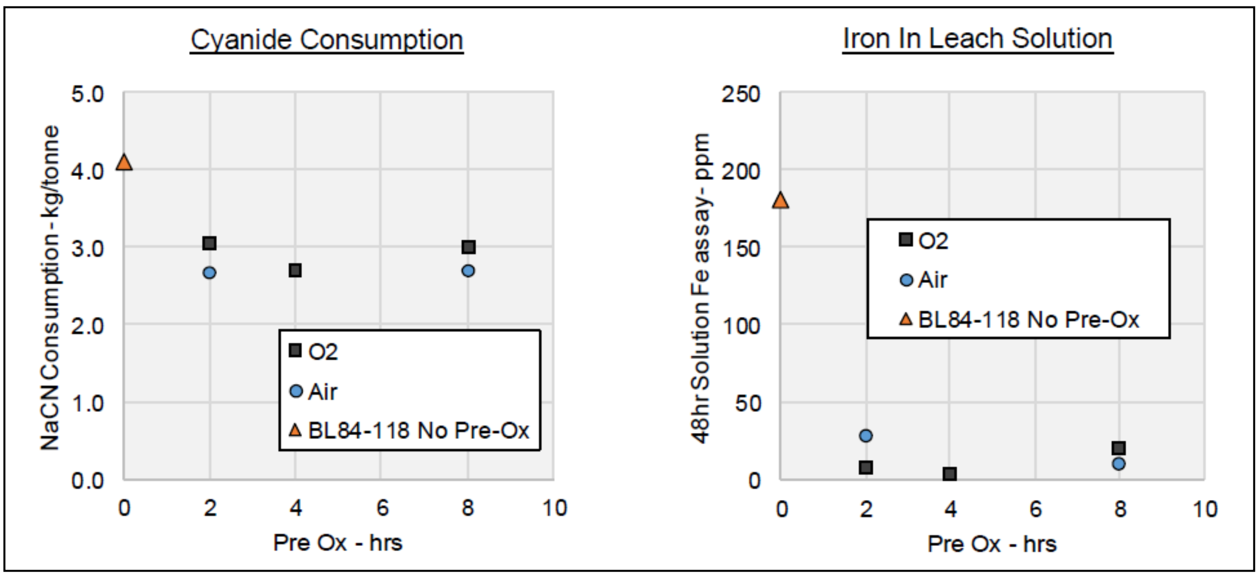
Based on the reduction of cyanide consumption and reduction of iron in the effluent, the use of a two-hour pre-oxidation step was included in the process design. Oxygen was selected for use in the pre-oxidation stage because it was shown to be beneficial to gold extraction in the kinetic leaching stages.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

Figure 13.8: The Effect of Pre-Oxidation on Gold and Silver Extraction



Source: Base Met (2017b)

Figure 13.9: The Effect of Pre-Oxidation on Cyanide Consumption and Iron Content



Source: Base Met (2017b)

13.2.5.5 CIL Oxygen Consumption

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

CIL. The results were combined with the projected plant mass balance to generate the oxygen consumptions estimates shown in Table 13.14.

Table 13.14: Projected Oxygen Consumption for Pre-Oxidation and CIL

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

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

13.2.5.6 Leach Variability Test Work

A leach test was performed on each of the variability composites based on the results from Marc master composite testing. The test conditions included: two-hour pre-oxidation, 500 ppm NaCN in solution, pH 11, primary grind P_{80} of 25 µm, 30 g/L carbon and 250 g/t $Pb(NO_3)_2$. The results are presented in Table 13.15.

Table 13.15: Leach Results for BL0184 Variability Composites

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

Source: Base Met (2017b)

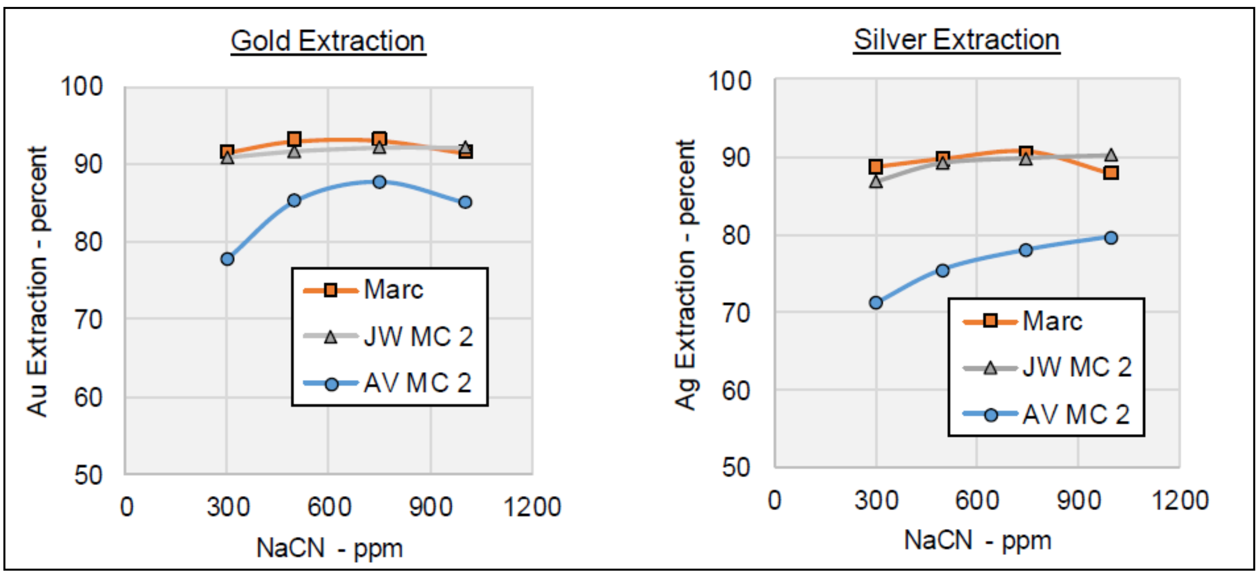
13.2.5.7 Leach Optimization Test Work

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

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

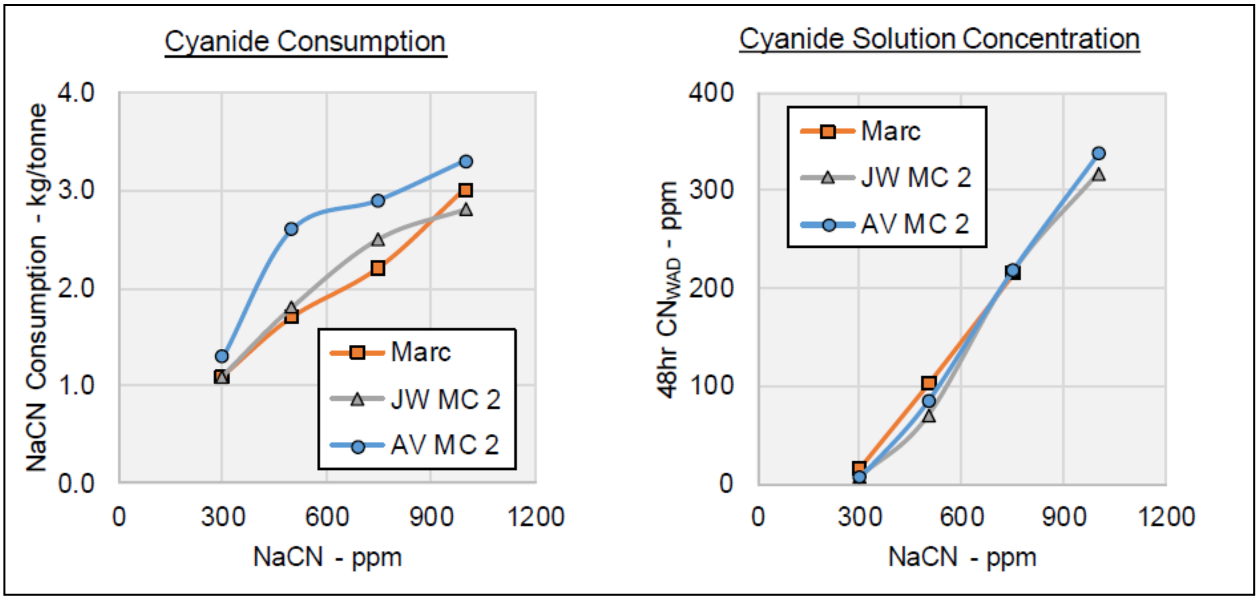
The effect of cyanide dosage on gold and silver extraction rate is presented in Figure 13.10. The resulting cyanide consumption and effluent levels are summarized in Figure 13.11. The trade-off between gold and silver extraction and cost of the process will be dominated by extraction of gold at current metal and reagent pricing. Current data would suggest operating at 500 ppm cyanide for the Marc zone and 750 ppm for the other zones.

