PROJECT 3CS012.00

RED MOUNTAIN PROJECT
ENGINEERING STUDY

2741

Prepared for:

SEABRIDGE GOLD INC.
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AUGUST, 2003

RED MOUNTAIN PROPERTY, Skeena Mining Division, BC

Claim #	Claim Name	Size (Units)	Expiry Date	Claim #	Claim Name	Size (Units)	Expiry Date
250331	Montreal No. 1	1	2006-01-27	253111	Sarah 5	4	2005-09-15
250332	Montreal No. 2	1	2006-01-27	253112	Sarah 6	12	2004-09-15
250333	Montreal No. 3	1	2006-01-27	253114	Sarah 8	12	2005-09-15
250334	Montreal No. 4 & 5	1	2006-01-27	253115	Sarah 9	20	2005-09-15
250335	Montreal No. 6	1	2006-01-27	253119	Vera 1	20	2005-09-16
250336	Montreal No. 7	1	2006-01-27	253131	Vera #10	20	2005-09-24
250781	Kim No. 1	1	2005-09-26	253158	Oro I	18	2006-09-16
250782	Kim No. 2	1	2005-09-26	253159	Oro II	18	2007-09-16
250783	Kim No. 3	1	2005-09-26	253160	Oro III	12	2008-09-16
250784	Kim No. 4	1	2005-09-26	253161	Oro IV	20	2006-09-23
250785	Kim No. 5	1	2005-09-26	253162	Oro V	20	2007-09-23
250786	Kim No. 6	1	2005-09-26	253163	Oro VI	20	2008-09-23
250787	Kim No. 7	1	2005-09-26	253172	Sarah I	20	2005-09-26
250788	Kim No. 8	1	2005-09-26	253173	Sarah II	15	2005-09-26
250789	Kim No. 9	1	2005-09-26	253236	Vera 5	4	2005-09-17
250790	Kim No. 10	1	2005-09-26	253778	Montreal No. 8	1	2006-03-22
250791	Kim No. 11	1	2005-09-26	255098	Gold Spot	1	2006-09-21
250792	Kim No. 12	1	2005-09-26	320189	Sabina 1	12	2005-08-03
250793	Kim No. 13	1	2005-09-26	320735	Oro Fr.	1	2006-09-06
250794	Kim No. 14	1	2006-09-26	320737	Theresa	20	2005-09-02
250795	Pam 1	20	2006-09-26	320867	Janet 1	5	2005-09-14
250796	Pam 2	1	2005-09-26	320868	Janet 2	5	2005-09-14
251627	Bon Accord No. 2	1	2005-01-19	320869	Windy	3	2005-09-14
251628	Bon Accord No. 3	1	2006-01-19	320870	Anita Fr.	1	2005-09-14
251629	Bon Accord No. 4	1	2006-01-19	320929	Michaela	6	2005-08-30
251630	Bon Accord No. 5	1	2005-01-19	320930	Ren	5	2005-09-02
251631	Bon Accord No. 6	1	2005-01-19	320932	Stimpy	6	2005-09-02
251632	Bon Accord No. 7	1	2005-01-19	320992	Sandra Fr.	1	2005-09-06
251633	Bon Accord No. 8	1	2005-01-19	321028	Sharon Fr.	1	2005-09-07
251660	Bon Accord	1	2005-02-16	321029	Rose	3	2005-09-20
251661	Bon Accord No. 1	1	2005-02-16	321646	Kim Fr.	1	2005-10-12
251662	Bon Accord No. 9	1	2005-02-16	324637	Desi 1	4	2005-03-27
251663	Bon Accord No. 10	1	2005-02-16	324638	Desi 2	4	2005-03-27
252153	Hrothgar	20	2006-07-11	328212	Pamvera Fr.	1	2005-07-18
252217	Willoughby 3	20	2005-09-21	328214	Bon Fr.	1	2005-07-18
252943	Dixie 1	18	2005-07-15	338971	Bromley	6	2005-08-17
252944	Dixie 2	15	2005-07-15	340214	Dixon 2 Fr.	1	2005-09-10
252945	Dixie 3	15	2005-07-15	343046	Vermillion #1	4	2005-01-18
252946	Dixie 4	18	2005-07-15	343047	Vermillion #2	12	2006-01-18
252990	Lisa 1	20	2005-08-12	395135	Gold Valley 1	12	2005-07-14
252991	Lisa 2	20	2005-08-12	395136	Gold Valley 2	12	2005-07-14
252992	Lisa 3	20	2005-08-12	395137	Otter 1	12	2005-07-14
252993	Lisa 4	20	2005-08-12	395138	Otter 2	12	2005-07-14
252994	Lisa 5	15	2005-08-12	396491	CB-1	16	2005-09-21
252995	Lisa 6	20	2005-08-12	404734	Windsor	1	2005-07-21
252996	Lisa 7	20	2005-08-12	404735	Windsor No. 2	1	2005-07-21
252997	Lisa 8	15	2005-08-12	404736	Laura	1	2005-07-21
253082	Janine 1	16	2005-09-08	404737	Last Chance	1	2005-07-21
253083	Janine 2	12	2005-09-08	404738	Raven No. 1	1	2005-07-21
253084	Janine 3	20	2005-09-08	404739	Raven No. 2	1	2005-07-21
253085	Janine 4	20	2005-09-08	404740	Raven No. 3	1	2005-07-21
253105	Vera 3	8	2006-09-17	404741	Raven No. 4	1	2005-07-21
253106	Vera 4	18	2005-09-16	404742	Raven Fr.	1	2005-07-21
253108	Vera 7	8	2005-09-16	404743	Windsor Fr.	1	2005-07-21
253109	Sarah 3	6	2004-09-15				
253110	Sarah 4	2	2005-09-15	***Total Number of Units =***		***832***	

Detailed Breakdown of Costs for Engineering Study - Red Mountain

Tasks	**Professional Time**	Jan	Feb	March	April	May	June	July	Aug	Totals
Resource and Reserves	Michael Michaud	$2,080.00	$1,560.00	$260.00						$3,900.00
Mine Planning	Ken Reipas	$4,900.00	$14,980.00	$10,640.00	$8,540.00	$8,990.00				$48,050.00
	Andrew Bradfield		$1,160.00	$290.00	$4,330.00	$2,030.00				$7,810.00
Assessing Exploration Potential	John-Francois Couture			$1,800.00	$4,040.00					$5,840.00
Metallurgical/Processing	John Starkey			$8,000.00						$8,000.00
Review of Tailings Costs	Maritz Rykaart				$1,840.00	$230.00			$460.00	$2,530.00
Closure/Permitting	Kelly Sexsmith		$402.00	$57.50	$517.50		$57.50	$57.50		$1,092.00
	Cam Scott				$225.00	$1,575.00				$1,800.00
Reporting	Ken Reipas					$2,000.00	$4,760.00		$1,050.00	$7,810.00
	John-Francois Couture				$1,000.00					$1,000.00
	John Starkey					$1,000.00				$1,000.00
	Michael Michaud					$520.00				$520.00
Subtotal										***$89,352.00***
Reporting Expenses										
Courier								$111.33	$4.49	$115.82
Reproductions										$0.00
Subtotal										***$115.82***
Total										$89,467.82
GST										$6,262.75
Total										$95,730.57

Notes:

Backup for the above expenditures is on file at SRK Consulting

Tasks were not tracked separately. Therefore, the breakdown between analysis and reporting tasks is approximate.

CERTIFICATE AND CONSENT

To Accompany the Red Mountain Project Engineering Study, August 2003

I, Ken S. Reipas, residing at 43 Deverell Street, Whitby, Ontario, Canada. do hereby certify that:

1) I am a Principal Mining Engineer with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 602, 357 Bay Street, Toronto, Ontario.

2) I am a graduate of Queen's University with a B.Sc in Mining Engineering in 1981, and have practiced my profession continuously since 1981.

3) I am a Professional Engineer registered with the Professional Engineers of Ontario (PEO).

4) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Red Mountain Project or securities of Seabridge Gold Inc.

5) I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6) I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7) I have not had any prior involvement with the property that is subject to the technical report.

8) I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9) Steffen Robertson and Kirsten (Canada) Inc. was retained by Seabridge Gold Inc. to prepare a preliminary assessment (engineering report) for the Red Mountain Project, Stewart, B.C., in accordance with National Instrument 43-101. The following report is based on our review of project files, and discussions with project personnel.

10) I was author of the report.

11) I hereby consent to use of this report for submission to any Provincial regulatory authority.

Ken Reipas

LICENSED PROFESSIONAL ENGINEER
K. S. REIPAS
PROVINCE OF ONTARIO

Toronto, Canada
September, 2003

Ken S. Reipas, P.Eng.
Principal Mining Engineer

STATEMENT OF QUALIFICATIONS

I, Ken Reipas of 43 Deverell Street, Whitby, ON, L1R 1W2
hereby certify:

I am a graduate of Queen's University with a B.Sc. degree in Mining Engineering, 1981.

I am presently employed as a Mining Engineering Consultant by Steffen Robertson and Kirsten (Canada) Inc. of 900 – 1066 West Hastings Street, Vancouver, British Columbia V6E 3X2

I have been active as a Mining Engineer in Canada for the past 23 years.

I personally worked on the engineering study for the property.

This report may be used by Seabridge Gold Inc. for any and all corporate purposes.

Ken Reipas

Signed by: Ken Reipas
Principal Mining Engineer

Dated at Toronto, Ontario, March 31, 2004.

PROJECT 3CS012.00

RED MOUNTAIN ENGINEERING STUDY

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EXECUTIVE SUMMARY

Introduction

In January 2003, Seabridge Gold Inc. ("Seabridge") commissioned Steffen Robertson and Kirsten (Canada) Inc. ("SRK") to complete an Engineering Study on Seabridge's Red Mountain underground gold project located 18km east of the town of Stewart, B.C. The objectives of the study were to build on previous project work to identify the best project development approach, and to assess the current economics of the project.

The Red Mountain property includes 97 contiguous mineral claims covering an area of approximately 19,280 hectares. The project area covers rugged mountainous terrain with steep to precipitous slopes and elevations ranging between 599 and 2,100m above sea level. Alpine glaciers and ice fields are abundant and cover approximately one third of the project area.
The coastal climate is characterized by very heavy snowfall. Access to the site is by helicopter.

Work by Previous Owners

Between 1991 and 1994, previous owner Lac Minerals delineated a sizeable gold-silver resource through diamond drilling and subsequently drove a 1700m decline to facilitate drilling, and to obtain a bulk sample. The project was sold to Royal Oak Mines in 1995, and in the following year the underground development was extended by 305m and additional surface and underground drill programs were completed.

In 1994, a feasibility study was partially completed by Rescan Engineering Limited for Lac Minerals. In 2001, previous owner North American Metals Corp. ("NAMC") undertook limited engineering studies of project development alternatives. Seabridge acquired the Red Mountain Project from NAMC in February 2002.

Project Development Alternatives

SRK reviewed previous studies and evaluated the development alternatives relating to various aspects of the project. The best development alternatives are:

- Road access to the site. A road was designed by NAMC in 2001 to access the project site. Alternatives that were discarded on the basis of costs were; aerial tramway, tunnel and shaft, and ongoing helicopter support.
- A seasonal operation from May to October was selected in favor of year-round operations on the basis of safety and reliability.
- An on-site mill using a grinding and cyanidation leaching (CIP) circuit was selected. The alternative of using flotation to produce a sulphide concentrate for offshore marketing

was discarded on the basis of poor economics caused by lower overall gold recovery and smelting costs.

- A conventional type of mill was selected instead of a portable type due to the tonnage required (1000tpd) and the very fine grind needed.
- The full use of backfill was selected to optimize the mining recovery of the resources. Minimizing backfill was considered to reduce costs, but the possible savings are not enough to justify the lower mining recovery that results.

Mineral Resources

In 2000, NAMC completed a comprehensive review of the project and a validation of the geological database. Several new technical studies were carried out leading to the creation of a revised resource model, which SRK used as the basis for the current study.

SRK determined a 6g/t Au cut off grade based on a gold price of US$375/oz and preliminary estimates of the site operating costs totaling CDN$82 per tonne. Resources above cut off are shown in Table I.

TABLE I: Mineral Resources Above Cut Off

Application of Cut Off Grade							
	M&I Total			All Categories			
	Tonnes (000's)	Au g/t	Au Oz	Tonnes (000's)	Au g/t	Au Oz	Ag g/t
Resources (no cut off)	**1596.1**	**7.80**	**400,429**	**1,941.2**	**7.74**	**482,987**	**26.2**
Resources (> 6 g/t Au)	**1058.9**	**9.22**	**313,863**	**1,216.6**	**9.14**	**357,573**	**28.7**
Gold Ounce Recovery			**78%**			**74%**	
Resources (< 6 g/t Au)	537.2	5.01	86,566	724.6	5.38	125,414	22.1
Gold Ounce Loss			22%			26%	

SRK decided to include inferred resources in the study because of the high degree of confidence that additional drilling will upgrade this material to the indicated category. Resources above cut off grade in all categories (measured, indicated and inferred) total 1,216.6kt at grades of 9.14g/t gold and 28.7g/t silver.

"Mineable Tonnes"

The term "mineable tonnes" in this report refers to mineral resources, including inferred mineral resources, to which SRK has applied factors for cut off grade, dilution, and mining recovery. As such, these do not qualify as reserves under N.I. 43-101.

"Mineable tonnes" shown in Table II were estimated by a process that applied dilution and mining recoveries to the individual deposits comprising the resources above cut off. The parameters were based on preliminary mining designs for longhole, and cut and fill methods.

TABLE II: "Mineable Tonnes"

"Mineable Tonnes" - All Categories (thousands)			
	Tonnes	**Au g/t**	**Ag g/t**
Resources (> 6 g/t Au)	**1,216.6**	**9.14**	**28.7**
Mining Recovery	89%		
Recovered Tonnes	1,081.2	9.13	28.9
Dilution Percent	14%		
Dilution Tonnes	180.7	0.55	n/a
"Mineable Tonnes"	**1,261.9**	**7.90**	**24.7**
Basis of Mine Plan			

Project Description

The 1000tpd trackless underground mine (owner operated) will be accessed through the existing exploration decline and a new portal at 1650m elevation. Production tonnes by mining method are planned at 65% longhole, 29% cut and fill, and 6% development muck. Longhole stope tonnes vary from 18 to 35kt. Backfill will consist of both cemented and unconsolidated waste rock, and no development waste rock will remain on surface at project completion. Mucking to a main muck pass will be accomplished by 6yd and 4yd scoops. An underground rock breaker will size the muck to be hauled out of the mine in 20t trucks.

A 1000tpd gold-silver recovery mill will reclaim feed from a run-of-mine stockpile. The mill process will include a normal two-stage SAG/Ball mill grinding circuit with high capacity thickening, cyanide leaching, carbon-in-pulp adsorption, carbon elution and regeneration, electro-winning and refining, and cyanide destruction with SO_2 and air. Metallurgical recoveries are estimated at 90% for gold and 80% silver.

Mill tailings will be discarded to an earth and rock constructed tailings dam (with water reclaim facilities) near the mill. The tailings facility, located in the Red Mountain cirque, follows the design used in the 1994 feasibility study. It is the recommendation of SRK that a further study be undertaken by Seabridge to assess alternative tailings facility locations and designs which could materially reduce the capital cost of the tailings storage facitity.

The project schedule has the new access road opened up for traffic by the end of the first operating season (mid-May through October). The pre-production period extends through seasons 2 and 3 with mill construction, tailings dam construction, and development of the underground mine. Eight seasons of production (157.5kt/a) follow in seasons 4 to 11. Mine closure is planned for season 12.

Project Costs

Estimated site operating costs are shown in Table III. Due to the seasonal nature of the project, SRK envisions that it may be difficult for the owner to fully supply all manpower each spring through an annual hiring campaign. For this reason, a labour premium was added to the operating costs, such that funds would be available to utilize contractor supplied labour as necessary.

TABLE III: Red Mountain Operating Costs

Site Operating Cost	
	Base Case
Mining	35.91
Milling	21.11
General & Administration	10.05
Additional Labour	1.39
Total Cost per Tonne	$ 68.46

Total estimated capital costs are shown in Table IV, exclusive of off-site owner's costs and working capital. The pre-production portion of capital is $60.3million. Included in this estimate is $9.2 million for the tailings dam. As indicated above, SRK believes that further work should be performed on assessing alternate tailings dam locations and designs which could result in a material decrease in the capital cost of the tailings facility.

TABLE IV: Red Mountain Capital Costs

Capital Costs (thousands)	Total
Feasibility Study, Permitting	1,400
Access Road	7,000
Power Line	1,600
Mine:	
Preproduction Development	6,673
Mobile Equipment	7,766
Other Mine Equipment	3,259
Mill	21,911
General and Administrative:	
Preproduction Costs	1,706
Surface Equipment	976
Labour Premium	312
Tailings Dam	9,200
Camp, Office, Dry Buildings	603
Sustaining Capital	2,100
Reclamation	260
Salvage	(3,000)
TOTAL CAPITAL	**61,765**

Economic Modeling

The term "base case" refers to the project as described in this study. It represents SRK's best estimate of gold production, and capital and operating costs. The gold price assumption in the model was varied to determine the "break even" gold price for the project. At a zero discount rate, the base case model indicates a break even project is achieved at a gold price of US$359/oz. At a 5% discount rate, the base case model indicates a break even project is achieved at a gold price of US$399/oz.

Base case economic results are shown in Table V for various gold prices.

TABLE V: Gold Price Sensitivity

Red Mountain Economic Model Results								
Gold Price	US$/oz	**350**	**375**	**400**	**425**	**450**	**475**	**500**
NPV (0%)	$M	(3.6)	6.4	16.3	26.3	36.2	46.1	56.1
NPV (5%)	$M	(13.7)	(6.7)	0.3	7.4	14.4	21.4	28.5
NPV (10%)	$M	(19.3)	(14.1)	(9.0)	(3.9)	1.2	6.3	11.5
IRR	%	(1.0)	2.0	5	8	11	13	16

Sensitivity analyses were also run for these two scenarios; firstly a 50% increase in "mineable tonnage", and secondly cost reductions of 15% for both capital and operating costs. At a 5% discount rate, the break even gold price was reduced from US$399/oz to US$350 and US$338 respectively.

Risks and Opportunities

The main risks associated with the project are related to:

- The construction and operation of the mine access road, which must traverse rugged mountainous terrain.
- A lack of continuity in the workforce due to the seasonal operation.
- Project economics requiring a higher gold price than currently exists.
- The tailings facility, which must retain a water cover on the tailings in perpetuity.

With follow up work, these opportunities could prove beneficial to the project:

- *Exploration to increase the potentially economic mineralization.*
- A revised design for the cirque tailings facility to reduce its capital cost and its long-term costs after closure.

Conclusions

SRK concludes that the project requires a minimum gold price of US$400/oz to be economically viable (yielding an IRR of 5%), and that higher gold prices are required to provide an attractive rate of return.

Significant improvements in project economics would be required to achieve economic viability at the current gold price of US$360/oz. For example, at US$360/oz gold, cost reductions of 15% for both capital and operating costs result in an IRR of 8%.

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RED MOUNTAIN
ENGINEERING STUDY

1. INTRODUCTION

1.1 Terms of Reference

Seabridge Gold Inc. ("Seabridge") and Steffen Robertson and Kirsten (Canada) Inc. ("SRK") met on January 8, 2003 to discuss the requirements for an Engineering Study on Seabridge's Red Mountain Project near Stewart, B.C. Subsequently, SRK submitted a proposal to Seabridge to complete an engineering and economic study on the project, and this was accepted by Seabridge on January 24, 2003.

SRK's proposal outlined the objectives of the study as:

- Review the findings of previous studies.
- Evaluate the different options available regarding site access, infrastructure, mining and processing.
- Assess the economic viability of the project in today's market.
- Identify what further improvements, including the gold price, may be required to enhance the project viability.
- Identify the risks and opportunities presented by the project, and any gaps in project data.
- Identify further fieldwork and investigations required to support a feasibility study.
- Evaluate the exploration potential of the project, including upgrading of the current inferred resources, and the possible discovery of addition economic mineralization.

Unless otherwise noted, currency amounts are in 2003 Canadian dollars, and metric units of measure are used.

1.2 Project History

The area surrounding the Red Mountain Project has been subject to sporadic exploration in the 1960s and 1970s, primarily for porphyry-molybdenum deposits.

Bond Gold Canada Inc, ("Bond Gold") became involved in the Skeena Mining Division in late 1988 through an option from Wotan Resources Corporation ("Wotan") to acquire seven claim blocks (ORO I through VI and Hrothgar). The first high grade gold-silver samples were collected at Red Mountain during the early part of the 1989 program on what was to become the Marc Zone. The discovery was made by tracing the source of auriferous floats uphill to bedrock.

During the period 1989-1991, Bond Gold assembled a very large land package surrounding the Red Mountain discovery through three distinct option agreements and claim staking. Public assessment file records indicate that during this period Bond Gold carried out reconnaissance exploration over much of this area, including: prospecting, reconnaissance geology, geochemical sampling, airborne and ground geophysical surveys and a limited amount of diamond drilling.

Between 1991 and 1994, following the acquisition of Bond Gold, Lac Minerals ("Lac") delineated a sizeable gold-silver resource through diamond drilling and subsequently drove a decline (1,700m) to facilitate drilling access and collect a bulk sample for metallurgical studies. Mine development and environmental baseline studies were initiated in 1993 through late 1994, when the project was put on hold by Barrick Gold Corporation ("Barrick"), following the acquisition of Lac Minerals. From 1989 through 1994, a total 406 surface and underground boreholes were drilled on the property, including 368 drilled within the limited footprint area of the Marc, AV, JW, AV-JV Tails and 141 gold-silver zones.

The project was sold to Royal Oak Mines ("Royal Oak") in August 1995. The following year, Royal Oak expanded the underground development (305m) and conducted surface (22 holes) and underground (15 holes) drill programs targeting the extensions of known mineralization outside the resource volumes and other nearby targets (23 holes).

In 2000, upon acquisition of the project from Price Waterhouse Coopers, North American Metals Corporation ("NAMC") completed a comprehensive review of the project and validation of the geological and environmental database. Several new technical studies were carried out leading to the creation of a revised resource model. Seabridge acquired the Red Mountain Project from NAMC in February 2002.

1.3 Project Description

1.3.1 Location

Red Mountain is located within the Boundary Range of the northwestern British Columbia Coast Mountains, approximately 18km east of the town of Stewart. The Red Mountain Project lies within the Skeena Mining Division between the Cambria Ice Field and the Bromley Glacier. The centroid of the project is located at latitude 55°57' N and longitude 129°42' W.

1.3.2 Physiography and Climate

The Red Mountain Project covers rugged mountainous terrain with steep to precipitous slopes and elevations ranging between 599 and 2,100m above sea level. The tree line occurs at approximately 1,300m elevation. Areas below the tree line are forested while higher elevations are characterized by bare rock, talus slopes and intermittent alpine vegetation. Alpine glaciers and ice fields are abundant and cover approximately one third of the project area. On-site infrastructure is located above the tree line.

The area is characterized by a coastal climate and vegetation, and receives very heavy snowfall. Temperatures at Red Mountain are moderated by the coastal influence. On-site temperature data collected between 1993 and 1994 indicate a mean average temperature of 0.1°C and varying between -25°C in winter and 20°C in summer. Wind conditions add a significant wind chill factor throughout the year. In more sheltered locations, hourly average wind speeds regularly exceed 10m/s and instantaneous wind speeds in excess of 30m/s have been observed.

1.3.3 Local Resources and Infrastructure

The town of Stewart has a population of approximately 600. The town of Terrace, located approximately 316km from Stewart, has a population of 21,000. Stewart is serviced by a paved airstrip and is located at the head of the Portland Canal, a 120km long fjord that remains ice-free year-round. The Stewart Bulk Terminal has a dock capacity of 800 tonnes per hour and currently handles ore from the Eskay Creek mine.

1.3.4 Land Tenure, Ownership and Royalties

A comprehensive review of land tenure and discussion of ownership and royalty agreements was presented by Craig (2002). Only the salient features are presented below.

The Red Mountain property includes 97 contiguous mineral claims (773 modified grid units and 2 post claims) covering an area of approximately 19,280 hectares. Five of the mineral claims comprising 86 units have been legally surveyed and are ready to be taken as lease. These are the ORO I, ORO IV, ORO VI, Hrothgar and part of the Vera 3 claims. The known gold-silver resource on this project (Marc, AV and JW zones) are located in the northwest corner of the ORO I mineral claim.

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

The Red Mountain Project is wholly owned by Seabridge. It is subject to the payment of productions royalties on the key "Wotan claims" group (seven claims: ORO I-VI and Hrothgar; and from any other property within a two kilometer area of influence around those claims) and to the payment of an annual minimum royalty of $50,000.

Production from the "Wotan claims" is subject to two separate royalties aggregating 3.5% of net smelter return (NSR) comprising a 1.0% NSR payable to Barrick and a 2.5% NSR payable to Wotan.

The Barrick 1.0% NSR royalty is applicable to all existing claims at the time the property was sold to Royal Oak in August 1995. The Red Mountain Project was assembled by Bond Gold through three option agreements exercised by its successor Lac Minerals. Each agreement provides for NSR royalties such that the bulk of the property has stacked royalty obligations ranging from 2.0% to 6.5%. Certain peripheral non-core claims staked by Bond Gold or Lac Minerals carry a 1.0% NSR royalty, and three none-core claims staked by Royal Oak are free of royalty. The principal underlying agreements were discussed by Craig (2002) and are not presented here.

1.3.4 Current Site Facilities and Condition

The information presented in this section is based on descriptions in the NAMC 2001 reclamation plan, and on discussions with previous project personnel. Refer to Figure 7.2.

The portal for the exploration decline is sealed with a wooden door to prevent access. To the south of the portal, located along the top of the ridge there is a service area and rock dumps. The old service area consists of an assembly of sea containers, which were originally used as a shop area. There are two main rock dumps, one is located near the portal and the second is located 250 m south of the portal. These rock dumps contain rock from the underground development. The piles were started in 1993 and the last waste rock was added in the summer of 1996.

There are several pieces of mobile equipment currently stored on top of the main rock dump and other equipment is stored in the service area sea containers. The mobile equipment includes an unknown number of 15 tonne trucks and scoops but there are no scissor lifts. Some of the mobile equipment was initially stored in the exploration decline but that equipment was brought out to surface in 2000. The current condition of the equipment is not well known

NAMC included the following table in their 2001 reclamation plan.

TABLE 1.1: Existing Mine Equipment

Unit	Number	Condition
Unimog personnel carrier	1	Poor condition
Cat D7G Dozer	1	Operable
Compressor GD 750 cfm	1	Diesel – poor condition
Compressor GD 800 cfm	1	Diesel – engine seized
Cat 500 kW generator	2	Generator end missing from one unit
Cat 973 track loader	1	Engine full of water
BR 400 Snow Cat	1	Operable
Air trac drill	1	Needs new hoses
Kubota backhoe	1	Needs engine work

The existing camp area consists of wooden exploration camp buildings, a helipad (a wooden structure) and a steel Quonset hut hanger. These facilities are located along side Goldslide Creek in the cirque below the Red Mountain deposit. There is approximately 0.5 ha disturbed area in the camp.

All of the camp buildings are temporary wood frame buildings with no permanent foundations. They were tents originally with walls built around them. The buildings were last used in 1996 and have suffered weather related wear and tear over the years.

The camp has deteriorated to the point that actions to preserve it would not be warranted.

A lower portal collar was excavated by Lac at their proposed upper tram station, as part of their development work. The excavation did not advance beyond the collar excavation.

There is a network of narrow, steep tote roads, which connect the camp, the lower portal location and the exploration portal location. Some of the roads along the talus slope immediately below the main portal location are being reclaimed naturally as the talus slope moves each year.

For the purposes of this study, SRK assumes that the Cat D7 dozer shown in Table 1.1 is useable.

1.3.5 Environmental Liabilities

Craig (2002) reports that a cash reclamation bond of $1.5million (since reduced to $1.0million) was posted with the provincial government against the Red Mountain Project. This bond can be recovered pending the remediation of certain environmental issues, including the 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 the removal of the camp and equipment from the site. Craig (2002) furthers report that there are no other known environmental issues.

1.4 Sources of Information

The Red Mountain Project files stored in Seabridge's Toronto office comprised the main source of information. These documents were authored by a variety of consultants and previous owners (North American Metals Corp, Royal Oak Mines Ltd, and Lac Minerals), and included items such as permits, maps, drill logs, sample results, geological reports, engineering reports, tailings studies and air photos. These documents are catalogued and stored in a data room. Seabridge provided SRK with an indexed list of these files. Refer to the list of references at the end of this report.

Previous engineering studies include a partially completed 1994 Feasibility Study by Rescan Engineering Limited ("Rescan") of Vancouver, BC. (Note: At the time of the 1994 study it was thought that there was a resource that would yield 6 to 7 million "mineable tonnes", and a production rate of 2000 tonnes per day. It is now known that the "mineable tonnes" are approximately 20% of this, as described herein.)

SRK also obtained useful project information through communications with previous project personnel familiar with the project and the site conditions.

SRK did not visit the project site specifically for the purpose of this study, although one member of the SRK project team has completed fieldwork at the site in the past.

1.5 Basis of the Study

This report is considered a Preliminary Assessment, as defined by N.I. 43-101, because it includes an economic evaluation partly based on inferred resources.

SRK has deliberately avoided the use of the term "reserves" in this report. The term "mineable tonnes" in this report refers to mineral resources, including inferred mineral resources, to which SRK has applied factors for cut off grade, dilution, and mining recovery. As such, these do not qualify as reserves under 43-101. (A Pre-feasibility Study, as defined under 43-101, would be required to estimate reserves.)

2. ACCESS TO SITE

2.1 Historical Access by Helicopter

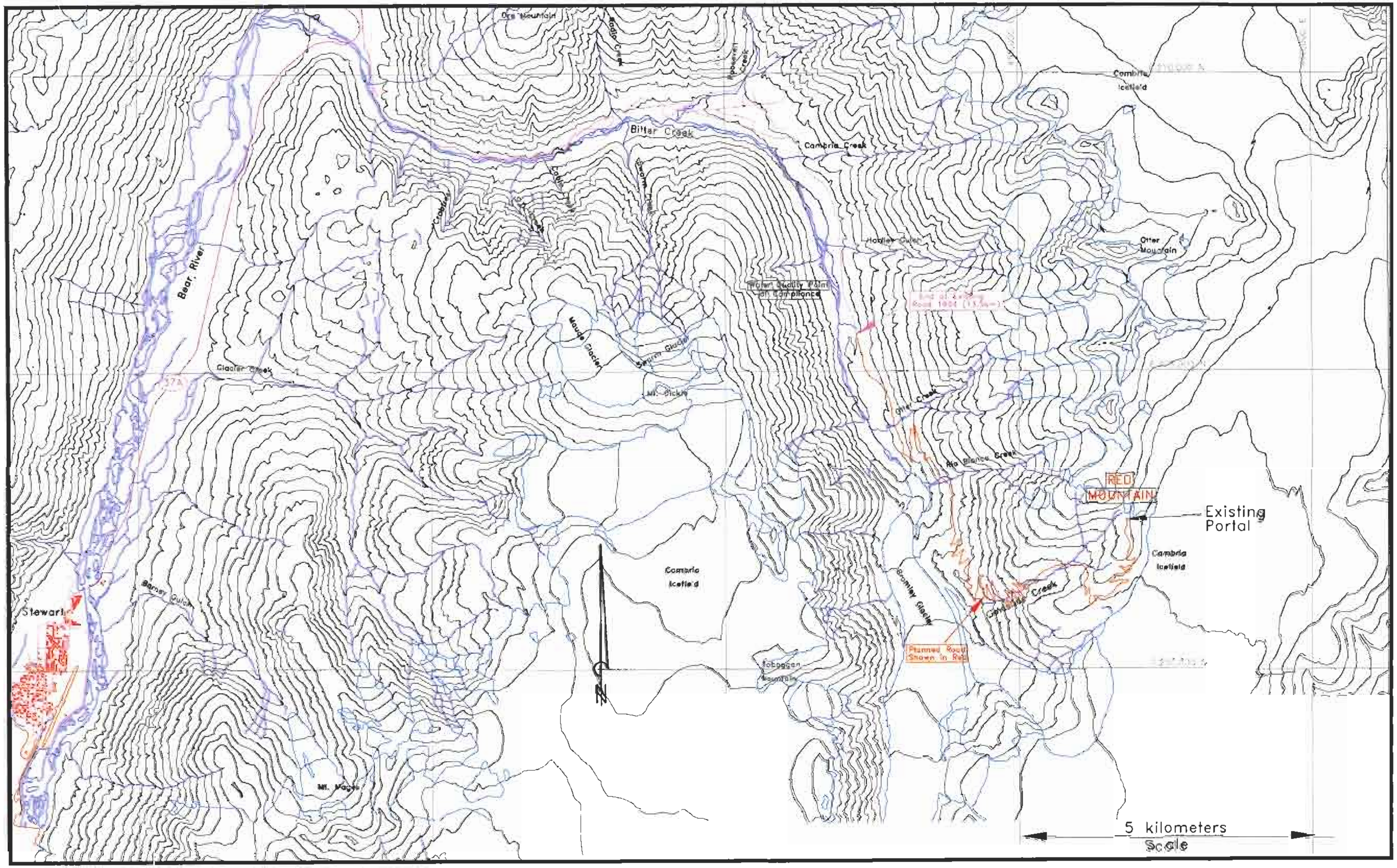
Currently, the Red Mountain Project site is only accessible by helicopter, and this has been the means of transportation that supported all previous exploration and development work.

Helicopter support utilized a staging area 10km north of Stewart at an old dry log sorting yard, located next to highway 37A, at the entrance to Bitter Creek valley. Helicopters would fly loads in following Bitter Creek valley first east, then south to the project site at roughly 1865 meters elevation. Turn around time ranged from 25 to 30 minutes per load.

Mainly Bell B205 medium lift machines were used to transport 1590kg (3500lb) loads that were prepared on surface by a forklift with a built in weigh scale. There were also requirements for light lift helicopters (B206) for moving people, and occasional needs for a heavy lift Sikorski helicopter. SRK was advised by previous project personnel that the cost of helicopter support for a typical seasonal exploration drilling and underground development program was in the order of $2 million (2003 dollars).

In 1994, Lac Minerals Ltd. ("Lac") began construction of a 14.5km access road along Bitter Creek to the bottom station of a planned tramway. The earthworks were mostly completed for the first 13.5km, with temporary timber bridges crossing creeks. The road has not been completed or maintained since that time. The alignment of the existing road is shown on Figure 2.1. It ends at a point approximately 1.8km northwest of Otter Creek.

Figure 2.1: Red Mountain Access Road



2.2 Previous Studies of Access Methods

In order to further develop the Red Mountain Project, previous engineering studies have considered alternate methods of access to the deposit. These have been reviewed by SRK. Some of the basic alternatives such as the use of a tramway to access the upper mountain, have more than one variation when the assumed mill location is changed from upper mountain to off-site. The tramway will handle production tonnage in one case, but not the other.

SRK presents below the alternatives that were deemed most suitable to the project. These are described here because SRK re-evaluated these options (section 2.3 below) to determine the best method of access.

In addition to the possible continued use of helicopter support (alternative 4 in SRK's re-evaluation), the alternatives previously studied include:

1) A road from highway 37A, along Bitter Creek valley, all of the way to the project site. The mill would be located on the upper mountain.
2) A road along Bitter Creek valley to the base of Red Mountain with a service tramway (no production) utilized to access the upper mountain project area where the mill would be located.
3) A road along Bitter Creek valley to the base of Red Mountain with a tunnel driven well into the mountain and a vertical shaft up to the project area. A muck pass would be driven parallel to the shaft, implying milling off-site.

These alternatives are further described below. Alternatives (2) and (3) are described fully in the report, "Transportation Alternatives – Order of Magnitude Study", February 1994, by Rescan Engineering Ltd.

