PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

Technical Report for the Tulsequah Chief Project of Northern British Columbia, Canada

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Contents

Tables

Figures

NOTICE

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

Chieftain Metals Inc. is authorized to file this report as a Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

1 Executive Summary

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Chieftain Metals Inc. (Chieftain) to carry out a feasibility study of the Tulsequah Chief project. The project encompasses two advanced stage polymetallic massive sulphide deposits known as the "Tulsequah Chief" and "Big Bull" deposits. The feasibility study focuses on the Tulsequah Chief deposit only. The project is located in northern BC, Canada on the banks of the Tulsequah River, approximately 97 km south of the town of Atlin.

This technical report summarizes the results of the feasibility study and is prepared according to the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

The Tulsequah Chief project design is based on underground mining and the construction of on-site processing facilities and infrastructure to support an ore throughput rate of 2,000 t/d.

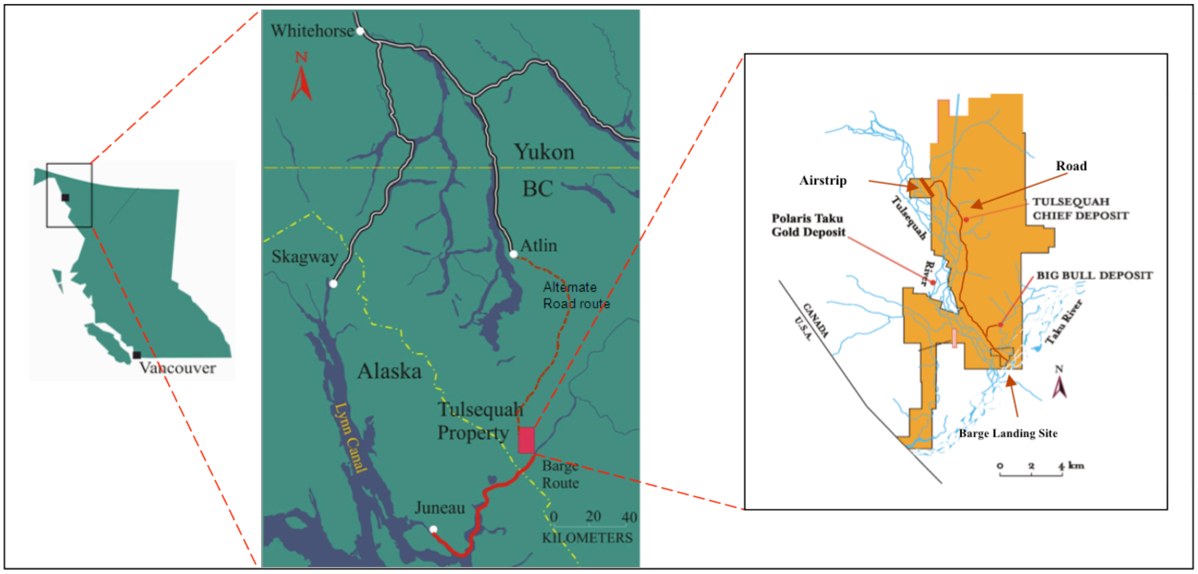
JDS managed the feasibility study and completed the mining, infrastructure and economics sections of the report. JDS was assisted by several principal-designated subcontractors providing report information as noted below:

- SRK Consulting (Canada) Inc. (SRK) Vancouver, Dr. Gilles Arseneau: property description, geology and mineral resource estimate.
- Ken Sangster and Associates Ltd. (Sangster): mineral processing, metallurgical testing and recovery methods.
- Marsland Environmental Associates (MEA): environmental and permitting.
- Klohn Crippen Berger Ltd. (KCB): geotechnical design for tailings management facility and potentially acid generating and pyrite facilities.
- Dave West Consulting (Dave West): underground geotechnical requirements analysis.
- Kovit Engineering Limited (Kovit): paste backfill testing assessment.

1.2 Property Description & Ownership

Chieftain's Tulsequah property is located at 58°43'N and 133°35'W in northwestern BC, as shown on Figure 1-1. The property is located 97 km south of the town of Atlin, BC (59°35'N, 133°40'W), which is the nearest Canadian community. Juneau (58°18'N, 134°24' W), the capital of Alaska, is situated 64 km southwest of the property. The property is accessible by air from both Atlin and Juneau, and by water during high-water periods from Juneau. The exploration base camp is situated on the east bank of the Tulsequah River at an elevation of 55 metres above sea level (masl).