Figure 13.10: The Effect of Cyanide Concentration on Gold and Silver Extraction



Source: Base Met (2017b)

Figure 13.11: The Effect of Cyanide Concentration on Cyanide Consumption and Effluent Level

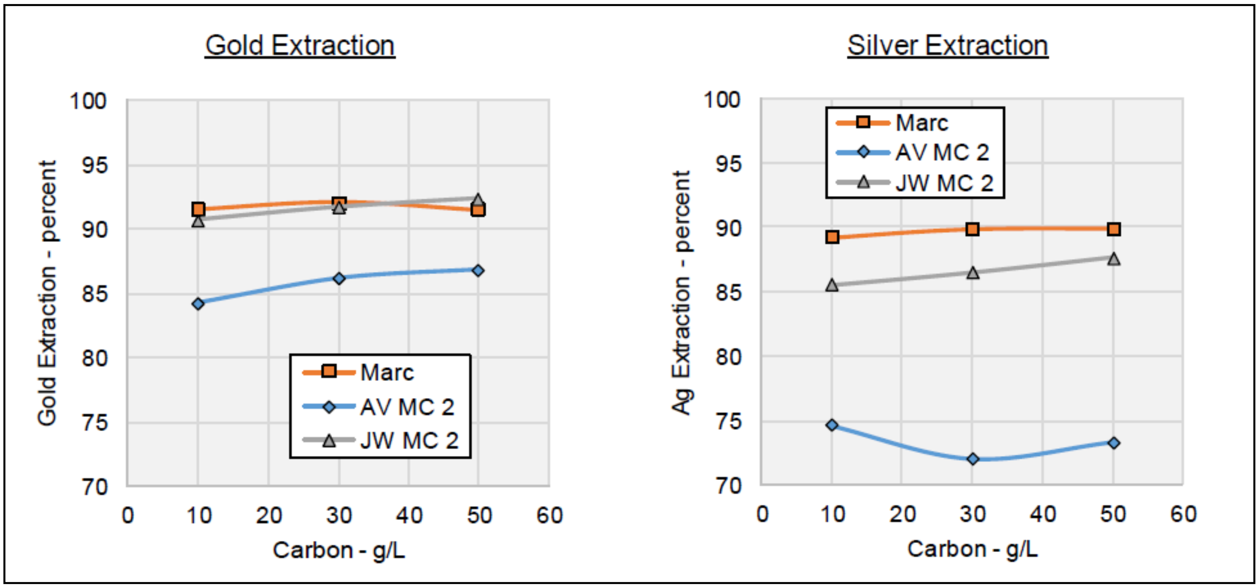


Source: Base Met (2017b)

A series of tests were performed on each composite to determine the impact of carbon loading. Three levels were tested on each composite; 10, 30, and 50 g/L. The effect on gold and silver extraction is shown in Figure 13.12. The resulting cyanide consumption and effluent levels are summarized in Figure 13.13. The carbon loading rate had a measurable effect on gold and silver extraction rates, with higher loading values resulting in the best extraction rates.

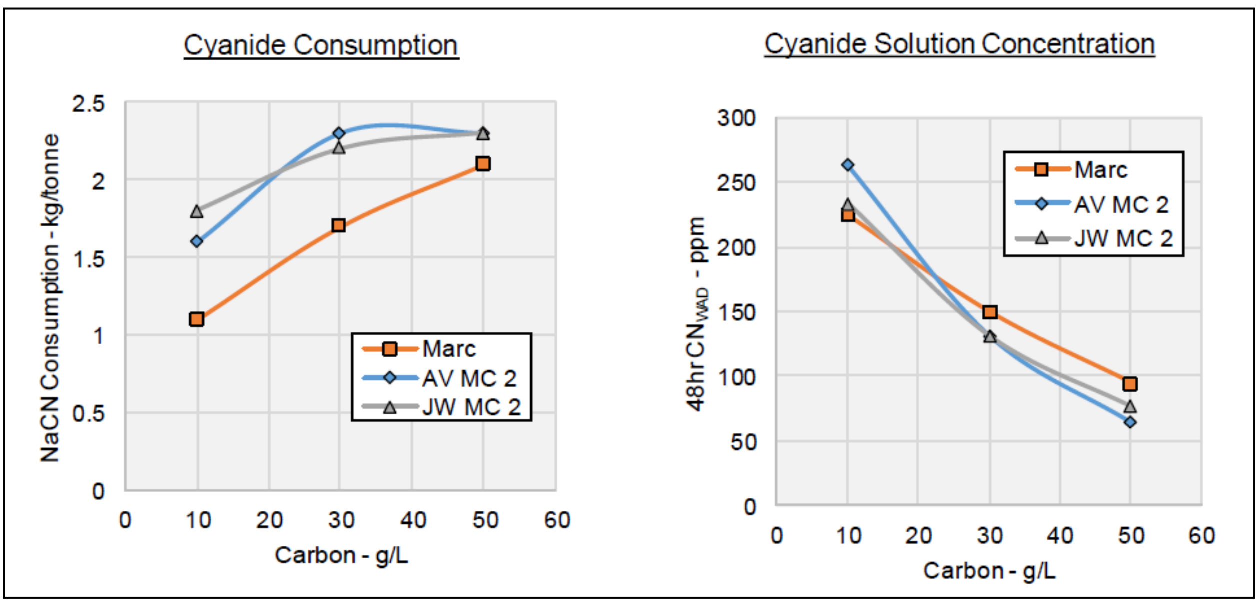
Carbon loading also affected the cyanide consumption and the final effluent CN_{WAD} values. Cyanide consumption was reduced as the carbon loading was reduced. Conversely, the final CN_{WAD} values were inversely proportional to carbon loading. The minor increase in metallurgical performance and the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

Figure 13.12: The Effect of Carbon Loading on Gold and Silver Extraction



Source: Base Met (2017b)

Figure 13.13: The Effect of Carbon Loading on Cyanide Consumption and Effluent Level



Source: Base Met (2017b)

13.2.5.8 Cyanide Destruction Testing

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

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

Cyanide destruction testing ran for seven days using the SO_2/air process. The feed had a measured CN_{WAD} level of 80 ppm and copper and iron levels of 41 and 16 ppm, respectively. The new conditions represented a large drop in CN_{WAD} and iron. Previous leach conditions resulted in CN_{WAD} levels over 269 ppm and iron levels of 200 ppm. The results are summarized in Table 13.16.

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

Table 13.16: Cyanide Destruction Test Results on Marc Master Composite

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

Source: Base Met (2017b)

13.3 PROCESS SELECTION

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

The levels of TOC observed in the majority of variability composites would not typically interfere with the leaching process; however, there were a few samples with higher levels of TOC: MV9 and MV10. If the carbon has high activity levels it may interfere with cyanide leaching and adsorption of gold and silver.

Based on test work results, some variability samples contained highly active organic carbon, which drastically affected gold leach recovery, including a test with decreasing gold recovery over time. This indicates that organic carbon may be preg-robbing, reducing the amount of leached gold adsorbing onto activated carbon. The CIL process was selected to mitigate the risk of this preg-robbing material. Subsequent test work (see in Table 13.5) found improved recoveries with the addition of carbon during the leach.

13.4 RELEVANT RESULTS FOR PROCESS DESIGN

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

13.4.1 Comminution Design Criteria

Comminution test work was completed on 43 variability samples, representing the Marc, AV, and JW deposits. Ore hardness was highly variable with Bond ball mill work indices ranging from 14.8 to 27.3 kWh/t. The grinding circuit was designed to accommodate this variability, with mill sizing capable of handling the 75th percentile value of 21.7 kWh/t. A summary of the key comminution design criteria is presented in Table 13.17.

Table 13.17: Key Comminution Design Criteria

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

Source: JDS (2017)

13.4.2 Leach Design Criteria

Extensive cyanide leach test work was carried out on variability and master composites from the Marc, AV, and JW deposits. As a result of the optimization testing in BL0184, the leach process will include a pre-oxidation stage prior to CIL, which will reduce cyanide consumption, as well as cyanide destruction costs. The optimized conditions included two-hour pre-oxidation, 500 - 750 ppm NaCN concentration, pH 11, a primary grind P_{80} of 25 µm, 30 - 50 g/L carbon and 250 g/t $Pb(NO_3)_2$. A summary of the key process design criteria for the leach area is presented in Table 13.18.

Table 13.18: Key Leach Circuit Design Criteria

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

Source: JDS (2017)

13.4.3 Cyanide Destruction Design Criteria

The cyanide destruction test work conditions were optimized in BL0184 using Marc master composite sample at the optimized leach test conditions. A summary of the key process design criteria for cyanide destruction is presented in Table 13.19.

Table 13.19: Key Cyanide Destruction Circuit Design Criteria

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

Source: JDS (2017)

13.5 PRELIMINARY RECOVERY ESTIMATE

The recoveries used for each zone were estimated based on optimized leach results from Marc Master Composite, AV Master Composite #2, and JW Master Composite #2. Cyanide concentration and carbon loading were used to find a balance between operating costs and metallurgical recovery. Marc was found to perform best at a NaCN concentration of 500 ppm, while AV and JW required a higher NaCN concentration of 750 ppm. Table 13.20 presents the preliminary recovery estimates used for economic projections. The 141 zone recoveries were calculated using a weighted average of test work results, while the overall recovery is a weighted average of recovery by zone and projected zone tonnage.

Table 13.20: Preliminary Recovery Projections

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

Source: JDS (2017)

14.0 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

The mineral resource model prepared by ACS utilised a total of 683 drill holes, 190 of which were drilled by IDM, 12 in 2014, 62 in 2016 and 116 in 2017. The resource estimation work was completed by Dr. Gilles Arseneau, P. Geo. (APEGBC) an appropriate independent "qualified person" within the meaning of NI 43-101. The effective date of the Mineral Resource statement is June 15, 2018.

This section describes the resource estimation methodology and summarizes the key assumptions considered by ACS. In the opinion of ACS, the resource evaluation reported herein is a reasonable representation of the gold and silver mineral resources found at the Red Mountain gold project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (2003) and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

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

14.2 RESOURCE ESTIMATION PROCEDURES

The resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Validation of wireframe models for the boundaries of the gold mineralization;
- Definition of resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades; and
- Preparation of mineral resource statement.