2.2.1 Alternative (1) Road

In 2000, NAMC prepared a road design to access the existing Red Mountain portal at elevation 1865m. It is comprised of 13.5km of existing logging road (built by Lac) to be upgraded, and 18.5km of new road to the upper mountain. Refer to Figure 2.1. The planned road passes through very rugged and steep terrain with significant avalanche hazards present during several months of the year. A total of 6 bridges are required, ranging from 10 to 25 meters in length. The road design features:

- Grades not exceeding 10%
- Approximately 40 switchback curves required

- The new section of road climbs 1465m vertical
- Minimum finished width of 5m
- 8m wide pullouts every 1000m

The full specifications for the road are included in Appendix A. In early 2001 several road construction companies submitted cost estimates for the complete road construction in response to a bid package prepared by NAMC. Refer to Appendix A.

2.2.2 Alternative (2) Road and Service Tramway

This alternative is comprised of 13.5km of existing logging road to be upgraded, 2km of new road to the location of the lower tramway terminal, and the service tramway system to the upper mountain. This case considers a service tramway only, not for moving production tonnes. This implies a mill located on the upper mountain. The UTM coordinates of the proposed lower tramway terminal (Figure 2.1) are:

North	6,203,550m
East	453,075m
Elevation	542m

The upper tram terminal was planned at an elevation of 1742m and the overall tramway length was 2,900m.

2.2.3 Alternative (3) Road, Tunnel and Shaft

This alternative is comprised of 13.5km of existing logging road to be upgraded, 2km of new road to the location of the collar of a tunnel into Red Mountain, a 4.5m diameter bored tunnel 3,500m in length at +10% grade, a 686m vertical shaft, and a 702m muck pass. The tunnel starts in Bitter Creek valley at the same location as the lower tram terminal described above.

Ore would be transferred through the muck pass to trucks hauling in the inclined tunnel to an off-site mill location.

2.3 SRK Current Study of Access Methods

SRK selected the four access alternatives described above to be re-evaluated in light of the current understanding of the project. There have been two important changes since the original access evaluation work was done in 1994, these being; the "mineable tonnage" is several times smaller than originally thought, and, detailed work has been done on designing a road to the upper mountain, making it a credible alternative.

From a technical point of view, all four alternatives are feasible. The objective of SRK's re-evaluation was to select the best alternative based on capital and operating cost comparisons. From the start it was assumed that the road, alternative (1), was the optimum, and it was desired that the other alternatives only be re-evaluated sufficiently to reject them on the basis of either capital or operating costs. Details of the re-evaluation of access methods are included in Appendix B.

The cost comparisons were done over the expected life of the mine in 2001 dollars, corresponding to the date of the road cost estimate by NAMC. Contingencies were adjusted to make a fair comparison.

Table 2.1 is a summary of the capital and operating cost estimates used in the re-evaluation. It demonstrates that the road is the best option economically.

TABLE 2.1: SRK Comparison of Access Alternatives

Red Mountain Access Alternatives
Cost Summary Table

Alternative		Pre-Production Capital Cost CDN$ Thousand	LoM Operating Cost CDN$ Thousand
(1)	Road access	$ 6,800	$ 6,807
(2)	Tramway	$ 16,007	N/A
(3)	Tunnel/Shaft	$ 20,932	N/A
(4)	Helicopter	$ 4,000	$ 23,000

Notes: Operating costs for alternatives (2) and (3) are not needed.
Capital costs are in 2001 dollars and include 12.5% contingency.
(blue) considered a worst case (high) estimate
(yellow) considered a best case (low) estimate

Conclusion: Alternative (1) Road is the most economically attractive.

The comparison shown in Table 2.1 was done at the start of the current study with only a preliminary understanding of the project life and operating costs. The conclusion would not change if it were updated to reflect final estimates of capital and operating costs, and mine life.

3. GEOLOGY

The regional geology of the Red Mountain area has been described elsewhere (Greig et al, 1994; Alldrick, 1993 and Rhys et al, 1995, Craig, 2001, Craig, 2002 and references therein). The following description is a summary assembled from these sources.

3.1 Regional Geology

Red Mountain is located near the western margin of the Stikine terrain in the Intermontane Belt. In the Stewart area, the Skitina terrain comprises three main stratigraphic assemblages: 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. Primary textures are commonly preserved and regional metamorphism has reached lower greenschists mineral assemblages.

Several distinct intrusive suites intrude the volcano-sedimentary units. Late Triassic calc-alkaline intrusions, coeval with the Stuhini Group rocks, form the Stikine plutonic suite; Early to Middle Jurassic intrusions, roughly coeval with the Hazelton Group rocks, have important economic implications for gold mineralization in the Stewart area, including the Red Mountain gold-silver deposits; 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 and molybdenum deposits.

The Red Mountain area lies along the western edge of a complex, northwest-southeast trending, doubly-plunging structural culmination, which was formed during the Cretaceous when rocks of the Stuhini, Hazelton and Bowser Lake groups were folded and/or faulted.

The tectonic history of northwestern British Columbia in the Red Mountain area during the Mesozoic Era is characterized by the progressive docking of several distinct terrains against ancestral North America and involved the formation of marginal basins, sedimentation and volcanic arc magmatism and related complex deformation. During the Tertiary, the Red Mountain area was subject to extensional block faulting.

3.2 Property Geology

Red Mountain Project area is underlain by folded Middle to Upper Triassic and Early Jurassic sedimentary and minor volcanic strata that are intruded by Early Jurassic plutons, sills and dikes known as the Goldslide intrusions, and by Tertiary intrusions. Stratified rocks comprise a sequence of Triassic chert and fine-grained siliciclastic rocks gradationally overlain by Early Jurassic clastic and volcaniclastic rocks.

The Goldslide intrusions comprise a suite of extensive and variably hydrothermally-altered sub-volcanic sills, dikes and irregular intrusive bodies which intrude the Triassic and lower parts of the Early Jurassic stratified sequences. The intrusions have been subdivided into three texturally and chemically distinctive phases (Rhys, 1995): the Hillside porphyry, the Goldslide porphyry and the Biotite porphyry.

The Hillside porphyry is a medium-grained hornblende and plagioclase-phyric porphyry occurring extensively on the south ridge and the east side of Red Mountain as discordant intrusive bodies.

The Goldslide porphyry (197.1+/-1.9 Ma; U/Pb zircon; Rhys et al, 1995) is a hornblende-biotite +/- quartz porphyry intrusion underlying most of the Red Mountain cirque. The Goldslide porphyry is distinguished from the Hillside porphyry by distinct habit of hornblendes and plagioclase phenocrysts, and the common presence of quartz and biotite phenocrysts. The presence of Goldslide porphyry dikes cross-cutting Hillside porphyry and xenoliths of Hillside porphyry within the Goldslide porphyry indicate that the Goldslide porphyry is the younger of the two phases.

Sills of Biotite porphyry (201+/-1 Ma U-Pb zircon; Rhys et al., 1995) intrude cherty sediments on the west side of Red Mountain. The Biotite porphyry, although texturally similar to the Hillside porphyry, contains distinctive biotite phenocrysts and a greater proportion of groundmass. Hornblende and plagioclase phenocrysts are also smaller in size and shapes than in the Goldslide porphyry.

Intrusive breccias, breccia dikes and sills, highly disrupted bedding in sedimentary carapace, and country rock rafts are common features associated with the contacts of Goldslide intrusions, primarily the Hillside porphyry, and subordinately the Goldslide porphyry. Clasts of the Goldslide porphyry, Hillside porphyry and Biotite porphyry in the overlying volcaniclastic sequence and the geochemical similarity of volcanic rocks and the Goldslide intrusions suggest that the Goldslide intrusions are sub-volcanic high level intrusions that were comagmatic with some of the volcanic rocks.

A Tertiary stock and several types of mafic dikes intrude the Goldslide intrusions and all stratified rocks on Red Mountain. The McAdam Point Stock (45+/-2 Ma; Ar-Ar biotite; Schroeter et al., 1992) is a small Tertiary intrusion occurring at the south end of Red Mountain and extends across the east arm of Bromley glacier. It is a medium to coarse-grained biotite quartz monzonite with common K-feldspar megacrysts. The stock is associated with a 500 to 800m wide biotite hornfels thermal aureole imparting a brown to purple tint to all pre-Tertiary rocks.

Structural features in the Red Mountain area are consistent with a deformation sequence involving the development of an early widespread hydrothermal system, followed by at least one phase of folding, and displacement along northeast and northwest-trending faults.

Mesoscopic folds affect the entire Triassic/Jurassic succession on Red Mountain. The geometry and inferred timing of the folds and related fabric development suggest they are coeval the Cretaceous-Early Tertiary Skeena Fold Belt deformation. Folds have moderate to steep north to northwest-plunging axes, generally steep limb dips, and open to tight, locally isoclinal, forms. Bedding is generally upright. In the Red Mountain area, asymmetry of minor folds varies from clockwise on the west side of the mountain to counter clockwise on the east and together with bedding facing directions, suggests the presence of the large-scale north-northwest-trending Bitter Creek antiform (Greig et al., 1994). The Red Mountain deposits lie at the core of the Bitter Creek antiform. A west to southwest-dipping axial planar slaty cleavage affects Triassic strata and the Hillside porphyry and the Goldslide porphyry and crenulates hydrothermal veinlets, but it is crosscut by the McAdam Point stock and related dikes and veins. Elsewhere rocks are unfoliated, except near shear zones.

Minor fold axes in Early Jurassic strata near the summit of Red Mountain generally plunge northwest, and seldom to the northeast. The fold patterns may reflect the complex strain patterns developed around the Goldslide intrusions during a single phase of progressive deformation and folding.

At least two phases of faulting affect Red Mountain lithologies. The earliest faults are steep northwest-dipping semi-brittle shear zones that form prominent lineaments on

Red Mountain. These structures include the Goldslide and Rick faults that displace the gold-silver zones. The structures have phyllitic foliations, cataclastic textures and exhibit right lateral sense of displacement with a normal component. The timing of displacement with respect to folding is uncertain.

North to northwest trending, moderately to steeply southwest and northeast-dipping faults are developed throughout the Red Mountain area and are locally associated with mafic dikes. They cut the McAdam Point stock and all other structures, including northwest-dipping shear zones. The faults exhibit shallow dipping internal fabric, hydrothermal alteration and are associated with more steeply dipping hydrothermal veins suggesting a normal sense of displacement.

3.3 Mineralization

In the Skeena Mining Division the most economically significant metallic mineralization consists of gold-silver hydrothermal mineralization spatially and genetically related to Early Jurassic calc-alkaline intrusions and volcanic edifices. Subordinate porphyry-style molybdenum-gold mineralization with associated silver-lead-zinc veins is genetically related to Middle Eocene quartz-monzonite intrusions intruded along the eastern margin of the Coast Plutonic Complex. Both mineralization styles occur within the Red Mountain property.

Porphyry-style molybdenum and gold mineralization is associated with the Tertiary McAdam Point Stock located in the southern portion of the property. Quartz sulphide veins occur throughout the stock and country rock and the most significant mineralization is restricted to within 25m of the contact. Although local high-grade gold across less than one meter widths are reported, cursory examination of these occurrences by Bond Gold apparently did not warrant further work (Vogt, 1991).

All of the Mineral Resources estimated for the Red Mountain deposit occur in three main orebodies (Marc, AV and JW zones) interpreted to have formed during a single geological event in the Early Jurassic but later separated by Tertiary extensional block faulting. They are located within a large gossanous area extending over much of Red Mountain and hosting several other gold-silver showings (141, Brad, MCEX, Darb, Cornica, Dicksito).

The sulphide-rich gold-silver mineralization is closely related to the emplacement of the sub-volcanic and polyphased felsic Goldslide intrusions intruding the Lower

Jurassic volcano-sedimentary sequence. In the Marc, AV and JW zones the gold-silver mineralization occurs in irregular sulphide-rich stockwork forming northwesterly trending crudely tabular zones located within the carapace of the Goldslide intrusion, in the Hillside porphyry sub-phase, volcano-sedimentary rafts and intrusive breccias. Taking into account the block faulting affects, the original mineralized sheet had a strike length of approximately 600 to 700 meters, a dip length of approximately 100 to 200 meters and a true thickness varying between less than one to more than 40m.

The gold and silver-bearing sulphide stockwork depicts complex internal patterns resulting from a staged hydrothermal system transgressing stratigraphy and involving repeated episodes of veining and brecciation. Pyrite is the dominant sulphide with subordinate, locally important, pyrrhotite and sphalerite. The stockwork consists of pyrite microveins, coarse-grained pyrite veins, masses and breccia matrix developed in zones of strong muscovite alteration. Vein thickness ranges between <10cm to 80cm. Vein spacing varies between 2-10 veins per meter.

The sulphide stockwork is typically surrounded by a thicker weakly auriferous zone of disseminated pyrite, pyrrhotite and locally sphalerite alteration. This alteration envelope displays a concentric zoning pattern, with pyrrhotite disappearing sharply away from the gold-silver stockwork, typically across less than a meter. The relationship with sphalerite remains elusive.

This geological model implies a strong spatial and genetic relationship between the gold-silver stockwork mineralization and the Goldslide intrusions. As such, structural and geological controls on the emplacement (and regional distribution) of these Lower Jurassic intrusive bodies are very important exploration targeting tools within this district.

4. MINERAL RESOURCES

4.1 Mineral Resource Estimation and Classification

The most recent resource estimate for the Red Mountain project was completed by NAMC in 2001 (Craig, 2001). Previous reporting (Craig, 2002) has described the extensive QA/QC programs utilized to ensure the reliability of the data, which included the use of sample duplicates, standards and blanks and also twinned holes. SRK concurs that the data is reliable for resource estimation.

Prior to completing the resource estimate, NAMC spent a significant amount of time and effort to better understand the geology and controls on mineralization in order to construct a three-dimensional model constraining the gold mineralization. Only continuous zones of mineralization were included in the resource estimate, while isolated zones of mineralization were not. Although it could be argued that the exclusion of these isolated zones is conservative, SRK is of the opinion that this is appropriate, and that only with additional information could a resource be defined in these areas. However, in the opinion of SRK, this additional mineralization is not expected to materially alter the deposit economics.

With the construction of the geologic model, gold grades were then interpolated into a block model using ordinary kriging, while removing post-mineralization fault movements across the Marc and AV zones where possible. Grades in the JW zones were interpolated separately because of insufficient data. Semivariogram modeling and geostatistical analysis were conducted to determine interpolation parameters and to evaluate capping procedures. SRK is satisfied with this approach and the resultant parameters defined.

In the opinion of SRK, the mineralization is adequately drilled to define continuous relatively regular shapes between holes to classify the majority of the resources in the AV and Marc zones as indicated, with a drill spacing of primarily 25 meters. The JW zone was classified as inferred, which SRK agrees with, where the drill spacing is approximately 50 meters. However, SRK believes that the there exists a high probability that additional drilling will confirm the widths and grades intersected in the JW zone and not significantly change the overall zone geometry. As such, SRK recommends that this mineralization be included in the current economic study.

The mineral resources for the Red Mountain project are summarized in Table 4.1.

TABLE 4.1: Red Mountain Mineral Resources: Capped Model, 0.0g/t Au Cut Off

	Volume (M^3 x 1000)	Tonnes (x 1000)	Au (g/t)	Ag (g/t)	Au (Oz)	Ag (oz)
Marc & Av Zones						
Measured	419	1,257	8.01	31.27	323,700	1,263,700
Indicated	112	337	7.04	21.78	76,300	236,000
Total	531	1,594	7.80	29.27	400,000	1,499,700
Marc, AV & JW Zones						
Inferred	115	346	7.45	12.36	82,900	137,500
Total	115	346	7.45	12.36	82,900	137,500
Marc Zone						
Measured	238	715	8.89	38.96	204,400	895,600
Indicated	7.08	21.2	9.09	26.49	6,200	18,100
Inferred	0	0	0	0	0	0
Total	245	736	8.89	38.6	210,600	913,700
AV Zone						
Measured	181	542	6.85	21.13	119,400	368,200
Indicated	105	316	6.90	21.46	70,100	218,000
Inferred	0.446	1.34	10.52	41.20	500	1,800
Total	286	859	6.87	21.28	190,000	588,000
JW Zone						
Inferred	115	345	7.44	12.25	82,500	135,900

* Totals for volume and tonnage have been rounded to the nearest 1000 M^3 and 1000T respectively. Calculations for Au and Ag ounces have been rounded to the nearest 100 oz. (After Craig, 2001).

5. PROJECT DEVELOPMENT ALTERNATIVES

This section describes the methods employed by SRK to select the best alternatives relative to:

- Type of mineral processing plant. Three alternatives were considered.
- Operating season. Two alternatives were considered.
- Electrical power supply. Two alternatives were considered.

5.1 Mineral Processing Alternatives

5.3.1 Previous Studies

The most recent technical/economic project development studies on Red Mountain were completed by NAMC in 2000. At that time NAMC evaluated three alternatives named A, B and C, described below. SRK reviewed the work and determined that the three alternatives were applicable to the current mineral resource estimate, and that a re-evaluation was needed to select the best alternative.

All three of the alternatives were based on establishing road access to the mine site.

Case A involved trucking the run-of-mine feed to the Premier Mill near Stewart, BC. The mill is owned by Boliden-Westmin (Canada) Limited, ("Boliden") and is currently inactive. An agreement would have to be negotiated with Boliden for the use of the mill. Provisions were made for modifications to the mill and for creating a separate tailings impoundment area within the existing tailings pond. A key aspect of this case was the trucking cost from mine to mill. NAMC obtained preliminary quotes for the trucking.

Case B involved on-site milling utilizing flotation to produce a sulphide concentrate that was to be trucked to Stewart and shipped to smelters for processing. The bulk of the material processed would report to tailings, stored in a new tailings facility to be constructed on-site. The quantity of material to truck would be greatly reduced compared to case A. A key aspect was the marketability of the gold-rich sulphide concentrate, and the smelter terms that could be negotiated. NAMC thoroughly investigated this aspect.

Case C considered an on-site cyanidation mill with gold bullion being produced. Again, tailings would report to a new on-site tailings facility. There would be no product to truck other than the bullion. A key aspect of this case was though to be the increased difficulty in permitting due to the use of cyanide in the process. Also, much finer grinding was required compared to case B.

5.3.2 Current Study

In order to evaluate these alternatives, SRK prepared simple economic models for each that compared the estimated revenue to the costs on a per tonne basis. The difference indicated the cash operating margin per tonne. Considering the tonnes of resources above cut off grade, an estimate was derived for the cash generated over the life-of-mine. Capital cost estimates were then taken into account and the alternatives were compared. The calculation sheets used are included in Appendix C.

The results of the comparison indicated that Alternative B (sulphide concentrate) was not competitive with A and B, and that alternative C (on-site cyanidation mill) was the best, but only slightly better than A (Premier mill option).

Alternative B suffered from these main drawbacks:

- Flotation test work showed that a concentration ratio of no better than 4 could be achieved. This meant that one quarter of the mill feed tonnage was subject to transportation charges and smelting costs.
- The smelter terms likely to be obtained were relatively expensive and indicated a poor recovery of the contained gold value.

Regarding alternative A, SRK contacted Boliden to determine the status of the Premier mill. SRK was informed that the mill had been sold, subject to a due diligence review, and that the mill would be dismantled and removed from site. Based on this information, no further work was done on developing the details of case A.

This study is based on alternative C, with an on-site mill using a cyanide leaching process.

5.2 Operating Season

5.2.1 Previous Studies

The operating season is an aspect of the project that must be carefully considered due to the harsh winter climate with heavy snowfalls and high winds. Debris avalanche and snow avalanche hazards exist along much of the access road route. Certain areas of the project site itself are subject to snow avalanche hazards.

Several reports have been prepared in the past dealing with various aspects of the natural hazards along the proposed road, at the project site, and as they may affect the tailings storage facility. One of the more recent and relevant reports is, "Engineering Feasibility Assessment – Routing and Costing for the Red Mountain Mine Access Road, Stewart, BC.", Golder Associates (Golder), August 2000.

Previous project development scenarios have assumed either year-round or seasonal mining operations for Red Mountain. The 1994 feasibility study by Rescan was based on a year-round operation, while the more recent year 2000 work by NAMC was based on a seasonal operation of approximately 5 operating months.

5.2.2 Current Study

For this current study, SRK has selected a seasonal mining operation, from mid-May to October with allowances for annual road opening, equipment start up, and shut down in the late fall. Refer to Table 5.1.

TABLE 5.1: Operating Season

RED MOUNTAIN SEASONAL SCHEDULE								
		May	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Seasonal Total**
Calendar Days	days	31	30	31	31	30	31	184
Road Opening	days	14						14
Fall closure	days						5	5
Avail. Work Days	days	17	30	31	31	30	26	165
Mill Availability	%	93.6%	93.6%	93.6%	93.6%	93.6%	93.6%	
Mill Capacity	tpd	1020	1020	1020	1020	1020	1020	
Tonnes Milled	t	16,227	28,636	29,591	29,591	28,636	24,818	**157,500**

The schedule provides for 157,500t milled per season during 5.5 months of operating time. The length of the season was based on discussions with previous project personnel familiar with seasonal conditions on the mountain. SRK considered it prudent to avoid winter operations due to:

- Increased hazards posed to personnel

- High cost of road maintenance and expected road closure delays
- Requirements for winterizing facilities
- Increased winter operating costs due to heating

The summary section of Golder's August 2000 report discusses the hazards along the road alignment and states, "The planned seasonal use of the road will significantly reduce the impact of these hazards on road operations."

Cost provisions for road opening and road maintenance are in section 10 Operating Costs.

5.3 Power Supply

Two sources of electrical power supply have been studied: diesel generators, and BC Hydro.

5.3.1 Diesel Generators

A set of four CAT3516B diesel generators rated at 1,145kWe, 1,431kVA, 600V, 3 Phase, 60Hz, 1,200RPM for continuous base-load service would be required to supply the estimated project constant load of 2.7MW. One unit would be on standby at all times. The capital cost to purchase, install and commission the generator sets is presented in Table 5.2.

TABLE 5.2: Diesel Generator Capital Costs

Item	Price/Unit	No. Units	Cost
CAT3516B Gen Set	$537,050	4	$2,148,200
Weather Enclosure	$341,550	1	$341,550
Switchgear Unit	$391,000	1	$391,000
Switchgear Enclosure	$110,400	1	$110,400
Engineering/Test/Commission	$70,150	1	$70,150
Diesel Storage Tank & Piping	$90,000	1	$90,000
Total Capital			**$3,151,300**

The estimated power cost as supplied by the diesel generators is $0.1545/kWh, as presented in Table 5.3.

TABLE 5.3: Diesel Generator Operating Costs

Item	Cost/Operating Season	Cost/Hour	Cost/kWh
Diesel Fuel @ $0.55/litre	$526,720	$133.01	$0.1425
Oil	$12,474	$3.15	$0.0034
Preventive Maintenance	$17,147	$4.33	$0.0046
Components	$9,227	$2.33	$0.0025
Overhauls	$5,584	$1.41	$0.0015
Total Operating Cost	**$571,151**	**$144.23**	**$0.1545**

5.3.2 BC Hydro

The capital cost to install a power line from the BC Hydro grid to the mine site is estimated to be $1.60 million, as presented in Table 5.4. The table also includes $65,000 for a 250kW generator to provide standby electricity in case of a power outage.

A study by Ian Hayward International Ltd. in 1994 concluded that it would cost $1.362 million to install a 34.5kV line from a point 0.5km from the existing Aiyansh-Stewart 138kV transmission line to the mine site. This was to supply 10mVa of power to the mine site. In discussions with BC Hydro, SRK was advised that if the installed power at the mine site is only 4mVa, a 3 phase distribution line of 25kV would be sufficient, which would be less expensive to install, and the cost of substations would be greatly reduced. SRK's cost estimate includes switchgear at the mine site, and helicopter support to install the line up the valley wall, plus an allowance for inflation and other miscellaneous costs.

SRK has made the assumption that BC Hydro would cover the cost of the substation needed at the start of the new line (estimated cost of $200,000). This is the same assumption that the 1994 feasibility study costs were based on.

TABLE 5.4: Power Line Capital Costs

Item	Cost
Overhead Power Lines	$1,217,000
Helicopter Support	$117,000
Switchgear	$185,000
Other Miscellaneous	$81,000
Standby 250kW Generator	$65,000
Total	**$1,665,000**

BC Hydro provided a quote for electrical power charges for 25kV line service. The average cost for power at Red Mountain is estimated at $0.036/kWh, as presented in Table 5.5. The BC Hydro quote is included in Appendix D.

TABLE 5.5: Electrical Power Operating Costs from BC Hydro

Item	Charge	Period	Yearly Cost
Basic Charge	$4.15	Monthly	$49.80
Monthly Demand Charge	$165.05	Monthly, May to Sept	$907.79
Monthly Demand Charge	$12.22	Monthly, Oct to April	$79.43
Energy Charge	$69,373	Monthly, May to Sept	$381,552
Total			**$382,589**
Based on 10,600 MWH/season			**$0.036/kWh**

5.3.3 Conclusion

Table 5.6 presents a comparison of the two power supply alternatives. It is clear that installing a power line is the most economic alternative.

TABLE 5.6: Electrical Power Trade-Off Study Results

Option	Capital Cost	Operating Cost
Power Line	$1,600,000	$0.0360/kWh
Generators	$3,151,300	$0.1545/kWh

6. "MINEABLE TONNES"

6.1 Cut Off Grade

The selected processing option (discussed in section 5.1) is a full cyanide leach gold-silver recovery mill, located on the upper mountain. Based on this alternative, a mining cut off grade was estimated, using preliminary estimates of operating costs shown in Table 6.1.

TABLE 6.1: Preliminary Operating Costs

Preliminary Operating Costs	Cost per Tonne
Mine	$ 40.00
Road Maintenance	$ 5.00
Processing	$ 21.89
General & Admin	$ 15.33
Total CDN$	$ 82.22

These costs were based on previous work by NAMC, and adjusted based on SRK experience. Other key inputs to the cut off calculation included:

- Silver grade fixed at 26.2 g/t, at a fixed price of US$4.80/oz
- Dilution set at 15%
- Gold and silver process recoveries of 88% and 80% respectively

The gold price was varied to obtain cut off grades at different prices. The results are shown in Table 6.2. The cut off grade calculation sheet is included in Appendix E.

TABLE 6.2: Cut Off Grade Results

Cut Off Grade	
Au US$/Oz	Au G/t
$ 375.00	5.92
$ 400.00	5.55
$ 425.00	5.21

This range of gold prices was selected based on previous work by others and on SRK's knowledge of the project.

The Gemcom block model was used to highlight blocks with in-situ grades of 6g/t Au and higher, and cross sections were plotted for manual review and mark up.

Considering that the current resources are reported by NAMC at a 0.0g/t Au cut off grade, and also that the deposit consists primarily of a high grade gold, pyrite-rich core that grades into a lower grade, pyrrhotite-rich margin, SRK defined regular zones of mineralization above a 6g/t Au cut off grade within the solid body models, considering whether these zones could be separated during mining given the mining method selection and mining selectivity. The zones of contiguous mineralization below the cut off grade occurred primarily along the footwall of the mineralized zones.

The sections marked up during the manual application of the cut off criteria were then used to construct new three dimensional solids of the deposits representing above cut off grade material. The results are shown in Table 6.3.

TABLE 6.3: Resources Above Cut Off

Application of Cut Off Grade							
	M&I Total			All Categories			
	Tonnes (000's)	Au g/t	Au Oz	Tonnes (000's)	Au g/t	Au Oz	Ag g/t
Resources (no cut off)	**1596.1**	**7.80**	**400,429**	**1,941.2**	**7.74**	**482,987**	**26.2**
Resources (> 6 g/t Au)	**1058.9**	**9.22**	**313,863**	**1,216.6**	**9.14**	**357,573**	**28.7**
Gold Ounce Recovery			**78%**			**74%**	
Resources (< 6 g/t Au)	537.2	5.01	86,566	724.6	5.38	125,414	22.1
Gold Ounce Loss			22%			26%	

Resources above cut off grade in all categories (measured, indicated and inferred) total 1,216.6 kt at grades of 9.14g/t gold and 28.7g/t silver.

During the application of the cut off grade, SRK noted that in some areas, the gold mineralization was quite erratic within a zone and therefore, the cut off grade could not be applied in a realistic manner that could be achieved during mining. In such areas, the grade was estimated over the entire volume of the zone. However, additional information during mining could better define zones of mineralization above and below cut off grade, and thus improve the overall average gold grade in these areas while lowering the tonnage.

6.2 Conceptual Mining Methods

The next step in determining the "mineable tonnage" was to select conceptual mining methods suitable to the deposits and to estimate factors for dilution tonnes, dilution grades, and mining recoveries. This process was closely tied to backfill planning; its type and availability, and how much backfill would be required. Also at this stage, a preliminary assessment was made of the need for any major pillars to ensure stability.

A considerable amount of conceptual planning was required at this stage due to the nature and complexity of the deposits, with the "mineable tonnage" distributed into several separate deposits as shown in Table 6.4. It is important to note that these individual deposits were not caused by the application of a cut off grade, but are due to the nature of the mineralization.

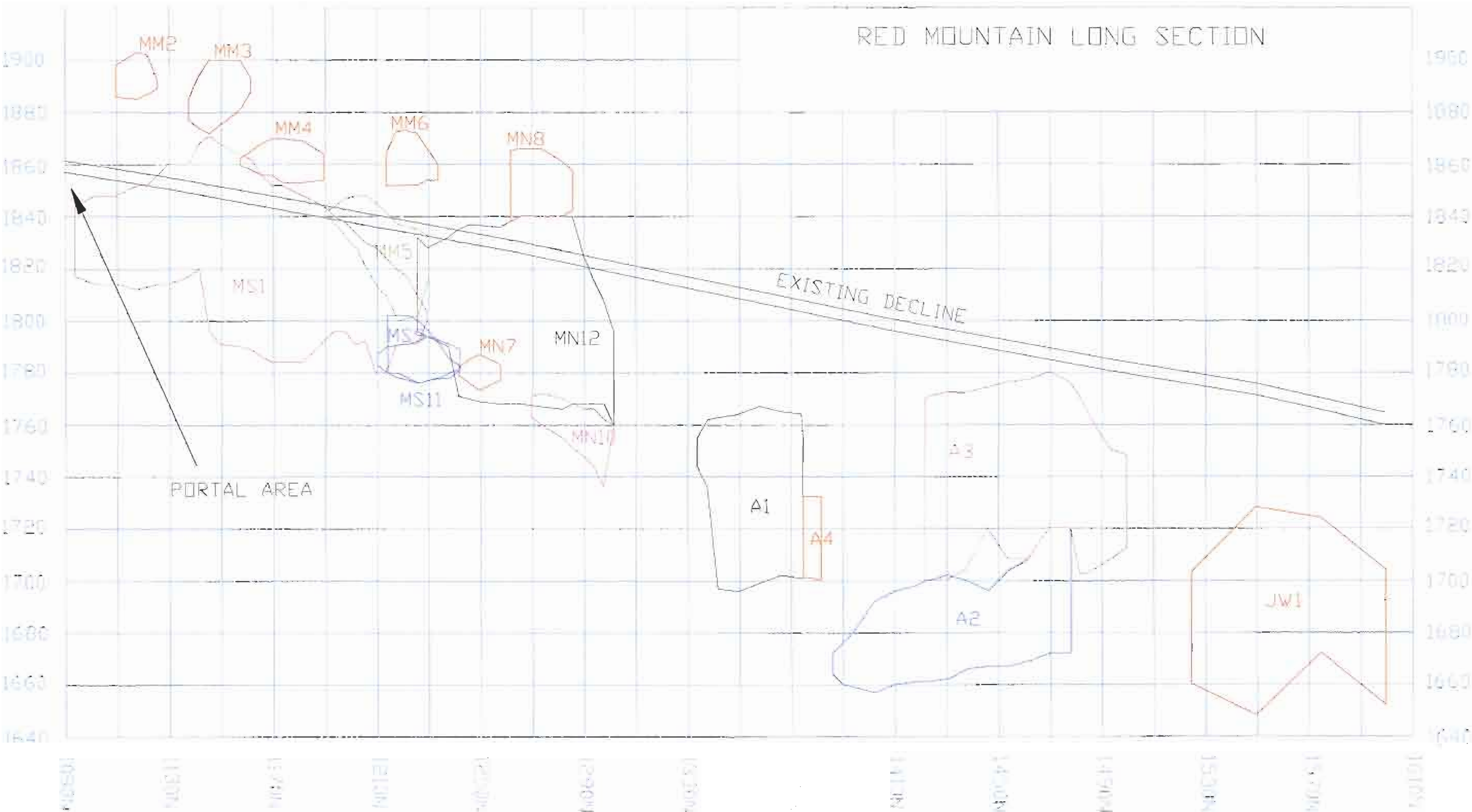
TABLE 6.4: Distribution of Resources Above Cut Off

Distribution of Resources Above Cut Off - All Categories					
		Gemcom Resources 6g/t cutoff			
Deposit Name on Long Section	Strike Leng. (m)	kTonnes	Gold g/t	Silver g/t	Mining Method
MS1 LH	136	211.3	8.57	34.1	longhole
MS1 CF		40.0	8.57	34.1	cut & fill
MM2	16	7.9	10.04	24.2	crown pillar
MM3	24	8.7	13.23	36.0	longhole
MM4	32	13.4	6.81	33.6	longhole
MM5 LH	40	17.6	10.05	35.3	longhole
MM5 CF		11.8	10.05	35.3	cut & fill
MM6	20	6.5	8.31	53.9	cut & fill
MS9	28	9.1	10.67	21.4	cut & fill
MS11	32	12.3	8.75	22.3	long hole
MN12	76	210.7	11.45	52.9	longhole
MN7	16	3.3	7.78	13.8	rm & pillar
MN8	24	7.6	12.85	53.2	cut & fill
MN10	32	20.1	13.30	22.6	cut & fill
MARC Total		580.3	9.99	40.3	
A1	40	115.8	8.57	23.4	longhole
A4	8	10.0	7.52	27.4	longhole
A2	92	77.2	10.50	21.2	cut & fill
A3 LH	78	260.7	7.57	16.6	longhole
A3 CF		15.0	7.57	16.6	cut & fill
AV Total		478.7	8.28	19.2	
JW1	75	157.7	8.62	15.0	cut and fill
Total Above Cut Off		**1,216.6**	**9.14**	**28.7**	

Table 6.4 includes resources from all categories. The Marc Deposit naming conventions are MN = Marc north, MM = Marc mid, MS = Marc south. Longhole (LH) and cut and fill (CF) designations were added in cases where more than one mining method was needed within a deposit.

Figure 6.1 is a long section through the center line of the existing decline, looking mine grid west. It illustrates the nature of the deposit.

Figure 6.1: Red Mountain Resources Above Cut Off



There are 20 individual deposits, or mining areas, for planning purposes (Table 6.4). For each of these the following was determined:

- Average true thickness and geometry
- The appropriate mining method
- Estimated dilution from hangingwall, footwall and backfill
- Dilution grade, considering gold only
- Percentage of mining void that required backfilling
- Quantities of backfill and whether or not binder was needed
- Requirements for major pillars to ensure stability
- Estimated mining recovery
- Diluted, recovered tonnes and grades for gold and silver

Several sketches were prepared to illustrate various aspects of the conceptual mining methods such as sublevel positions, backfilled areas, stope boundaries and sequencing, and recovery/loss of mineralization.

Longhole stoping was utilized as much as possible, with both up hole and down hole drilling planned. Smaller deposits were conceptually recovered as a single stope while the larger deposits were sub-divided into individual stopes, with a maximum sublevel spacing of 30m.

Cut and fill mining was envisaged as being fully mechanized utilizing 4 meter overhand cuts.

The mining recovery and dilution estimates derived were applied to the resources of Table 6.4 to estimate the "mineable tonnage". This exercise of conceptual planning was done twice, with differing backfill assumptions, as described in the next section.

6.3 Backfill Study

During the development of conceptual mining methods, it was known that the use of backfill should be carefully examined because:

- Mining recovery of the resources would be greatly enhanced through the use of backfill.
- Mill tailings were deemed unsuitable for hydraulic backfill due to the fine grind of 80% passing 37 microns.

- Paste backfill was an option, but the capital cost per tonne would be very high with a relatively small resource.
- Waste rock backfill was an option, but it was also deemed to be expensive considering the equipment operating time required to place it underground.
- The relatively small deposit could not support a high capital cost.