14.3 DRILL HOLE DATABASE

The drilling database consists of historical drilling most of which has been carried out by LAC in the early 1990s. Between 2000 and 2001, North American Metals Corporation (NAMC) relogged all of the mineralized intervals and carried out an extensive database validation of the drill database. Banks Island Gold drilled two holes in the Marc zone in 2013 and IDM drilled five holes in the deposit in 2014, three holes targeting the 141 zone and two holes targeting the AV zone. IDM drilled 62 holes in 2016 to better defined the mineralization, collect some samples for metallurgical tests and upgrade some of the inferred mineralization to indicated. IDM also drilled seven exploration holes targeting other areas on the Red Mountain gold project in 2014.The 2017 drill program concentrated on upgrading the mineral resource classification for the Marc, AV and JW and better defining the SF zone. Table 14.1 summarizes the drill holes used for each mineralized zone estimated.

Table 14.1: Drill Hole Used in Resource Estimate Update

Zone	Number of Holes	Metres
Marc	222	31059
Marc FW	28	5389
Marc Outliers	17	2673
Marc NK	10	2056
AV	74	19958
JW	48	12523
JW HW	8	2442
JW FW	3	672
141	78	21986
Smit	78	26936
Bray	20	17147
Cambria	18	4439
Chicka	5	892
SF	13	4580
Total	622	152752

There is a total of 58,057 records in the assay database, of these 3,232 represent samples taken from the mineralized horizons.

Table 14.2 summarises the basic statistical data for all the assays in the database. Table 14.3 summarises the gold assays contained within the mineralized zones and Table 14.4 summarises the silver assays.

Table 14.2: Basic Statistical Information for all Assays in Database

Zone	All	All
Assays	**Au (g/t)**	**Ag (g/t)**
Valid cases	72043	72043
Mean	0.91	3.10
Variance	79.74	657.77
Std. Deviation	8.93	25.65
Variation Coefficient	9.84	8.29
Minimum	0.00	0.00
Maximum	1400.00	2152.00
1st percentile	0.01	0.00
5th percentile	0.01	0.00
10th percentile	0.02	0.00
25th percentile	0.03	0.10
Median	0.11	0.40
75th percentile	0.38	1.10
90th percentile	1.23	3.00
95th percentile	2.87	8.79
99th percentile	14.20	50.80

Table 14.3: Basic Statistical Information of Gold Assays within the Mineralized Zones

Zone	Marc	AV	JW	141	All Other Zones
Assays	**Au (g/t)**	**Au (g/t)**	**Au (g/t)**	**Au (g/t)**	**Au (g/t)**
Valid cases	1942	684	250	338	1098
Mean	13.06	11.01	8.83	4.33	3.59
Variance	1570.25	2964.55	134.37	112.68	48.94
Std. Deviation	39.63	54.45	11.59	10.62	7.00
Variation Coefficient	3.03	4.95	1.31	2.45	1.95
Minimum	0.00	0.04	0.09	0.01	0.02
Maximum	1400.00	1320.66	75.50	169.30	110.50
1st percentile	0.10	0.09	0.26	0.09	0.03
5th percentile	0.56	0.60	0.71	0.28	0.18
10th percentile	1.04	1.12	1.32	0.45	0.33
25th percentile	2.72	2.48	2.72	1.09	0.90
Median	5.65	4.30	5.06	2.24	1.80

75th percentile	13.09	7.92	9.37	4.00	3.76
90th percentile	28.03	17.79	18.44	9.29	8.22
95th percentile	45.71	27.40	33.76	15.54	11.71
99th percentile	109.46	110.70	60.89	38.68	26.10

Table 14.4: Basic Statistical Information of Silver Assays within the Mineralized Zones

Zone	Marc	AV	JW	141	All Other Zones
Assays	**Ag (g/t)**	**Ag (g/t)**	**Ag (g/t)**	**Ag (g/t)**	**Ag (g/t)**
Valid cases	1942	684	250	338	1098
Mean	49.49	22.43	31.62	7.41	7.25
Variance	14666.16	1725.77	6566.77	297.02	1687.72
Std. Deviation	121.10	41.54	81.04	17.23	41.08
Variation Coefficient	2.45	1.85	2.56	2.33	5.67
Minimum	0.00	0.00	0.00	0.00	0.00
Maximum	2152.00	504.20	889.00	203.30	789.60
1st percentile	0.05	0.05	0.00	0.00	0.00
5th percentile	1.30	0.25	0.05	0.00	0.00
10th percentile	2.98	1.12	0.05	0.10	0.05
25th percentile	8.40	4.50	3.39	0.80	0.29
Median	21.45	10.25	11.55	2.40	0.80
75th percentile	44.23	23.60	28.37	6.65	2.66
90th percentile	94.91	52.05	57.75	16.40	9.38
95th percentile	174.11	79.10	124.50	29.14	24.31
99th percentile	557.59	221.68	494.64	76.83	100.12

14.4 DESIGN OF MODELLING CRITERIA

A significant amount of time and effort was invested during the 2000 field season to develop modelling criteria for the mineralization at Red Mountain. Areas of investigation included general lithology, nature of sulphide occurrences, relationship of pyrite to gold grade and structural control on mineralization.

The results of the studies suggested that the following were important modelling criteria:

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

Average pyrite content in lower gold grade sections of the stockwork is at least 4%. Outside the stockwork limits, sulphide occurs as disseminations and sparse micro-veinlets with an average pyrite content of 1.5%.

3. The shift from a pyrite-dominated stockwork to a pyrrhotite-dominated alteration halo is sharp and often corresponds to a 1 g/t gold outline, except in rare cases where pyrrhotite abundance, style and gold content mimics the pyrite stockwork.
4. The cumulative thickness of pyrite in a given interval has the best correlation to gold grade regardless of the width or number of veins and represents the most important data that can be collected to constrain gold distribution. The data collected suggest that cumulative pyrite thickness could be used to delineate high and low-grade domains.
5. Brecciation of pyrite veins is also related to gold distribution and can be measured by qualitative measurements, although in practical terms such measurements are time-consuming and very subjective.

After the compilation of the 2016 drilling, IDM decided to review and modify the geological wireframes defining the mineralized zones at Red Mountain. While a similar geological approach to the 2000 modelling was followed, a stronger emphasis was placed on including grade that may have been excluded because of strict geological modelling rules. Furthermore, the base cut-off was raised from a nominal 1 g/t to 2.5 g/t. Some of the lesser defined zones remained modelled at a 1 g/t cut-off.

Following the 2017 drilling, IDM re-evaluated the geological model for the mineralization at Red Mountain and proposed a different geological model that incorporated both stratigraphic as well as structural controls. The re-interpretation resulted in only minor changes to the Marc, AV and JW but to significant changes to the 141, and the 132 Zone (now named Smit). The new interpretation seems to open new ground for exploration but still needs additional drilling to be fully confirmed.

14.5 SOLID MODELLING

New three-dimensional solids were generated for the mineralized zones using the following process:

- Cross-sections were plotted at 25 m intervals showing all surface and underground diamond drill holes. The sections were plotted with one side of the drill hole trace showing the primary lithology and its modifiers, and the other side showing the assay interval and gold grade.

ACS reviewed all of the three-dimensional solids prior to resource estimation and agrees with the general modelling criteria selected. The outlines are generally based on gold cut-off that for the most part coincide with the limits of pyrite and pyrrhotite stockwork. The boundaries of the stockwork are very abrupt in some places and gradational into the wall rock in others. The stockwork outlines often, but not always, corresponded to areas of intense quartz sericite alteration that give the rock a bleached appearance.

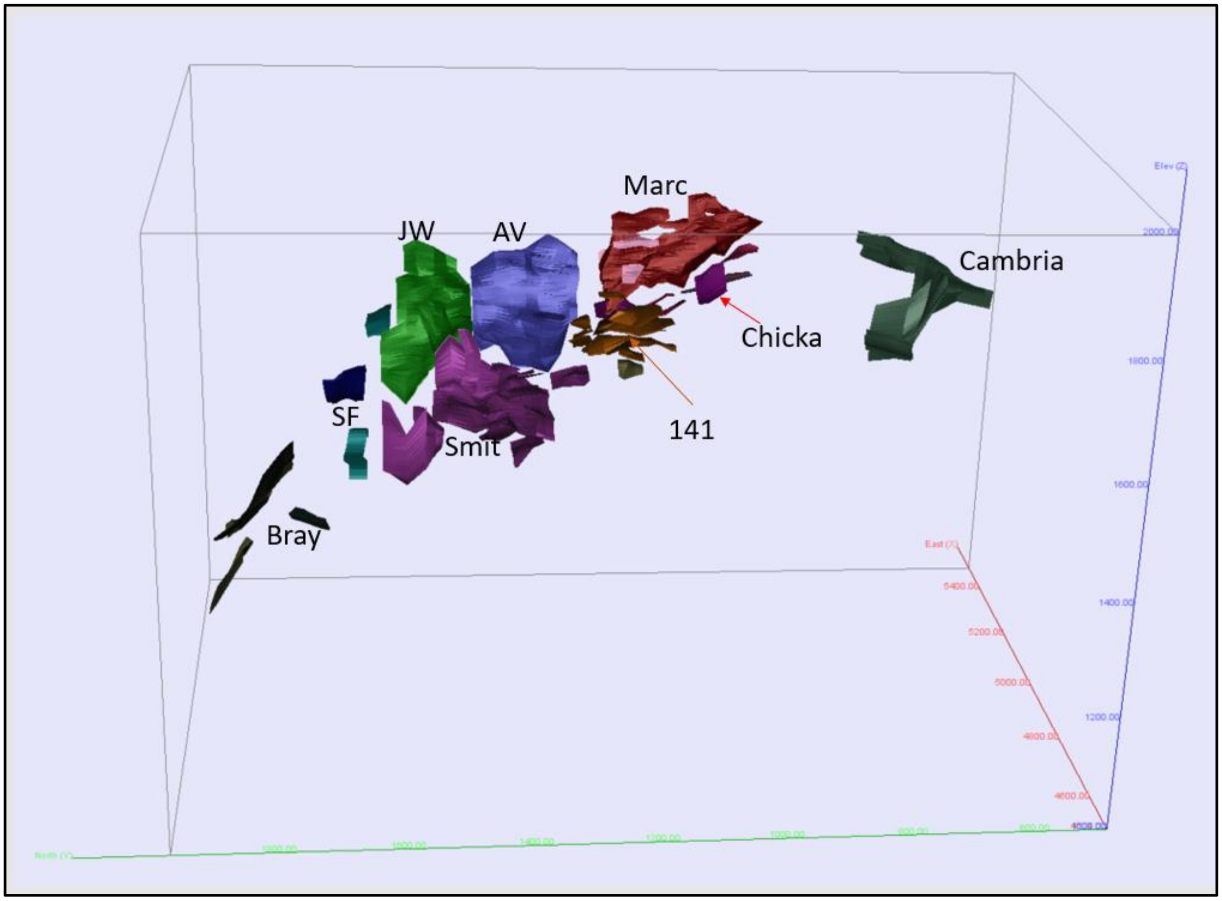
The outlines were derived from vertical sections in Geovia software. The vertical section outlines were digitized as closed polylines that were snapped to the actual 3D locations of the drill holes. The closed polylines were then "wobbled" (splined) in order to smooth the transition to off-section drill holes while maintaining the integrity of the interpretation.

Each wireframe was assigned a unique rock code as outlined in Table 14.5 and shown on Figure 14.1.

Table 14.5: Rock Codes Assigned to Wireframes

Zone	Rock Code
Marc	101
AV	201
JW	301
Marc Footwall	102 & 105
Marc Outliers	103
Marc NK	104
JW FW	302
JW HW	303
141	401 & 402
Smit	501
Cambria	601
Chicka	701
Bray	801 & 802
SF	901 & 902

Figure 14.1: 3D Perspective View Looking East Showing Location of Mineralized Zones



Note: Markers on axes are 200 m apart
Source: ACS (2018)

14.6 BULK DENSITY

The bulk density of the Red Mountain gold deposits has been evaluated by LAC in 1993-94 when 4,225 specific gravity determinations estimated from drill submitted to the Eco-Tech lab in Stewart. In 2000, NAMC collected 58 samples that were subjected to bulk density analysis. IDM has been routinely collecting bulk density data as part of their drilling programs. Collectively, there are 4,810 bulk density readings in the drill hole database, 1,452 are from sample intervals within the solids used for resource calculation. Average specific gravity values for different subsets of the entire data set are given in Table 14.6.

Table 14.6: Bulk Density Sample Results

Zone	# Samples	Minimum	Maximum	Average
All samples	4810	1.44	4.87	2.88
Within mineralized zones	1452	2.03	4.87	3.00
Marc zone	1022	2.03	4.43	2.96
AV zone	225	2.73	4.42	3.01
JW, JW FW & JW HW	53	2.67	4.39	3.00
Marc FW	14	2.73	3.86	3.05
Marc Outliers	15	2.74	3.05	2.90
141	26	2.70	3.46	2.89
Smit	84	2.72	4.87	3.46
Cambria	11	2.96	4.10	3.28
Chicka	0	No Data	No Data	No Data
Bray	0	No Data	No Data	No Data
SF	2	3.66	3.66	3.66
Waste	3358	1.44	4.37	2.83

14.7 TOP CUT APPLIED TO ASSAY DATA

Block grade estimates may be unduly affected by high grade outliers. Therefore, assay data were evaluated for high grade outliers. Based on the analysis of the assay distribution, ACS decided that capping of high grade assays was warranted. ACS evaluated each of the mineralized lenses separately to define appropriate capping level for each zone prior to compositing. Table 14.7 summarises the various gold capping levels and Table 14.8 summarises the silver capping levels.

Table 14.7 Gold Assay Capping Levels

Rock Code	Zone Name	Capping Level (g/t)	Number of Assays Capped
101	Marc	75	32
102 & 105	Marc FW	75	1
103	Marc Outliers	No Capping	0
104	Marc NK	No Capping	0
201	AV	55	22
301	JW	35	12
302 & 303	JW FW & HW	35	2
401	141	40	3
501	Smit	No Capping	0
601	Cambria	No Capping	0
701	Chicka	No Capping	0
801 & 802	Bray	55	1
901 & 902	SF	55	1

Table 14.8 Silver Assay Capping Levels

Rock Code	Zone Name	Capping Level (g/t)	Number of Assays Capped

101	Marc	500	21
102 & 105	Marc FW	No Capping	0
103	Marc Outliers	No Capping	0
104	Marc NK	No Capping	0
201	AV	200	7
301	JW	200	7
302 & 303	JW FW & HW	200	3
401	141	200	1
501	Smit	No Capping	0
601	Cambria	No Capping	0
701	Chicka	No Capping	0
801 & 802	Bray	No Capping	0
901 & 902	SF	No Capping	0

14.8 COMPOSITE STATISTICS

14.8.1 Composite Statistics

All assay data were composited to a fixed length prior to estimation. ACS evaluated the assay lengths for the various deposits and found that most samples had an average length of less than 1.5 m. ACS therefore decided to composite all assay data to 1.5 metres prior to estimation. Table 14.9 summarizes the basic statistical data for capped gold composites used in the resource estimates and Table 14.10 shows the statistics of the silver composited data.

Table 14.9: Descriptive Statistics of 1.5 m Gold Composites

	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)	Au (g/t)
Zone	**Marc**	**AV**	**JW**	**141**	**102 & 105**	**103**	**104**	**302 & 303**	**501**	**601**	**701**	**801 & 802**	**901 & 902**
Valid cases	1365	478	168	268	50	35	18	25	492	28	14	77	80
Mean	10.96	7.62	7.90	3.58	5.90	4.94	7.82	6.94	2.46	3.68	6.73	3.77	3.18
Variance	158	84.05	47.88	19.09	114	34.50	33.00	29.11	6.16	19.22	40.51	29.38	37.13
Std. Deviation	12.57	9.17	6.92	4.37	10.67	5.87	5.74	5.39	2.48	4.38	6.36	5.42	6.09
Variation Coefficient	1.15	1.20	0.88	1.23	1.81	1.19	0.73	0.78	1.01	1.19	0.95	1.44	1.92
Minimum	0	0.04	0.78	0.00	0.16	0.08	0.68	0.12	0.00	0.29	1.13	0.01	0.00
Maximum	75	55.00	35.00	35.14	74.98	33.74	20.81	22.69	24.48	21.51	19.18	38.45	46.44
1st percentile	0.17	0.015	0.78	0.00	0.16	0.01	0.86	0.14	0.13	0.01	1.22	0.01	0.00
5th percentile	0.72	0.85	1.29	0.00	0.42	0.13	1.61	0.15	0.40	0.39	1.51	0.07	0.02
10th percentile	1.43	1.58	2.12	0.37	0.93	0.58	1.67	1.12	0.65	0.67	1.62	0.31	0.04
25th percentile	3.10	2.96	3.40	1.09	2.23	1.78	3.24	2.97	1.09	1.24	2.46	0.81	0.14
Median	6.54	4.79	5.40	2.31	3.48	3.65	6.04	4.94	1.69	1.66	3.39	2.31	1.30
75th percentile	13.51	7.88	9.71	4.32	6.02	6.53	11.40	10.56	2.83	4.89	12.80	4.55	4.02
90th percentile	26.12	18.05	16.67	8.82	9.23	9.62	16.73	13.89	5.39	8.96	16.96	8.09	7.98
95th percentile	39.33	23.66	25.22	12.55	18.43	17.13	116.95	20.25	7.65	16.81	18.29	15.36	13.11
99th percentile	61.47	55.00	34.99	22.49	21.36	26.70	20.03	20.73	12.84	18.69	19.00	23.77	23.54

Table 14.10: Descriptive Statitics of 1.5 m Silver Composites

	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)
Zone	**Marc**	**AV**	**JW**	**141**	**102 & 105**	**103**	**104**	**302 & 303**	**501**	**601**	**701**	**801 & 802**	**901 & 902**
Valid cases	1365	478	168	268	50	35	18	25	492	28	14	77	80
Mean	44.09	20.79	23.92	6.91	10.92	21.64	8.83	43.02	1.58	4.07	2.54	9.62	3.85
Variance	4834.00	753.33	1187.3	178.75	2198.0	4900.00	70.62	4450.0	24.93	12.23	10.76	238.07	236.43
Std. Deviation	69.53	27.45	34.46	13.37	46.88	70.00	8.40	66.71	4.99	3.50	3.28	15.43	15.38
Variation Coefficient	1.58	1.32	1.44	1.94	4.29	3.24	0.95	1.55	3.16	0.86	1..29	1.60	4.00
Minimum	0.00	0.00	0.00	0.00	0.30	0.00	0.05	0.00	0.00	0.53	0.20	0.03	0.00
Maximum	500.00	200.00	200.00	106.09	332.93	416.29	31.79	185.00	72.33	15.80	11.30	75.72	122.03
1st percentile	0.05	0.05	0.00	0.00	0.30	0.00	0.24	0.00	0.00	0.63	0.21	0.11	0.00
5th percentile	1.82	0.36	0.05	0.00	0.30	0.00	0.99	0.00	0.05	0.70	0.24	0.60	0.00
10th percentile	3.43	1.71	0.05	0.00	0.64	0.40	1.05	0.00	0.05	0.96	0.25	0.60	0.00
25th percentile	10.26	5.20	4.51	0.71	0.99	1.80	3.23	2.46	0.26	1.42	0.32	1.11	0.00
Median	23.60	11.85	13.93	2.31	1.65	4.69	6.69	4.83	0.51	3.08	0.80	3.44	0.16
75th percentile	45.86	25.45	25.66	6.79	5.42	14.74	10.42	84.29	1.29	5.93	4.58	11.75	1.30
90th percentile	93.20	50.41	52.72	19.69	18.56	37.49	26.92	176.56	2.92	8.84	6.12	23.71	3.73
95th percentile	158.20	79.38	92.65	30.09	25.05	130.70	27.18	183.02	4.94	12.69	8.05	54.95	25.00
99th percentile	388.40	152.88	200.00	89.29	183.97	295.0	30.87	183.42	24.56	13.94	10.65	68.75	67.64

14.9 SPATIAL ANALYSIS

Spatial continuity of gold and silver was evaluated with correlograms developed using SAGE 2001 version 1.08. The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram is close to zero is called the "range of correlation" or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample or composite.