Due to these considerations, a trade off study was done on the use of backfill. The conversion process was completed twice, taking the resources of Table 6.4 through to a "mineable tonnage" under two different backfill assumptions:

1) Minimum use of unconsolidated waste rock backfill. Pillars were created to maintain stability. Minimize capital and operating costs.
2) Full use of waste rock backfill, including cement binder as needed, to maximize the recovery of the resources.

The increased use of backfill resulted in more "mineable tonnes" and extra cash flow (after site costs) which was compared to the extra costs for utilizing more backfill, including backfill plant capital and cement costs, as shown in Table 6.5.

In Table 6.5 the revenue from the extra ounces (measured and indicated) recovered is based on US$400/oz gold and an estimated site operating cost of CDN$75.00 per tonne.

Table 6.5 demonstrates that backfill should be used (option 2) to maximize the recovery of the resources, and this is the basis for the "mineable tonnes" summarized in the next section. In addition to the results of the economic analysis, it is also a fact that the full use of backfill will result in better stope stability than the pillars required under option 1. Option 2 represents a lower risk, with more surety of achieving planned production.

TABLE 6.5: Backfill Trade Off Results

Analysis of Backfill Use at Red Mountain				
		Option 1 Minimum Backfill		Option 2 Max. Au Oz Recovery
Backfill requirements:				
Unconsolidated waste	kt	214.6		285.7
Cemented waste 4.5%	kt	-		258.6
Total backfill	kt	214.6		544.3
Gold Recovered:				
"Mineable Tonnes"	kt	880.6		1,096.5
Gold Grade	g/t Au	8.31		8.04
Gold Ounces	Au Oz	235,254		283,371
Recovery of gold in resource	%	75.0%		90.3%
Extra Cash Flow:				
Extra Ounces with (2)	Au Oz		48,117	
Cash Flow on extra ounces	**CDN$ M**		**$ 9.9**	
Extra costs for (2) Backfill				
Capital:				
One portable u/g backfill plant	CDN$ M		$ 0.3	
LoM Operating Cost:				
Extra waste rock $7 per t	CDN$ M		$ 2.3	
Cement cost $230 per t	CDN$ M		$ 2.7	
Extra Capital + Operating	**CDN$ M**		**$ 5.3**	

6.4 "Mineable Tonnes"

Red Mountain current "mineable tonnes" are shown in Table 6.6 based on all categories of resources; measured, indicated, and inferred. (The inferred resources are related to the JW1 deposit exclusively.) The parameters used are:

- Cut off grade of approximately 6g/t Au at US$375 gold
- Overall mining recovery of 89% of tonnes
- Overall dilution of 14% at < 1g/t Au

TABLE 6.6: "Mineable Tonnes"

Dilution, Recovery and "Mineable Tonnage"							
	Mining	Mining	Dilution	Dilution	"Mineable Tonnage"		
Deposit Name	Method	Recovery	Au g/t	Percent	Tonnes	Au g/t	Ag g/t
MS1 LH	longhole	90%	1.5	13%	217,388	7.64	29.79
MS1 CF	cut & fill	85%	-	14%	39,617	7.35	29.22
MM2	crown pillar						
MM3	longhole	90%	-	16%	9,388	11.06	30.10
MM4	longhole	80%	-	17%	12,979	5.64	27.81
MM5 LH	longhole	80%	-	14%	16,359	8.66	30.41
MM5 CF	cut & fill	85%	-	14%	11,665	8.61	30.20
MM6	cut & fill	85%	-	22%	7,017	6.49	42.10
MS9	cut & fill	89%	-	18%	9,929	8.72	17.49
MS11	long hole	85%	-	15%	12,353	7.43	18.95
MN12	longhole	92%	1.0	12%	220,178	10.17	46.53
MN7	rm & pillar	80%	-	21%	3,362	6.15	10.90
MN8	cut & fill	75%	-	28%	7,912	9.20	38.08
MN10	cut & fill	85%	-	14%	19,906	11.42	19.38
MARC Total		**88%**	**0.84**	**13.2%**	**588,052**	**8.77**	**35.37**
A1	longhole	95%	1.5	11%	123,528	7.77	20.86
A4	longhole	95%	2.0	12%	10,751	6.80	24.07
A2	cut & fill	85%	-	23%	85,353	8.07	16.28
A3 LH	longhole	92%	1.0	13%	275,006	6.70	14.48
A3 CF	cut & fill	80%	-	13%	13,771	6.60	14.47
AV Total		**91%**	**0.81**	**14.1%**	**508,409**	**7.19**	**16.53**
JW1 Total	cut and fill	**85%**	**-**	**19.0%**	**165,397**	**6.99**	**12.13**
Total "Mineable"		**89%**		**14.3%**	**1,261,858**	**7.90**	**24.73**

Table 6.7 provides an overview of the conversion process from in-situ resources through to "mineable tonnes".

TABLE 6.7: Overview of Conversion Process

Red Mountain Resources and "Mineable Tonnes"

(Thousands of tonnes) Full use of backfill

	Measured & Indicated						M&I Total			Inferred			All Categories		
	MARC			AV						JW			TOTAL		
	Tonnes	Au g/t	Ag g/t	Tonnes	Au g/t	Ag g/t	Tonnes	Au g/t	Au Oz	Tonnes	Au g/t	Ag g/t	Tonnes	Au g/t	Ag g/t
Resources (no cut off)	**736.7**	**8.89**	**38.6**	**859.4**	**6.87**	**21.3**	**1596.1**	**7.80**	**400,429**	**345.1**	**7.44**	**12.3**	**1,941.2**	**7.74**	**26.2**
Resources (> 6 g/t Au)	**580.3**	**9.99**	**40.3**	**478.7**	**8.28**	**19.2**	**1058.9**	**9.22**	**313,863**	**157.7**	**8.62**	**15.0**	**1,216.6**	**9.14**	**28.7**
Resources (< 6 g/t Au)	156.4	4.81	32.2	380.8	5.09	23.9	537.2	5.01	86,566	187.4	6.45	10.0	724.6	5.38	22.1
Mining Recovery	88%			91%						85%			89%		
Recovered Tonnes	510.2	10.00	40.8	436.9	8.26	19.2				134.0	8.62	15.0	1,081.2	9.13	28.9
Dilution Percent	13%			14%						19%			14%		
Dilution Tonnes	77.9	0.67	n/a	71.5	0.65	n/a				31.4	-	n/a	180.7	0.55	n/a
"Mineable Tonnes"	**588.1**	**8.77**	**35.4**	**508.4**	**7.19**	**16.5**	**1,096.5**	**8.04**	**283,371**	**165.4**	**6.99**	**12.1**	**1,261.86**	**7.90**	**24.7**

7. MINING

7.1 Geotechnical

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

The geotechnical work consisted of compiling geotechnical data collected from diamond drill logs and the mapping of surface and underground exposures, specifically the 1830m level exploration development. Rockmass classification data from the 1830m level was compared to the information from core logs to provide confidence in the interpretation of ground conditions throughout the Marc and AV zones. A lower level of confidence in estimating ground conditions for the area identified as the lower AV zone was noted.

Overall the rock mass classification broadly describes the ground conditions in the Marc zone as "good" according to the CSIR/RMR system and "fair to good" according to the NGI/Q system. The gold mineralization is the most competent rock type and the hanging wall is the least competent.

TABLE 7.1: Rockmass Classification Summary for Marc Zone

Area	RMR	Q
Hangingwall	68	12.5
Footwall	77	20
Ore	80	45

Far field estimates of the in-situ stress field indicate that the major principle stress is not oriented favorably with respect to the orebody. The major horizontal stress acts normal to the strike of the deposit and will contribute to higher stresses in rib and sill pillars. However, given that the depth of mining is shallow, ground problems related to high stress are not expected. If mining induced stresses exceed the strength of a pillar, it would exhibit yielding, rather than rock burst type behavior.

The rock mass classification data was used as an input for the design of excavation spans. The procedure followed the empirical approach (stability graph) developed by Matthews and Potvin, which also incorporates factors such as geometry of the opening, the orientation of geological structure, stress conditions and the rate of mining (stand up time). The stability graph method indicates stable spans of 15 to 20m in width.

Stable access development spans can be maintained in the Marc zone through the installation of patterned rockbolts. Areas which will require supplementary support result from the formation of wedges at the intersection of joint sets in the Marc zone, and the local intersection of major continuous fault structures. Additional local support such as shotcrete or cablebolts may be required for these areas.

7.2 Production Rate

These considerations were included in assessing the production rate; effect on process plant capital cost, mine life, rule of thumb vertical fall rate, stope cycle times and number of independent work places.

The seasonal operating schedule selected has the effect of more than doubling the "normal " mine life, lengthening the time needed to pay back pre-production capital. SRK felt that the production phase of the mine life should not be more than about 8 seasons (years).

Table 7.2 shows the application of a production rate rule of thumb, which states that a suitable production rate is equivalent to a fall rate in the range of 40 to 50 vertical meters per year. (Higher production rates are possible with more pre-development to create more independent work places.)

From Table 7.2 SRK selected a production rate of 1000 tonnes per day to achieve a reasonable mine life. This was considered acceptable and achievable on the basis of an analysis of independent work places and stope cycle times.

Table 7.3 shows that there are two main, independent production areas supplemented by smaller deposits. The two main production areas, Marc and AV are each comprised of two deposits being mined by the longhole method with reasonable stope sizes.

TABLE 7.2: Production Rate Analysis

Mine Production Rate

Tonnes per vertical meter:

Upper elevation	1880m	
Lower elevation	1660m	
Vertical meters	220m	
"Mineable tonnes"	1,261,858	tonnes
Tonnes per vertical meter	5,736	

Estimated annual production rate:

Vertical Fall Rate m/year		Tonnes per Year	**Tonnes per Day**	Continuous Mine Life: Years	Seasonal Mine Life: Seasons
40	Rule of	229,000	**654**	5.5	12.0
50	Thumb	287,000	**820**	4.4	9.6
60		344,000	**983**	3.7	8.0
70		402,000	**1,149**	3.1	6.8
80		459,000	**1,311**	2.7	6.0

TABLE 7.3: Production Areas For 1,000TPD

Major Independent Production Sources

Source	Deposit	Tonnes	Average T/Stope	Distribution TPD
(1) Marc	MS1 LH	217,388	21,000	
	MN12	220,178	30,000	
		437,566		399
(2) AV	A1	123,528	31,000	
	A3 LH	275,006	35,000	
		398,534		363
(3) Smaller C&Fill stopes			n/a	238
			Total =	1,000

The typical cycle time for a 30,000t longhole stope was estimated at 100 days by considering each step of the mining cycle including backfilling. This results in an average production rate of 300tpd. In each of the two main production areas, more than one stope can be cycled at a time, easily achieving the targets shown in Table 7.3.

Representing smaller deposits, the cycle time for a 6,800t cut in a 4.4m wide cut and fill stope was estimated at 31 days including backfill jammed tight to the back, resulting in an average production rate of 220tpd.

These analyses confirmed that a production rate of 1,000tpd should be achievable.

7.3 Mine Shift Schedule

The mine is scheduled to operate on the basis of two 10-hour shifts per day, seven days per week. There will be three mine crews following this schedule, resulting in the statistics shown in Table 7.4.

TABLE 7.4: Mine Shift Schedule

Mine Manpower Statistics				
Days per season		165days	23.6	weeks
Shift per day	(10 hrs)	2shifts		
Shifts per season		330shifts		
Number of crews		3crews		
Shifts per crew per season		110		
Shifts per miner per season		110 includes	15.7	O/T shifts
Hours per miner per season		1100 includes	157	O/T hours
Mine operating hours per week:		7days		
		2shifts/day		
		10hrs/shift		
		140mine operating hours per week		
Mine operating hours per Month:		30days		
		2shifts/day		
		10hrs/shift		
		600mine operating hours per month		
Hours per miner per week		46.7 hrs worked per week on average		
Hours per miner per month		200 hrs worked per month on average		
Shifts per miner per month		20 shifts worked per month on average		

It was assumed that the miners will want to work more than 40 hours per week due to the seasonal schedule.

7.4 Mine Layout

The mine layout was prepared as a 3 dimensional model using Gemcom software. The model includes:

- Surface topography
- Ore deposit solids
- Major faults
- Existing exploration decline
- Planned ramps, levels, cross cuts and raises
- Mine grid coordinate system

Figure 7.1 is a 3-dimensional model of the underground mine, looking west (UTM).

Surface features of the mine site were designed in AutoCAD. Figure 7.2 is a plan view of the mine site on upper Red Mountain. This figure shows the relationship between the UTM grid and the mine grid, as well as the main surface infrastructure discussed in the following sections.

Underground access is through two portals; the existing decline at 1860m elevation, and a new portal at 1650m elevation (mine coordinates 1170N, 4185E). There are no raises connecting to surface.

The existing exploration decline is approximately 5m wide by 4m high, at a grade of –17%.

There is only one general area safe from avalanche hazard, which is the ridge northeast (UTM grid) of the existing exploration camp. This dictated the location of the surface mill. The new portal is designed to provide an efficient haulage route to the mill area. The portal elevation was selected to provide gravity drainage from most of the underground mine to prevent mine flooding during the winter.

Figure 7.3 is a photograph of Red Mountain showing the area of much of the proposed mine infrastructure.

Figure 7.1: Red Mountain Underground Mine Model

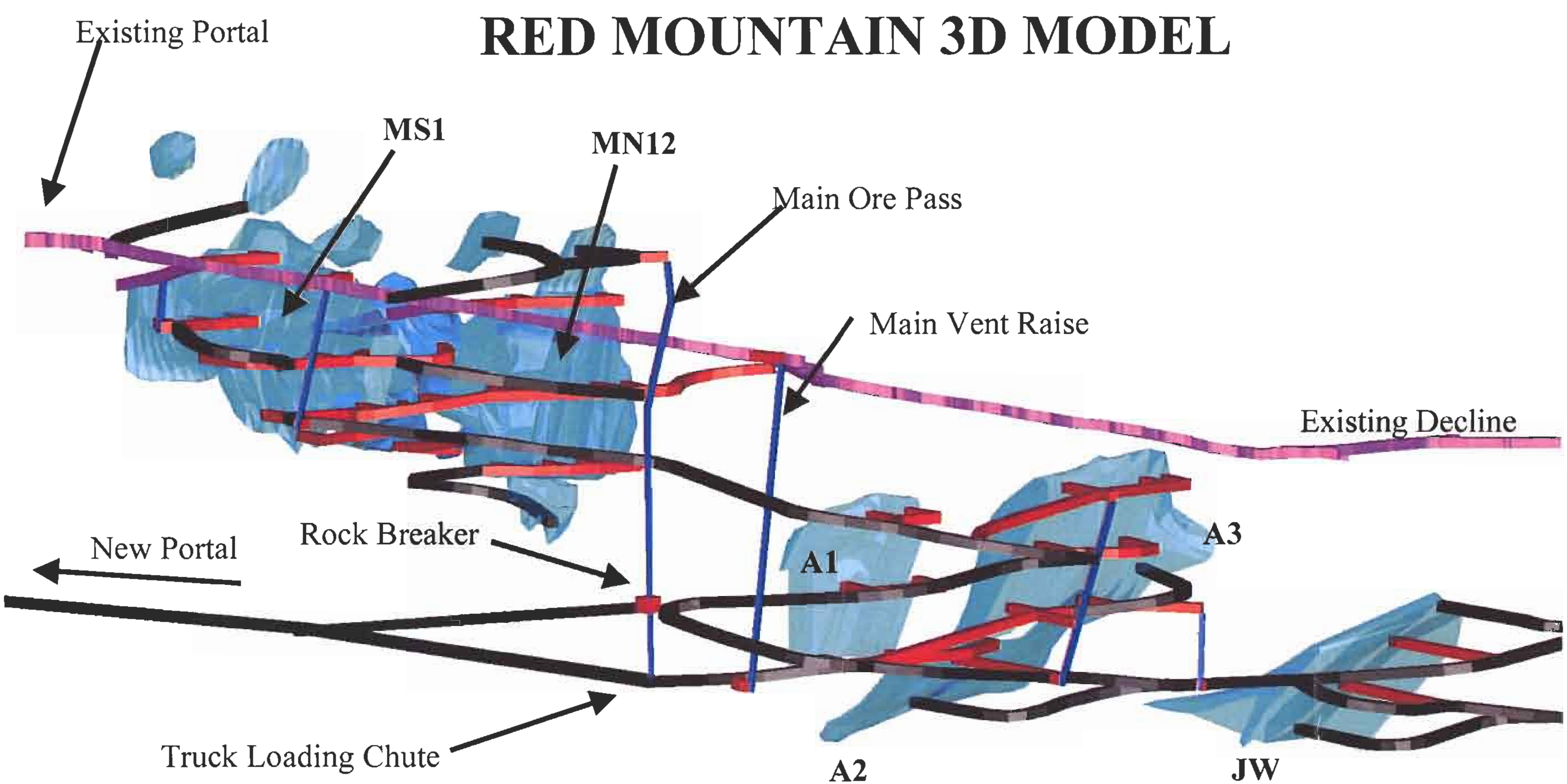


Figure 7.2: Red Mountain Site Plan

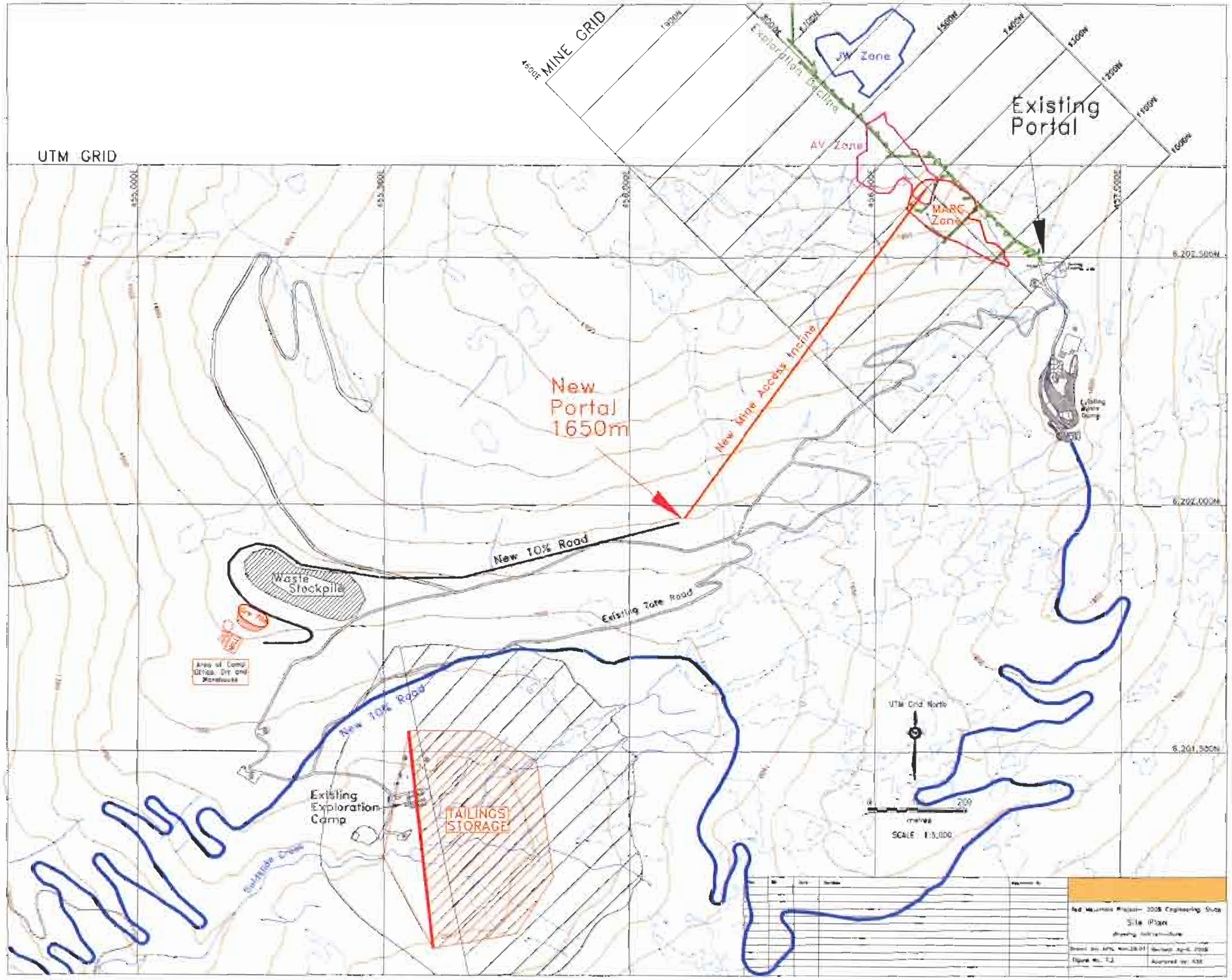


Figure 7.3: Red Mountain Photo

Photo looks mine grid east into Red Mountain cirque. Approximate mill location shown. Arrows indicate: Red: Exploration portal, Yellow: New 1650m Portal, Green: Existing exploration camp just left of Goldslide Creek.

From the new portal, the access drift (main haul route) extends 625m at +5.7% grade to an intersection. From that point, the main haulage continues for 225m at +2.0% to a truck loading chute at 1690m elevation.

Ramps are generally sized at 4.4m wide by 4.0m high to accommodate the 20t trucks employed. Ramp grades range from 12% to 15%. Levels and cross cuts are sized at 4m x 4m in waste, and safety bays are required. Level spacing is variable, up to a maximum of 30m.

Where longhole stoping is planned, development in the deposit at each level is sized at 5m wide x 4m high. This development has been estimated, but it is not shown in the model as 3D solids.

7.5 Development Summary

Table 7.5 is a summary of all lateral development required over the life of the mine. It is totaled showing main ramps separately from the three main deposits; Marc, AV and JW. For each deposit, quantities of lateral mineralized and waste development are tabulated, and waste development is sub-divided into ramp, level, cross cut, and secondary cross cut. Secondary cross cut is comprised mainly of slashing to establish successive accesses to cut and fill stopes after the primary access cross cut is driven to the first cut.

Table 7.5 indicates the allowances that have been used to account for development details not included in the Gemcom model. Quantities of line advance development for ramps levels and cross cuts have been increased by the percentages shown.

On the right side of the table, ratios are shown for "mineable tonnes" per meter of lateral development. This provides a measure of how development intensive each deposit is.

Life of mine raising (not shown in Table 7.5) requirements total 478m, with 304m of ventilation raising and 174m for a main muck pass.

TABLE 7.5: Lateral Development Summary

Red Mountain Lateral Development									
	Lateral Waste Meter Details								
Area	Ramp meters	Levels meters	Primary Cross Cut meters	Secondary Cross Cut meters	Total Waste m	Total Ore m	Total O + W Lateral meters	Mineable Tonnes kt	Ratio Tonnes per Meter t/m
	+15%	+30%	+15%	← factors to cover misc development					
Total Main Ramps	2,493	-	-	-	2,493		2,493	excluded from below	
Marc Total	593	803	895	372	2,664	706	3,369	588.1	175
AV Total	460	684	409	160	1,713	592	2,305	508.4	221
JW Total	707	-	138	198	1,043	-	1,043	165.4	159
MINE TOTAL (m)	4,254	1,487	1,442	730	7,913	1,298	9,211	1,261.9	188

7.6 Mining Methods

"Mineable tonnage" is recovered by the methods shown in Table 7.6.

TABLE 7.6: Mining Methods

Distribution by Mining Method		
	%	kt
Development	6%	77.9
Longhole	65%	820.1
Cut & Fill	29%	363.9
Total	100%	1,261.9

Longhole stoping was maximized to take advantage of higher productivity and lower costs compared to cut and fill. Four of the largest deposits account for most of the longhole tonnage; MS1, MN12, A1 and A3. These deposits were divided into individual stopes sized at 15m to 20m in strike length, with sublevels spaced at intervals of 15m to 30m as dictated by deposit geometry and drill limitations. Individual stope tonnages range from 18,000t to 35,000t.

For each of the four deposits noted above, stope sequencing was determined to provide good mining flexibility and to minimize requirements for cemented backfill.

Longhole drilling of mainly down holes 76mm (3 inch) diameter are planned at a drilling factor of 9 tonnes per meter drilled. Fan drilling is the most common case, with parallel holes in the slot area. Slots will be opened by drop raising. Anfo is assumed as the main blasting agent.

Mucking will be accomplished by 6 yard scoops equipped with remote controls. Backfill will be waste rock, both cemented and uncemented, dumped by scoop or truck into the stope.

The longhole method has also been applied to smaller deposits such as MM3, MM4, and MM5. These deposits can be opened as a single stope, using up holes from a mucking horizon. In some cases, cut and fill mining is used to level up the bottom section of the deposit, or to reduce the up hole length needed to reach the upper limit of the deposit.

Overhand cut and fill mining is planned in 4m lifts, with uncemented rock backfill being placed in each completed cut and jammed tight to the back by a "rammer jammer" attachment on a 4 yard scoop. Other methods of leveling the backfill such as using a small dozer were considered, but they are not well suited to Red Mountain due to the moderate dip and/or limited deposit widths where cut and fill is planned.

Mucking in the cut and fill stopes is achieved with 4 yard and 6 yard scoops, depending on the width. Three of the cut and fill stopes will require cut widths of down to 3m to control dilution. These are MM6, MN8, and A2. The total tonnage in the narrow sections is estimated at 20,000t representing 5.5% of total cut and fill mining. The 4 yard scoop specified is 2.1m wide, therefore it may be necessary to mine some of these narrow cuts with a 2 yard scoop.

7.7 Mine Equipment

Mine mobile equipment requirements are listed in Table 7.7.

For most of the equipment shown, requirements (units purchased and operating hours) have been based on estimated productivities and scheduled quantities of work. In the case of scoops and trucks for example, all haulage requirements for mineralization and waste tonnage, in and out of the mine, have been modeled for cycle times and productivities. The number of units needed have been derived based on the mechanical availability and utilization factors that can be achieved. Schedules for annual equipment purchases and annual equipment operating hours provide input to the capital costs and operating costs in the economic model.

Referring to Table 7.7, a personnel carrier is provided to transport miners into the mine from the mine dry. Some of the other mobile equipment will be parked on surface at shift change, such as trucks, grader, mechanics vehicles, etc., and these will also be used to transport miners into the mine.

A hiab truck and forklift will be used to unload supplies and transport them into the mine.

Definition drilling requirements are described in section 7.8.8. A small electric diamond drill is provided, drilling AQ core.

TABLE 7.7: Mobile Mining Equipment

Mine Mobile Equipment	
Function	**Units**
Supervision	
Kubota Tractor	1
Personnel Transport	
Getman carrier	1
Nipping	
Getman hiab	1
Forklift (surface)	1
Definition drilling	
Gopher drill (AQ)	1
Development	
Jumbo 2 boom	1
6 YD Scoop	1
Scissor lift	1
Truck 20T	1
Production & Backfill	
Jumbo 2 boom	1
Scissor lift	2
6 YD Scoop	2
4 YD Scoop	2
Truck 20T	2
LH Drill (76mm)	2
Anfo Truck	1
Road Maintenance	
Grader	1
Construction	
Getman hiab	1
Maintenance	
Kubota Tractor	1
Mechanics hiab	1
Scissor lift	1
Backfill Supply (surface)	
980 Loader	1
D7 Dozer	1

For the functions of development, production and backfilling, equipment requirements are variable, and units will move among these functions over time. These requirements have been carefully scheduled to determine equipment needs. For example; initially two independent development crews are needed, but later when cut and fill production is starting up, development requirements taper off to one crew, freeing a jumbo for cut and fill mining.

All bolting will be done from scissor lifts using stoppers and jacklegs. One longhole drill can handle all of the production drilling requirements, but a second unit is specified for backup and for utility work such as drain holes and electrical holes. One anfo loading truck is provided to handle longhole explosives loading and some cut and fill loading. The rest of the explosives loading will be done using small portable anfo loaders.

One grader is provided and it will also be responsible for the mine access road.

The construction crew is provided with a hiab truck for bulkhead construction, repair work, and small projects.

The mechanics will utilize a tractor and a hiab truck. The electricians are provided with a scissor lift to allow access to the back for electrical installations.

On surface, a 980 front end loader will load trucks with development waste to be back hauled into the mine as backfill. The D7 dozer will be used on the surface waste piles during pre-production when the piles are being built up.

7.8 Mine Services

7.8.1 Site Access and Transportation

Access to the site will be by road as described in section 2.2.1. From Stewart, BC it is 12km along highway 37A to the mine road turn off at Bitter Creek. The new gravel road from highway 37A to the mine dry will be an additional 30km. Mine employees will be transported to and from Stewart each shift in vans or busses, with an estimated one way travel time of 40 minutes. For safety reasons, personal vehicles will be prohibited on the mine road.

The road will be opened each spring beginning May 1. A period of 10 to 14 days has been allowed to open the road, working from both ends with D7 and D8 dozers. Personnel will be flown in by helicopter to begin work on the upper road snow clearing. A small camp is provided at the mine site for these workers during road opening.

During the operation of the road, avalanche control measures will be used to reduce this risk. Avalanche technicians will be flown in by helicopter to assess conditions and bring down potential avalanches using explosives. These provisions are included in the general and administrative operating costs, section 10.3.

7.8.2 Electrical Power Supply

Total installed power at the mine site is estimated to be 3.9MW, as presented in Table 7.8. The total amount of power expected to be drawn at any one time is 2.7MW.

TABLE 7.8: Power Supply Requirements

Area	**Installed Power (kW)**	**Derating Factor**	**Power Drawn (kW)**
Mine	1,261	59%	742
Surface Operations	97	73%	71
Mill	2,500	75%	1,875
Total	**3,858**	**70%**	**2,688**

Power will be supplied to site by an overland power line, connected to the BC Hydro grid. A 25kV line approximately 18km in length will supply 4mVa of power to the mine site. The new branch line will be connected to a 138kV line that runs along the Bear River valley from Meziadin to Stewart. A 0.5km line will be installed to a 138/25kV substation that will feed the 18km power line to the mine/mill site. The first 15km of the line will parallel the proposed access road. The final 3km from the valley floor to the mine site will be constructed at pre-selected structure sites using helicopter support. The line will end at a 25kV/4160V substation at the mill.

A 250kW generator will be installed at the mill for standby power in case of a power interruption in the BC Hydro supply. The generator will be capable of supplying power to the thickener, leach tank agitators, and other necessary items such as lighting.

7.8.3 Compressed Air, Water

Compressed air requirements will be modest due to the use of electric/hydraulic underground equipment; the development jumbos and the main longhole drill. Maximum air consumption is estimated at approximately 0.5 cubic meters per second

(1,000cfm). Pneumatic equipment includes stoppers, jacklegs, face pumps, shotcrete machine, ventilation doors and chute/chain controls.

Compressed air will be provided by two, 1032cfm, rotary screw, electrically powered compressors, with one operating and one on standby.

During pre-production, prior to the power line being commissioned, provisions have been made for three rental diesel compressors, with one at each portal and one on standby.

The underground water supply will be sourced from the unused, flooded portion of the existing exploration decline (the portion north of about 1400N). Once the main ventilation raise is connected to the exploration decline, a water supply line will be installed in the raise to supply the lower mine area.

SRK has not determined what the best temporary water supply is for the first few months of development on the lower (1650m portal) mine access. This will be best determined once crews are on-site. One option is to pipe water along the tote road from the exploration portal to the 1650m portal.

7.8.4 Mine Dewatering

Mine water will be handled without clarifying it, using portable submersible pumps located in several sumps throughout the mine. All mine water will be handled through the lower portal at 1650m elevation, and directed into the tailings pond.

Pumping power requirements are expected to be low since most of the mine is well above 1650m elevation. Most of the water will be handled by small submersible pumps of 5 -10kW and 100 - 150mm diameter piping. Some of the water will be drained from sumps through drain holes and into the drainage piping system.

During the winter period, the deepest portion of the mine will be allowed to flood, while most of the mine will be drained by gravity to the tailings pond.

7.8.5 Ore and Waste Handling

As shown in Figure 7.1, a main muck pass is located centrally and all muck will pass through a rock breaker installed at 1720m elevation. The muck pass above the rock breaker will be equipped with control chains to control the flow of muck onto the

300mm x 380mm grizzly. The rock breaker will accept muck from the muck pass above, or from trucks or scoops dumping directly onto the grizzly.

Below the rock breaker, a 25 meter section of the muck pass will be slashed out to dimensions of 4m x 4m, creating a small storage bin above the pneumatic truck loading chute at 1690m elevation at the bottom of the main muck pass. 20t trucks will haul from the 1690m chute to 1650m portal and to the surface storage pile located next to the mill. A reclaim system will draw feed for milling from the stockpile.

Table 7.9 shows the quantities of development waste rock that must be handled over the life of the mine.

TABLE 7.9: Life of Mine Development Waste Rock Handling

Total Waste Rock from Development		
Created:	During pre-production	141,915
	During production	256,617
	Total	398,531
Waste Handling		
To:	Upper waste pile	55,814
	New lower waste pile	110,517
	Remains underground	232,200
	Total	398,531

All of the waste shown in Table 7.9 is planned for use as mine backfill. The existing upper waste pile on the ridge, next to the collar of the exploration decline, is estimated at roughly 90,000 tonnes. Refer to Figure 7.2. It will also consumed as backfill such that no development waste will remain on surface at the end of the mine life.

Waste will be placed into stopes by scoop and truck, and this work has been modeled to determine operating hours and costs. The upper and lower surface waste piles will be hauled into the mine by truck.

7.8.6 Backfill Supply and Delivery

The quantities of cemented and uncemented backfill required over the mine life are shown in Table 7.10, as well as the sources of supply.

The quantity shown in the ridge (upper) stockpile of 145.8kt includes the 90kt that is already piled there.

Basically the backfill comes from three sources; the existing ridge stockpile, waste from planned underground mine development, and waste excavated from a surface borrow area near the lower waste pile. At the lower mine portal, a 980 front end loader will fill the 20t trucks, while at the upper portal, a 4 yard scoop will perform the loading.

TABLE 7.10: Life of Mine Backfill Requirements

Backfill Required for Mining			**Thousands of Tonnes**	Distribution
Upper Mine:	uncemented		149.6	
	cemented		135.9	
Lower Mine:	uncemented		235.4	
	cemented		122.6	
Total Backfill Required			**643.5**	100%
Supplied from:				
1Existing ridge stockpile, hauled in existing decline			145.8	23%
2Underground development during production			232.2	36%
3Hauled in through new 1650 portal				
	3aMine development waste	110.5		
	3bSurface borrow material	155.0		
		265.5	265.5	41%
Total Backfill Supplied			**643.5**	100%

All of the quantities shown on Tables 7.9 and 7.10 have been scheduled on a monthly basis for planning purposes, to estimate equipment requirements. Backfill requiring jamming tight to the back in cut and fill stopes was identified.

Cemented backfill will be prepared underground, near the empty stopes, using a portable backfill plant consisting of an automated cement slurry mixing and delivery system. A normal Portland cement slurry will be sprayed onto the waste rock as it is dumped into the stopes by truck or scoop. Cement will be delivered underground in one tonne tote bags.

The portable plant includes a cement hopper and metered feed auger to the mixing tank. The cement slurry is piped up to 75m to a spray bar actuated by a pull cord at the dump point. A backfill plant operator is included in mine manpower.

7.8.7 Ventilation

Total mine ventilation requirements are estimated at 164 cubic meters per second (cms) or 347,000cfm, utilizing the existing exploration decline as the main air intake and the new 1650 portal/access drift as the exhaust air route.

The overall air flow rate has been estimated by assessing the mobile equipment fleet diesel engine power ratings, and by comparing to other mines. Refer to Appendix F for details.

The mine ventilation network has not been modeled, and the main ventilation fan requirements are based on comparisons to other mines and SRK's experience. The main fans are specified as two, 1.5m (60 inch) diameter axivane fans in parallel, each powered by a 150kW (200HP) motor. These fans are to be installed underground in the exploration decline, just inside the existing portal. The fans will be housed in a bulkhead to be constructed in a by-pass drift, with air flow in the decline controlled by two sets of ventilation doors. A preliminary drawing of this arrangement is in Appendix G.