Directional correlograms were generated for composited data at 30 degree increments along horizontal azimuths. For each azimuth, correlograms were calculated at dips of 0, 30 and 60 degrees. A vertical correlogram was also calculated, using the information from these 37 correlograms. Sage then determines the best fit model using the least square fit method. The correlogram model is described by the nugget (Co), the variance contribution of the two nested structure (C1, C2) and the range of each of the structures.

Experimental correlograms were obtained for drill hole directions for which sufficient data existed for the mineralized zones at Red Mountain. The Marc zone is the most densely drilled and provides the greatest opportunity for determining the short-range character of the correlogram. Correlograms developed from the Main Marc zone was utilized for all the Marc lenses. Correlograms were also derived for AV, JW, 141 and Smit zones.

Because of the sparse drilling in the Cambria, Chicka, Bray and SF zones, ACS decided to estimate these zones with inverse distance to the second power (ID^2) as they contained insufficient data to develop any significant correlograms.

Table 14.11 summarises the correlogram parameters used to estimate gold and silver in the block model.

Table 14.11: Correlogram Parameters Used for Grade Estimation

Zone	Metal	Model Type	Nugget (C_0)	C_1 & C_2	Rotation			Range		
					(Z)	(Y)	(Z)	Rot X	Rot Y	Rot Z
Marc	Au	Exponential	0.2	0.452	-55	29	47	4	4	25
				0.347	-55	29	47	10	156	34
	Ag	Exponential	0.2	0.8	-66	26	61	6.5	38	21
				NA	NA	NA	NA	NA	NA	NA
AV	Au	Exponential	0.2	0.537	72	51	23	14	8	7
				0.262	72	51	23	16	12.5	41
	Ag	Exponential	0.25	0.75	14	-42	-14	52	26	8
				NA	NA	NA	NA	NA	NA	NA
JW	Au	Exponential	0.3	0.542	-80	14	49	4	22	4

				0.157	-80	14	49	12	89	154
	Ag	Exponential	0.1	0.748	-12	29	-5	4	12	74
				0.151	-12	29	-5	39	123	156
141	Au	Exponential	0.35	0.65	31	-86	-6	10	32	5
				NA	NA	NA	NA	NA	NA	NA
	Ag	Exponential	0.27	0.611	27	46	-17	27	60	5
				0.118	27	46	-17	28	120	78
Smit	Au	Exponential	0.5	0.5	22	34	4	85	18	7
				NA	NA	NA	NA	NA	NA	NA
	Ag	Exponential	0.2	0.57	-28	-13	55	15	80	18
				0.23	-28	-13	55	59	135	62

14.10 BLOCK MODEL

A 3D block model was created using Geovia GEMs Version 6.8.1 to represent the lithological and structural characteristics specific to the Red Mountain deposit. This model was used as a framework for the grade model, which relied on geostatistical analysis of the sample data and a detailed understanding of the geology to produce an estimate of the contained resource.

The parameters for the block model are listed in Table 14.12. Block model coordinates are in local grid coordinates to be consistent with historical data. Block size was set to 4 m x 4 m x 4 m to better define the mineralized zones and to stay consistent with previous resource estimates. The rock type element in the block model was coded for all zones using a 0.001% selection process. The rock and percent models were then updated with specific codes for each of the mineralized zones as outlined in Table 14.5 above. All waste blocks were assigned a default rock code of 99.

Table 14.12: Model Parameters for the Red Mountain Block Model

Coordinates		Origin Coordinates	Block Size (m)	Number of Blocks
Axis	Model			
Easting	Column	4500	4	250
Northing	Row	500	4	375
Elevation	Level	1000	4	250

Gold grades were interpolated within the individual zones using ordinary kriging or ID^2 and multiple passes as outlined in Table 14.13. Grades were only interpolated into blocks if the blocks had not been interpolated by a previous pass.

Table 14.13: Interpolation Parameters Used for Grade Interpolation

Zone	Pass	Rotation	Search Ellipse Size	No of composites	Max no per hole

		Z	Y	Z	X	Y	Z	Min	Max	
Marc	1	0	-75	0	30	30	10	5	15	3
Marc	2	0	-75	0	60	60	15	2	15	1
Marc	3	0	-75	0	20	20	20	2	15	none
AV	1	0	-50	0	30	30	10	5	15	3
AV	2	0	-50	0	60	60	15	2	15	1
AV	3	0	-50	0	20	20	20	2	15	none
JW	1	0	-36	0	30	30	10	5	15	3
JW	2	0	-36	0	60	60	15	2	15	1
JW	3	0	-36	0	20	20	20	2	15	none
141	1	0	33	0	30	30	10	5	15	3
141	2	0	33	0	60	60	15	2	15	1
141	3	0	33	0	20	20	20	2	15	none
Marc FW	1	40	50	0	30	30	10	5	15	3
Marc FW	2	40	50	0	60	60	15	2	15	1
Marc FW	3	40	50	0	20	20	20	2	15	none
Marc FW_2	1	40	-20	0	30	30	10	5	15	3
Marc FW_2	2	40	-20	0	60	60	15	2	15	1
Marc FW_2	3	40	-20	0	20	20	20	2	15	none
Marc Outlier	1	0	-60	0	30	30	10	5	15	3
Marc Outlier	2	0	-60	0	60	60	15	2	15	1
Marc Outlier	3	0	-60	0	20	20	20	2	15	none
Marc NK	1	46	23	0	30	30	10	5	15	3
Marc NK	2	46	23	0	60	60	15	2	15	1
Marc NK	3	46	23	0	20	20	20	2	15	none
JW FW	1	-48	-39	0	30	30	10	5	15	3
JW FW	2	-48	-39	0	60	60	15	2	15	1
JW FW	3	-48	-39	0	20	20	20	2	15	none
JW HW	1	-45	-58	0	30	30	10	5	15	3
JW HW	2	-45	-58	0	60	60	15	2	15	1
JW HW	3	-45	-58	0	20	20	20	2	15	none
Smit	1	90	20	0	30	30	10	5	15	3
Smit	2	90	20	0	60	60	15	2	15	1
Smit	3	90	20	0	20	20	20	2	15	none
Cambria	1	-45	-75	0	30	30	10	5	15	3
Cambria	2	-45	-75	0	60	60	15	2	15	1
Cambria	3	-45	-75	0	20	20	20	2	15	none
Chicka	1	-25	-25	30	30	30	10	5	15	3
Chicka	2	-25	-25	30	60	60	15	2	15	1

Chicka	3	-25	-25	30	20	20	20	2	15	none
Bray	1	76	56	0	30	30	10	5	15	3
Bray	2	76	56	0	60	60	15	2	15	1
Bray	3	76	56	0	20	20	20	2	15	none
Bray_2	1	0	8	0	30	30	10	5	15	3
Bray_2	2	0	8	0	60	60	15	2	15	1
Bray_2	3	0	8	0	20	20	20	2	15	none
SF	1	-45	-20	0	30	30	10	5	15	3
SF	2	-45	-20	0	60	60	15	2	15	1
SF	3	-45	-20	0	20	20	20	2	15	none
SF_2	1	0	50	0	30	30	10	5	15	3
SF_2	2	0	50	0	60	60	15	2	15	1
SF_2	3	0	50	0	20	20	20	2	15	none

Bulk density was interpolated using Inverse distance weighted to the second power. For those blocks that had insufficient density data to generate a block estimate, the block densities were assigned the average density for the rock type as defined in Table 14.14.