During the development of the mine, it will be a priority to drive the main ventilation raise shown in Figure 7.1. It establishes an important flow through ventilation circuit between development work in the upper mine (existing decline) and the lower mine (new 1650 portal). This work is included in the development and production schedule indicating that flow through ventilation is achieved prior to the start of production.

A total of 5 ventilation raises are included in the mine model, Figure 7.1. Work area ventilation will be provided as needed by 6 auxiliary fans (56kW) included in mine capital.

7.8.8 Definition Drilling

A small amount of underground definition drilling has been scheduled for two purposes:

- To fill in the JW deposit to a 15m x 20m pattern to support mine planning

- To provide a capability to test any areas of uncertainty in and around planned stopes (in-stope drilling)

To put this in perspective, the total drilling scheduled requires less than one drill operating one shift per day (ranges from 0.50 to 0.76 operating drills). An electric powered Gopher drill utilizing AQ core size is assumed. It can be moved in a scoop bucket after disassembly.

The JW deposit drilling must be completed mainly during seasons 8 and 9 of the production schedule, once suitable drill position has been developed underground. The total drilling is estimated at 1,595m with average 55m hole lengths.

The in-stope drilling is set at 270m per month during the production period. This is equivalent to one drill working half of the time on day shift.

7.8.9 Equipment Maintenance

The maintenance shop will be an ATCO Fold-A-Way® building, 25m x 37m, located on surface near the mill. The structure will be placed on a concrete slab, and erected using a crane. An overhead crane inside the shop will be supported on a steel beam structure, and a fire suppression system will be installed. The shop will be equipped with the usual equipment and tools to allow for a preventative maintenance program.

It is intended that the maintenance program will track the status of all mobile equipment, (such as operating, standby, downtime) to allow mechanical availabilities to be monitored and to provide a basis for planning PM work and major component changeouts. Equipment component overhauls will be carried out off-site. A work order system will be used to record parts and labour costs to each PM or repair job. An oil sampling and analysis program will be set up.

Due to the seasonal nature of the mining operation, there is an opportunity to perform maintenance work on the mobile equipment during the off season, if required. This could be done by moving some of mining equipment and the necessary tools to the Stewart warehouse/office building owned by Seabridge. A small number of mechanics could be retained beyond the normal end of the operating season, and they would be supported by the purchasing and warehosing functions located in the same building.

7.8.10 Mine Dry, Offices and Camp

These structures will be located in the area next to the mill.

The mine dry will be built from two ATCO wash car units, with a total area of 15m x 36m. The dry will be capable of handling 80 people at one time. The units are skid-mounted and can be transported by truck to site.

An office complex will be constructed from four ATCO office skid-mounted trailers, providing a total office space of 15m x 50m. The complex will contain offices for the Mining Department, the Geology Department, and Site Management.

Another ATCO office trailer will be part of the complex to provide a First Aid/Mine Rescue office.

A 16-man camp will be installed to provide sleeping quarters during start-up operations at the beginning of each working season, prior to road opening. The camp will consist of two 8-man ATCO Wet Sleeper units and one ATCO Kitchen trailer. The term Wet means that washroom facilities are built into each trailer.

7.8.11 Warehousing

Seabridge currently owns a warehouse/office building in Stewart, BC, located within a fenced yard. This facility will be utilized for the project along with an additional warehouse trailer at the mine site. Two ATCO units will serve as an on-site warehouse, providing 24m x 36m of space. The on-site warehouse will contain a supply of the most common consumable items. Large, and less-consumed items will be kept in the warehouse in Stewart.

A 10-tonne delivery truck will be purchased to deliver supplies from Stewart to the mine site. Materials management and purchasing functions will be located in the Stewart office.

7.9 Production Schedule

The production schedule for the underground mine (mine schedule) was prepared as a monthly schedule, assuming the mining was to be done on a continuous basis. This approach facilitated the scheduling process. The physical quantities from this monthly schedule were then divided into seasonal (5.5month) totals for input into the overall project schedule (project schedule).

The methodology, key assumptions and results of the mine schedule are discussed below. Month 1 in the mine schedule is defined as the month in which the new portal is established at 1650m elevation. Services including compressed air, water, and electricity would be set up next to the new 1650m portal, and the existing portal, during this same month. The start of season 1 in the project schedule is defined as the start of access road construction in May of that year.

7.9.1 Mine Schedule Methodology

Preparation of input data for the mine scheduling process included estimating quantities of development, lateral development advance rates, "mineable tonnes" and grades by deposit, maximum limiting production rates, ventilation raising requirements, muck pass requirements, and advance rates for the alimak raising.

Preliminary scheduling indicated clearly that mine development would have to proceed on two fronts; through the existing exploration decline, and through the new lower portal. The mine was considered split into two parts for scheduling purposes; an upper mine and a lower mine, which would ultimately be joined as the mine development advanced. Referring to Figure 7.1, the upper mine is to the left of the main muck pass, while the lower mine, accessed through the new 1650m portal, is to the right.

From the 3D mine model, the quantities of development related to bringing each individual deposit into production were determined, and the interrelationships among various ramps, levels and cross cuts were determined in terms of allowable sequencing.

The most critical faces were identified in the upper and lower mine areas. These were:

- Lower mine: main access drift towards main muck pass/vent raise area.
- Upper mine: 1802m level development leading into main ramp.

The assumed advance rate on these two single faces impacts directly on the length of time required to reach production readiness. Logically, all other activities are dependent on having the main access routes in place. An advance rate of 174m line advance per month was selected for the lower mine face, and 130m/mo. for the upper mine.

A provision for ramp cut outs (second faces) such as remuck bays was set at 15% (in addition to line advance). When this is added to the monthly line advance, the development totals become; lower mine at 200m/mo and upper mine at 150m/mo, as shown in the mine schedule. It was determined that the advance on the upper mine face was not as critical as in the lower mine.

The lower mine, line advance rate requires 50 rounds per month at 3.5m advance per round. With two 10hr shifts per day, there are 60 shifts per month to accomplish these rounds, normally cycling a round each shift. SRK believes that this is reasonable given that good ground conditions are expected. In general, miners have been scheduled to achieve 1.3m per man-shift (10hr shift basis), but extra men have been added to the mine schedule through to the end of month 10 to help push the critical faces.

During this period, each single face has two miners assigned each shift, equipped with a 2-boom jumbo, two 6 yard scoops, a 20t truck, and a scissor lift. A mechanic is assigned to each shift, and electricians are scheduled on day shift.

"Mineable tonnes" and grades were input to the schedule for each deposit. Estimated stope production rates were input to the schedule as constraints.

As the development and production schedule was created, an effort was made to smoothen out the peaks and valleys in the development advance required per month, particularly during the pre-production period. Higher grade deposits were selected for early production where possible.

In creating the production schedule, steady mill feed was not started until:

- The initial deposits to be mined were accessed by ramp, and appropriate level development was completed to start production.
- The main ventilation raise was driven, connecting the upper and lower mine areas, creating up a flow through ventilation circuit.
- The main muck pass and rock breaker were constructed.

7.9.1 Schedule Results

Highlights of the schedule include:

- The pre-production period is defined as season 1 through to the end of season 3 in the project schedule. This corresponds to the end of month 9 in the monthly mine schedule.
- 7.9 years of full production of 157.5kt/a (1000tpd) starting in season 4 through season 11. Total tonnes mined is 1,261.9kt.
- Mine production starts at the end of season 3, and full production is achieved at the start of season 4.
- The mill is commissioned at the end of season 3.
- Total development waste is 398.5kt During pre-production, 141.9kt of this total is generated over an 8 month period at an average rate of 590tpd.
- Mine development ratios are 0.32 waste tonne per mineralized tonne, and 137 mineralized tonnes per meter of lateral development (lateral development here includes all ramps).

Table 7.11 shows the project schedule, based on a seasonal operation. It includes the results of the monthly mine schedule shown on a seasonal basis. (The monthly mine schedule is included in Appendix H.)

The project schedule reflects the access road being useable by the end of the first season, with completion of the road to specification at the start of the second season. This is a key assumption in the project schedule. Establishing a useable tote road in the first season was discussed with one of the road construction companies that provided a construction cost estimate in 2001 (they worked on the Eskay Creek mine road). The company representative expressed a high level of confidence that the road could be established for traffic in the first season.

Construction of the power line will be started in season 1 and completed in season 2.

Season 2 and most of season 3 are available for the construction of the mill and the tailings dam.

TABLE 7.11: Project Schedule

RED MOUNTAIN PROJECT SCHEDULE														
Season		**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**Total**
		← Pre-Production →			← Production Period →								
Road Construction													
Mill Construction													
Tailings Pond													
Power Line													
Pre-production U/G Dev.'t													
U/G Development													
Ore Dev.'t	m		-	227	376	100	100	403	57	-	35	-	1,298
Waste Dev.'t	m		1,225	1,743	1,062	423	658	847	690	519	483	267	7,913
Total Lateral	m		1,225	1,970	1,438	523	758	1,251	747	519	518	267	9,211
Ore Production													
Development	t			13.6	22.6	6.0	6.0	24.2	3.4	-	2.1	-	77.9
Longhole	t			3.7	108.9	151.5	138.0	98.6	141.7	96.6	37.7	43.4	820.1
Cut & Fill	t			7.0	24.0	-	13.5	34.8	12.3	60.9	117.7	93.8	363.9
Total	t			22.4	157.5	157.5	157.5	157.5	157.5	157.5	157.5	137.1	1,261.9
Grade													
Gold	g/t			8.07	8.49	8.80	8.65	7.75	7.23	7.34	7.44	7.41	7.90
Silver	g/t			28.3	30.7	34.0	29.8	23.3	21.9	19.1	18.0	20.0	24.7

8. MINERAL PROCESSING

8.1 General Description

The mill will be designed to process 1000 tonnes per day (nominal capacity) of gold-silver feed in a normal two-stage SAG/Ball mill grinding circuit with high capacity thickening, cyanide leaching, carbon-in-pulp adsorption, carbon elution and regeneration, electro-winning and refining, and cyanide destruction with SO_2 and air. After processing, the tailings will be discarded to an earth and rock constructed tailings dam with water reclaim facilities, located near the mill.

The mill will operate 24 hours per day and 7 days per week for 5 ½ months each summer, or a total of 165 days per year on average. Availability while operating has been estimated to be 93.6% to allow for maintenance and other non-scheduled downtime due to power outages or other unforeseen events. The design throughput rate is 45 t/h for equipment and process piping design.

In keeping with the low capital cost design objective, the grinding, leaching, CIP and gold refining circuits will be controlled manually by the operator. Thickening, carbon stripping, electro-winning, carbon regeneration, and cyanide destruction, will be automated to protect the high value of the carbon and adsorbed precious metals that are handled in these sections.

Selected slurry streams including cyclone overflow, leach tailings and CIP tailings, will be sampled for monitoring of the operation and for metallurgical accounting purposes.

The process facilities will be in an enclosed weatherproof building located below the feed stockpile where the mine haulage trucks dump. Certain of the process equipment items will be located outdoors next to the mill building. These items include the feed thickener and the five leach tanks. Preliminary sketches of the mill arrangement are included in Appendix I.

8.2 Metallurgy

Extensive metallurgical test work was conducted at Brenda Process Technology's laboratories just prior to the Rescan Feasibility Study that was done in the mid 1990's.

This work indicated that the Red Mountain deposit is amenable to whole-ore direct cyanidation for the extraction and recovery of gold and silver. This process has been recommended for this present study.

The Brenda work indicated that the expected recovery of gold and silver using this process would average 89% and 81% respectively for the three samples in equal portions.

Three different zones (Marc, AV and JW) were tested and the gold recoveries varied from 84 to 92% while silver recoveries varied from 72 to 85%. Because of the proportions of feed planned from each zone, Rescan's estimate for plant recovery of gold and silver of 90% and 80% respectively, appears to be reasonable and will be used in the current study. Average head grades for the Brenda test work were 11g/t of gold and 31g/t of silver, slightly higher than the average grade for the present "mineable tonnes" which grade 7.9g/t Au and 24.7g/t Ag.

Losses in the leached product will be difficult to reduce because of the presence of gold and silver tellurides and 1 to 2 micron gold particles encapsulated in the pyrite. Since the deposit will assay about 13% sulphur, about 25% of the deposit occurs as pyrite. All zones were shown to be grind sensitive and Rescan recommended a very fine grind for good liberation. The present design has used a specification of 80% minus 37 microns for the two-stage grind.

The AV zone was found to contain lower precious metal grades than the other two zones and the highest levels of tellurides and is clearly the most difficult of the three zones to treat, requiring high lime above saturation for best recovery and the use of aeration with atmospheric air to enhance recovery. High lime can be justified because of reduced antimony in the leachate but the oxygen is not required for the other two zones.

Grinding test work was done by A.R. MacPherson Consultants Ltd. And Brenda. Both techniques showed a grinding work index of 17.4 kWh/t for the Marc zone from the Red Mountain deposit. Other tests indicated the work index would be just over 19 kWh/t. A value of 19 kWh/t was used for the present study. It must be noted however that for hard ores like this, a better criteria for determining the SAG energy requirements is to utilize a limited suite of carefully chosen samples for direct SAG testing using a lab test such as the SAG Power Index (SPI) available through

Minnovex. This program alerts the designer to the hardness variability and precisely defines the hardest feed and its location. This program needs a budget of about \$20,000 for the sampling, testwork and design of the grinding circuit if suitable drill core is available. If not, several carefully placed holes are required and the approach to grinding design will need to be carefully reviewed.

The reason for this recommendation is to provide enough grinding energy so that scheduled production can always be achieved. In the present environment of seasonal operation, the plant cannot afford to miss throughput objectives during the first season of operation due to unexpected harder than average feed being fed to the mill.

Carbon test work indicated that good loading characteristics can be expected using CIP technology. In addition, base metal loadings on the carbon were low, indicating good performance on recovering the precious metals from the carbon can be expected.

Detoxification test work was done by INCO Engineering and Technical Services using the patented SO_2/air cyanide destruction process and gave normal results.

Another test work program was completed by Beattie Consulting Ltd. in January 2001. The purpose of this study was to examine flotation as a process option because it could be done at a coarser size and produce a saleable concentrate.

The work was conclusive that rougher flotation recovery would be unlikely to exceed the recovery that could be obtained by direct cyanidation of a fine grind. In addition, the high sulphur content of the deposit made it impossible to achieve good recovery without recovering over 25% of the weight as rougher concentrate and attempts to clean the rougher concentrate showed large losses in the cleaner tailings.

The treatment of rougher concentrate could not be shown to operate economically, either by direct sale to a smelter or by fine grinding and cyanidation of the flotation concentrate. The economics of sale to a smelter would consume in the order of one third of the value contained in the concentrate, in shipping and smelting costs because of the high weight recovery. The alternative cyanidation of the float concentrate was economically unattractive because a second recovery loss in the leach tailings of 5% would occur, in addition to the flotation loss of about 10% of the gold.

From an operating perspective, flotation is less attractive as well. In the event that the operators were not attentive on a 24-hour basis, large increments of recovery could be lost until the circuit stability was restored. This will not happen in a leach circuit.

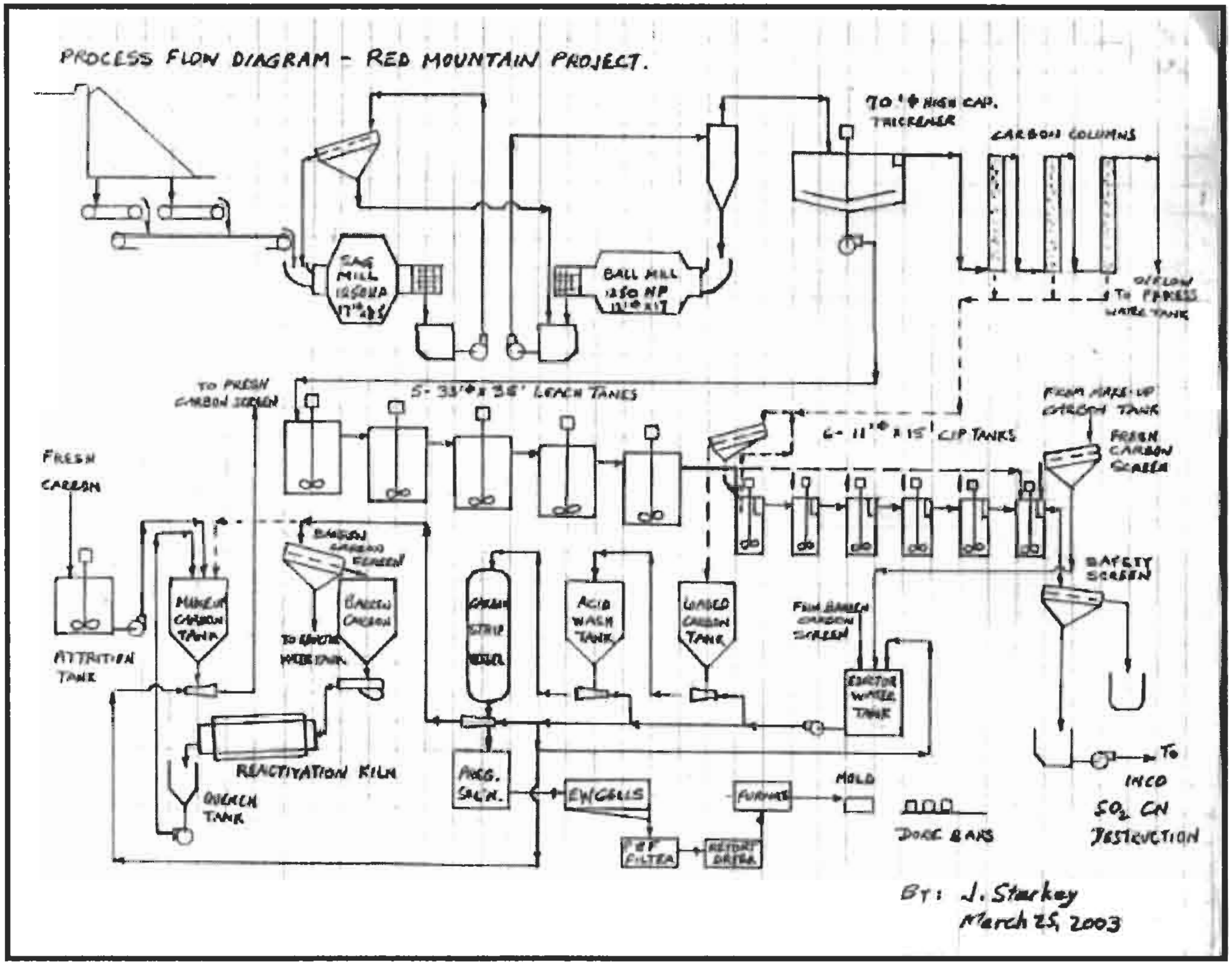
SRK is therefore confident that the recommended process is competitive with the best alternative methods for processing the Red Mountain deposit, from both an income and capital cost perspective. This is not to say that more test work, as suggested by Bateman engineers who market modular plants, could not develop another way to successfully treat this ore, but it is unlikely that such a process, if discovered, will produce better overall recovery, or can be built at a lower cost.

8.3 Process Description

In the underground mine, mill feed will be sized by a hydraulic rock breaker working on a 300 x 380mm grizzly. Refer to Figure 8.1. The target size for the mill feed is 80% passing 150mm. The grizzly undersize will then be hauled from the mine (1690m chute) and dumped on the mill stockpile. Two 1m by 3m apron feeders will withdraw the surface stockpile and will transfer it to the 900mm wide conveyor that feeds the SAG mill, inside the mill.

Grinding will be first done in a 5.18m diameter x 2.59m (17' x 8.5') EGL 932 kW SAG mill operating in closed circuit with a 1.22m x 2.44m (4' x 8') long vibrating screen with 750 micron slotted openings in a polyurethane deck. Screen feed will be pumped from the basement level in a 125mm x 100mm (5" x 4") dedicated all metal slurry pump to the screen. Oversize will be returned by gravity to the SAG mill, while the screen undersize will flow by gravity to the ball mill discharge pump box. The fine grinding will be done in a 3.66m diameter x 5.18m (12' x 17') long 932 kW ball mill operating in closed circuit with a cluster of 250mm diameter cyclones. The cyclones will be fed with a 250mm x 200mm (10" x 8") rubber lined slurry pump.

Figure 8.1: Simplified Process Flow Diagram



Cyclone overflow will flow by gravity to a 28 mesh trash screen for removal of tramp material then to an outdoor high capacity 21.3m diameter thickener to raise the density to 45% solids for leaching. Thickener overflow will pass through three carbon columns in series to recover any soluble values and the water will then be sent to the process water tank for reuse along with water reclaimed from the tailings dam.

The thickened slurry will be pumped to a series of 5 - 10m diameter x 10.67m high leach tanks in series that will provide 48 hours of leach time at high pH >11. Lime slurry and cyanide solution will be metered into the leach circuit to maintain proper leaching conditions. Air will be added through spargers as required to provide oxygen for the more difficult to treat ores which require elevated oxygen levels for best results.

After leaching the slurry will flow by gravity into 6 - 3.35 m x 4.57m high, CIP tanks. The last stage CIP tank, overflows to a 20 mesh safety screen to recover any values adsorbed on the carbon fragments that have broken down in the CIP tanks. Tailings from the safety screen will be pumped to the INCO SO_2 /Air cyanide destruction unit and then to the tailings dam for final disposal.

Fresh carbon will be prepared for addition to the circuit by adding it to an attrition tank for pre-conditioning and storage in a make-up holding tank. Carbon transfers to and from the CIP tanks and between vessels in the stripping circuit will be done using eductors operating in closed circuit with the eductor water tank. The fresh carbon will be added to the last tank of the CIP circuit by dewatering the educted carbon on a small 2.44m x 1.22m vibrating screen.

Carbon will be advanced in the CIP circuit using recessed impeller pumps to minimize the attrition wear on the partially loaded carbon. Carbon from the first tank will be dewatered on a loaded carbon screen and the carbon will be placed in a loaded carbon tank ready for acid washing, stripping and regeneration in a 2t per batch packaged plant supplied by Summit Valley. About one strip cycle per day will be required.

Carbon stripping will be done using a caustic cyanide solution at elevated temperature and pressure in a 2t per batch stripping vessel. The solution containing the gold and silver values will then be treated in electrowinning cells for recovery of the precious metals on stainless steel mesh cathodes. The values will be recovered as sludge that will be smelted to doré bullion in an electric induction furnace.

Regeneration of the carbon will be done in an oil heated, horizontal rotary kiln. After regeneration the carbon will be quenched and educted to the make-up carbon tank for reuse in the CIP circuit.

Three water systems will be provided to supply fresh, fire protection and process water to the plant. Process water will be supplied by reclaiming as described above, with a fresh water make-up allowance of up to 15% of total process water requirements to allow for losses and contingent items. Reagent systems will be provided to supply NaCN, Lime, Flocculant, Sulphur, $CuSO_4$, NaOH, and HCl for the process plant. Other reagents for gold smelting including borax, $NaNO_3$ and silica will also be provided.

9. WASTE STORAGE

9.1 Tailings Storage Facility

The tailings storage facility described below is based on the cirque tailings disposal option presented in the 1994 feasibility study. It is however, much smaller than the facility envisioned in the 1994 study, due to the smaller "mineable tonnage" in the current study. Assumptions regarding the tailings containment are:

- Ore delivery rate of 1000tpd
- Ore delivery for 5 ½ months of each year (mid May to October)
- Total mine life of 8 years or 1.26 million tonnes processed
- Assumed in place density of tailings, 1.36t/m^3.
- Total volume of tailings storage required, 1.9 million m^3.
- Use the Cirque tailings dam as proposed by Klohn-Crippen for tailings storage.
- Final dam elevation of 1467m.
- Final tailings deposition elevation of 1460 m (allows for 2 m permanent water cover and 5 m freeboard).

SRK notes the following concerns with respect to using the cirque tailings dam:

- The final dam will be 42 m high and which makes it a substantial structure which may result in permitting complications due to the potential risks associated with it.
- The dam is in a narrow valley that receives substantial snowfall and the spring-melt is significant – the potential for failure is thus real.
- The final closure of the tailings dam using a permanent water cover entails keeping a water retaining structure in perpetuity. This means that there will be a permanent requirement for annual dam safety inspections. The alternative would be to construct a dry cover, which has substantial implications with regard to closure costs (assuming a 1m compacted cover and 142,000m^2 surface area, would constitute a closure cost in the order of $2 million, over and above all the other costs).

Considering the volume of tailings to be disposed of, it is likely that a side valley impoundment may be both environmentally and economically more attractive. This alternative was not evaluated by SRK under the current scope of work.

9.2 Waste Rock Storage

There is an existing pile of mine development waste on the ridge near the exploration portal. This is shown in the site plan, Figure 7.2. Project data indicates that 90kt are currently stored there. There are 5kt adjacent to the portal and 85kt stored 250m south of the portal. SRK verified that the stated amount is in general agreement with the volume of the existing underground excavation.

The Red Mountain mine will both create waste rock from mine development, and consume it as mine backfill, being a net consumer of waste rock over its projected life. During pre-production seasons 2 and 3, an additional quantity of roughly 50kt will be added to the existing ridge waste stockpile. Also, during the same period, about 100kt of waste will be stored in a new pile in the general area of the mill as shown in Figure 7.2. Any drainage water escaping from these piles will be directed by gravity into the tailings pond area.

Both of the waste storage piles discussed above will be completely used up as mine backfill. The waste will be loaded into 20t trucks and hauled to empty underground stopes as needed. This activity is included in the mine operating cost estimate. Mine backfill requirements exceed the available mine development waste, therefore some waste rock will be excavated from a small surface borrow area.

10. OPERATING COSTS

The Red Mountain site operating cost is presented in three components; mine, mill, and general and administrative (G&A). The mine includes mine supervision, engineering, and geology functions. It includes the operating and maintenance costs for all equipment directly related to the underground mine. The mill includes all operating and maintenance costs directly related to milling. G&A costs include costs related to the surface site, surface equipment operating and maintenance costs, access road maintenance, and other normal G&A functions.

All operating cost estimates are based on an owner operated mine site, with a seasonal mining operation. Off-site owner's costs are not included.

Mine staff salaries are intended to reflect industry norms, while hourly wage rates have been set to be comparable to unionized mine sites in B.C. There may be difficulties in re-hiring a workforce each season to operate the mine. Some positions may have to be temporarily filled by personnel hired from mining contractors. To address this concern, the following provisions are made in the operating cost estimate:

- Operating costs include a labour allowance of an extra 14% (explained below) of the fully loaded normal operating labour cost. This amounts to $5.9million, or $4.69/t, over the life of the mine.
- This includes eight key positions that will be filled by full time employees. These are shown in Table 10.1
- Many of the operating crews are scheduled to work 46.7 hours per week, making the seasonal positions more attractive in terms of earning potential.

The 14% premium on the total labour cost is composed of the extra costs for paying 8 staff positions on a full year basis, plus a 15% premium added onto the fully loaded labour rates for development and production miners, and mechanics and electricians. For example, fully loaded hourly wage rates, including the 15% premium, include the following; shiftboss $51.26, development miner $53.39, and mechanic $43.42. These wage rates are intended to provide the flexibility of employing contractor supplied labour in certain instances, should it be necessary.

The provision of 15% on Red Mountain labour is applied in the economic model as a separate line item, and is excluded from the operating costs discussed in this section.

TABLE 10.1: Full Time Staff

Full Time Employees	
	LoM Labour Cost (Thousands)
General Manager	1,144
Personnel Manager	1,053
Mine superintendent	1,170
Maintenance G.F.	936
Mill G.F.	833
Chief Engineer	1,040
Senior Accountant	*761*
Senior Environmental Coordinator	845
Total (thousands)	$ 7,782

Operating cost estimates are based on electrical power supplied by B.C. hydro at an average cost of $0.036 per kWh. The project enjoys a low power cost by completely avoiding power use during the peak winter season.

Fuel for mine equipment is assumed to cost $0.60 per liter.

Mine, mill and G&A operating cost estimates are discussed below, including manpower estimates for each area.

10.1 Mine

The underground mine operating cost includes the manpower shown in Table 10.2

TABLE 10.2: Typical Mine Manpower

Typical Mine Manpower	
Mine Supervision	6
Mine Operations:	
Development	7
Stoping	16
Truck Haulage	6
Services	9
Maintenance Supervision	3
Maintenance	14
Engineering & Geology	9
Total Mine Department	**70**

Mine manpower requirements were detailed for each year through pre-production and production. Refer to Appendix J. Numbers of men were based on labour productivities and on previously calculated "number of operating units", such as for trucks, scoops, LH drills. Some positions, such as many of the mine services, were estimated based on SRK experience. In the case of productivity based estimates, it was assumed that men at work equaled 90% of men on the payroll, to account for non-productive absences from the workplace such as vacation, sickness, training, meetings, etc.

To cover equipment start up and shut down each season, an additional thirty 10hr man-shifts are provided, over and above the normal season, for mechanics and electricians to work on the equipment.

The average productivities shown in Table 10.3 were used to estimate mine labour requirements. They are all on a 10hr shift basis.

TABLE 10.3: Average Productivities Planned

Average Productivities		
Diamond drilling	18.0	m/ms
Lateral Development	1.3	m/ms
Raising	1.0	m/ms
Longhole Drilling	75.0	m/ms
Utility Drilling	50.0	m/ms
Longhole Blasting	400.0	t/ms
Cut & Fill Stoping	100.0	t/ms
Mucking - 6YD	508.0	t/ms
Mucking - 4YD	408.0	t/ms
Truck to Mill	440.0	t/ms

Notes:
m/ms = meters per 10hr man-shift
t/ms = tonnes per 10hr man-shift

The mine labour costs include provisions for an underground bonus system that will provide clear performance objectives and scales of monetary reward to those underground workers whose efforts and efficiency exceed the levels normally expected for the regular wages alone. It is envisioned that the system will include measures of quality as well as quantity, and will ensure that worker safety is not compromised in the interest of increased productivity.

Table 10.4 shows the equipment operating costs used in the mine operating cost estimate. These costs apply to operating hours and include maintenance parts, some off-site rebuild labour costs, fuel, lube, tires, wear parts, and for drills, feed and drifter maintenance. These costs exclude the maintenance labour costs that are represented by the mechanics labour in the manpower estimate.

TABLE 10.4: Equipment Operating Costs

Equipment Costs per Operating Hour	
2 Boom Jumbo	$ 37.53
4 Yard Scoop	$ 38.00
6 Yard Scoop	$ 44.31
20 Tonne Truck	$ 39.58
Scissor Lift	$ 13.98
Anfo Truck	$ 18.93
Personnel Carrier	$ 13.11
Hiab Truck	$ 15.71
D8 Dozer	$ 58.68
980 Loader	$ 42.74
16H Grader	$ 50.44

The mine operating cost estimate averages $35.91 per tonne during the production period, seasons 4 through 11, as shown in Table 10.5. Mine costs in pre-production seasons 2 and 3 are capitalized.

The costs shown in Table 10.5 are generally a build up of labour plus materials and supplies for each function shown. Materials costs are estimated based on experience, comparisons to other projects and mines, and by contacting suppliers.

Electrical power costs and equipment operating and maintenance costs are shown as separate lines, and have not been allocated to individual functions.

The average direct unit cost for lateral development of a 4.4 x 4.0m drift is $1140 per meter, broken down; $445/m labour, $460/m materials, and $235 equipment operating.

TABLE 10.5: Mine Operating Cost Estimate

U/G MINE OPERATING COST SUMMARY

Function	Pre-production			Operating									
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**Total**	**$/T**
Mine Supervision	Road	174	251	356	356	356	356	356	356	356	356	2,849	2.30
Electrical Power	Const.	-	80	116	116	116	116	116	116	116	101	912	0.74
Diamond Drilling	-	-	7	52	52	52	52	68	80	62	15	434	0.35
Development	-	1,152	1,992	1,265	464	760	1,090	646	444	456	228	5,354	4.32
Longhole Mining	-	-	26	450	598	567	430	595	437	274	267	3,619	2.92
Cut and Fill Mining	-	-	77	297	33	179	468	196	795	1,516	746	4,230	3.41
Truck Haulage	-	54	104	206	231	230	219	228	242	246	211	1,813	1.46
Backfilling	-	19	28	333	631	627	432	536	529	230	168	3,486	2.81
Construction	-	78	90	157	157	157	157	157	157	157	147	1,247	1.01
Mine Services	-	-	104	447	447	447	447	447	443	435	415	3,527	2.85
Equip. Operating	-	974	1,057	1,690	1,596	1,641	1,724	1,677	1,736	1,857	1,699	13,619	10.99
Technical Services	-	159	247	428	428	428	428	428	428	428	428	3,422	2.76
TOTAL	**-**	**2,610**	**4,062**	**5,798**	**5,108**	**5,560**	**5,920**	**5,449**	**5,762**	**6,133**	**4,781**	**44,512**	**35.91**

10.2 Process Plant

The mill operating cost estimate is shown in Table 10.6.

TABLE 10.6: Mill Operating Cost Estimate

Red Mountain Milling Cost			
Mill Capacity	1,000 tpd		
Milled per Season	157,500 t		
Item	**Description**	**Seasonal Cost ($thousands)**	**Cost per Tonne $/t**
Labour	32 Employees	1,055	6.70
Supplies	per Rescan '94	1,925	12.22
Power	$0.036/kWh	267	1.70
Fuel	$0.50/t	79	0.50
Total		**$ 3,326**	**$ 21.11**

Labour requirements, presented in Table 10.7, are based on the process flowsheet. The cost of supplies includes reagents, and this unit cost has been taken from the 1994 feasibility study by Rescan, as their estimate was reviewed and it appears reasonable. The power demand was estimated from the list of mill equipment developed for the capital cost estimate. Average power draw was taken as 75% of installed power. Power is supplied at a low unit cost by BC Hydro.

TABLE 10.7: Mill Labour Requirements

Mill Personnel	
Staff Positions	
Mill Supervision	5
Maint. Supervision	1
Assay/Metallurgy	3
	9
Hourly Positions	
Operations	11
Maintenance	6
Sampler/Lab Tech.	6
	23
Total Mill	**32**

10.3 General and Administrative

The G&A manpower estimate for a typical season is shown in Table 10.8

TABLE 10.8: G&A Manpower

General and Administrative Manpower	
Administration	
General Manager	1
Personnel Manager	1
Clerical	1
Safety	
First Aiders	2
Environmental	
Senior Environmental Co-coordinator	1
Environmental Technician	1
Accounting	
Senior Accountant	1
Accounts Payable/Clerical	1
Payroll Administrator	1
Materials Management	
Purchasing Agent	1
Buyer	1
Warehouseman	3
Surface Crew	
Site Maintenance	2
Road Maintenance	3
Transportation	
Bus drivers	2
Total G&A	**22**

Table 10.9 shows the general and administrative (G&A) cost estimate. The average operating cost for seasons 4 through 11 is $10.05 per tonne. The costs shown in pre-production seasons 1, 2 and 3 are capitalized.

In addition to the personnel shown, Administration includes office supplies, insurance, communications, property tax, and fees/permits.

TABLE 10.9: G&A Operating Cost Estimate

G&A OPERATING COST SUMMARY

Function	Pre-production			Operating								Total	$/T
	1	2	3	4	5	6	7	8	9	10	11		
Administration	-	4	153	300	300	300	300	300	300	300	300	2,400	1.94
Safety	-	-	35	70	70	70	70	70	70	70	70	563	0.45
Environmental	-	85	90	142	142	142	142	142	142	142	142	1,133	0.91
Accounting	-	-	119	142	142	142	142	142	142	142	142	1,133	0.91
Materials Mgmt	10	20	106	212	212	212	212	212	212	212	212	1,692	1.37
Surface Crew	-	134	217	276	276	276	276	276	276	276	276	2,204	1.78
Surf Equip Operating	-	265	345	366	366	366	366	366	366	366	317	2,879	2.32
Transportation	-	29	57	57	57	57	57	57	57	57	57	458	0.37
Temporary Services	-	38	-	-	-	-	-	-	-	-	-	-	-
TOTAL	**10**	**575**	**1,121**	**1,564**	**1,564**	**1,564**	**1,564**	**1,564**	**1,564**	**1,564**	**1,515**	**12,463**	**10.05**

Safety includes salaries plus safety supplies. Environmental includes salaries and provisions for consulting services and analytical services. Accounting includes some supplies in addition to the salaries for the personnel shown. Materials management includes salaries, freight, and operating costs for the warehouse/office in Stewart.

The surface crew is responsible for site maintenance and road maintenance. Cost provisions in addition to wages included with this function are:

- Materials for road/bridge maintenance, and/or building maintenance.
- Equipment rentals including a crane for bridge work and a 980 loader for road work, (only when the 980 loader owned by the mine is unavailable).
- Outside services including avalanche technicians and BC Hydro crews for line repairs.
- A small amount for mine site building heat.

Surface Equipment Operating includes the equipment shown in Table 10.10.

TABLE 10.10: Surface Equipment

Surface Equipment		
Administration		
	Pick up trucks	4
Safety		
	Suburban	1
Materials Management		
	Supply Truck	1
Surface Crew		
	D8 Dozer	1
	D7 Dozer	1
	16G Grader	1
	980 Loader	1
	Small backhoe	1
	Helicopter	rental
Crew Transportation		
	Crew bus	2

All of the equipment shown in Table 10.10 is owned by the mine except the helicopter, which is rented for seasonal road opening and avalanche control. The D7 dozer is the existing unit at the site.

Transportation in Table 10.9 covers the salaries of two bus drivers. Their work schedules will be adjusted to suit the crew change requirements.

11. CAPITAL COSTS

Table 11.1 is the Red Mountain capital cost estimate. Pre-production capital is estimated at $60.3million and sustaining capital $4.2million. Working capital is not included. Capital items are discussed below.

11.1 Feasibility Study

A feasibility study would be required to support a production decision. The cost provision covers a small amount of fieldwork.

11.2 Permitting

This cost estimate is based on SRK's previous involvement with the Red Mountain Project. In October 2000, SRK issued a report for NAMC titled, "Review of Baseline Studies at Red Mountain". The report presented the results of a review of completed baseline studies, identified gaps, and determined what further data and analysis would be required to support a project application report, which is the first step in the formal permitting process. SRK's permitting cost estimate is based on our understanding of the Red Mountain Project as compared to our experience at other sites.

11.3 Access Road

In 2000, NAMC prepared a road design to access the existing Red Mountain portal. It is comprised of 13.5km of existing logging road to be upgraded, and 18.5km of new road to the upper mountain. Refer to report section 2.2.1 and to Figure 2.1. The full specifications for the road are included in Appendix A.

In early 2001 several road construction companies submitted cost estimates for the complete road construction in response to a bid package prepared by NAMC. Five quotations were received, ranging in cost from $4.4million to $9.3million. For the purposes of this report SRK has selected the quote from Don Hull & Sons of Terrace, B.C. of $6.8 million as representing the road cost in 2001. This figure has been rounded up to $7million for input to the economic analysis in 2003 dollars.

TABLE 11.1: Capital Cost Estimate

Red Mountain Capital Cost Estimate

Thousands of Dollars

	Pre-production			Production								Reclam	
Season	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**Total**
Feasibility Study	400												400
Permitting	1,000												1,000
Access Road	5,600	1,400											7,000
Power Line	480	1,120											1,600
Mine													
Preproduction Development	-	2,610	4,062										6,673
Mobile Equipment	-	4,541	3,008	217									7,766
Other Mine Equipment	-	1,238	2,022	-									3,259
Sub-total Mine	-	8,389	9,092	217									17,698
Mill	-	13,440	8,471	-									21,911
General and Administrative													
Preproduction Costs	10	575	1,121										1,706
Surface Equipment	-	873	103	-									976
Sub-total G&A	10	1,448	1,223										2,681
Labour Premium	-	120	192										312
Tailings Dam	-	3,900	3,300	2,000	-								9,200
Camp, Office, Dry Buildings	-	423	180	-									603
Sustaining Capital			100	250	250	250	250	250	250	250	250	-	2,100
Reclamation												260	260
Salvage												(3,000)	(3,000)
													-
TOTAL CAPITAL	**7,490**	**30,240**	**22,558**	**2,467**	**250**	**250**	**250**	**250**	**250**	**250**	**250**	**(2,740)**	**61,765**
	Pre-production =		**60,288**		Sustaining =		**4,217**						

11.4 Power Line

The capital cost for the power line is based on studies completed in 1994 and on recent discussions with B.C. Hydro. Previous studies incorporated costs for a transmission line system operating at 34.5kV, able to supply 10MW. It was assumed in the 1994 feasibility study that B.C. Hydro would pay for a sub-station required at the start of the new transmission line.

The current study requires only a 25kV three-phase distribution line system capable of 4MW. The capital cost estimate of $1.6million is discussed in section 5.3.2. SRK makes the assumption that B.C. Hydro will pay for the substation (valued at approximately $200thousand) at the start of the new line.

11.5 Mine

The pre-production development costs for seasons 2 and 3 are copied from Table 10.5. The main activity during this period is lateral development, with 2,923m being driven. These costs include some of the underground construction activity, and in season 3 include stope preparation and the start of production stoping.

The underground equipment capital estimate is shown in Table 11.2. Note that development and production equipment will move back and forth between these functions during the mine life. Remanufactured units have been selected for most of the fleet. These are priced at 65% of new. The jumbos and the longhole drill are electric/hydraulic units.

Other equipment for the mine includes all other items required to operate the mine including tools, equipment, surface buildings and infrastructure. Individual items costing more than $200,000 are; air compressors, underground rock breaker, backfill plant, main ventilation fans, surface maintenance shop, mine electrical distribution, and surface warehouse (shared with the mill). A complete list is provided in Appendix K.

TABLE 11.2: Underground Equipment Capital Cost Estimate

Underground Equipment Capital				
Function / Unit	Number	Unit Cost Thousands	Status	Total Cost Thousands
Supervision				
Kubota Tractor	1	55	new	55
Personnel Transport				
Getman carrier	1	175	rebuilt	175
Nipping				
Getman hiab	1	175	rebuilt	175
Forklift (surface)	1	90	new	90
Definition drilling				
Gopher Drill	1	150	new	150
Development				
2-Boom Jumbo	1	399	rebuilt	399
6 YD Scoop	1	399	rebuilt	399
Scissor lift	1	175	rebuilt	175
Truck 20T	1	404	rebuilt	404
Production & Backfill				
2-Boom Jumbo	1	399	rebuilt	399
6 YD Scoop	2	614	new	1,228
4 YD Scoop	1	478	new	478
4 YD Scoop	1	311	rebuilt	311
Truck 20T	2	622	new	1,244
LH Drill	1	318	rebuilt	318
Utility Drill	1	72	rebuilt	72
Anfo Truck	1	175	rebuilt	175
Scissor lift	2	175	rebuilt	350
Road Maintenance				
Grader	1	247	rebuilt	247
Construction				
Getman hiab	1	175	rebuilt	175
Maintenance				
Kubota Tractor	1	55	new	55
Mechanics hiab	1	175	rebuilt	175
Scissor lift	1	175	rebuilt	175
Backfill Supply (surface)				
980 Loader	1	344	rebuilt	344
D7 Dozer	1		existing	
Total Underground Equipment				**$ 7,766**

11.6 Mill

The following method was used to develop the mill capital cost estimate.

A process flowsheet was prepared to describe the selected process in enough detail to enable sizing and costing of all of the major equipment items, and to list the installed power that would be needed on each piece of process equipment. In the case of the gold recovery system, a total bulk price for the CIP screens, carbon handing, stripping and regeneration, was obtained from Summit-Valley who are specialists in this area.

Quotations were then solicited from major vendors to get budget pricing for these items. Contractors were contacted in order to get bids for fabricating thickener, leach and CIP tanks and pump boxes of various sizes. These informal quotations, by phone, E-mail and fax, were used to estimate the total equipment cost (and the total connected power used in the operating cost estimate).

Industry standard factors were then selected for providing the building and civil construction, installation, piping, instrumentation and electrical facilities needed to support the equipment. This procedure gave the total direct cost for the process facilities. To this, estimates were added for construction indirects and EPCM, based on the project complexity and industry standards.

Investigations were held concerning the use of a modular design and/or used equipment as a way to reduce the capital cost. The modular design was not accepted due to the size of the grinding mills that are required. Concerning used equipment, the maximum equipment savings could not exceed $2million overall. This is because the installation costs would be about the same using either new or used equipment.

The capital cost shown in Table 11.1 reflects a $2million reduction in cost by maximizing the use of used mill components.

11.7 General and Administrative

The pre-production support costs for seasons 1 through 3 are copied from Table 10.9. The support functions included in G&A are; administration, environmental monitoring, materials management, surface crew (mainly access road maintenance), employee transportation, and temporary services (electricity and compressed air).

The surface equipment capital estimate is shown in Table 11.3. Remanufactured units have been selected for many units. These are priced at 65% of new.

TABLE 11.3: Surface Equipment Capital Cost Estimate

Surface Equipment Capital				
Unit	Number	Unit Cost Thousands	Status	Total Cost Thousands
Pick up trucks	4	30	new	120
Suburban	1	40	new	40
Supply Truck	1	100	new	100
D8 Dozer	1	466	rebuilt	466
16G Grader	1		use mine unit	-
980 Loader	1		use mine unit	-
Small backhoe	1	100	rebuilt	100
Helicopter	1		rental	-
Crew bus	2	75	new	150
Total Surface Equipment				**976**

11.8 Labour Premium

The concept of a 15% premium on certain labour costs is discussed in section 10. The premium is also applied during pre-production, in which case the cost is capitalized as shown in Table 11.1.

11.9 Tailings Facility

The capital cost estimate for the tailings impoundment of $9.2million is based on:

- Construction quantities used as supplied by Klohn-Crippen in the 1994 feasibility study.
- Staged construction considered: Season 2 (pre-production) build dam to 1452m, Season 3 (pre-production) raise dam to 1464m, Season 4 (1st year of production) raise dam to 1467m elevation.
- Tailings impoundment costs are all inclusive assuming an outside contractor supplying his own equipment and consumables.
- Tailings impoundment costs include provisions for a 2m permanent water cover needed for closure.
- Tailings impoundment costs include engineering, construction supervision and permanent instrumentation costs.
- Costs include local quarrying of all materials and crushing and screening for finer materials and riprap.
- Costs exclude closure and rehabilitation of quarry areas.
- Mobilization and demobilization will be done by road – no helicopter support.

SRK compared the current cost estimate of CDN$9.2million to the estimate for the starter dam included in the 1994 feasibility study of US$11.4million (based on a contractors cost estimate). The starter dam in the 1994 study was built to 1464m while the current study requires a final elevation of 1467m. The 1994 study assumed heavy lift helicopter support during construction of the starter dam.

11.10 Camp, Offices and Dry Buildings

The surface maintenance shop, warehouse, office complex, first aid office, dry, small camp and kitchen will all be supplied by ATCO. Most will be trailers that can be connected to one another, with the exception of the shop, which will be a Fold-A-Way® structure and will require a concrete slab. The trailers will be skid-mounted and will only require level ground. All of the ATCO units can be transported by truck to/from site.

SRK has based the cost of the units on new prices quoted by ATCO, including taxes and transportation to site.

11.11 Sustaining Capital

This provision is included to cover unspecified equipment replacements and small items not listed in the capital cost estimate. There is no other contingency applied to the capital estimate.

11.12 Reclamation

Reclamation activities are described in section 12. The closure cost estimate of $1.26million is based on a best case scenario, and it includes post-closure monitoring of the site. This cost will be largely offset by the existing closure bond that has been posted, valued at $1.0million.

11.13 Salvage Value

Salvage value has been estimated by comparison to other projects. It is assumed that most of the salvage value of $3million would come from mine mobile equipment, mill components, and maintenance shop equipment.

12. MINE CLOSURE

12.1 Introduction

The Red Mountain closure cost estimate is based on SRK's judgement and experience with the closure of similar projects in B.C and the Yukon.

The closure plan has been developed on the basis of the following general assumptions:

- Site conditions that will influence closure costs are assumed to be relatively positive, i.e. favourable conditions within the range of realistic possibilities have been assumed.
- The existing infrastructure (camp and maintenance shop) will be removed prior to or during mine operation.
- The impacts of oil spills and other similar events will be rectified during mine operation.
- Based on available data, the waste rock and tailings are assumed to be potentially acid generating.
- All waste rock will be placed underground during mine operation.
- The spillway required at the tailings impoundment for closure will be designed according to the closure-based design flood.
- The closure measures implemented to prevent and/or mitigate the consequences of acid generation will be successful.
- No significant environmental issues will exist beyond closure, and the site will be abandoned based on 15 years of monitoring results.

Further comments related to the closure plan and the estimated costs are provided below.

12.2 Disturbed Areas

The disturbed areas at the mine site are grouped as follows:

- Infrastructure, including the plant facilities, maintenance shops, camp, mining equipment, fuel storage, explosives magazine and the roads
- Underground workings and portal areas
- Waste rock areas
- Tailings facilities

- Quarries and/or borrow areas associated with dam construction and/or the mine backfill supply

12.3 Restoration Activities

The following general restoration activities are planned:

- The infrastructure elements will, where possible, be removed from the site and buried. It may be possible to salvage some of the components of the plant and camp, but this option has been ignored for costing purposes. The gravel access road, about 32 km long, will be deconstructed in accordance with the Forest Practices Code.
- The rock exposed in the underground workings is potentially acid generating. Closure activities will focus on limiting the oxygen supply to the underground mine, and to the extent possible, flooding the underground workings. This will involve constructing concrete bulkheads in both the upper and lower mine access ramps.
- Sulphate reducing bacteria will be utilized to provide partial treatment of water which seeps through the bulkhead at the lower portal.
- As noted previously, all waste rock will be placed underground prior to closure. Surface areas disturbed by the placement and removal of waste rock will be revegetated.
- The tailings will be flooded with 2m (minimum) of water to prevent acid generation.
- Areas used to provide material for underground backfill will be revegetated.

12.4 Post-closure Monitoring

The post-closure monitoring program will include the execution and reporting of annual safety inspections and annual water quality monitoring. These costs are included in the closure estimate as net present values, assuming the post-closure monitoring program is required for 15 years. As with the closure costs, it is assumed that there are no significant water quality issues that would necessitate monitoring beyond 15 years. Cost provisions also include an allowance for management and administration duties by the owner in relation to these programs and liaison with regulatory authorities.

12.5 Risks

There are risks associated with the concept of a permanent water cover on the tailings impoundment. A water cover implies, by necessity, a water retaining dam. There are significant issues associated with the permitting and maintenance of a water dam and these issues have cost implications. Further studies could lead to an alternative, low permeability soil cover as the preferred cover option. For the current study, SRK has followed the design approach used in the 1994 Rescan feasibility study.

13. PROJECT ECONOMICS

13.1 Base Case Model

Table 13.1 shows the base case economic model for the Red Mountain Project in 2003 dollars. The base case represents SRK's best estimate of gold production and capital and operating costs. The gold price assumption was varied to determine the "break even" gold price for the project. At a zero dicount rate, the base case indicates a "break even" project with a gold price of US$359/oz. At a 5% discount rate, the base case indicates a "break even" project with a gold price of US$399/oz.

The details of the base case model are as follows:

- Tonnage mined and milled is 1,261.9kt at grades of 7.90g/t Au and 24.7g/t Ag.
- Mining rate of 1000tpd, or 157,500t per season.
- Mill recoveries of 90% for gold and 80% for silver.
- US dollar exchange rate of 1.43 (0.70)
- Silver price set at US$5.00/oz.
- Site operating cost of $68.46/t milled. Plus a 3.5% NSR royalty.
- Refining cost of $5.70/oz.
- Total capital cost of $61.8 million as shown in Table 11.1
- Cash flow is pre-tax, and does not include financing or corporate costs.

At the time of writing (late July 2003) the current metal prices are approximately US$360/oz gold and US$5.00/oz silver, with an exchange rate of 1.40. Under these conditions the project is presently not economically viable, based on the development approach and cost structure incorporated in the base case model.

TABLE 13.1: Red Mountain Economic Model

RED MOUNTAIN ENGINEERING STUDY		Pre-production			Production								Reclamation	
		1	2	3	4	5	6	7	8	9	10	11	12	Total
Production	**Unit**													
Ore Milled	tonnes	0	0	22,290	157,500	157,500	157,500	157,500	157,500	157,500	157,500	137,100	0	1,261,890
Gold Head Grade	g/t	0	0	8.07	8.49	8.80	8.65	7.75	7.23	7.34	7.44	7.41	0	7.90
Mill Gold Recovery	%	0%	0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0%	90.0%
Recovered Gold	oz	0	0	5,202	38,697	40,087	39,418	35,341	32,937	33,443	33,916	29,403	0	288,446
Silver Head Grade	g/t	0	0	28.32	30.67	33.96	29.83	23.34	21.86	19.07	18.00	20.02	0	24.73
Mill Silver Recovery	%	0%	0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	0%	80.0%
Recovered Silver	oz	0	0	16,235	124,240	137,586	120,839	94,551	88,540	77,265	72,900	70,604	0	802,761
Recovered Gold Equivalent	**oz**	**0**	**0**	**5,406**	**40,255**	**41,812**	**40,933**	**36,527**	**34,047**	**34,411**	**34,830**	**30,288**	**0**	**298,510**
Revenue														
Gold Price	US$/oz	0	0	399	399	399	399	399	399	399	399	399	0	399
Silver Price	US$/oz	0	0	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	0	5.00
Exchange Rate	US$/C$	0	0	1.43	1.43	1.43	1.43	1.43	1.43	1.43	1.43	1.43	0	1.43
Gross Revenue - Gold	C$000	0	0	2,964	22,048	22,840	22,459	20,136	18,767	19,055	19,324	16,753	0	164,347
- Silver	C$000	0	0	116	887	983	863	675	632	552	521	504	0	5,734
- Total	C$000	0	0	3,080	22,936	23,823	23,322	20,812	19,399	19,607	19,845	17,257	0	170,081
Refining Cost	C$000	0	0	31	230	239	234	209	195	197	199	173	0	1,706
Net Smelter Return	**C$000**	**0**	**0**	**3,049**	**22,706**	**23,584**	**23,088**	**20,603**	**19,204**	**19,410**	**19,646**	**17,084**	**0**	**168,375**
Operating Cost														
Mining	C$/tonne	0	0	0	36.81	32.43	35.30	37.59	34.60	36.58	38.94	34.87	0	35.91
Milling	C$/tonne	0	0	0	21.11	21.11	21.11	21.11	21.11	21.11	21.11	21.11	0	21.11
General & Administration	C$/tonne	0	0	0	9.93	9.93	9.93	9.93	9.93	9.93	9.93	11.05	0	10.05
Labour Premium	C$/tonne	0	0	0	1.51	1.11	1.31	1.53	1.30	1.43	1.73	1.14	0	1.39
Total	**C$/tonne**	**0**	**0**	**0**	**69.36**	**64.59**	**67.66**	**70.16**	**66.94**	**69.06**	**71.72**	**68.17**	**0**	**68.46**
NSR Royalty Payable 3.5%	C$000	0	0	107	795	825	808	721	672	679	688	598	0	5,893
Annual Operating Cost	**C$000**	**0**	**0**	**107**	**11,719**	**10,998**	**11,464**	**11,771**	**11,215**	**11,556**	**11,983**	**9,944**	**0**	**90,757**
Cash Cost/Ounce Gold Equivalent	US$/oz	0	0	0	204	184	196	226	231	235	241	230	0	213
Total Capital Cost	**C$000**	**7,490**	**30,240**	**22,558**	**2,467**	**250**	**250**	**250**	**250**	**250**	**250**	**250**	**-2,740**	**61,765**
Total Cost/Ounce Gold Equivalent	US$/oz				247	188	200	230	236	240	246	236	0	358
Cash Flow	**C$000**	**-7,490**	**-30,240**	**-19,615**	**8,519**	**12,337**	**11,374**	**8,582**	**7,739**	**7,604**	**7,413**	**6,890**	**2,740**	**15,853**
Cumulative Cash Flow	C$000	-7,490	-37,730	-57,345	-48,826	-36,489	-25,115	-16,533	-8,793	-1,190	6,223	13,113	15,853	
NPV @ 5%	**C$000**	**$0**												
NPV @ 10%	**C$000**	**-$9,253**												
IRR	**%**	**5%**												

13.2 Sensitivity Analyses

The base case economic model was used to perform sensitivity analyses on the following variations to the base case:

- Capital savings through tailings facility redesign
- Variations in the price of gold
- "Mineable tonnage" increase of 50%
- Operating and capital costs both reduced by 15%

These are discussed below.

13.2.1 Tailings Facility Redesign

This study incorporates a scaled down version of the tailings facility design that was proposed in the 1994 Red Mountain feasibility study. The design suffers from a high capital cost caused by the requirements for a permanent 2m water cover, and 5m of freeboard.

It is SRK's opinion that a more economical design may be possible, given the greater flexibility afforded by the current reduced tailings storage volume requirements, compared to the 1994 feasibility study.

For this sensitivity analysis, it is assumed that the tailings facility capital cost can be reduced from the current estimate of $9.2million, to $6.0million.

The impact on project economics includes:

- The break even gold price is reduced from US$399/oz to US$389/oz. (At 5% discounting)
- Pre-production capital is reduced from $60.9 to $58.1million.
- Sustaining capital is reduced from $4.2 to $3.2million.
- The project IRR is increased by approximately 1% compared to the values of Table 13.2

13.2.2 Gold Price Sensitivity

Table 13.2 shows the project sensitivity to the price of gold. Only the gold price has been varied, while other economic parameters reflect the base case as described in preceeding section 13.1.

TABLE 13.2: Gold Price Sensitivity

Red Mountain Economic Model Results								
Gold Price	US$/oz	**350**	**375**	**400**	**425**	**450**	**475**	**500**
NPV (0%)	$M	(3.6)	6.4	16.3	26.3	36.2	46.1	56.1
NPV (5%)	$M	(13.7)	(6.7)	0.3	7.4	14.4	21.4	28.5
NPV (10%)	$M	(19.3)	(14.1)	(9.0)	(3.9)	1.2	6.3	11.5
IRR	%	(1.0)	2.0	5	8	11	13	16

13.2.3 "Mineable Tonnage" Increase

For this analysis, the "mineable tonnage" was increased by 50% to a total of 1,892.8kt at the same metal grades as the base case. The production rate was increased to 1500tpd (236,250t per season) to retain the same mine life.

Revisions were made to operating costs by considering the portion of each that was either fixed, or variable with the mining rate. The revised operating costs are shown in Table 13.3

TABLE 13.3: Operating Cost Comparison for Increased Tonnage

Operating Cost Comparison		
	Base Case	**Increased Tonnage**
Mining	35.91	34.10
Milling	21.11	20.34
General & Admin	10.05	7.84
Labour Premium	1.39	1.39
Total Cost per Tonne	$ 68.46	$ 63.66

Each component of the base case capital cost estimate was reviewed to determine how much it would have to be increased to support a 50% higher production rate. The increases that were applied varied from zero to 50%, and averaged 25%, bringing the capital cost estimate up to $76.9million (from $61.8million).

These changes result in:

- A break even gold price of US$350/oz at a discount rate of 5%. (The cumulative discounted cash flow equals zero.)

- The project IRR is increased by 7% compared to the values of Table 13.2
- At a US$400/oz gold price, the NPV(10%) increases from ($9.0) to $3.9million.

13.2.4 Reduction in Operating and Capital Costs

Both operating and capital costs were reduced by 15%. The revised operating cost (before royalty) was $58.19/t and the total capital was reduced to $52.5million.

These changes result in:

- A break even gold price of US$338/oz at a discount rate of 5%. (The cumulative discounted cash flow equals zero.)
- The project IRR is increased by approximately 8% compared to the values of Table 13.2
- At a US$400/oz gold price, the NPV(10%) increases from ($9.0) to $4.8million.

14. PRELIMINARY ASSESSMENT OF EXPLORATION POTENTIAL

As part of the engineering study commissioned by Seabridge, SRK examined the available exploration data for the project in order to comment on any opportunity to increase the project resource base through exploration. A complete description of the assessment of exploration potential is presented in the SRK memorandum, "Preliminary Assessment of Exploration Potential of the Red Mountain Project", April 11, 2003, which is included in Appendix L. Below, a summary of the contents of the assessment is provided.

14.1 Introduction

This preliminary assessment is based on a cursory examination of sparse archived geological reports made available by Seabridge, discussions with previous project personnel, and on a review of public information available on the BG geological survey website.

The following aspects of the project were investigated:

- Review of available documentation on the project;
- Gap analysis on available geological information;
- Historical exploration work conducted over the project area and immediate surroundings;
- Nature, scale and extend of past exploration work performed over the project area, including: geophysical, geological and geochemical surveys and diamond drilling;
- Review of geological/structural interpretations with emphasis on the nature, and characteristics of the gold mineralization, the geological/structural controls on its distribution and potential targeting tools for exploration;
- Local deposit-scale and regional exploration potential.

14.2 Assessment Summary

Exploration efforts have focussed primarily on the immediate area surrounding of the known resource but no current compilation of borehole data exists. It is therefore not possible to provide detailed comments on the potential of adding to the current resource base through additional drilling. However, the high-grade precious metals mineralization at Red Mountain is enclosed within broader lower-grade alteration envelopes that are quite extensive. Several isolated high-grade intercepts exist outside the resource volumes and may have been overlooked. There is an opportunity to re-evaluate such isolated drill intercepts with the objective to delineate small high-grade volumes within the reach of proposed underground development. Also, several previous authors have pointed out that a re-evaluation of the down-dip extension of the AV and JW zones (AV/JW Tails zone) and the 141 zone is warranted. In order to properly evaluate these opportunity a thorough re-evaluation of all drilling results should be completed.

On the property scale, reconnaissance exploration appears to have covered accessible areas. No obvious exploration targets have the capability to increase the resource base in the short term. The exploration strategy included airborne magnetometer and electromagnetic surveys, follow-up ground geophysics surveys, prospecting and sampling, geological mapping and geochemical sampling. It appears that only limited diamond drilling was conducted, primarily on geophysical targets. Results indicate that sulphide mineralization was generally encountered with barren to low gold grades. A third of the project area is covered by ice and thus has not been explored. Attempts to image the geology underneath ice fields with remote sensed data and determine locally the thickness of the ice cover have produced mixed results.

The exploration potential of the property remains excellent, but will require a fair commitment to test adequately. Exploration in the Red Mountain area will face serious logistical challenges owing to steep mountainous topography and widespread ice fields and glaciers. A thorough review of exploration work in the area is warranted to produce a current compilation of all exploration data and revise the regional structural interpretation. New exploration targets could be identified by integrating the current knowledge about the geological setting of the Red Mountain deposit.

14.3 Recommendations

Recommendations include the following:

- At the scale of the deposit, several recommendations were expressed in recent reports prepared after the acquisition of the property by NAMC. In particular, Craig (2001) pointed out that the AV Tails and 141 zones have been tested by scarce drilling and therefore deserve re-evaluation.
- Additional studies were also recommended to continue to address the structural controls on distribution of the stockwork mineralization and derive predictive targeting tools.

- It is recommended that surface and underground drilling information along the periphery of all known gold-bearing zones be carefully reviewed to ensure that all gold-bearing structures have been properly closed out by drilling.
- In addition, within the area covered by surface and underground drilling, isolated high-grade drill intercepts should be also carefully evaluated for the potential to delineate smaller mineralized zones. It is considered very likely that additional follow-up exploration drilling around isolated intercepts will result in the delineation of small high-grade mineralized shoots.
- An independent review of down-hole geophysical surveying is also warranted to examine if any geophysical targets have been missed and whether the surveying has adequately sterilized the targeted drill area. This review should also consider alternative down-hole surveying techniques (EM, IP) that could offer a wider detection radius around boreholes or a better response to Red Mountain mineralization style.
- At the scale of the property, it is strongly recommended that all exploration work and other regional geological and geochemical data be compiled into a GIS environment. This compilation should also list all sulphide-gold occurrences encountered within the project area and provide an assessment of follow-up work, if warranted. Given the good understanding of the setting of the Red Mountain deposit, it is very likely that additional exploration targets can be identified.
- Subsequently, available data should be integrated with remote sensed data (airborne geophysics) to derive a new regional structural interpretation for the area.

15. RISKS AND OPPORTUNITIES

This section discusses the inherent risks and opportunities of the project development approach described in this Engineering Study.

15.1 Risks

15.1.1 Access Road

The access road presents risks during its construction and operation. During construction the risks are delays to the expected construction schedule and cost overruns. During operation the risks are loss of availability (temporary closure affecting production), safety, and operating costs.

Potential problems during road construction would most likely be related to road segments that are planned to traverse the very steep and rugged mountainous terrain at the upper end of the road. Road construction will be very challenging in this terrain.

During its operating life the road will be exposed to a number of natural hazards, and these are described by Golder in their year 2000 study of the access road,

"In general, the access road will be exposed to existing natural landslides or unstable terrain above and along the access road; debris flow and debris flood hazards on tributary streams that cross the road alignment; snow avalanches; erosion of the toe of fills by Bitter Creek or tributary streams; localized raveling from cut slopes; and localized rockfall from high rock cuts and natural rock slopes above the road."

SRK has addressed these concerns in the current study. There are road construction companies that are confident the road can be built as designed and scheduled, and they have supplied budget pricing. SRK did not rely on the low bid. The impact of the natural hazards has been reduced through the planned seasonal use of the road. Operating costs include substantial provisions for road maintenance.

15.1.2 Workforce

The current study is based on an owner operated mine, with the owner's full time employees (8 in total) responsible for hiring a seasonal workforce each spring. The risks that SRK perceives relative to this operating mode are workforce availability, training, and labour cost overruns. It is possible that the planned production rate could be affected by these risks. This would most likely be caused by manning shortfalls and/or equipment problems related to the lack of continuity and consistency in the training of the maintenance and operating employees. Cost overruns could be caused by the necessity to rely heavily on mining contractors to provide manpower, or to take on certain functions on a unit cost basis.

SRK has addressed these concerns in the current study by including a 14% premium on labour unit costs compared to the unit costs that would be applicable to a year-round operation. This means that the labour cost provisions in the current study allow a portion of the planned workforce to be paid at contractor equivalent wage rates, if necessary. Refer to section 10.

15.1.3 Project Economics

The Red Mountain Project, as it now stands, and with the development approach described herein, would require a relatively high price of gold to make it economically attractive. The risk is that the gold price would have to be maintained for at least an eight year duration. Forward sales could reduce this risk.

15.1.4 Tailings Storage Facility

There is a long-term cost risk associated with the proposed tailings facility (same design as the 1994 feasibility study). To prevent acid generation in the pond, a water cover is required in perpetuity. The tailings dam must be viewed as a water retention dam, and it will require inspections and maintenance work over the long-term. This fact is compounded by the high cost of access to the dam, which will either involve ongoing road maintenance costs or helicopter support.

15.2 Opportunities

15.2.1 Exploration

Exploration efforts have focussed primarily on the immediate area surrounding the known resource but no current compilation of borehole data exists. It is therefore not possible to provide detailed comments on the potential of adding to the current resource base through additional drilling. However, the gold mineralization at Red Mountain is enclosed within alteration envelopes that are quite extensive.

Several isolated high-grade intercepts exist outside the resource volumes and may have been overlooked. There is an opportunity to re-evaluate such isolated drill intercepts with the objective of delineating small high-grade volumes within the reach of proposed underground development. Also, several previous authors have pointed out that a re-evaluation of the down-dip extension of the AV and JW zones and the 141 zone is warranted.

On the property scale, the exploration potential remains excellent, but will require a fair commitment to test adequately because of steep mountainous topography and widespread ice fields and glaciers. SRK believes that new exploration targets can be identified by undertaking the work programs discussed in section 14.

15.2.2 Tailings Storage Alternatives

SRK believes that an alternate design for the cirque tailings impoundment may be possible, given the reduced volume of tailings compared to the scale of the project envisioned in the 1994 feasibility study. It would be located on one of the side slopes of Goldslide Creek, but would not cross the path of the creek. Field reconnaissance and conceptual design work are required to assess this possibility.

The potential advantages include a reduced capital cost and less onerous long-term requirements after closure.

16. CONCLUSIONS AND RECOMMENDATIONS

16.1 Conclusions

(1) Based on the development approach and cost structure incorporated in the base case economic model, at a 5% discount rate the project will break even at a gold price of US$399/oz. The base case represents SRK's best estimate of gold production and capital and operating costs.

(2) The sensitivity analyses completed on the project economics indicate that significant improvements would be needed to bring the project to viability in today's market conditions of US$360/oz gold. A 50% increase in "mineable tonnage" reduced the 5% discounted break even gold price from US$399/oz to US$350/oz. A sensitivity run incorporating 15% reductions in both operating and capital costs reduced break even from US$399/oz to US$338/oz.

(3) Regarding exploration to increase "mineable tonnage" near the planned mine workings, SRK has identified the general nature of these opportunities.

On the property scale, SRK concludes that the exploration potential remains excellent, but will require a fair commitment to test adequately because of steep mountainous topography and widespread ice fields and glaciers.

(4) SRK has drawn several conclusions about the best alternatives for the project development in the areas of site access, milling approach, operating season, and use of backfill. These are described herein.

16.2 Recommendations

SRK recommends that Seabridge consider undertaking the following work:

(1) Detailed compilation of exploration history to identify targets.

(2) Limited fieldwork to assess an alternate design for the cirque tailings facility.

This report, **3CS012.00, Red Mountain Project, Engineering Study**, June 2003, has been prepared by:

STEFFEN. ROBERTSON AND KIRSTEN (CANADA) INC.

Ken Reipas

Ken Reipas, P.Eng.
Principal Mining Engineer

CERTIFICATE AND CONSENT

To Accompany the Red Mountain Project Engineering Study, August 2003

I, Ken S. Reipas, residing at 43 Deverell Street, Whitby, Ontario, Canada, do hereby certify that:

1) I am a Principal Mining Engineer with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 602, 357 Bay Street, Toronto, Ontario.

2) I am a graduate of Queen's University with a B.Sc in Mining Engineering in 1981, and have practiced my profession continuously since 1981.

3) I am a Professional Engineer registered with the Professional Engineers of Ontario (PEO).

4) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Red Mountain Project or securities of Seabridge Gold Inc.

5) I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6) I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7) I have not had any prior involvement with the property that is subject to the technical report.

8) I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9) Steffen Robertson and Kirsten (Canada) Inc. was retained by Seabridge Gold Inc. to prepare a preliminary assessment (engineering report) for the Red Mountain Project, Stewart, B.C., in accordance with National Instrument 43-101. The following report is based on our review of project files, and discussions with project personnel.

10) I was author of the report.

11) I hereby consent to use of this report for submission to any Provincial regulatory authority.

Ken Reipas

Toronto, Canada
September, 2003

Ken S. Reipas, P.Eng.
Principal Mining Engineer

CERTIFICATE AND CONSENT

To Accompany the Red Mountain Project Engineering Study, August 2003

I, Kelly Sexsmith, residing at 517 East 10th Street, Vancouver, British Columbia, Canada, do hereby certify that:

1) I am a Senior Geochemist with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 800, 1066 West Hastings Street, Vancouver, B.C.

2) I am a graduate of the University of British Columbia with a B.Sc in Geology in 1990, and a graduate of the Colorado School of Mines with an M.S. in Environmental Sciences in 1996, and have practiced my profession continuously since 1990.

3) I am a Professional Geologist registered with the Professional Engineers and Geoscientists of British Columbia (APEGBC).

4) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Red Mountain Project or securities of Seabridge Gold Inc.

5) I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6) I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7) I have prepared other independent reports on the potential environmental impacts of the Red Mountain Property that is subject to the technical report.

8) I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9) Steffen Robertson and Kirsten (Canada) Inc. was retained by Seabridge Gold Inc. to prepare a preliminary assessment (engineering report) for the Red Mountain Project, Stewart, B.C., in accordance with National Instrument 43-101. The following report is based on our review of project files, and discussions with project personnel. I personally visited the project site in August 2000 and August 2003 to collect water samples and examine waste rock, and am familiar with the facilities that are currently on site.

10) I was co-author of the report, responsible for report section 11.2 Permitting.

11) I hereby consent to use of this report for submission to any Provincial regulatory authority.