Table 14.14: Block Model Default Densities by Rock Codes

Rock Code	Average Density
99	2.83
101	2.96
102	2.96
103	2.96
105	2.96
201	3.01
301, 302 & 303	3.00
401	2.89
501	3.4
601	3.28
701	2.89
801 & 802	2.89
901 & 902	3.28

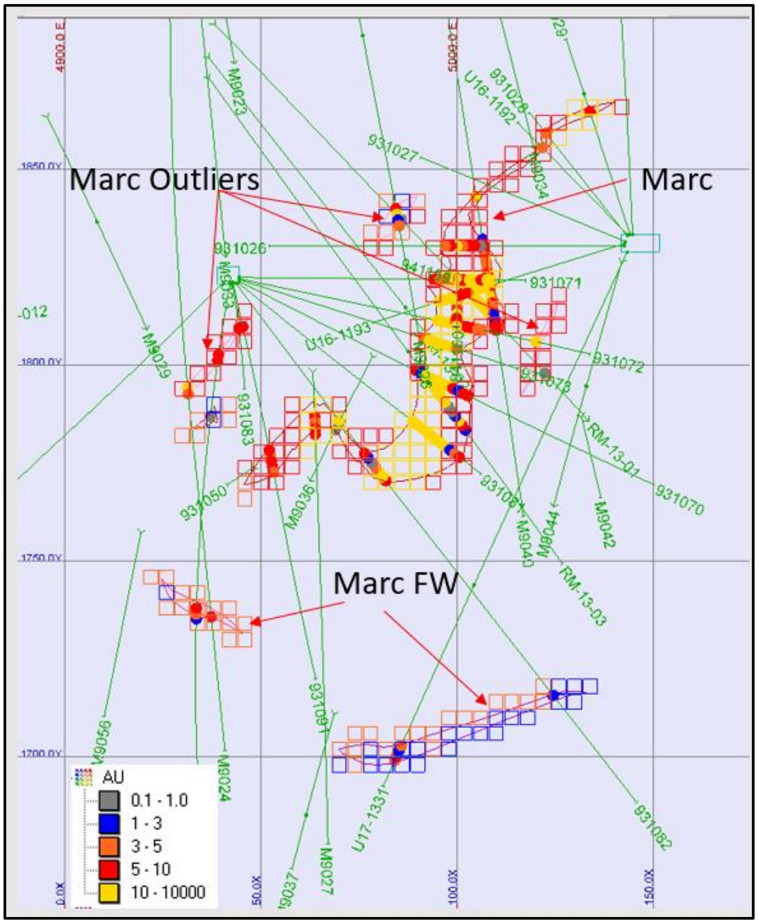
14.11 MODEL VALIDATION

The zones were validated by completing a series of visual inspections and by comparison of average assay grades with average block estimates along different directions - swath plots.

14.11.1 Visual Comparison

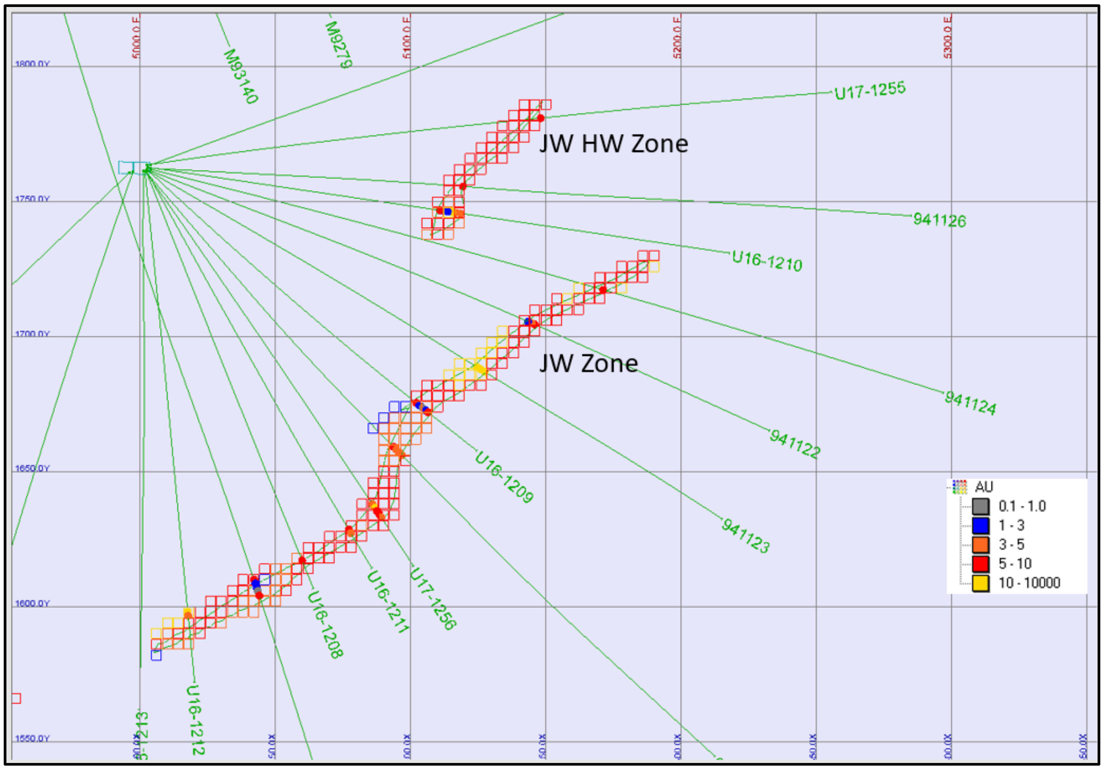
The model was checked for proper coding of drill hole intervals and block model cells. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values (Figure 14.2 and Figure 14.3).

Figure 14.2: Section 1250N Showing Block Drill Hole Composites and Estimated Gold Grades



Note: Grid lines are 50 m apart and blocks are 4 m by 4 m
Source: ACS (2018)

Figure 14.3: Section 1600N Showing Drill hole Composite and Estimated Gold Grades



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

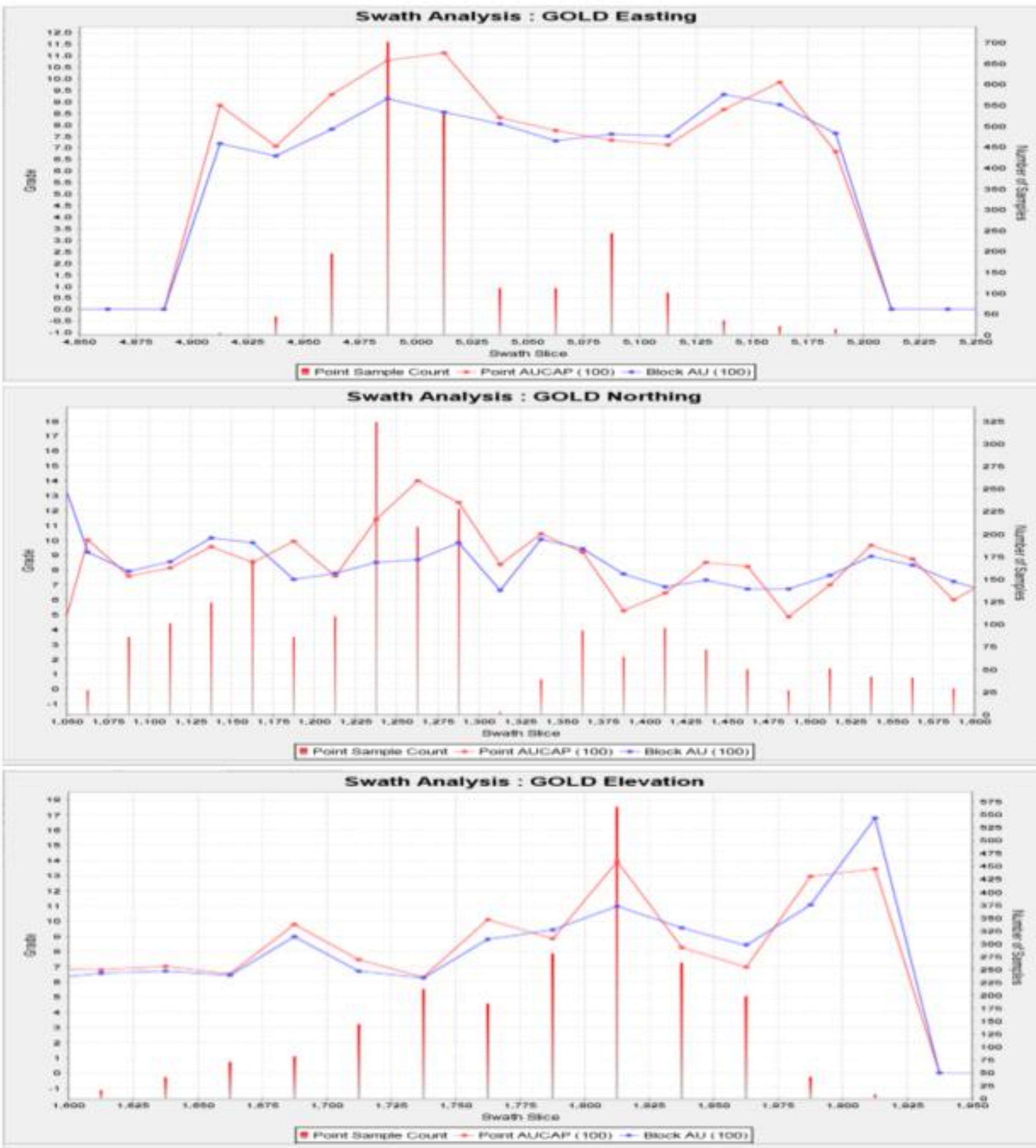
Source: ACS (2018)

14.11.2 Swath Plots

Average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south and horizontal (by elevation) swaths.

Figure 14-4 shows the swath Plot for gold for the Marc, AV and JW zones. On average, the estimated data agree well with the composited data with the estimated values being slightly more smoothed than the composite data.

Figure 14.4: Swath Plot for Gold for the Marc, AV and JW Zones



Source: ACS (2018)

14.12 RESOURCE CLASSIFICATION

Block model quantities and grade estimates for the Red Mountain gold project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (the CIM Definition Standards, May 2014) by Dr. Gilles Arseneau, P. Geo. (APEGBC), an independent "qualified person" for the purpose of NI 43-101.

Mineral resource classification is typically a subjective concept; however, industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

ACS is satisfied that the geological modelling reflects the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drill holes. Drilling samples were from sections spaced between 20 to 80 metres.