PROFESSIONAL PROVINCE OF BRITISH COLUMBIA GEOSCIENTIST

Kelly Sexsmith Sept 29/03

Vancouver, Canada
September, 2003

Kelly Sexsmith, P.Geo
Senior Geochemist

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Appendix A
NAMC Access Road Design Details

December 15, 2000

Don Hull & Sons
3998 Desjardins St.,
Terrace, B.C.
V8G 5K1

Attention: Keith Alexander or Lloyd Hull

Re: Red Mountain Access Road

Dear Keith or Lloyd;

Please find attached a discussion report on the preliminary design of an access road for the Red Mountain project, in Stewart, British Columbia. Included with the report are a set of drawings, a table summarizing estimated volumes along the road, and a collection of photographs taken along the road alignment.

As discussed with Dianne in your office, NAMC is completing a scoping study for the Red Mountain project, and would appreciate some contractor input into estimating the capital costs of the access road. Would you please review these documents and complete the cost estimate sheet, along with any comments you may have on the design. When a decision to construct the road is made, we will prepare and issue a complete tender package to qualified and interested contractors. Please note that the volumes are 'optimized' on a 500 metre section basis, where cut volumes with their expansion equal placed fill volumes.

If we have not talked yet, I would appreciate a phone call. Please contact me at our Vancouver office (604-684-9648) or at my home (604-599-7323). I look forward to your feedback, and thank you in advance for your effort. Have a good Christmas.

Sincerely,
NORTH AMERICAN METALS CORP.

Randy Smallwood, P.Eng.
Project Manager

Proposed Red Mountain Access Road

1.0 Introduction

North American Metals Corp. is conducting a scoping study on the Red Mountain gold deposit, near Stewart, British Columbia. Part of the study includes the development of access to the deposit, and this report details the preliminary layout of a seasonal access road to the mine site.

Several options are being considered for development of the Red Mountain deposit. Possibilities range from total processing of ore at the site, concentrating the ore at the site and shipping a concentrate off site for further refining, or hauling the ore off the site for processing elsewhere. Whichever option is chosen, the access road will form an integral component of a successful operation.

2.0 Location and Geography

The Red Mountain gold deposit is located in the headwaters of Bitter Creek, eighteen kilometres due east of the town of Stewart in north-western British Columbia. The centroid of the deposit sits at the 1750 metre elevation, but surface activities will be focused on a ridge around the portal for the existing exploration decline, at the 1865 metre elevation. Please see the attached location map (figure 1).

In 1994, Lac Minerals began construction of a 14.5 kilometre access road to the bottom station of their planned tramway. The bulk of the earthworks were completed for the first 13.5 kilometres, with temporary timber bridges crossing the creeks. Development did not continue in 1995, and all the bridges were pulled out. No further work has been completed on the access road.

To avoid the high costs and difficulties associated with operating an access road through the winter in this mountainous environment, North American Metals Corp. (NAMC) is considering a seasonal mining operation. Operations would start in May and end in October, resulting in a 150 day operating season.

3.0 Scope of Work

3.1 Current Roadway (km 0+00 to 13+500)

The current roadway, from 0.0 to 13.5 kilometres, is in reasonable condition considering that it has seen no maintenance or activity since 1994. Sluffing of the cut slopes between kilometre 5+100 and 6+100, and completion of the terminal moraine gravel excavation at kilometre 7+000 form the bulk of the required earthworks for completion of this section. Bridge installations at Radio, Roosevelt, Cambria, and Hartley Gulch

Creeks will also be required. In addition, clearing vegetation back from the roadways, the re-establishment of proper drainage through ditching and culvert maintenance, and resurfacing any sections of road damaged by water overflows over the past years will also be required.

The scope of Work for this section is summarized as follows;

- Removal of sluffed material from 5+100 to 6+100.
- Selectively armouring cut slopes between 5+100 and 6+100 with oversize materials to promote stability.
- Selectively armouring the base of the fill slope from 5+100 to 6+100 with oversize materials to restrict the possibility of undercutting erosion by Bitter Creek.
- Finish the cut into the morainal gravels at kilometre 7+000 to attain a maximum road gradient of 10%.
- Clear vegetation back a minimum of 5 metres from the edges of the roadway.
- Clean all ditches and re-establish or replace culvert crossings.
- Construct abutments and approaches for bridges across Radio Creek, Roosevelt Creek, Cambria Creek, and Hartley Gulch.
- Supply and install modular bridges capable of carrying standard highway rated loads as follows;
  - Radio Creek - 10 metres long
  - Roosevelt Creek - 25 metres (possibly two sections)
  - Cambria Creek - 15 metres long
  - Hartley Gulch - 15 metres long
- Redress the travel surface of the roadway.

3.2 New Roadway (km 13+500 to 31+860)

This section of the proposed access road climbs from the 400 metre elevation in the Bitter Creek valley up to the minesite, at the 1865 metre elevation. No construction or earthworks have been completed in this section.

The proposed alignment climbs up the east side of the Bitter Creek valley until Goldslide Creek, at which point the route climbs up to the Red Mountain Cirque. From there, the road wraps around the north and east side of the cirque, climbs up onto the ridge, and approaches the minesite. With grades not exceeding 10%, about 40 switchback curves will be required.

Route options south of Goldslide Creek and north on the west flank of Red Mountain were considered, but topographic and geotechnical considerations restricted options. The attached figures detail the proposed alignment.

Volumes have been estimated for each 500 metre sections along the proposed alignment. With respect to cut slopes, the very recent glacial activity over the entire corridor has left most outcrop a very competent, resistant rock that should stand well with proper blasting techniques. Sections of the access road that passed through mapped outcrop limits and observed outcrop were treated as a rock cut slope, with a 70 degree cut slope angle.

The rest of the alignment sits on common materials typically consisting of thin ablation tills, lateral moraines, talus, or surface felsimeer (weathered in situ bedrock). In areas where a mixture of common material and rock cut are expected, an average cut slope of 57.5 degrees was used. In areas with no expected rock cut, cut slopes were taken at 45 degrees. The following list details the design criteria and approach used.

- Fill slopes at 1.5 horizontal to 1.0 vertical (34 degrees).
- Cut slopes in rock at 0.36 h to 1.0 v (70 degrees).
- Cut slopes in common at 1.0 h to 1.0 v (45 degrees).
- Cut slopes in mixed common and rock areas at 0.64 h to 1.0 v (57.5 degrees).
- In areas where the natural slope was steeper than 1.5 h to 1.0 v, a full bench cut was incorporated into the design.
- A rock cut bank bulk density of 2.5 tonnes per cubic metre.
- A common cut bank bulk density of 2.0 tonnes per cubic metre.
- A fill placed bulk density of 2.0 tonnes per cubic metre.
- An optimized material balance within each 500 metre section, where excess cut materials are used as fill elsewhere within the section. We forecast that 20% of all materials will have to be moved more than 100 metres, but none should be moved more than 500 metres.

A total of 96,000 cubic metres of rock cut and 78,000 cubic metres of common material will provide 198,000 cubic metres of fill. Material distribution is detailed by 500 metre section on the attached table.

The scope of Work for the access road from km 13+500 to 31+860 can be summarized as follows;

- Clearing of vegetation from 13+500 to 24+500. Clearing must be the larger of 20 metres or the road prism plus 3 metres above the cut slope and 3 metres below the fill slope. All vegetation must be disposed off as per forest code guidelines for methods of burning, burial or storage, but must not be placed in the road subgrade. There is no appreciable vegetation above km 24+500 on the alignment.
- Grubbing and stripping of topsoil for the entire road prism area from 13+500 to 26+000. Topsoil must be stockpiled and then spread over fill slopes to promote re-vegetation and slope stability. There is no appreciable topsoil above km 26+000 of the alignment.
- Development of the road subgrade, involving drilling and blasting of rock, and excavation of common materials. All placed materials shall be compacted.

Volumes are summarized on the attached table.

- Excavate and establish a road drainage system with ditches and culverts. Culverts should be spaced appropriately along the roadway, with ditch blocks and discharge slope armouring.
- Construct pullouts along the roadway. Average spacing of 1,000 metres, with maximum spacing of 1,500 metres. Pullouts shall provide an additional 3.0 metres of travel width, and be 30 metres long with 15 metre tapers at each end.
- Construct abutments and approaches for bridges across Otter Creek and Rio Blanco Creek.
- Supply and install modular bridges capable of carrying standard highway rated loads as follows;
  - Otter Creek - 10 metres long
  - Rio Blanco Creek - 15 metres long

  The Rio Blanco bridge should be a removable structure, as avalanche intensity may warrant removal of the bridge through the winter.
- Surfacing of the entire roadway with select granular material to a minimum 300 mm thickness, 5 metres wide. Assume that borrow sources will be within 5,000 metres along the roadway.

4.0 Conceptual Specifications

While the jurisdiction for road permitting is not yet certain (Ministry of Mines or Ministry of Forests), the finished road shall comply with B.C. Ministry of Forests Class 5 Forestry Road Standards.

- Sustained grades are not to exceed 10%, as per the attached design. Maximum allowable grades in pitches of 100m or less with tapered approaches shall not exceed 12%.
- Average design speed for the roadway shall be 50 km/hr.
- All traffic on the roadway will be two-way radio controlled, with required call-outs at regular intervals.
- The minimum finished travel width of the road shall be 5 metres.
- Pullouts shall be stationed on average every 1,000 metres, with a maximum distance between pullouts of 1,500 metres. Pullouts shall be 30 metres long with 15 metre tapers at each end. Pullouts shall be 3 metres wide, for a total driving width of 8 metres.
- On blind corners and whenever the radius of a curve is below 50 metres, the roadway shall be widened out as required for the safe travel of vehicles and equipment. On switchback curves of 15 - 20 metre radius, the minimum roadway width shall be 10 metres. On switchback curves of less than 15 metres, the roadway width shall taper into a minimum 15 metre width.

- Surfacing of the roadway shall be a minimum 300 mm thick compacted select granular material.
- The minimum clearing width shall be 20 metres, or 3 metres beyond the road prism, whichever is larger.

5.0 Schedule

With a successful scoping study, NAMC intends to proceed into feasibility and permitting. Current project scheduling has detailed design of the road and road permitting occurring through the 2001 summer season, and construction beginning early in 2002. As the road plays an important role in the development of the minesite with respect to moving equipment and consumables, NAMC hopes to have the road access to the minesite completed by the end of the 2002 season, with final dressing in 2003.

It should be noted that once construction reached the Red Mountain cirque (26+500), access over the existing tote roads will allow construction on the rest of the road to advance from two fronts.

Red Mountain Access Road Cost Estimate

#	Item	Description	Road Distances From	To	Quantity	Units	Unit Rate	Total Costs
1		**Completion of existing roadway, summarized as follows** • includes strip and stockpile of sluffed materials from 5+100 to 6+100. • completion of road cut in terminal moraine gravels ar 7+000, estimated at 10,000 m^3. * Cut and fill slope armouring for slope stabilization, erosion protection from 5+100 to 6+100. * re-establishment of drainage system, including cleaning out ditches, re-establishing or replacing culverts, and armouring and ditchblocking as * Redress the travel surface of the roadway * Clear vegetation 5 metres back from roadway.	**0+000**	**13+500**	Lump Sum			$ ________
2		**Clearing Vegetation** • standing trees. • inside of curves. • Grubbing of entire road prism area. * Minimum 20 metres back from centre point of switchback curves. • No significant vegetation above kn 24+500.	**13+500**	**24+500**	22.2	ha	$ ______ / ha	$ ________
3		**Grubbing and Stripping Topsoil** * Road prism grubbed, topsoil stripped and stockpiled apprpriately along corridor. • No appreciable topsoil above km 26+000.	13+500	26+000	18.5	ha	$ ______ / ha	$ ________
4		**Excavation and Placement of Materials** * Optimized balance of cut and fill volumes. * Certain areas of the road will require a full bench cut, as natural slopes are steeper than the long term stable fill slope angle of 1.5/1.0. * For estimating purposes, assume that 20% of borrow materials will have to be moved > 100 metres but < 500 metres. • All materials must be placed and compacted to form a long term stable subgrade and slope. Design specifications for the fill slopes are based on 1.5/1.0. • Payment based on cut bank volumes						
	4.1	Rock Cut Excavation • excavation.	13+500	31+860	95,854	m^3	$ ______ / m^3	$ ________
	4.2	Common Material Excavation • includes rippable rock, colluvium, glacial tills, current waste dump materails.	13+500	31+860	77,809	m^3	$ ______ / m^3	$ ________

Red Mountain Access Road Cost Estimate

#	Item / Description	Road Distances From	To	Quantity	Units	Unit Rate	Total Costs
5	**Surfacing**	13+500	31+860	27,000	m^3		
	• Entire road subgrade capped with a 300 mm compacted granular bed.					$ ________ / m^3	$ ________
	* For estimating purposes, assume that haulage from borrow sources will be < 1,000 metres						
6	**Pullouts**	13+500	31+860	18	pullouts	$ ________ / pullout	$ ________
	• Maximum spacing 1,500 metres, average spacing 1,000 metres.						
	* 25 metre length where roadway is 8 metres wide, with 15 metre flares on each end.						
7	**Culverts**	13+500	31+860	1,500	linear metres	$ ________ / m	$ ________
	* average diameter of 800 mm, minimum 600 mm, maximum 1200 mm.						
	* includes materials and installation, ditch and discharge armouring						
8	**Slope Armouring**	13+500	31+860	5,000	m^2	$ ________ /m^2	$ ________
	* erosion and drainage protection						
	• 80% +150mm, >300mm thick, shot rock from road cuts						
9	**Bridge Construction**	at the creek crossings					
	• abutment timber and/or concrete plus all fill materials within 10 metres of bridge deck			2	10m bridges	$ ________ / bridge	$ ________
	• 6 bridge required all together						
	* Load criteria based on standard highway ratings			3	15m bridges	$ ________ / bridge	$ ________
	• includes all costs of temporary bridge crossing during construction						
				1	25m bridges	$ ________ / bridge	$ ________
10	**Other Items**						
	________________						$ ________
	________________						$ ________
11	**Contingency**						
	* reflects the confidence level in the above estimates				percent	________ %	$ ________

Total Access Road Construction Cost Estimate $ ________

NORTH AMERICAN METALS CORP.

MEMORANDUM

To:	Dennis Bergen, John Kalmet, Bob Gilroy, Dunham Craig	
From:	Randy Smallwood	Date: February 17, 2001
Re:	Red Mountain Access Road Cost Estimate	

Gentlemen:

Please find attached a summary of the contractor estimates for the Access Road construction at the Red Mountain project.

The preliminary design and contractor estimate package was delivered to all contractors in December, 2000. In all, we had five contractor estimates returned, and their numbers are presented on the attached table.

Respecting the wide range of results, running from $4.4 to $9.3 million, and the current level of design, I feel we should discount the highest and lowest bids, and average the rest for a pre-feasibility capital cost estimate. This leaves the two Terrace based logging road contractors and Andy Alsager, a group that I also feel are the most qualified to complete the work, and that put the most time into their estimate. The resulting average cost estimate is $ 5,975,689.

If the results of the pre-feasibility study indicate that Red Mountain should proceed into the feasibility stage, I would like to invite all of the above contractors, and any other qualified contractors, for a site visit in July or August of 2001. Feasibility level design work should also be completed on the bridges and the final alignment chosen and surveyed. The centre line of the entire roadway should be cut, with topographic surveys completed to firm up volumetric estimates.

Randy Smallwood

Red Mountain Access Road Cost Estimate - Contractor Comparison

#	Item Description	Quantity	Units	Soucie Construction Unit Rate	Soucie Construction Total Costs	Pelly Construction Unit Rate	Pelly Construction Total Costs	Don Hull & Sons Unit Rate	Don Hull & Sons Total Costs	Bear Creek Contracting Unit Rate	Bear Creek Contracting Total Costs	Andy Alsager Unit Rate	Andy Alsager Total Costs
1	Completion of existing roadway	Lump Sum			$ 194,000		$ 1,300,000		$ 900,000		$ 675,000	Erosion Prote $ 400,000 Finish Cut $ 100,000 Rest of Road 13.5k @ $120/m	$ 2,000,000
2	Clearing Vegetation	22.2	ha	$ 5,163 / ha	$ 114,620	$ 6,000 / ha	$ 135,000	$ 10,951 / ha	$ 243,112	$ 7,000 / ha	$ 155,400	/ ha	
3	Grubbing and Stripping Topsoil	18.5	ha	$ 4,338 / ha	$ 80,250	$ 15,000 / ha	$ 277,500	$ 10,951 / ha	$ 202,594	$ 12,000 / ha	$ 222,000	/ ha	
4	Excavation and Placement of Materials												
	Rock Cut	95,853.9	m^3	$ 25 / m^3	$ 2,396,350	$ 25 / m^3	$ 2,396,350	$ 23 / m^3	$ 2,216,144	$ 15 / m^3	$ 1,437,810	/ m^3	
	Common Material	77,808.7	m^3	$ 5 / m^3	$ 389,045	$ 15 / m^3	$ 1,167,135	$ 9 / m^3	$ 661,377	$ 10 / m^3	$ 758,638	/ m^3	
5	Surfacing	27,000.0	m^3	$ 7 / m^3	$ 189,000	$ 20 / m^3	$ 540,000	$ 10 / m^3	$ 256,500	$ 12 / m^3	$ 324,000	/ m^3	
6	Pullouts	18.0	pullouts	$ 2,000 / pullout	$ 36,000	$ 5,000 / pullout	$ 90,000	$ 11,111 / pullout	$ 200,000	$ 2,500 / pullout	$ 45,000	/ pullout	
7	Culverts	1,500.0	linear metres	$ 85 / m	$ 127,500	$ 500 / m	$ 750,000	$ 400 / m	$ 600,000	$ 300 / m	$ 450,000	/ m	$ 293,000
8	Slope Armouring	5,000.0	m^2	$ 7 /m^2	$ 35,000	$ 15 /m^2	$ 75,000	$ 32 /m^2	$ 161,200	$ 15 /m^2	$ 75,000	/m^2	
9	Bridge Construction	2.0	10m bridges	$ 45,500 / bridge	$ 91,000	$ 100,000 / bridge	$ 200,000	$ 75,000 / bridge	$ 150,000	$ 37,500 / bridge	$ 75,000	$ 39,000 / bridge	$ 78,000
		3.0	15m bridges	$ 68,250 / bridge	$ 204,750	$ 150,000 / bridge	$ 450,000	$ 100,000 / bridge	$ 300,000	$ 52,000 / bridge	$ 156,000	$ 48,000 / bridge	$ 144,000
		1.0	25m bridges	$ 113,750 / bridge	$ 113,750	$ 300,000 / bridge	$ 300,000	$ 150,000 / bridge	$ 150,000	$ 84,000 / bridge	$ 84,000	$ 91,000 / bridge	$ 91,000
10	Other Items											Overlanding 4.5k @ $130/m	$ 585,000
						Overhead	$ 750,000			Bridge Freight	$ 40,000	0-5 7k @ $150/m	$ 1,050,000
												5-10 2.5k @ $185/m	$ 462,500
										Camp, Support	$ 100,000	10-15 2k @ $220/m	$ 440,000
												Endhaul 2k @ $350/m	$ 700,000
11	Contingency		percent	10%	$ 397,127	10%	$ 843,099	12.5%	$ 755,116	15%	$ 689,677	%	
	Total Access Road Construction Cost Estimate				**$ 4,368,392**		**$ 9,274,084**		**$ 6,796,043**		**$ 5,287,525**		**$ 5,843,500**
	Actual Contractor Estimate (some totals did not match sum of items)												$ 5,806,000

Contractor Notes:	Soucie Construction	Pelly Construction	Don Hull & Sons	Bear Creek Contracting	Andy Alsager
	Based in Stewart	Based in Whitehorse	Based in Terrace	Based in Terrace	Based on the Gulf Islands (D. Craig reference)
	Completed first phase of construction for Lac	Better known as a Rock contractor versus a Road Contractor	Built Eskay Creek Mine Access Road, working with the Tahltans	Strongly recommended by Randy Daggitt of Titan Explosives	road $0.5m with more detail on end hauling requirement, and has contingency built into his numbers
	Probably the most familar with the condition of the current road	Site Visit in August 2000	Bid on Previous Round of work, and on subsequent closure work (never completed). Several Site Visits.	Large road contractor, primarily focussed on Logging Roads	Used road profiles spaced every 500 metres along new road construction. Prefers linear metre costing.
	Some history with MOF	Bridge estimates without consulting bridge supplier	described as largest Road Contractor in Northern B.C.	Has not been to site	Has not been to site

Appendix B
SRK Re-evaluation of Access Alternatives

Red Mountain Access Alternatives

Methods of access considered:

(1) Road to the top of the mountain. 32 kilometers in length from the paved highway 37A .

13.5 kilometers of existing logging road must be upgraded
18.5 kilometers of new road is needed to get to the top of Red Mountain.

(2) Tramway. It starts in Bitter Creek valley at:

North	6,203,550	UTM coordinates
East	453,075	
Elevation	542	

and goes to the top of the mountain, not far from the proposed portal.
To access the bottom tram terminal, the following road work is needed:

13.5 kilometers of existing logging road must be upgraded
2.0 kilometers of new road is needed

(3) Tunnel and Shaft. The tunnel starts in Bitter Creek valley at the same point as the lower tram terminal shown above.
Road work needed to access this point is the same as in (2) above.

(4) Helicopter Supported Mining.
All men, equipment and supplies would be flown in, with no other method of access.
The staging area for preparing loads would be along highway 37, near Bitter Creek.

Alternative (1) Road Access

Preliminary engineering work has been done on the road alignment.
Specifications are: Sustained grades not to exceed 10%
Maximum grades of 12% for 100m or less
Minimum finished travel width 5m
Road drawings have been prepared. Cut and fill volumes have been determined.

Capital Cost Estimate
A detailed description of the road, including pictures along the route, was sent to five contracting companies in December 2000 to obtain cost estimates.

Five responses were recieved, ranging from $4.4 to $9.3 million.

For the purpose of this current study, the following cost estimate is being used:

$6.8 million
Don Hull & Sons, Terrace , BC
one of the largest road contractors in northern BC
have made several site visits
built Eskay Creek mine access road

Their cost estimate is presented below, broken up into the two road segments that are needed for this trade-off study:

			CDN$ Thousands
15.5 kilometers to the start point of the tram and the tunnel	Upgrade existing road		900
	5 bridges		500
	2km new road		300
	Sub-total		1,700
	Contingency 12.5%		213
	TOTAL	**$**	**1,913**
16.5 kilometers the remaining distance from Bitter Creek valley up to the portal area:			
	1 bridge		100
	16.5 km new road		4,244
	Sub-total		4,344
	Contingency 12.5%		543
	TOTAL	**$**	**4,887**
32 kilometers in total	**Combined Total**	**$**	**6,800**

year 2001 dollars

Alternative (2) Tramway

The capital cost of this alternative is the cost of the tramway itself, and the cost of the access road to the bottom terminal.

Capital costs for a service tramway (no ore transportation) were presented in the report, "Order of Magnitude Study - Transportation Alternatives", 1994

To bring the 1994 costs up to 2001 dollars, to compare to the road capital cost, inflation of approximately 16% must be added.
The same contingency is used here, 12.5%

Tramway Capital Cost Estimate

			CDN$ Thousands
Tramway			10,800
Inflation amount	16%		1,728
Sub-total			12,528
Contingency 12.5%			1,566
Tramway Total			14,094
Plus access road to lower terminal			1,913
TOTAL		**$**	**16,007**
			year 2001 dollars

Alternative (3) Tunnel and Shaft

This option also requires an ore pass in parallel to the tunnel.
The development work for this option is:

4.5m diameter bored tunnel, 3500 meters long, +10%
686 meter shaft
702 meter ore pass

Capital costs for this alternative were presented in the report, "Order of Magnitude Study - Transportation Alternatives", 1994

To bring the 1994 costs up to 2001 dollars, to compare to the road capital cost, inflation of approximately 16% must be added.
The same contingency is used here, 12.5%

Tunnel and Shaft Capital Cost Estimate

			CDN$ Thousands
Tunnel, shaft, ore pass, etc			14,574
Inflation amount	16%		2,332
Sub-total			16,906
Contingency 12.5%			2,113
Tramway Total			19,019
Plus access road to portal location			1,913
TOTAL		**$**	**20,932**
			year 2001 dollars

Alternative (4) Mining with Helicopter Support

Of the first 3 alternatives described, the access road is clearly the best option based on the capital costs. The differences in operating costs among these 3 alternatives cannot change this conclusion, and they will not be examined and compared in this trade-off study.

Helicopter support, alternative (4) is more difficult to compare to alternative (1) access road.

Capital Cost
The capital cost of alternative (4) is the helicopter costs for completing the pre-production works.

This cost is difficult to estimate, and may be very attractive compared to the cost of the alternative (1) access road.

For one field season of helicopter support at Red Mountain, the cost is roughly $2 million. This supports a diamond drilling program and limited underground development.

Based on this, a very rough estimate for Red Mountain pre-production helicopter support is at least $ 4,000 thousand.

Operating Cost Estimates (1) and (4)
There are very large operating cost differences between alternatives (1) and (4) and these must be examined, and compared.

Road Operating Cost Estimate (worst case - high estimate)
Based on seasonal production of 194,000 tonnes

			CDN$ Thousands				
Road Maintenance	L.Sum		900	per season			based on factoring 1.8 million per year contractor estimate
Avalanche control			101				helicopter 8 hrs per week for 3 months, plus powder
Mine freight costs	$1.00/t		194				a high estimate only
Crew transport			166				two crew busses with drivers full time
Total road costs		$	1,361	per season	$	7.02	per tonne mined

Helicopter Support Operating Cost Estimate

		CDN$ Thousands		
Limited development & diamond drilling	$	2,300	per season	based on detailed discussion with Vancouver Island Helicopters
Supporting 1200tpd mining and milling would be double this (best case -low)		2.0	factor	
Helicopter support for production	$	4,600	per season	
Operating Cost Difference (1) vs (4)	**$**	**3,239**	per season	
Life of mine (minimum)		5	seasons	
LOM operating cost difference	**$**	**16,193**	over life of mine	

Conclusion: Alternative (4) is eliminated on the basis of operating costs. Even if there were free helicopter support for pre-production and a zero capital cost, this conclusion would stand.

Appendix C
Worksheets: Mineral Processing Alternatives

Operating Cost: Previous work NAMC June 2000

Case A: Premier Mill, Haul road up mountain

Seasonal mining/milling 830tpd

							Contained Grams	Contained Ounces
In-stu ore mined	tonnes		1,000	tonnes				
above cut off	gold grade		9.14	Au g/t			9,140	
	silver grade		28.70	Ag g/t			28,700	
Dilution	percent		15.00%		estimated dilution			
	tonnes waste		176	tonnes			0	
Tonnes Mined	tonnes		1,176	tonnes				
	gold grade		7.77	Au g/t			9,140	293.9
	silver grade		24.40	Ag g/t			28,700	922.8
Gross revenue	Gross revenue				$	187,671		
	Gross revenue per tonne				$	159.52		
Mill Feed	tonnes		1,176	tonnes				
	gold grade		7.77	Au g/t			9,140	
	silver grade		24.40	Ag g/t			28,700	
Mill Recovery	gold		88%					
	silver		80%					
Recovered Metal	gold		in dore				8,043	258.6
	silver		in dore				22,960	738.3
Refining Charges	Gold	$	6.00	US$/payable oz				
	Silver	$	0.45	US$/payable oz				
	Gold refining cost				$	2,387		
	Silver refining cost				$	511		
	Total refining charge				$	2,898		
	Cost per tonne mined				$	2.46		
Metal Prices	gold			US$/oz	$	400.00		
	silver			US$/oz	$	4.80		
	exchange					0.65		
Revenue	gold revenue				$	159,153		
	silver revenue				$	5,452		
	Total revenue				$	164,605		
	Revenue per Tonne Mined				$	139.91		
NSR Calculation	*Revenue per Tonne Mined*				$	139.91		
	minus: refining				$	2.46		
	NSR per tonne mined				**$**	**137.45**		

Oper. Costs			**LOM Avg.**				
Mine			40.00		road maint	$	5.00
Roads/Ore haul			31.22	←	haulage	$	26.22
Mill			19.90				
Admin			13.93				
Total CDN$		$	105.06	per tonne			
NSR Royalty 3%	plus	$	3.15				
Total Oper Cost		**$**	**108.21**	per tonne			
Operating margin		$	29.24	per tonne mined			

Operating Cost: Previous work NAMC June 2000

Case B: Flotation Mill, Haul road up mountain

Seasonal mining/milling 830tpd

Haul sulphide concentrate to tidewater

Assume:
concentration ratio of 4
gold recov to conc 90%
mill cost 90% of A

					Contained Grams	Contained Ounces
In-stu ore mined	tonnes	1,000	tonnes			
above cut off	gold grade	9.14	Au g/t		9,140	
	silver grade	28.70	Ag g/t		28,700	
Dilution	percent	15.00%		estimated dilution		
	tonnes waste	176	tonnes		0	
Tonnes Mined	tonnes	1,176	tonnes			
	gold grade	7.77	Au g/t		9,140	293.89
	silver grade	24.40	Ag g/t		28,700	922.83
Gross revenue	Gross revenue			$ 187,671		
	Gross revenue per tonne			$ 159.52		
Mill Feed	tonnes	1,176	tonnes			
	gold grade	7.77	Au g/t		9,140	
	silver grade	24.40	Ag g/t		28,700	
Concentration Ratio		4.0				
Gold and Silver Recovery		90%				
Conc. Produced	tonnes	294.1	tonnes			
	gold grade	27.97	Au g/t		8,226	
	silver grade	87.82	Ag g/t		25,830	
Tails Produced	tonnes	882.4	tonnes			
	gold grade	0.93	Au g/t		823	
	silver grade	2.93	Ag g/t		2,583	
Conc. Haulage	Land cost per tonne conc		CDN$/DMT	$ 26.22		
	Total cost in country			$ 7,712		
	Ocean freight		US$/DMT	$ 30.00		
	Total ocean freight cost			$ 13,575		
	Cost per DMT conc			$ 72.37		
	Cost per tonne mined			$ 18.09		
Conc. Smelting	TC rate	$ 110.00	US$/DMT			
	Exchange	0.65				
	TC rate	$ 169.23	CDN$/DMT			
	TC Cost			$ 49,773.76		
	Cost per tonne mined			$ 42.31		
					Gram	Ounce
Smelter Payable	*gold*	97.5%		payable	8,020	257.9
	silver	90.0%		payable	23,247	747.5
Refining Charges	Gold	$ 6.00	US$/payable oz			
	Silver	$ 0.45	US$/payable oz			
	Gold refining cost			$ 2,381		
	Silver refining cost			$ 517		
	Total refining charge			$ 2,898		
	Cost per tonne mined			$ 2.46		
Metal Prices	gold		US$/oz	$ 400.00		
	silver		US$/oz	$ 4.80		
	gold revenue			$ 158,701		
	silver revenue			$ 5,520		
	Total revenue			$ 164,221		
	Revenue per Tonne Mined			$ 139.59		
NSR Calculation	Revenue per Tonne Mined			$ 139.59		
	minus:	Conc. Haulage		18.09		
		Smelting		42.31		
		Refining		2.46		
	NSR per tonne mined			**$ 76.72**		

Oper. Costs		**LOM Avg.**		
Mine		40.00		
Road maint		5.00		
Mill (90% of case A)		17.91		less grinding
Admin		13.93		
Total CDN$		$ 76.85	per tonne	
NSR Royalty 3%	plus	$ 2.31		
Total Oper Cost		**$ 79.15**	per tonne	
Operating margin		$ (2.43)	per tonne mined	

Operating Cost: Previous work NAMC June 2000

Case C: Cyanidation Mill, Haul road up mountain

Seasonal mining/milling 830tpd

						Contained Grams	Contained Ounces
In-stu ore mined	tonnes	1,000	tonnes				
above cutoff	gold grade	9.14	Au g/t			9,140	
	silver grade	28.70	Ag g/t			28,700	
Dilution	percent	15.00%		estimated dilution			
	tonnes waste	176	tonnes			0	
Tonnes Mined	tonnes	1,176	tonnes				
	gold grade	7.77	Au g/t			9,140	293.9
	silver grade	24.40	Ag g/t			28,700	922.8
Gross revenue	Gross revenue			$	187,671		
	Gross revenue per tonne			$	159.52		
Mill Feed	tonnes	1,176	tonnes				
	gold grade	7.77	Au g/t			9,140	
	silver grade	24.40	Ag g/t			28,700	
Mill Recovery	gold	88%					
	silver	80%					
Recovered Metal	gold	in dore				8,043	258.6
	silver	in dore				22,960	738.3
Refining Charges	Gold $	6.00	US$/payable oz				
	Silver $	0.45	US$/payable oz				
	Gold refining cost			$	2,387		
	Silver refining cost			$	511		
	Total refining charge			$	2,898		
	Cost per tonne mined			$	2.46		
Metal Prices	gold		US$/oz	$	400.00		
	silver		US$/oz	$	4.80		
	exchange				0.65		
Revenue	gold revenue			$	159,153		
	silver revenue			$	5,452		
	Total revenue			$	164,605		
	Revenue per Tonne Mined			$	139.91		
NSR Calculation	Revenue per Tonne Mined			$	139.91		
	minus: refining			$	2.46		
	NSR per tonne mined			**$**	**137.45**		
Oper. Costs		**LOM Avg.**					
Mine		40.00					
Road maint		5.00					
Mill (110% of case A)		21.89		due to being on mountain			
Admin (110% of case A)		15.33					
Total CDN$		$ 82.22	per tonne				
NSR Royalty 3%	plus	$ 2.47					
Total Oper Cost		**$ 84.69**	per tonne				
Operating margin		$ 52.77	per tonne mined				

Appendix D
BC Hydro Power Supply Quote

BChydro THE POWER IS YOURS

Fax Cover Sheet

TO: Andrew Bradfield DATE: April 29/2003

FAX: 416-601-9046

PHONE:

FROM: Reg Durrell
Prince George
BC Hydro
Box 6500
Prince George,

TEL: (250) 561-4820

FAX: 866-266-6366

As per your conversation with Terry Shulz, attached are the energy rates for 25kv service

E-MAIL:

If you are experiencing any difficulty in receiving this transmission, please call: 250 561-4822

No. of pages (including cover page): 4

MESSAGE

SCHEDULES 1200, 1201, 1210, 1211

GENERAL SERVICE (35 kW and over)

Availability: For all purposes. Supply is 60 hertz, single or three phase at secondary or primary potential. The Authority reserves the right to determine the potential of the service connection.

Applicable in: Rate Zone I.

Rate: Basic Charge $4.15 per month

Demand Charge

First 35 kW of billing demand per month Nil
Next 115 kW of billing demand per month @ $3.32 per kW
All additional kW of billing demand per month @ $6.37 per kW

plus

Energy Charge

First 14800 kW.h per month @ 6.49¢ per kW.h
All additional kW.h per month @ 3.12¢ per kW.h. C

Discounts

1. A discount of 1½% shall be applied to the above rate if a customer's supply of electricity is metered at a primary potential.

2. A discount of 25¢ per kW of billing demand shall be applied to the above rate if a customer supplies transformation from a primary potential to a secondary potential.

3. If a customer is entitled to both of the above discounts, the discount for metering at a primary potential shall be applied first.

SCHEDULES 1200, 1201, 1210, 1211

GENERAL SERVICE (35 kW and over) (Cont'd)

Billing Codes:

Schedule 1200 applies if a customer's supply of electricity is metered at a secondary potential and the Authority supplies transformation from a primary potential to a secondary potential.

Schedule 1201 applies if a customer's supply of electricity is metered at a primary potential and the Authority supplies transformation from a primary potential to a secondary potential.

Schedule 1210 applies if a customer's supply of electricity is metered at a secondary potential and the customer supplies transformation from a primary potential to a secondary potential.

Schedule 1211 applies if a customer's supply of electricity is metered at a primary potential and the customer supplies transformation from a primary potential to a secondary potential.