ACS considers that blocks in the Marc, AV and JW zones estimated during pass one and from at least 3 drill holes could be assigned to the Measured category. All other blocks interpolated during pass 1 in the Marc, AV and JW zones were assigned to the Indicated category. Blocks estimated with at least 5 holes during pass 2 in all zones were classified Indicated. All other estimated blocks were classified as Inferred.

14.13 MINERAL RESOURCE STATEMENT

CIM Definition Standards defines a Mineral Resource as:

"a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling".

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, ACS considers that major portions of the Red Mountain deposits are amenable for underground extraction by long hole stoping method.

In order to determine the quantities of material satisfying "reasonable prospects for economic extraction", ACS assumed a minimum mining cut off of 3 g/t gold representing an approximate mining cost of $160 Canadian and a minimum mining width of 2 m. The reader is cautioned that there are no mineral reserves at the Red Mountain gold project.

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

ACS considers that the blocks with grades above the cut-off grade satisfy the criteria for "reasonable prospects for economic extraction" and can be reported as a Mineral Resource. Mineral resources for each deposit at the Red Mountain gold project are summarized in Table 14.15.

Table 14.15: Red Mountain Mineral Resource Statement at a 3 g/t Gold Cut-off Effective June 15, 2018

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	715,100	10.65	41.46	244,800	953,300
Indicated	9,300	11.02	45.63	3,300	13,700
Inferred	0			0	0
AV Zone					
Measured	837,200	7.75	19.77	208,700	532,200
Indicated	116,500	8.47	20.81	31,700	78,000
Inferred	3,200	9.32	12.27	900	1,200
JW Zone					
Measured	275,600	7.96	20.07	70,500	177,800
Indicated	150,500	7.24	18.48	35,000	89,400
Inferred	4,900	8.83	16.88	1,400	2,600
141 Zone					
Indicated	234,700	4.86	7.04	36,700	53,100
Inferred	18,000	4.67	3.86	2,700	2,200
Smit					
Indicated	241,400	4.54	4.64	35,200	36,000
Inferred	48,100	5.28	2.26	8,200	3,500
Marc Footwall					
Indicated	28,600	5.76	10.79	5,300	9,900
Inferred	21,400	4.61	1.95	3,200	1,300
Marc Outlier Zone					
Indicated	12,100	5.24	28.64	2,000	11,100
Inferred	0			0	0

Marc NK Zone					
Indicated	37,500	7.4	8.26	8,900	9,900
Inferred	500	6.79	8.19	100	100
JW HW					
Indicated	39,900	5.66	32.28	7,300	41,400
Inferred	2,100	7.22	3.55	500	200
Bray					
Indicated	57,100	5.68	10.43	10,400	19,100
Inferred	73,800	4.66	7.49	11,100	17,800
Chicka					
Indicated	15,800	9.46	3.82	4,800	1,900
Inferred	600	5.3	1.57	100	0
JW FW					
Inferred	4,800	16.09	33.78	2,500	5,200
SF					
Inferred	54,600	6.88	17.55	12,100	30,800
Cambria					
Inferred	84,000	6.89	4.54	18,600	12,300
Total Measured & Indicated	**2,771,300**	**7.91**	**22.75**	**704,600**	**2,026,800**
Total Inferred	**316,000**	**6.04**	**7.6**	**61,400**	**72,000**

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Mineral reserves can only be estimated based on the results of an economic evaluation as part of a preliminary feasibility study or feasibility study. As such, no Mineral Reserves have been estimated by ACS. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined, however, ACS is of the opinion that it is reasonable to expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Mineral resources that are not mineral reserves have no demonstrated economic viability.

14.14 GRADE SENSITIVITY ANALYSIS

The mineral resources at the Red Mountain are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model quantities and grade estimates of the measured and indicated resource are presented in Figure 14.5 and the inferred resources are presented in Figure 14.6. The reader is cautioned that the grade and tonnages presented in these figures should not be misconstrued as a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Figure 14.5: Grade Tonnage Curve for Measured and Indicated Mineral Resource at Red Mountain

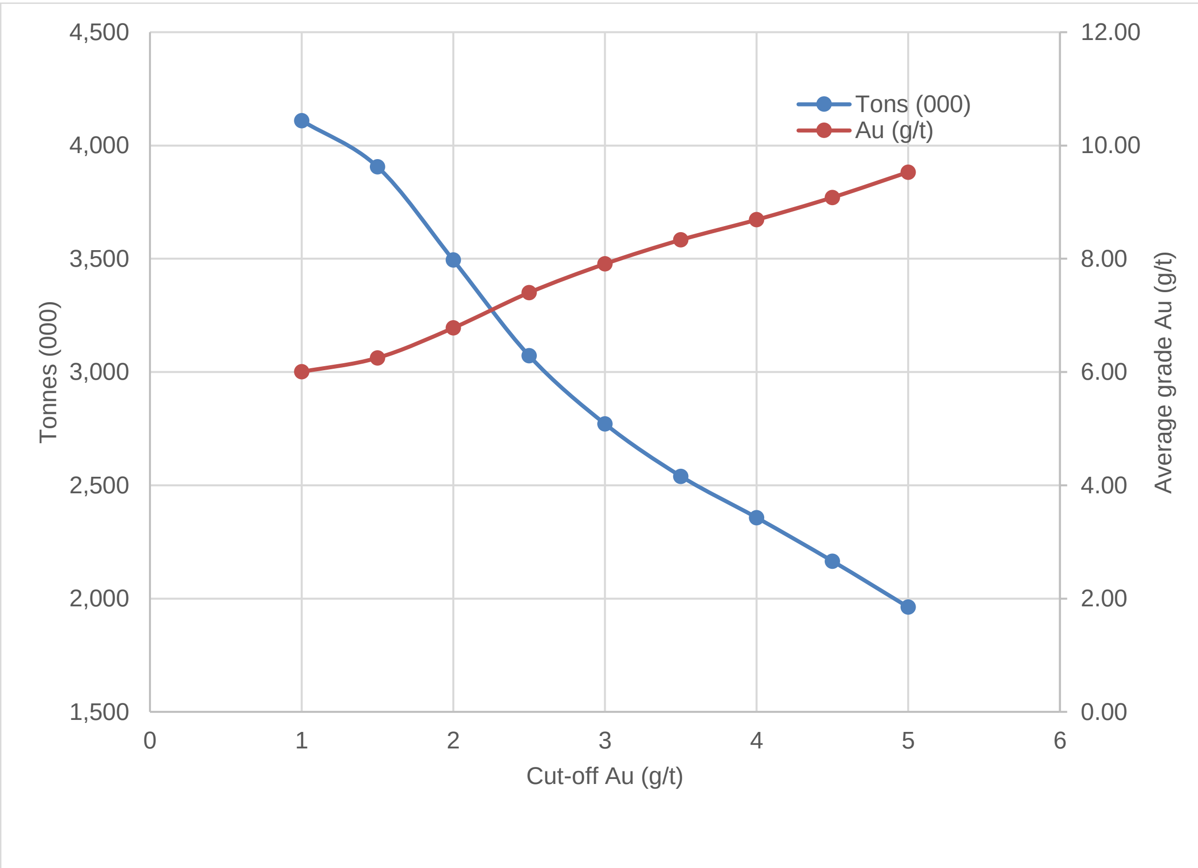
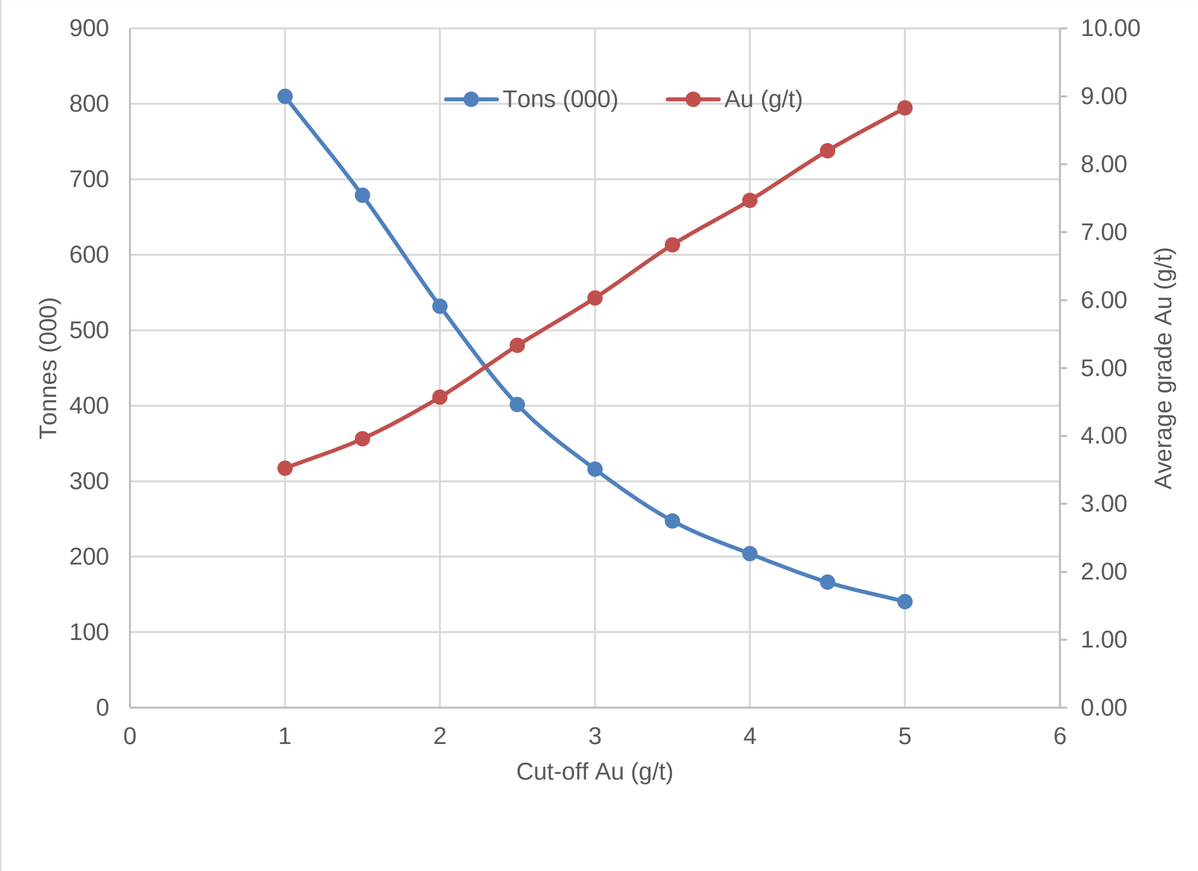