Monthly Minimum Charge:

The greater of:

1. Twelve dollars and twenty-two cents ($12.22) per month, or r
2. 50% of the highest maximum demand charge billed in any month wholly within an on-peak period during the immediately preceding eleven months. For the purpose of this provision an on-peak period commences on 1 November in any year and terminates on 31 March of the following year.

o

SCHEDULES 1200, 1201, 1210, 1211

GENERAL SERVICE (35 kW and over) (Cont'd)

Special Condition:	A demand meter will normally be installed. Prior to the installation of such a meter, or if such a meter is not installed, the demand for billing purposes shall be the assessed demand estimated by the Authority.
Special Condition: (Closed)	Where electricity is supplied under Schedule 1272 for air conditioning of, or as a principal space heating fuel for the premises being supplied, and, in addition,

1. all metering is at the secondary potential on the load side of the customer's transformers, and
2. the customer provides the necessary transformers and all associated equipment other than meters, and
3. electricity supplied for purposes within Schedule 1272 is metered through a separate single meter, and separately billed on that schedule,

then all other electricity supplied at the same premises shall be metered through a separate single meter and separately billed on Schedule 1211.

This condition is available only to a customer in those premises with respect to which the Authority, prior to 1 April 1969, agreed that service would be provided under this condition and only with respect to the load which the Authority, prior to 1 April 1969, agreed to provide with service.

Appendix E
Cut Off Grade Worksheet

Red Mountain Cut Off Grade Estimate

Operating Cost:	Previous work NAMC June 2000						
	Case C: Cyanidation Mill, Haul road up mountain						
	Seasonal mining/milling 830tpd						
						Contained	Contained
In-stu ore mined	tonnes	1,000	tonnes			Grams	Ounces
	gold grade	5.21	Au g/t			5,215	
	silver grade	26.20	Ag g/t			26,200	
Dilution	percent	15.00%		estimated dilution			
	tonnes waste	176	tonnes			0	
Tonnes Mined	tonnes	1,176	tonnes				
	gold grade	4.43	Au g/t			5,215	167.7
	silver grade	22.27	Ag g/t			26,200	842.4
Gross revenue	Gross revenue			$	115,859		
	Gross revenue per tonne			$	98.48		
Mill Feed	tonnes	1,176	tonnes				
	gold grade	4.43	Au g/t			5,215	
	silver grade	22.27	Ag g/t			26,200	
Mill Recovery	gold	88%					
	silver	80%					
Recovered Metal	gold	in dore				4,589	147.6
	silver	in dore				20,960	674.0
Refining Charges	Gold $	6.00	US$/payable oz				
	Silver $	0.45	US$/payable oz				
	Gold refining cost			$	1,362		
	Silver refining cost			$	467		
	Total refining charge			$	1,829		
	Cost per tonne mined			$	1.55		
Metal Prices	gold		US$/oz	$	425.00		
	silver		US$/oz	$	4.80		
	exchange				0.65		
Revenue	gold revenue			$	96,481		
	silver revenue			$	4,977		
	Total revenue			$	101,458		
	Revenue per Tonne Mined			$	86.24		
NSR Calculation	Revenue per Tonne Mined			$	86.24		
	minus: refining			$	1.55		
	NSR per tonne mined			**$**	**84.69**		

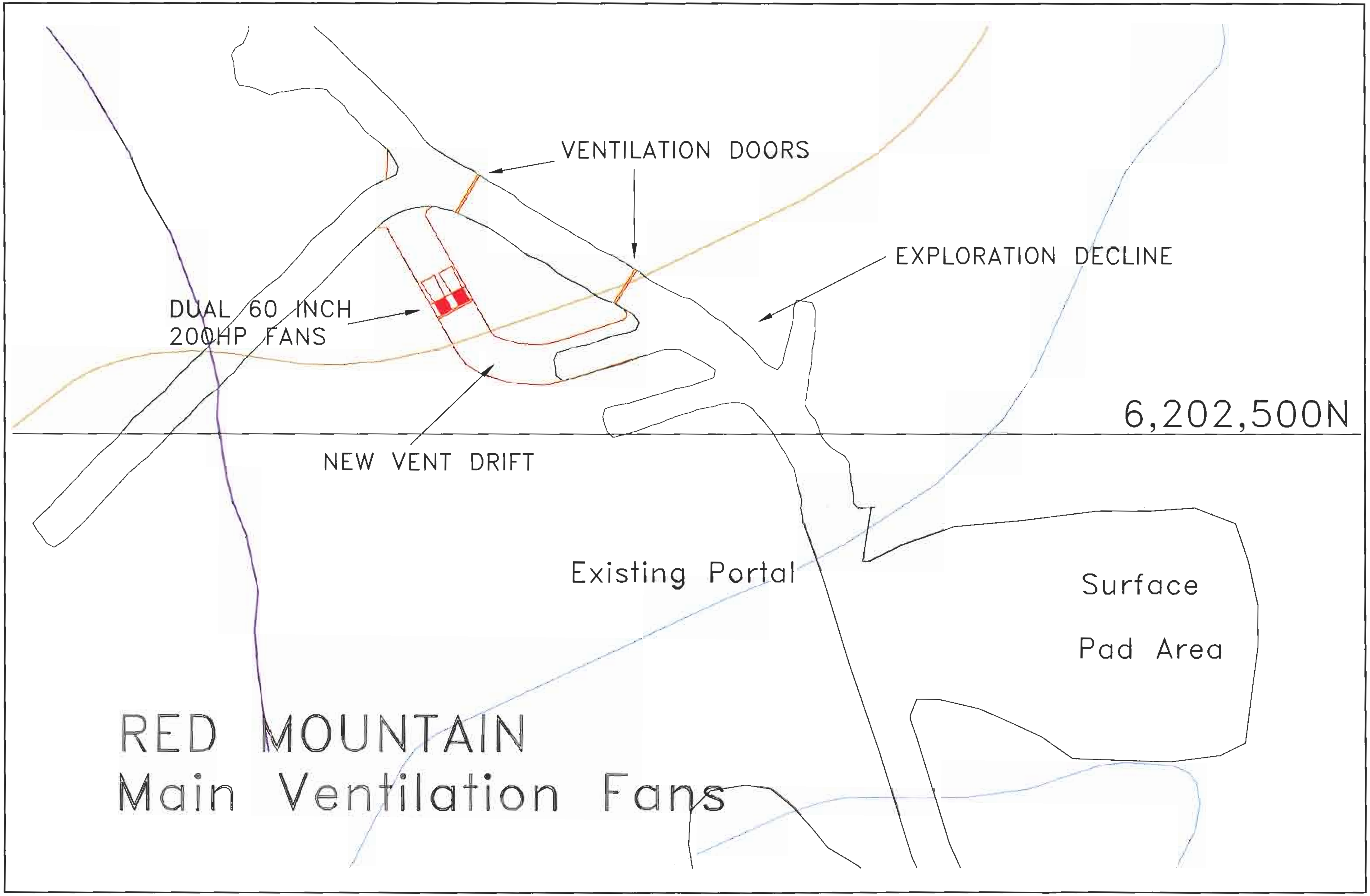
Oper. Costs		**LOM Avg.**		
Mine		40.00		
Road maint		5.00		
Mill (110% of case A)		21.89		due to being on mountain
Admin (110% of case A)		15.33		
Total CDN$	$	82.22	per tonne	
NSR Royalty 3% plus	$	2.47		
Total Oper Cost	**$**	**84.69**	per tonne	
Operating margin	$	-	per tonne mined	

Appendix F
Mine Ventilation Requirements

Red Mountain Ventilation Requirements

Equipment Requirements UNDERGROUND	No. Units	Ventilation Required per Unit cfm	Ventilation Required @ 100% cfm	Derating Factor %	Ventilation Required Derated cfm	
Supervision						
Kubota Tractor	1	12,000	12,000	50%	6,000	
Personnel Transport						
Getman carrier	1	12,000	12,000	50%	6,000	
Nipping						
Getman hiab	1	12,000	12,000	50%	6,000	
Definition drilling						
Gopher Drill	1	8,000	8,000	100%	8,000	
Development						
Jumbo	1	10,000	10,000	50%	5,000	
6 YD Scoop	2	35,000	70,000	100%	70,000	
Scissor lift	1	12,000	12,000	50%	6,000	
Truck 20T	1	35,000	35,000	50%	17,500	
Production & Backfill						
Jumbo C&F	1	10,000	10,000	100%	10,000	
6 YD Scoop	1	35,000	35,000	100%	35,000	
4 YD Scoop	2	18,000	36,000	100%	36,000	
Scissor lift	1	12,000	12,000	50%	6,000	
Truck 20T	2	35,000	70,000	100%	70,000	
LH Drill	1	10,000	10,000	100%	10,000	
Anfo Truck	1	12,000	12,000	50%	6,000	
Road Maintenance						
Grader	1	25,000	25,000	100%	25,000	
Construction						
Getman hiab	1	12,000	12,000	50%	6,000	
Maintenance						
Kubota Tractor	1	12,000	12,000	50%	6,000	
Mechanics hiab	1	12,000	12,000	50%	6,000	
Scissor lift	1	12,000	12,000	50%	6,000	
Total	23	341,000	429,000		**346,500**	**cfm**
					163.7	**cms**
		Air Velocity	4.0 x 5.0 m drift		1,610	
		Air Velocity	4.5 x 5.0 m drift		1,431	fpm

Appendix G
Main Ventilation Fan Installation


VENTILATION DOORS
EXPLORATION DECLINE
DUAL 60 INCH
200HP FANS
6,202,500N
NEW VENT DRIFT
Existing Portal
Surface
Pad Area
RED MOUNTAIN
Main Ventilation Fans

Appendix H
Monthly Mine Production Schedule

LOWER MINE

DEVELOPMENT

Capital		Meters	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
Portal to 1685el Junction		720	portal	200	200	200	120																																															
Junction to R.Breaker 1720el		274					80	194																																														
Excavate for R.Breaker		23						6	17																																													
Junction to Chute 1690el		259								30	175	54																																										
O/P 1720 to 1800		85							70	15																																												
O/P + Bin 1690 to 1720		34								34																																												
Install R.Breaker + Chains											install																																											
A1A3 Ramp 1720-1712		89							89																																													
1315VR 1712 to 1815		105							40	65																																												
A1A3 Ramp 1712-1690		189							131	58																																												
1315VR 1695 to 1712		17									17																																											
A1A3 Ramp 1720-1730		106							33	73																																												
A1A3 Ramp 1730-1741		98									25	73																																										
A1A3 Ramp 1741-1758		138										98	40																																									
A1A3 Ramp 1758-1772		138											113	25																																								
A1 1700 Dev		57								57																																												
A1 1730 Dev		74								45					29																																							
A1 1758 Dev		88														40	48																																					
A3 LH Dev All		808											97	100	93						50	50	50	50	50	30	30	30	30	30	30	30	30	28																				
A3 CF		37												37																																								
A4 Dev All		59																	59																																			
A2 Dev All		431																										50	50	50	50	50	50	50	50	31																		
JW Dev Lower		339																															29	30	40	40	40	40	40	40	40													
JW Dev Upper		368																														18	30	40	40	40	40	40	40	40	40													
Total Capital		4,274																																																				
Operating		Meters																																																				
A1	LH	178									50	50	prep		35	43																																						
A4	LH																																																					
A3 LH	LH	414														80	80	prep								30	30	30	30	30	30	30	30	14																				
A3 CF	C&F																																																				
A2	C&F	160																																				15	15	15	15	15	15	15	15	15	15	10						
JW Lower	C&F	168																																								10	15	18	18	18	18	18	18	18	17			
JW Upper	C&F	168																																								10	15	18	18	18	18	18	18	18	17			
Total Oper		1088																																																				

PRODUCTION

Lower		stope t	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
A1	LH	112,824												11.8	11.9	8.0	10	10.8	10.8	10.8	10.8	10	10	8																														
A4	LH	10,751																																																			10.8	
A3 LH	LH	250,190																						1.9	8.9	12	12	12	12	14.3	14.9	14	14	14	14	14	14	14	14	14	14	7.6										4.2	10.4	
A3 CF	C&F	13,771												4.0	4.0	3.0	2.8																																					
A2	C&F	85,353																																				5	7	7	7	7	6	6	7	7	7	6	3.8	5.0	4.8			
JW Lower	C&F	82,699																																								2.7	8	8	6.9	8	8	7	7	8	8	8	3.1	
JW Upper	C&F	82,699																																								2.7	8	8	8	8	8	7	7	8	8	8	2	
		638,286																																																				

LOWER MINE TOTALS

Development	month 1	2	3	4	5	6	7	6	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
Ore meters	-	-	-	-	-	-	-	-	50	50	-	-	35	123	80	-	-	-	-	-	-	-	-	30	30	30	30	30	30	30	30	14	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste meters	0	200	200	200	200	200	250	263	200	225	250	162	122	40	48	0	59	0	50	50	50	50	50	30	30	80	80	80	80	98	139	148	130	111	80	95	95	95	95	35	45	51	51	51	51	46	36	36	34	0	0	0
Total lateral meters	0	200	200	200	200	200	250	263	250	275	250	162	157	163	128	0	59	0	50	50	50	50	50	60	60	110	110	110	110	128	169	162	130	111	80	95	95	95	95	35	45	51	51	51	51	46	36	36	34	0	0	0
Raising meters	0	0	0	0	0	0	110	114	17	0	0	0	0	0	0	0	0	0	0	0	0	80	0	0	0	0	0	0	0	0	30	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Waste tpd	-	329	329	329	329	329	476	499	339	370	411	266	200	66	79	-	97	-	82	82	82	129	82	49	49	131	131	131	131	161	246	243	214	182	131	156	156	156	156	57	74	84	84	84	84	76	59	59	56	-	-	-
Production																																																				
Longhole tonnage (kt)	-	-	-	-	-	-	-	-	-	-	-	12	12	8	10	11	11	11	11	10	10	10	9	12	12	12	12	14	15	14	14	14	14	14	14	14	14	14	14	8	-	-	-	-	-	-	-	-	-	4	21	-
C&Fill tonnage (kt)	-	-	-	-	-	-	-	-	-	-	-	4	4	3	3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5	7	7	7	12	22	22	22	23	23	20	18	21	21	16	5	-
Total Ore tonnage (kt)	-	-	-	-	-	-	-	-	-	-	-	16	16	11	13	11	11	11	11	10	10	10	9	12	12	12	12	14	15	14	14	14	14	14	14	19	21	21	21	20	22	22	22	23	23	20	18	21	21	20	26	-

UPPER MINE

DEVELOPMENT

month

Capital	Meters	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
MN12 1772 Dev	202					75									30	30	30	30	7								
MN12 1802 Dev	341	setup	150	50	50									30	30	31											
MN12 1838 Dev	144					48	30						36	30													
A1A3 Ramp 1796-1772	187			80	100	27																					
MS1 CFU Dev	59																				59						
MS1 CFL Dev	62						82																				
MS1 LH 1796 Dev	132			40			20															30	20	22			
MS1 LH 1818 Dev	169								15	40												20	30	30	34		
MS1 LH 1832 Dev	120																							20	20	20	20
MS1 LH 1847 Dev	87							40															20	27			
MS1 LH 1858 Dev	73																										
MS1 1802-1816	130						38	42	50																		
MS1 1816-1832	163											15	50	35	35	28											
MM3	95																										
MM4	48																										
MS11	53									53																	
MM5 LH	37																										
MM5 CF	84																										
MM6	189																										
MS9	53							18	15	20																	
MN7	37									37																	
MN8	117																										
MN10	212																			50	50	50	50	12			
Total Capital	2,772																										

Capital	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
MN12 1772 Dev																										
MN12 1802 Dev																										
MN12 1838 Dev																										
A1A3 Ramp 1796-1772																										
MS1 CFU Dev																										
MS1 CFL Dev																										
MS1 LH 1796 Dev																										
MS1 LH 1818 Dev																										
MS1 LH 1832 Dev	20	20																								
MS1 LH 1847 Dev																										
MS1 LH 1858 Dev		73																								
MS1 1802-1816																										
MS1 1816-1832																										
MM3			50	45																						
MM4					26	20																				
MS11																										
MM5 LH																37										
MM5 CF										44	40															
MM6													24	30	30	30	30	25								
MS9																										
MN7																										
MN8																	17	25	25	25	25					
MN10																										
Total Capital																										

Operating		Meters	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
MS1 LH	LH	381										30	85	prep														50
MN12	LH	250								80	prep	50					20	20	20	20	20	20						
MM3	LH	15																										
MM4	LH	25																										
MS11	LH	-																										
MM5 LH	LH	35																										
MM5 CF	C&F	24																										
MS1 CFU	C&F	50																										
MS1 CFL	C&F	10										10																
MM6	C&F	80																										
MS9	C&F	80																										
MN7	C&F	-																										
MN8	C&F	80																										
MN10	C&F	48																							12	12	12	12
Total		1,078																										

Operating		27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
MS1 LH	LH	50	40	50	50	28																					
MN12	LH																										
MM3	LH					15																					
MM4	LH							25																			
MS11	LH																										
MM5 LH	LH																	35									
MM5 CF	C&F																				12	12					
MS1 CFU	C&F			12	12	14	12																				
MS1 CFL	C&F																										
MM6	C&F																						25	30	25		
MS9	C&F			30	30	20																					
MN7	C&F																										
MN8	C&F																						20	25	25	10	
MN10	C&F	12																									
Total																											

PRODUCTION

Upper		stope t	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
MS1 LH	LH	194,540																8.0	6.0	8.0	6.0	8.8	8.0	8.0	9.1			
MN12	LH	205,202											12.0	12.0	12.0	11.8	11.2	12.0	12.0	12.0	12.0	12.0	12.0	12.0	12.0	10.2	10.2	10.2
MM3	LH	8,476																										
MM4	LH	11,491																										
MS11	LH	12,353										5.0	5.0	2.4														
MM5 LH	LH	14,247																										
MM5 CF	C&F	11,665																										
MS1 CFU	C&F	25,751																										
MS1 CFL	C&F	13,866										8.0	7.9															
MM6	C&F	7,017																										
MS9	C&F	9,929																										
MN7	C&F	3,362										3.4																
MN8	C&F	7,912																										
MN10	C&F	19,906																								6.0	6.0	3.0
Total		545,716																										

Upper		27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
MS1 LH	LH					2.6	6.9	11.4	9.8	12.0	7.5	9.0	9.0	9.0	10.0	8.0	8.0	6.0	7.0	7.0	8.0	7.4					
MN12	LH	10.1	9.7																								
MM3	LH						3.2	3.1	2.2																		
MM4	LH								4.0	4.0	3.5																
MS11	LH																										
MM5 LH	LH																							6.0	5.0	3.3	
MM5 CF	C&F																				2.0	5.0	4.7				
MS1 CFU	C&F			7.3	7.4	6.0	5.1																				
MS1 CFL	C&F																										
MM6	C&F																						2.0	1.6	3.4		
MS9	C&F			3.0	3.8	3.1																					
MN7	C&F																										
MN8	C&F																						2.3	1.6	2.4	1.8	
MN10	C&F	3.1	1.8																								
Total																											

UPPER MINE TOTALS

month

Development	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Ore meters	-	-	-	-	-	-	-	80	-	80	85	-	-	-	20	20	20	20	20	20	-	-	-	-	-	50
Waste meters	-	150	150	150	150	150	100	80	150	10	15	86	95	95	89	30	30	7	50	109	100	120	111	66	32	32
Total lateral meters	0	150	150	150	150	150	100	160	150	90	100	86	95	95	109	50	50	27	70	129	100	120	111	66	32	82
Raising meters	0	0	0	0	0	0	0	0	0	0	40	0	0	0	0	19	0	0	0	0	0	0	53	0	0	0
Waste tpd	-	246	246	248	248	246	164	131	246	16	48	141	156	156	146	61	49	11	82	179	164	197	214	108	53	53
Production																										
Longhole tonnage (kt)	-	-	-	-	-	-	-	-	-	5	17	14	12	12	11	18	16	18	18	19	20	20	21	10	10	10
C&Fill tonnage (kt)	-	-	-	-	-	-	-	-	-	9	8	-	-	-	-	-	-	-	-	-	-	-	-	8	6	3
Total Ore tonnage (kt)	-	-	-	-	-	-	-	-	-	14	25	14	12	12	11	18	16	18	18	19	20	20	21	16	16	13

Development	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52
Ore meters	50	40	50	50	41	-	25	-	-	-	-	-	-	-	-	-	35	-	-	-	-	-	-	-	-	-
Waste meters	32	93	92	87	60	32	-	-	-	44	40	-	24	30	30	67	47	50	25	37	37	45	55	50	10	-
Total lateral meters	82	133	142	137	101	32	25	0	0	44	40	0	24	30	30	67	82	50	25	37	37	45	55	50	10	0
Raising meters	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	15	0	0	0	0	0	0
Waste tpd	53	153	151	143	99	53	-	-	-	72	66	-	39	49	49	110	77	82	41	70	61	74	90	82	16	-
Production																										
Longhole tonnage (kt)	10	10	-	-	3	10	14	16	16	11	9	9	9	10	8	8	8	7	7	8	7	-	6	5	3	-
C&Fill tonnage (kt)	3	2	10	11	9	5	-	-	-	-	-	-	-	-	-	-	-	-	-	2	5	9	3	6	2	-
Total Ore tonnage (kt)	13	12	10	11	12	15	14	16	16	11	9	9	9	10	8	8	6	7	7	10	12	9	9	11	5	-

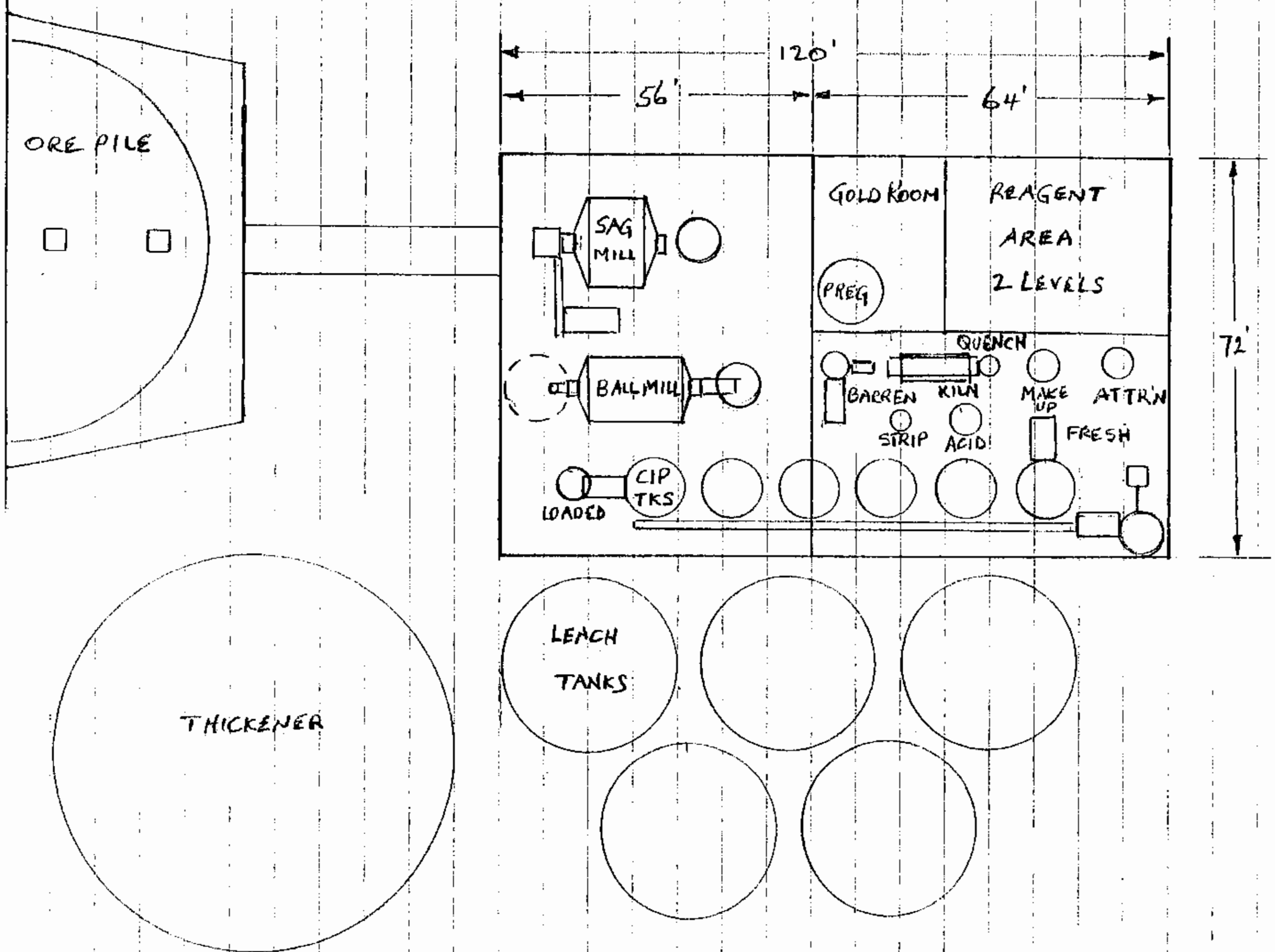
RED MOUNTAIN TOTALS

month

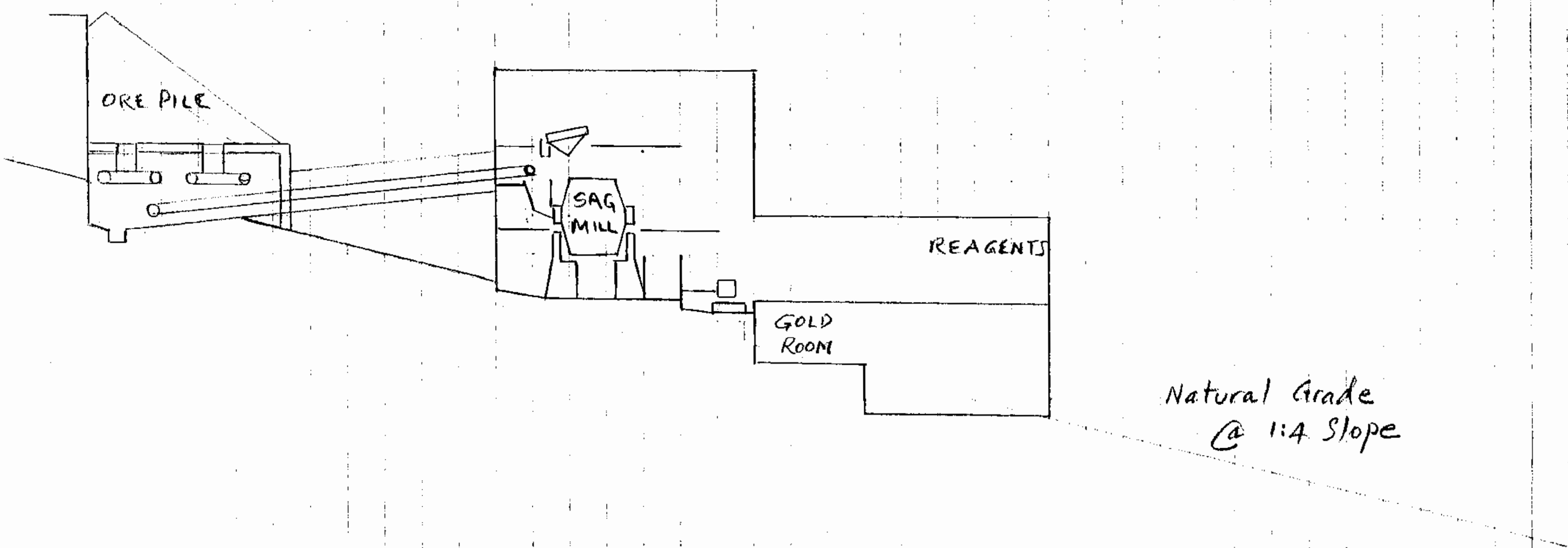
Development	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Ore meters	-	-	-	-	-	-	-	80	50	130	85	-	35	123	100	20	20	20	20	20	-	-	-	30	30	80
Waste meters	-	350	350	350	350	350	350	343	350	235	265	248	217	135	137	30	89	7	100	159	150	170	161	96	62	112
Total lateral meters	-	350	350	350	350	350	350	423	400	365	350	248	252	258	237	50	109	27	120	179	150	170	161	126	92	192
Raising meters	-	-	-	-	-	-	110	114	17	-	40	-	-	-	-	19	-	-	-	-	-	80	53	-	-	-
Waste tpd	-	575	575	575	575	575	640	631	585	386	459	407	356	222	225	61	146	11	184	261	246	327	298	158	102	184
Production																										
Development ore (kt)	-	-	-	-	-	-	-	4.8	3.0	7.8	5.1	-	2.1	7.4	6.0	1.2	1.2	1.2	1.2	1.2	-	-	-	1.8	1.8	4.8
Percent	-	-	-	-	-	-	-	100	100	35	17	-	7	25	20	4	4	4	4	4	-	-	-	6	6	16
Longhole tonnage (kt)	-	-	-	-	-	-	-	-	-	5.0	17.0	26.0	23.9	19.6	21.2	28.8	28.8	28.8	28.8	28.8	30.0	30.0	30.0	22.2	22.2	22.2
Percent	-	-	-	-	-	-	-	-	-	23	57	87	80	65	71	96	96	96	96	96	100	100	100	74	74	74
C&Fill tonnage (kt)	-	-	-	-	-	-	-	-	-	9.4	7.9	4.0	4.0	3.0	2.8	-	-	-	-	-	-	-	-	6.0	6.0	3.0
Percent	-	-	-	-	-	-	-	-	-	42	26	13	13	10	9	-	-	-	-	-	-	-	-	20	20	10
Total Ore tonnage (kt)	-	-	-	-	-	-	-	4.8	3.0	22.2	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0
Ore tpd	-	-	-	-	-	-	-	160	100	739	999	996	1000	999	999	1000	1000	1000	1000	1000	1000	1001	998	1000	1000	1000
Total material tpd	-	575	575	575	575	575	640	791	685	1125	1458	1406	1356	1221	1224	1061	1146	1011	1164	1261	1246	1327	1294	1158	1102	1184

Development	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52	TOTAL	
Ore meters	80	70	80	80	71	14	25	-	-	-	-	-	-	-	-	-	35	-	-	-	-	-	-	-	-	-	1,298	Ore meters
Waste meters	112	173	172	185	199	180	130	111	80	139	135	95	119	85	75	118	98	101	78	83	73	81	89	50	10	-	7,915	Waste meters
Total lateral meters	192	243	252	265	270	194	155	111	80	139	135	95	119	65	75	118	133	101	76	83	73	61	89	50	10	-	9,213	Total lateral meters
Raising meters	-	-	-	-	30	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15	-	-	-	-	-	-	478	Raising meters (ventilation)
Waste tpd	184	284	283	304	345	298	214	182	131	228	222	156	195	107	123	194	161	166	125	145	120	133	146	82	16	-		
Production																												
Development ore (kt)	4.8	4.2	4.8	4.8	4.3	0.8	1.5	-	-	-	-	-	-	-	-	-	2.1	-	-	-	-	-	-	-	-	-	77,880	Ore Dev't Tonnage
Percent	16	14	16	16	14	3	5	-	-	-	-	-	-	-	-	-	7	-	-	-	-	-	-	-	-	-	6%	of total production
Longhole tonnage (kt)	22.1	24.0	14.9	14.0	16.6	24.1	28.5	30.0	30.0	25.0	23.0	23.0	23.0	17.6	8.0	8.0	6.0	7.0	7.0	8.0	7.4	-	6.0	9.2	24.4	-	820,070	Longhole tonnage
Percent	74	80	50	47	55	80	95	100	100	83	77	77	77	59	27	27	20	23	23	27	25	-	20	30	78	-	65%	of total production
C&Fill tonnage (kt)	3.1	1.8	10.3	11.2	9.1	5.1	-	-	-	5.0	7.0	7.0	7.0	12.4	22.0	22.0	21.9	23.0	23.0	22.0	22.6	30.0	24.0	21.8	6.7	-	383,940	C&Fill tonnage
Percent	10	6	34	37	30	17	-	-	-	17	23	23	23	41	73	73	73	77	77	73	75	100	80	70	22	-	29%	of total production
Total Ore tonnage (kt)	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	31.0	31.2	-	1,261.9	Total Ore kt
Ore tpd	1000	1000	1000	1001	999	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1001	999	998	1034	1038	0		Ore tpd
Total material tpd	1184	1285	1283	1305	1343	1295	1213	1182	1131	1228	1222	1156	1195	1107	1123	1194	1161	1166	1125	1145	1121	1132	1145	1116	1055	0		Total material tpd

Appendix I
Preliminary Mill Sketches



RED MOUNTAIN - SKETCH 1
PRELIMINARY SKETCH
GENERAL ARRANGEMENT
PLAN VIEW
SCALE: 1" = 32'
By: J. Starkey
DATE: March 26, 2003



RED MOUNTAIN – SKETCH 2
PRELIMINART GENERAL ARRAGEMENT
SECTION THROUGH ℄ OF SAG MILL
SCALE: 1" = 32'
BY: J. Starkey
DATE: March 26, 2003

Appendix J
Yearly Mine Manpower Schedule

RED MOUNTAIN PROJECT

Mine Manpower

MINE MANPOWER ESTIMATE		1	2	3	4	5	6	7	8	9	10	11	Total
		Season											
Mine Supervision													
Superintendent			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Shiftboss			1.0	1.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	
Safety/Training				1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Mine/Maint Clerk				1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Development													
Diamond drill	18 m/ms		-	0.1	0.8	0.8	0.8	0.8	1.0	1.2	0.9	0.2	
Miners	1.3 m/ms		11.5	17.3	11.2	4.1	5.9	9.7	5.8	4.0	4.0	2.1	
Raise miner	1.0 m/ms		-	2.4	0.4	0.2	1.3	0.2	0.1	-	0.2	-	
Longhole Mining													
3" LH Drilling	75m/ms		-	0.1	1.6	2.3	2.1	1.5	2.1	1.4	0.6	0.6	
LH Blasting	400t/ms		-	0.1	2.8	3.8	3.5	2.5	3.6	2.4	1.0	1.1	
6 Yard Scoop			-	0.3	2.2	2.4	2.7	2.5	3.0	2.8	3.4	2.8	
Cut and Fill Mining													
Miners	100t/ms	-	-	0.7	2.4	-	1.4	3.5	1.2	6.2	11.9	9.5	
4 Yard Scoop			-	0.1	0.9	0.7	0.8	2.2	1.4	3.2	5.8	4.7	
Truck Haulage													
Trucks 20T			1.4	2.6	4.9	5.5	5.5	5.2	5.4	5.8	5.9	5.0	
Backfilling													
Operator	400t/ms		-	-	0.6	1.2	1.2	0.8	1.0	1.0	0.4	0.2	
FE Loader			-	-	0.1	0.8	0.7	0.4	0.7	1.2	1.3	1.5	
Dozer			0.5	0.8	0.3	-							
Construction													
Miners			2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	
Mine Services													
Utility drilling	50m/ms			0.1	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	
Bitman					0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	
Nipping				1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Blockholer					0.5	0.5	0.5	0.5	0.5	0.4	0.2	0.2	
Rock breaker					2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	
Grader				1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Rehab Miner					1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Pumpman					1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Maintenance Supervision													
Mech/Elec GF				1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Maint Supervisors					2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	
Mine Maintenance													
Dryman/Lampman			1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	
Mechanics			3.0	3.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	
Electricians			2.0	2.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	
Technical Services													
Chief Engineer			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Mining Engineer				1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	
Mine Technician					1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Chief Surveyor			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Surveyor					1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Geologist				1.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	
	Season	1	2	3	4	5	6	7	8	9	10	11	
TOTAL MANPOWER		-	25	44	70	63	67	71	67	70	76	69	