Figure 14.6: Grade Tonnage Curve for Inferred Mineral Resource at Red Mountain



The 2018 mineral resource estimate is reported at a 3.0 g/t Au cut-off grade. Cut-off grades may be re-evaluated considering prevailing market conditions (including gold prices, exchange rates and mining costs). Table 14.16 summarises the variability of the tonnage and grade at various cut-off selection. The mineral resource bases case is highlighted in bold and the reader is cautioned that the grade and tonnages presented in these figures should not be misconstrued as a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14.16: Variability of Mineral Resource at Various Cut-off Grades

Class	Cut Off	Tonnes	Au g/t	Ag g/t	Oz Au	Oz Ag
Measured	>4.0gpt	1,713,200	9.27	29.04	510,800	1,599,700
	>3.5gpt	1,780,400	9.07	28.59	519,000	1,636,300

	>3.0gpt	**1,827,900**	**8.92**	**28.30**	**524,000**	**1,663,300**
	>2.5gpt	1,853,600	8.83	28.16	526,300	1,678,400
	>2.0gpt	1,868,200	8.78	28.13	527,300	1,689,500
	>4.0gpt	644,300	7.13	14.98	147,700	310,200
	>3.5gpt	759,700	6.61	13.63	161,600	332,800
Indicated	**>3.0gpt**	**943,400**	**5.95**	**11.98**	**180,600**	**363,500**
	>2.5gpt	1,218,600	5.23	10.02	204,900	392,600
	>2.0gpt	1,626,000	4.48	8.01	234,200	418,700
	>4.0gpt	204,000	7.47	9.59	49,000	62,900
	>3.5gpt	247,400	6.81	8.76	54,200	69,700
Inferred	**>3.0gpt**	**316,000**	**6.04**	**7.60**	**61,400**	**77,200**
	>2.5gpt	401,500	5.33	6.78	68,800	87,500
	>2.0gpt	532,000	4.57	5.78	78,200	98,900

14.15 PREVIOUS MINERAL RESOURCE ESTIMATES

Mineral resources have been estimated for the Red Mountain gold project in the past. IDM reported mineral resources in a technical report dated March 1, 2017 with an effective date of January 23, 2017. The Mineral resources were included in a Feasibility Study report published by IDM on August 10, 2017. The 2017 mineral resources are summarized in Table 14.17.

Table 14.17: Prevoius Mineral Resource Statement for Red Mountain

Class	Tonnes	gold (g/t)	Ag (g/t)	Au Oz	Ag Oz
Measured and Indicated	2,074,700	8.75	24.8	583,700	1,655,700
Inferred	324,700	6.21	10.1	64,800	105,500

The previous mineral resources are presented here only as a means of comparing the previous estimate with the current estimate present in Table 14.15 above. As can be seen, the tonnage of the mineral resource has increased in the measured and indicated categories as a result of the 2017 drilling and the gold and silver grades have dropped slightly in all categories due to the inclusion of lower grade mineralized zones.

15.0 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

The following Section is taken from JDS (2017) with minor modifications. ACS takes responsibility for this section prepared by Gord Doerksen P. Geo. on behalf of JDS.

15.1 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 Piésold for Bond Gold, in 1993 and 1994 by Rescan for LAC Minerals, and in 1996 and 1997 by Royal Oak. Subsequently, many engineering and environmental studies have utilized this data. The historic environmental database was utilized for initiating an environmental assessment in 1996 by Royal Oak. The environmental studies included sampling and assessment of water quality, climate, hydrology, hydrogeology, wildlife and vegetation, fisheries, ARD/ML, terrain stability, socio-economics, and culture and heritage. The available information indicates that the effects of the Project on the environment can be mitigated to meet regulatory requirements.

IDM has completed a gap analysis of all previously available baseline studies; this resulted in additional studies being conducted in 2015 – 2017, which are summarized in Table 15.1. These additional studies were conducted in order to update the baseline to current environmental conditions, to address refinement of the Project design, and to reflect current regulatory requirements. The gap analysis indicated that additional information was required for the Project area atmosphere/climate, surface hydrology, aquatics, water quality, sediment quality, terrestrial wildlife, and fish habitat. IDM has completed comprehensive studies of rock geochemistry, archaeology and heritage resources, land use, cultural, and socio-economic baseline to characterize the regional human environment. Where available, traditional ecological knowledge functions were supplied as additional information for the assessment of the Project’s effects. Further, IDM Mining is currently working with NLG to finalize the ecological and socio-economic assessments that are required under Chapter 10, paragraphs 8(e) and 8(f) of the NFA. Mitigation measures are being developed in consultation with NLG through the environmental assessment process.

Table 15.1: Environmental Baseline Studies at Red Mountain

Baseline Component	Additional Information
Terrain and Physiography	New mapping completed to reflect changes as the glaciers of the Cambria Icefield have retreated. Updates completed to a natural hazards assessment in the Bitter Creek Valley proximal to the proposed mine, mill, waste storage facilities and access route.
Water Quality	Collected and updated water quality data to address gaps and project design refinements.
Climate	Meteorological station installed on site to extend historical data.
Hydrology	Extensive monitoring of local and regional stream flow to support project design, fisheries and water quality assessments.
Hydrogeology	Extensive monitoring of groundwater to support project design, fisheries and water quality assessments.
Wildlife and Vegetation	Detailed baseline studies of wildlife and vegetation for the Bitter Creek Valley to support the assessment of project effects.
Fisheries	Detailed baseline studies of fish and fish habitat for the Bitter Creek, Bear River and local creeks 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 be conducted in the summer/fall of 2016. This work will also support wastewater quality assessment and water quality predictions associated with various disposal options.
Terrain Stability Assessment	Detailed terrain stability assessments along road corridor and in the vicinity of project facilities have been completed.
Socioeconomics	Baseline information for the socio-community and economic characteristics of the area to support an assessment of project effects is being collected and assessed.
Culture and Heritage	An archaeological assessment was required pursuant to the *BC Heritage Act*, and was completed in 2015.
First Nations Interests	IDM has a comprehensive engagement and consultation program in place for First Nations in the project area. The primary work in this regard is with the Nisga'a and this began back in the spring of 2014 and is ongoing.

16.0 ADJACENT PROPERTIES

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

17.0 OTHER RELEVANT DATA & INFORMATION

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

18.0 INTERPRETATIONS & CONCLUSIONS

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 mineralized zone controls.

The Marc, AV and JW Zones host the main portion of the mineralized deposit and require no further drilling.

The 141 and Smit Zones are drilled at nearly a 25 to 50 m grid spacing and shows reasonably good geological and grade continuity yielding a large portion of the deposits in the indicated category. The zones will require infill drilling to confirm the geological continuity prior to mining. The infill drilling carried out in 2017 confirmed the geological and grade continuity of the AV and JW and was successful in upgrading the inferred resources in these zones to indicated classification and upgrading some indicated blocks to measured in the Marc Zone.

The Cambria, SF and JW FW are currently classified as inferred due to wider spaced drill density. The existing drill holes display reasonable geological continuity and it is reasonable to assume that the majority of the inferred could be converted to indicated with additional drilling.

The interpretation of high-amplitude folds suggests that potential to expand resources exists along the favourable brecciated horizon, including: the synform between the Marc and 141 Zones, extending north to the Smit Zone; the west portion of an antiform deforming the 141 Zone, to the east of the antiform deforming the Marc, AV, JW and SF Zones

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.

19.0 RECOMMENDATIONS

Further work is recommended by ACS for the Red Mountain gold project as follows:

- The 2017 feasibility study (FS) should be updated to include the new measured and indicated mineral resources defined as part of this report. The total estimated cost is expected to be in the order of $1.2 million for the feasibility update.
- A 10,000 metre drilling program of infill and step out drilling is recommended to be carried out to upgrade and expand recently identified inferred resources. Total estimated cost for this program is $5.0 million.
- Metallurgical test work should be carried out on samples from the Smit Zone as this new zone has not been tested in the past metallurgical test programs. Total estimated cost for this program is $100,000.

The total next phase exploration program for the Red Mountain Project is estimated at $6.3 million as summarized in Table 19.1.

Table 19.1: Budget of Recommended Exploration Program

Item	Estimated Costs
Feasibility Update	$1,200,000
10,000 m drilling (all inclusive)	$5,000,000
Metallurgical testing of Smit Zone	$100,000
Total Phase 1 Work Program	**$6,300,000**

20.0 UNITS OF MEASURE, ABBREVIATIONS AND ACRONYMS

Table 20.1: 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
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
wmt	Wet metric tonne

Table 20.2: Abbreviations & Acronyms

% or pct	Percent
AA	Atomic absorption
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
PPM	Parts per million
Project	Red Mountain Project
QA/QC	Quality assurance/quality control
QP	Qualified Person
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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