Mine Staff and Hourly Pay Rates

	10 hr shifts worked per season	O/T shifts worked per season	Annual Salary $	Base Wage $/hr	Burden on Base %	Bonus per 10hr shift	Cost per Man per Season	All in Cost per Hour
Mine Supervision								
Superintendent			90,000		30%		$ 117,000	$ 56.25
Shiftboss	110	15.7		32	30%		$ 49,029	$ 44.57
Safety/Training	110	15.7		32	30%		$ 49,029	$ 44.57
Mine/Maint Clerk	110	15.7		15	30%		$ 22,982	$ 20.89
Development								
Diamond drill	110	15.7		22	30%	50	$ 39,207	$ 35.64
Miners	110	15.7		24	30%	130	$ 51,071	$ 46.43
Raise miner	110	15.7		24	30%	120	$ 49,971	$ 45.43
Longhole Mining								
3" LH Drilling	110	15.7		24	30%	70	$ 44,471	$ 40.43
LH Blasting	110	15.7		24	30%	70	$ 44,471	$ 40.43
6 Yard Scoop	110	15.7		24	30%	70	$ 44,471	$ 40.43
Cut and Fill Mining								
Miners	110	15.7		24	30%	110	$ 48,871	$ 44.43
4 Yard Scoop	110	15.7		24	30%	70	$ 44,471	$ 40.43
Truck Haulage								
Trucks 20T	110	15.7		22	30%	50	$ 39,207	$ 35.64
Backfilling								
Operator	110	15.7		22	30%	50	$ 39,207	$ 35.64
FE Loader	110	15.7		23	30%		$ 35,239	$ 32.04
Dozer	110	15.7		23	30%		$ 35,239	$ 32.04
Construction								
Miners	110	15.7		22	30%	50	$ 39,207	$ 35.64
Mine Services								
Utility drilling	110	15.7		22	30%	50	$ 39,207	$ 35.64
Bitman	110	15.7		22	30%		$ 33,707	$ 30.84
Nipping	110	15.7		22	30%	50	$ 39,207	$ 35.64
Blockholer	110	15.7		22	30%	70	$ 41,407	$ 37.64
Rock breaker	110	15.7		22	30%	70	$ 41,407	$ 37.64
Grader	110	15.7		22	30%	50	$ 39,207	$ 35.64
Rehab Miner	110	15.7		24	30%	70	$ 44,471	$ 40.43
Pumpman	110	15.7		22	30%	50	$ 39,207	$ 35.64
Maintenance Supervision								
Mech/Elec GF			80,000		30%		$ 104,000	$ 50.00
Maint Supervisors	110	15.7		32	30%		$ 49,029	$ 44.57
Mine Maintenance								
Dryman/Lampman	110	15.7		20	30%		$ 30,643	$ 27.86
Mechanics	113	15.7		25	30%	30	$ 42,669	$ 37.76
Electricians	113	15.7		25	30%	30	$ 42,669	$ 37.76
Technical Services								
Chief Engineer			80,000		30%		$ 104,000	$ 50.00
Mining Engineer	110	15.7		26	30%		$ 39,836	$ 36.21
Mine Technician	110	15.7		21	30%		$ 32,175	$ 29.25
Chief Surveyor	110	15.7		26	30%		$ 39,636	$ 36.21
Surveyor	110	15.7		21	30%		$ 32,175	$ 29.25
Geologist	110	15.7		25	30%		$ 38,304	$ 34.82

Labour Costs per Season - Thousands

	1	2	3	4	5	6	7	8	9	10	11	Total	
	Season												
Mine Supervision													
Superintendent	-	117	117	117	117	117	117	117	117	117	117	1,170	
Shiftboss	-	49	49	147	147	147	147	147	147	147	147	1,275	
Safety/Training	-	-	49	49	49	49	49	49	49	49	49	441	
Mine/Maint Clerk	-	-	23	23	23	23	23	23	23	23	23	207	
Development													
Diamond drill	-	-	4	31	31	31	31	40	47	37	9	262	
Miners	-	588	684	571	207	301	496	296	206	206	106	3,860	
Raise miner	-	-	122	20	10	67	11	4	-	8	-	241	
Longhole Mining												-	
3" LH Drilling	-	-	2	72	101	92	66	94	64	25	29	546	
LH Blasting	-	-	4	122	170	155	111	159	108	42	49	921	
6 Yard Scoop	-	-	14	96	106	119	110	135	124	152	127	981	
Cut and Fill Mining													
Miners	-	-	34	119	-	67	172	61	301	561		1,334	
4 Yard Scoop	-	-	3	41	33	35	97	64	144	258	207	881	
Truck Haulage													
Trucks 20T	-	54	101	190	215	214	204	212	226	231	197	1,844	
Backfilling												-	
Operator	-	-	-	25	48	48	33	40	39	15	9	258	
FE Loader	-	-	-	5	28	25	15	26	41	45	52	236	
Dozer	-	19	28	12	-	-	-	-	-	-	-	59	
Construction												-	
Miners	-	78	78	78	78	78	78	78	78	78	78	784	
Mine Services												-	
Utility drilling	-	-	4	12	12	12	12	12	12	12	12	98	
Bitman	-	-	-	17	17	17	17	17	17	17	17	135	
Nipping	-	-	39	39	39	39	39	39	39	39	39	353	
Blockholer	-	-	-	21	21	21	21	21	17	8	8	137	
Rock breaker	-	-	-	83	83	83	83	83	83	83	83	663	
Grader	-	-	39	39	39	39	39	39	39	39	39	353	
Rehab Miner	-	-	-	44	44	44	44	44	44	44	44	356	
Pumpman	-	-	-	39	39	39	39	39	39	39	39	314	
Maintenance Supervision												-	
Mech/Elec GF	-	-	104	104	104	104	104	104	104	104	104	938	
Maint Supervisors	-	-	-	98	98	98	98	98	98	98	98	784	
Mine Maintenance												-	
Dryman/Lampman	-	31	31	61	61	61	61	61	61	61	61	552	
Mechanics	-	128	128	384	384	384	384	384	384	384	384	3,328	
Electricians	-	85	85	128	126	128	128	128	128	128	128	1,195	
Technical Services												-	
Chief Engineer	-	104	104	104	104	104	104	104	104	104	104	1,040	
Mining Engineer	-	-	40	80	80	80	80	80	80	80	80	677	
Mine Technician	-	-	-	32	32	32	32	32	32	32	32	257	
Chief Surveyor	-	40	40	40	40	40	40	40	40	40	40	398	
Surveyor	-	-	-	32	32	32	32	32	32	32	32	257	
Geologist	-	-	38	115	115	115	115	115	115	115	115	958	
	1	2	3	4	5	6	7	8	9	10	11	Total	
Total Labour Cost per Season	-	1,294	2,166	3,191	2,835	3,039	3,231	3,019	3,182	3,474	2,659	28,090	Total/Season
												24,630	Operating
												3,459	Pre-Prod
Labour cost per Tonne				20.26	18.00	19.30	20.51	19.17	20.20	22.05	19.39	$ 19.87	Operating

Appendix K
List of Other Mine Equipment

CAPEX ITEMS	CAPEX TOTAL	PREPRODUCTION PERIOD 1	2	3	COMMENTS
Other Mine Equipment					
survey equipment	$8,000		$8,000		One total station
anfo loader	$10,000		$10,000		Four units @ $2,500 each
swedish loader	$6,000		$6,000		Two units @ $3,000 each
shotcrete machine	$35,000		$35,000		Based on quote from ShotcretePlus, Sudbury
stopers and jacklegs	$30,000		$15,000	$15,000	6 sets of each @ $2,500 per unit
air compressors, 2 @ 1032cfm	$235,000		$235,000		Atlas Copco pricing
water pumps/dewater pumps	$150,000		$75,000	$75,000	Three pumps @ $15,000 each plus piping plus sumps
1690 chute and installation	$120,000			$120,000	based on Nordic Technology quote
rock breaker, grizzly, chains, bin	$400,000			$400,000	rock breaker quote, includes installation
backfill plant	$250,000			$250,000	quote from Theissen Equipment
rammer jammer c/w quick attach fittings	$30,000			$30,000	
remotes for scoops, 3ea	$150,000			$150,000	
portal construction	$40,000		$40,000		Estimate only
main vent fans and installation	$257,000		$128,500	$128,500	Includes fans, motors, installation, vent doors and extra Alimak cost of vent raise
auxiliary vent fans, 6ea.	$50,000		$50,000		
main ore pass	$159,000			$159,000	159 m raise (extra to cover alimak)
surface maint shop	$456,000		$228,000	$228,000	25m x 37m = 925sq.m @ $342/sq.m = $316,000 plus $140,000 shop tools, crane et
initial mine electrical distribution system costs	$300,000		$150,000	$150,000	Estimate for transformers,power cables, switch gear and installation
refuge station	$40,000			$40,000	
mine rescue equipment	$60,000			$60,000	Estimate only
mine rescue trailer	$35,000			$35,000	One ATCO unit
site preparation costs	$150,000		$75,000	$75,000	Construct 1.0km road, stockpile prep: Ore & Waste, level yard
warehouse - shared with mill	$212,000		$106,000	$106,000	Two ATCO units, 24m x 36m = 864sq.m total
surface powder magazines	$31,000		$31,000		One powder mag, one cap mag, one accessory mag
surface fuel storage	$45,000		$45,000		Based on Enviro Tank
Total Other Mine Equipment	**$3,259,000**	**$0**	**$1,237,500**	**$2,021,500**	
Camp, Office, Dry					
satellite phone	$10,000		$10,000		
office computers and software	$42,000		$42,000		Four computers @ $3,000 each plus $30,000 for software
offices	$140,000		$70,000	$70,000	Four unit ATCO complex @ $35,000 per unit
dry	$220,000		$110,000	$110,000	Two ATCO wash car units, 15m x 36m = 540sq.m total
small sleeping quarters	$70,000		$70,000		Two ATCO 10-man sleeper units
small kitchen facilities	$56,000		$56,000		One ATCO kitchen unit
diesel electric standby generator 250kW	$65,000		$65,000		AB estimate - power study
Total Camp	**$603,000**	**$0**	**$423,000**	**$180,000**	

Appendix L
SRK Memorandum – Preliminary Assessment of Exploration Potential

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MEMORANDUM

Toronto, April 11, 2003

From: Jean-Francois Couture

To: Ken Reipas

Object: Preliminary Assessment of Exploration Potential of the Red Mountain project.

Introduction

As part of an engineering study commissioned by Seabridge Resources Inc ("Seabridge") on the Red Mountain project, available exploration data for the project was examined in order to comment on any opportunity to increase the project resource base through exploration. This preliminary assessment is based on cursory examination of sparse archived geological reports made available by Seabridge, past workers and public information available from the BG geological survey website.

The review involved investigations on the followings aspects of the project:

- Review of available paper documentations on the project;
- Gap analysis on available geological information;
- Historical exploration work conducted over the project area and immediate surroundings;
- Nature, scale and extend of past exploration work performed over the project area, including: geophysical, geological and geochemical surveys and diamond drilling;
- Review of geological/structural interpretations with emphasis on the nature, and characteristics of the gold mineralization, the geological/structural controls on its distribution and potential targeting tools for exploration;
- Local deposit-scale and regional exploration potential.

…/2

Summary

Exploration efforts have focussed primarily on the immediate area surrounding of the known resource but no current compilation of borehole data exists. It is therefore not possible to provide detailed comments on the potential of adding to the current resource base through additional drilling. However, the high-grade precious metals mineralization at Red Mountain is enclosed within broader lower-grade alteration envelopes that are quite extensive. Several isolated high-grade intercepts exist outside the resource volumes and may have been overlooked. There is an opportunity to re-evaluate such isolated drill intercepts with the objective to delineate small high-grade volumes within the reach of proposed underground development. Also, several previous authors have pointed out that a re-evaluation of the down-dip extension of the AV and JW zones (AV/JW Tails zone) and the 141 zone is warranted. In order to properly evaluate these opportunity a thorough re-evaluation of all drilling results should be completed.

On the property scale, reconnaissance exploration appears to have covered accessible areas. No obvious exploration targets have the capability to increase the resource base in the short term. The exploration strategy included airborne magnetometer and electromagnetic surveys, follow-up ground geophysics surveys, prospecting and sampling, geological mapping and geochemical sampling. It appears that only limited diamond drilling was conducted, primarily on geophysical targets. Results indicate that sulphide mineralization was generally encountered with barren to low gold grades. A third of the project area is covered by ice and thus has not been explored. Attempts to image the geology underneath ice fields with remote sensed data and determine locally the thickness of the ice cover have produced mixed results.

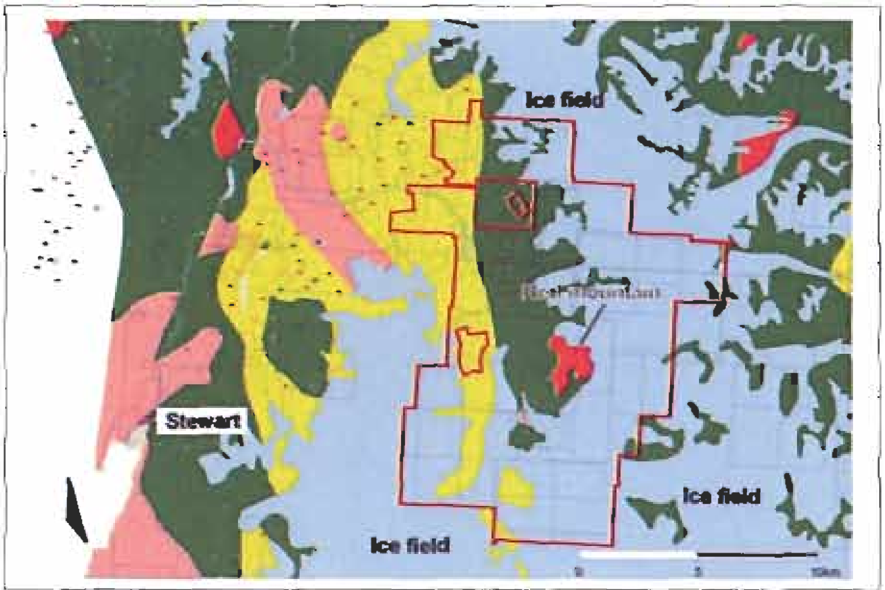
The exploration potential of the property remains excellent, but will require a fair commitment to test adequately. Exploration in the Red Mountain area will face serious logistical challenges owing to steep mountainous topography and widespread ice fields and glaciers. A thorough review of exploration work in the area is warranted to produce a current compilation of all exploration data and revise the regional structural interpretation. New exploration targets could be identified by integrating the current knowledge about the geological setting of the Red Mountain deposit.

Review of available data

The results of this review are further discussed below into three sections: (i) historical work and data gap analysis; (ii) geological/structural/geophysical characteristics of the Red Mountain deposit and insight for exploration targeting; (iii) suggested recommendations to consider before resuming exploration on the project.

i. Historical work and data gap analysis

The current Red Mountain Property (Lang, 2002, Technical Report) consists of 97 modified grid and 2 post claims covering a contiguous area of approximately 19,280 hectares. The project area is characterized by rugged mountainous terrain. Extremely steep to precipitous topography and abundant alpine glaciers create significant logistical challenges to exploration. In addition, a large percentage of the property (approximately one third of the project area) is covered by ice fields and alpine glaciers. In such areas, the geology is very poorly constrained.



Outline of the Red Mountain property on the regional geology of the Skeena Mining district and location of the Red Mountain deposit. Area coloured in cyan are ice covered (extracted from BC Geological Survey MapPlace website).

When Seabridge acquired the project from North American Metals Corporation ("NAMC"), not all project files were transferred. As a result, a number of historical reports describing previous exploration work and discussing results and recommendations were lost. This includes a number of assessment file reports and internal memorandum reports discussing exploration activities at Red Mountain and surrounding exploration properties. Assessment files reports can be recovered from government archives.

Although the history leading to the discovery and development of the Red Mountain deposit is well documented in several published public files, documentation of exploration work carried out on the large land package assembled around Red Mountain is far more elusive. A technical report on the Red Mountain project prepared in 2002 for Seabridge mostly deals with the deposit itself and contains little information pertaining to historical exploration work, other than generic description about work carried out by past project operators.

It is difficult to accurately describe all exploration work carried out in the Red Mountain area without an in-depth review of assessment records and compiling regional exploration data. Much of the documentation examined for this review relates primarily to the area surrounding the known resource.

The synoptic description of historical work was constructed from available assessment files, BC government web-based public files, from Rhys et al. 1995 and Craig, 2001. Incidentally, Dave Rhys who worked on the project for Lac Minerals until it was abruptly suspended by Barrick, apparently retains a number of paper and digital archives on this project. This data is available and should be retrieved.

The area surrounding the Red Mountain project has been subject to sporadic exploration in the 1960s and 1970s, primarily for porphyry-molybdenum deposits.

Bond Gold Canada, became involved in the Skeena Mining Division in late 1988 through an option from Wotan Resources to acquire 6 claim blocks (ORO I through VI and Hrothgar). The first high grade gold samples were collected at Red Mountain during the early part of the 1989 program on what was to become the Marc zone. The discovery was made by tracing the source of auriferous floats uphill to bedrock.

During the period 1989-1991, Bond Gold assembled a very large land package surrounding the Red Mountain discovery trough several distinct option agreements and claim staking. The land package examined by Bond Gold covers a much larger area then the current property limits and includes several historical Au-Ag-Cu-Mo showings. Public assessment file records indicate that during this period Bond Gold carried out reconnaissance exploration over much of this area, including: prospecting, reconnaissance geology, geochemical sampling, airborne and ground geophysical surveys and a limited amount of diamond drilling.

Between 1991 and 1994, following the acquisition of Bond Gold, Lac Minerals delineated a sizeable resource through diamond drilling and subsequently drove a decline to facilitate drilling access and collect a bulk sample for metallurgical studies. Mine development was initiated in 1993 through late 1994, when the project was put on hold by Barrick Gold Corp, following the acquisition of Lac Minerals. From 1989 through 1994, a total of 406 surface and underground boreholes were drilled on the property, including 368 drilled within the limited footprint area of the Marc, AV, JW, AV-JV Tails and 141 gold zones. The project was sold to Royal Oak Mines in 1995. The following year, Royal Oak expanded the underground development and conducted surface (22 holes) and underground (15 holes) drill programs targeting the extensions of known mineralization outside the resource volumes and other nearby targets (23 holes). Apparently, after 1993, all exploration work focussed on the immediate Red Mountain deposit area and there are no records of work carried out elsewhere on the property.

In 2000, upon acquisition of the project, NAMC completed a comprehensive review of the project and validation of the geological and environmental database. Several new technical studies were carried out leading to the creation of a revised resource model. New geological/structural studies enabled NAMC to revise the geological model for the deposit.

ii. Geological/structural and geophysical characteristics of the Red Mountain deposit

The Mineral Resources of the Red Mountain deposit occur in three main orebodies: Marc, AV and JW located within a large gossanous area hosting several other gold-silver showings (141, Brad, MCEX, Darb, Cornica, Dicksito). Over the years, numerous geological, structural, geophysical, metallurgical and petrography studies have addressed several aspects of this deposit in an attempt to constrain the controls on distribution of the gold-silver mineralization and elucidate the genesis of the deposit. A consensual model proposed by NAMC suggests that the three main gold-silver zones (Marc, AV and JW) were initially formed during a single geological event in the Early Jurassic (204Ma) but were later separated by extensional block faulting during the Miocene (20Ma).

The sulphide-rich gold-silver mineralization at Red Mountain is closely related to the emplacement of the sub-volcanic and polyphased felsic Goldslide intrusions intruding the Lower Jurassic volcano-sedimentary sequence. Most of known the gold-silver mineralization occurs in irregular sulphide-rich stockworks forming north-westerly trending crudely tabular zones located within the carapace of the Goldslide intrusion, in the Hillside porphyry sub-phase, volcano-sedimentary rafts and intrusive breccias.

The gold and silver-bearing sulphide stockwork depicts complex internal patterns resulting from a staged hydrothermal system transgressing stratigraphy and involving repeated episodes of veining and brecciation. Pyrite is the dominant sulphide with subordinate, locally important, pyrrhotite and sphalerite. The stockwork consists of pyrite microveins, coarse-grained pyrite veins, masses and breccia matrix developed in zones of strong muscovite alteration. Vein thickness ranges between <10cm to 80cm. Vein spacing varies between 2-10 veins per metre.

The sulphide stockwork is typically surrounded by a thicker weakly auriferous zone of disseminated pyrite, pyrrhotite and locally sphalerite alteration. This alteration envelope displays a concentric zoning pattern, with pyrrhotite disappearing sharply away from the gold-silver stockwork, typically across less than a metre. The relationship with sphalerite remains elusive.

This geological model implies a strong spatial and genetic relationship between the gold-silver stockwork mineralization and the Goldslide intrusions. As such, structural and geological controls on the emplacement (and regional distribution) of these Lower Jurassic intrusive bodies are very important to consider for targeting exploration within this district. Such controls have not been well constrained. In addition, the apparent lack of distinct structural and lithological controls on the distribution of high grade gold-silver stockworks has limited the ability to successfully target ongoing exploration outside the deposit area.

Geophysical work carried out on the project area

The gold-silver mineralization at Red Mountain is sulphide-rich and occurs as either stockwork veins or disseminations in sizeable altered rock bodies measuring several tens of meters in thickness and extending laterally over several hundred metres.

Early following the discovery of the Red Mountain deposit, orientation surveys were conducted to characterize the geophysical signature of this style of mineralization, determine whether it could be traced with geophysics and help identify follow-up regional geophysical targets on a regional Aerodat helicopter survey flown in late 1989 (5,220 line kilometres). The airborne survey data were reprocessed in 1993 and are available.

Follow-up geophysics between 1990 and 1993 included surface magnetometer, horizontal loop EM (Genie), UTEM, MELIS EM, gradient array IP surveys over selected portions of the property, and down-hole UTEM, MELIS EM and Wellmac physical property surveys. A review report (Rainsford, 1993) documents geophysical techniques deployed primarily over the Red Mountain area with insight to guide ongoing regional exploration. A number of geophysical anomalies were identified for follow-up consideration. It is not known if these recommendations were implemented in subsequent exploration programs.

The Marc Zone (and possibly some other mineralized zones: Meg, AV) is characterized by a distinct airborne magnetic low, which in drill core translate into a reduction in measured magnetic susceptibility. The reduction in magnetic susceptibility was interpreted to result from the destruction of magnetite by alteration but apparently may also emphasized zones of magnetic reversals suggesting that the alteration and mineralization processes may have locally reset remnant magnetism. This characteristic was used for targeting from the airborne survey.

Ground horizontal loop EM, UTEM, MELIS and IP surveys were successful in defining drilling targets, which resulted in the discovery of pyrrhotite stockwork mineralization locally auriferous (e.g. GY zone) and other base metal mineralization (e.g. UTEM zone). Ground surveys were also successful in locating EM conductor detected by the airborne EM survey. Limited borehole geophysical surveys were completed between 1990 and 1993. Mineralized drill intercepts generally produce weak to moderate “in hole” response, but apparently Marc Zone style mineralization may not be sufficiently conductive to generate strong off hole response. This could limit the usefulness of down-hole geophysics to trace lateral extensions of known mineralization or detect new mineralization around drill holes. However, several other down-hole geophysics techniques exist and could provide better opportunity to maximize the radius of influence of individual drill holes. In this respect additional testing is strongly warranted. It is not known whether additional down-hole geophysics surveying was carried out after 1993. The report list makes no mention of any geophysical work carried out after 1994. A new helicopter-borne EM-Mag-radiometric survey was flown by Geonex Aerodat in 1994. (is this a new survey or the reprocessing of the old 1989 survey?)

In general, geophysical techniques were successful in delineating widespread sulphide (especially disseminated pyrrhotite) mineralization but have failed to discriminate the smaller high grade gold-silver zones from the larger more extensive disseminated sulphide alteration.

iii. Recommendations

Geological model

The geology of the Red Mountain deposit is fairly well constrained and the geological model is robust. The gold mineralization is related to a magmatic hydrothermal developed during the emplacement of an Early Jurassic polyphase felsic sub-volcanic pluton. All known gold-silver mineralization occurs in widespread hydrothermal alteration zones developed primarily near the top of the intrusion and its Early Jurassic volcano-sedimentary carapace. Several studies have addressed local controls on the distribution of the gold mineralization, but attempts to derive predictive structural targeting tools have not been very successful. The high-grade gold and silver mineralization occurs within 100m of the top of the Goldslide porphyry and is spatially associated with inflections in the porphyry contact, where the alteration zone intersects breccia bodies and Early Jurassic sedimentary rafts. The upper contact zone of the Goldslide intrusion (or any other similar Early Jurassic intrusion in the area) thus represents the primary targeting criterion. The widespread hydrothermal alteration with disseminated pyrrhotite mineralization appears to be successfully imaged by a combination of airborne and ground geophysics, although geophysics cannot discriminate the gold and silver stockwork. The alteration and mineralization zones are dissected by younger Eocene normal faults, but their effect is only to displace earlier formed gold-silver mineralization.

Near deposit exploration

At the scale of the deposit, several recommendations were expressed in recent reports prepared after the acquisition of the property by NAMC. In particular, Craig (2001) pointed out that the AV Tails and 141 zones have been tested by scarce drilling and therefore deserve re-evaluation. Additional studies were also recommended to continue to address the structural controls on distribution of the stockwork mineralization and derive predictive targeting tools. Such studies should aim at integrating alteration geochemistry with local and broader-scale structural patterns in an attempt to determine controls on emplacement of the Goldslide intrusions.

It is recommended that surface and underground drilling information along the periphery of all known gold-bearing zones be carefully reviewed to ensure that all gold-bearing structures have been properly closed out by drilling. In addition, within the area covered by surface and underground drilling, isolated high-grade drill intercepts should be also carefully evaluated for the potential to delineate smaller mineralized zones. The current mine plan already schedules the development of fairly small mineralized pods (3,000 to 15,000 tonnes) which represent fairly small volumes. It is considered very likely that additional follow-up exploration drilling around isolated intercepts will result in the delineation of small high-grade mineralized shoots. The economic attractiveness of such targets will however decrease with increasing distance from planned underground development. Also, delineation of such small volume will require careful analysis of borehole data and tight drilling pattern.

An independent review of down-hole geophysical surveying is also warranted to examine if any geophysical targets have been missed and whether the surveying has adequately sterilized the targeted drill area. This review should also consider alternative down-hole surveying techniques

(EM, IP) that could offer a wider detection radius around boreholes or a better response to Red Mountain mineralization style.

Furthermore, as stated by Barclay (2000), it appears that several surface gold-silver occurrences in the vicinity of the resource area have not been fully evaluated and tested by diamond drilling. This emphasizes the necessity to re-evaluate all surface occurrences and borehole intercepts.

Property-scale exploration
At the scale of the property, it is strongly recommended that all exploration work and other regional geological and geochemical data be compiled into a GIS environment to help determine the extent of exploration outside the deposit area and analyse the nature of exploration targets tested. This compilation should also list all sulphide-gold occurrences encountered within the project area and provide an assessment of follow-up work, if warranted. Given the good understanding of the setting of the Red Mountain deposit, it is very likely that additional exploration targets can be identified.

Subsequently, available data should be integrated with remote sensed data (airborne geophysics) to derive a new regional structural interpretation for the area and help map geological and structural features possibly related to the emplacement of Early Jurassic intrusions. Since a very large portion of the property is overlain by ice fields and glaciers this regional structural interpretation may provide the only tool to evaluate those areas.

The rugged topography and extensive ice fields will present difficult logistical challenges, leading to slow progress and translating into more expensive exploration programs.

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Vogt, A., 1989. Assessment Report for the 1989 Geological/Geochemical Exploration and Diamond Drill Report, Willoughby Property, Skeena Mining Division. Assessment Report number 19474, British Columbia Ministry of Energy, Mines and Petroleum Resources, 30pages.

Vogt, A., 1990. Assessment Report for the 1990 Diamond Drill Program on the Red Mountain Property. Assessment Report number 20971, British Columbia Ministry of Energy, Mines and Petroleum Resources, 38 pages.

List of Assessment Reports filed with the B.C. Ministry of Energy, Mines and Petroleum Resources by Royal Oak, Barrick, Lac Minerals, and Bond Gold in the area surrounding the Red Mountain project.

Assessment reports filed by Royal Oak in the Skeena Mining District.

Report #	Claim Names	Property Name	Mining Divisions	NTS Maps	BCGS Map	MINFILE #	Latitude/ Longitude (NAD27)	General Work	Off Confidential	Mining Camp
25204	Entrance 2, Nelson 2-3	Nelson	Skeena	104A04E	104A003		56 03 00 129 31 00	Prospecting	1998-08-15	Stewart Camp
24947	Kim 14, Bon Fr., Bon Accord 2, Oro IV, Oro Fr., Bon Accord 6, Lisa 2	Red Mountain	Skeena	103P13E 103P13W 104A04E 104A04W	103P092	103P 086 103P 220 103P 221	55 58 00 129 42 00	Drilling, Geochemical, Physical	1998-03-26	Stewart Camp
24889	Kim 14	Vermillion	Skeena	103P13E	103P092		55 57 00 129 42 00	Drilling, Geochemical	1998-01-16	Stewart Camp

Assessment reports filed by Barrick in the Skeena Mining District.

Report #	Claim Names	Property Name	Mining Divisions	NTS Maps	BCGS Map	MINFILE #	Latitude/ Longitude (NAD27)	General Work	Off Confidential	Mining Camp
23996	Oro IV	Red Mountain	Skeena	103P13E	103P092	103P 220 103P 221	55 58 00 129 44 00	Drilling, Geochemical	1996-04-19	Stewart Camp
23927	Lisa 5-8	Lisa	Skeena	103P13E	103P092		55 54 48 129 42 30	Drilling, Geochemical	1996-04-19	Stewart Camp

Assessment reports filed by Lac Minerals in the Skeena Mining District

Report #	Claim Names	Property Name	Mining Divisions	NTS Maps	BCGS Map	MINFILE #	Latitude/ Longitude (NAD27)	General Work	Off Confidential	Mining Camp
23996	Oro IV	Red Mountain	Skeena	103P13E	103P092	103P 220 103P 221	55 58 00 129 44 00	Drilling, Geochemical	1996-04-19	Stewart Camp
23927	Lisa 5-8	Lisa	Skeena	103P13E	103P092		55 54 48 129 42 30	Drilling, Geochemical	1996-04-19	Stewart Camp
23854	Copper Queen (L. 4781), Castle Rock (L. 4784), Helena (L. 4783), Grand View (L. 4793), Enterprise (L. 5346), Heather (L. 5354), Red Top (L. 4803), Superior (L. 4801), Amazon (L. 4945)	Bear Pass	Skeena	104A04E 104A04W	104A012	104A 019 104A 020 104A 021 104A 022 104A 023 104A 024 104A 029	56 06 57 129 45 21	Geochemical, Geological	1996-04-05	Stewart Camp
23596	Nelson 1-3, Lisa 9-10	Nelson	Skeena	104A03W 104A04E			56 03 00 129 30 00	Geophysical	1995-08-15	Stewart Camp
23123	Lisa 9-10, Nelson 1-3	Nelson	Skeena	104A03W 104A04E		104A 099 104A 151 104A 152 104A 160 104A 167	56 03 00 129 30 00	Geochemical	1994-08-12	Stewart Camp
22598	Sarah 3-10, Oro 1-6, Dick 1-4, Hrothgar, Lisa 1-2, Lisa 4, Lisa 7-8, Vera 4-9, Willoughby	Red Mountain	Skeena	103P13E	103P092	103P 220 103P 221	55 57 00 129 42 00	Geological	1993-09-15	Stewart Camp
22570	Nelson 1-3, Lisa 9-10	Nelson	Skeena	104A03W	104A003		56 03 00 129 31 00	Geochemical, Geological	1993-08-17	Stewart Camp
22417	Oro I-IV, Hrothgar	Red Mountain	Skeena	103P13E	103P092	103P 086 103P 220 103P 221	55 57 00 129 42 00	Drilling, Geological, Geochemical	1993-07-10	Stewart Camp
20653	Mike 1-3, Goldfields 1-2, Sovereign, Georgia 2, Gem, Gem 1, Gem Fr., June 1-2, Sun, Jitney, Gold Fr., Top Fr., Danny Fr., Sovereign 1-2	Georgia River	Skeena	103O16E	103O090	103O 013	55 48 00 130 02 00	Drilling, Geochemical, Geological, Geophysical, Physical	1991-09-18	Stewart Camp
16148	Neekas	*	Skeena	103A08E	103A050	103A 003	52 28 36 128 10 00	Geochemical, Physical	1988-05-19	

Assessment reports filed by Bond Gold for the Skeena Mining District.

Report #	Claim Names	Property Name	Mining Divisions	NTS Maps	BCGS Map	MINFILE #	Latitude/ Longitude (NAD27)	General Work	Off Confidential	Mining Camp
22002	Shul 1-6, Dug 1-2	Shul	Skeena	104A04W	104A011		56 07 00 129 50 00	Geological, Prospecting	1992-09-30	Stewart Camp
21975	Sarah 1-2, Bon Accord 1-10, Montreal 1-8, Kim 1-14, Pam 1-2	Kai	Skeena	103P13W 104A04W	104A002	104A 059 104A 060	56 00 00 129 46 00	Geochemical, Geological	1992-09-26	Stewart Camp
21974	Alex 1-2	Alex	Skeena	104A04E	104A013		56 07 30 129 32 00	Prospecting	1992-09-30	Stewart Camp
21973	Vera 8	Vera	Skeena	104A04E 104A04W			56 02 30 129 30 00	Prospecting	1992-09-24	
21972	Janine 7-8	Janine	Skeena	103P13W	103P091		55 56 00 129 52 00	Prospecting	1992-09-24	
21966	Gold Spot, Gold Valley 6-7	Lehto	Skeena	104A04E	104A002		56 01 00 129 40 00	Prospecting	1992-09-18	Stewart Camp
21943	Janine 3	Janine	Skeena	103P13W	103P091		55 55 00 129 50 30	Prospecting	1992-09-09	
21942	Sarah 3-10	Sarah	Skeena	104A04W	104A002		56 02 45 129 46 00	Geochemical, Geological	1992-09-09	Stewart Camp
21941	Dick 1, Dick 4	Dick	Skeena	103P13W 104A04W	104A001		56 00 00 129 48 50	Prospecting	1992-09-09	Stewart Camp
21929	Irene 1	Irene	Skeena	103P13W	103P082		55 52 30 129 46 00	Geological	1992-09-09	
21813	Nelson 1-3, Lisa 9-10	Nelson	Skeena	104A04E	104A003		56 03 00 129 31 00	Prospecting	1992-08-14	Stewart Camp
21304	Bria 1-4, Field 1-3, Flat 1-4, Kit 1-4, Willoughby 24	Bria Wotan	Skeena	103P14W	103P083		55 51 00 129 26 00	Geochemical, Geological, Geophysical	1992-02-08	
21260	Hunter 1, Ore 4	Hunter	Skeena	104A04W	104A011		56 07 00 129 50 00	Drilling, Geochemical, Geological, Geophysical	1992-02-08	Stewart Camp
20971	Oro 1-IV, Hrothgar	Red Mountain	Skeena	103P13E	103P092	103P 220	55 57 00 129 42 00	Drilling, Geochemical	1991-09-17	Stewart Camp

Assessment reports filed by Bond Gold for the Skeena Mining District.

20200	Oro 1-6, Hrothgar, Bria 1-4, Camb, Cindy 1-10, Del, Dick 1-4, Dixie 1-4, Field 1-3, Flat 1-4, Gold Mountain 1-3, Janine 1-12, Pam 1-6, Sarah 1-10, Willoughby 1-50	Cambria	Skeena	103P13E 104A04E	104A002	103P 007 103P 078 103P 086 103P 220 103P 221 104A 049 104A 059 104A 060 104A 064 104A 069	56 00 00 129 37 00	Geophysical	1991-02-12	Stewart Camp
20133	Oro 1-6, Hrothgar	Red Mountain	Skeena	103P13W	103P092	103P 220 103P 221	55 57 00 129 42 00	Drilling, Geochemical, Geological, Physical	1991-07-09	Stewart Camp
19982	Bria 1-4, Field 1-3, Flat 1-4, Kit 1-4	Bria	Skeena	103P14W	103P083		55 53 00 129 27 00	Geophysical, Prospecting	1991-02-09	
19474	Del, Gold Mountain 3, Willoughby 1-4	Willoughby	Skeena	103P13E	103P093		55 58 00 129 35 00	Drilling, Geochemical, Geological	1990-09-12	Stewart Camp
19424	Nelson 1-3	Nelson	Skeena	104A03W 104A04E	104A003		56 03 00 129 31 00	Geochemical, Geological	1990-09-19	Stewart Camp