Figure 1-1: Project Location Map



Chieftain's property comprises 14 cell mineral claims and 25 overlapping crown granted mineral claims totalling 140.9 km^2. The Tulsequah Chief project (on which all the current economics are based) is located in Chieftain's property and composed only by the claims shown in Table 1-1.

Table 1-1: Chieftain's Mineral Claims

Property Area	Type	Tenure No.	Record No.	Area (ha.)	Expiry Date
Tulsequah Chief	Mineral Claim	590422		420	31-Dec-22
	Crown Grants		5669	7.99	03-Jul-13
	Crown Grants		5668	20.90	03-Jul-13
	Crown Grants		5676	14.16	03-Jul-13
	Crown Grants		5670	20.90	03-Jul-13
	Crown Grants		5679	9.70	03-Jul-13

All mineral claims are in good standing until 2022, except claim 1011222, which expires on July 16, 2013. Crown grants are maintained through annual tax payments due on July 2 of each year, and are in good standing through July 3, 2013. Chieftain holds a 100% interest in both the mineral claims and the crown grants. There are no back-in rights or royalties on any of Chieftain's mineral claims or crown grants.

1.3 Geology & Mineralization

The Tulsequah Chief deposit is dominantly underlain by rocks of the Devono-Mississippian to Permian-aged Mount Eaton group, which is a low metamorphic grade, island arc volcanic assemblage contained within the Stikine Terrane of northwest BC. These rocks are situated east of the Chief (Llewelyn) fault, and are predominately located north of the Taku River, and east of the Tulsequah River.

The mineral deposit consists of numerous stacked sulphide lenses developed within the basal stratigraphy or a rhyolite-rich sequence of volcanic flows and fragmental units. These felsic volcanics rest above a thick assemblage of mafic volcanics (primarily basalt, and basaltic andesite). Above the assemblage of rhyolitic volcanics, a mafic dominated sequence of basalt flows, breccias and sills, overlays the unit. Within the mine area, a thick diorite/gabbro sill, which is geochemically identical to the upper mafic volcanic units, intrudes the rhyolite above the sulphide deposits. Basaltic dykes recognized to be feeders to the thick sill, cut through the sequence. Late stage Sloko dykes of Tertiary age are associated with faults cutting all of the mine sequence rocks.

1.4 Exploration status

In 2011, Chieftain carried out a detailed drilling program focused at upgrading some of the inferred mineral resource to the indicated category. In total, 10 surface holes and 50 underground diamond drill holes totalling 22,630 m were completed and included in the resource calculation.

1.5 Mineral Processing & Metallurgical Testing

The metallurgical test program for the Tulsequah ore was initiated with the objective of determining a treatment methodology that would result in saleable concentrates at economic recoveries.

Production records from 1953—as well as work conducted by Beattie in 1993 through 1995, IMEL and G&T in 1996, and others—showed that the ore could be effectively processed, although attempts to replicate some of the work from the early 1990s did not achieve the high levels of concentrate grades and recoveries reported in that work. The lack of consistency between the various historical records contributed to Chieftain's decision to conduct its own sampling and test program. It should be noted that, at that time, arsenic was assumed to be well understood. The testwork demonstrated that the arsenic levels were consistent with previous work; however, its mineralogical distribution is now better understood.

ALS AMMTEC laboratory in Burnie, Tasmania, Australia was selected to carry out the work based on their proven expertise, their close association with the operations at Rosebery Tasmania (which is an analogous deposit to Tulsequah Chief), as well as their ability to carry out test programs relatively quickly when compared to other laboratories.

Extensive mineralogy was conducted as a precursor to establish the likely optimum for liberation and therefore grinding protocols. The key findings were that there is a critical relationship between grind size and effective liberation.

It is now clear that much of the previous work suffered from insufficient mineral liberation requiring the recovery of composite particles and over-grinding in subsequent phases creating excessive surface area that compromised flotation selectivity.

It also appears that previous testwork was hampered by a lack of appreciation of the tendency of this ore to oxidize sufficiently to effect metallurgical performance. Correct sample storage is extremely important in this regard to avoid sample degradation over the relatively protracted period of a testwork program.

Successful metallurgical testwork depends on a robust understanding of the mineral assemblages in the ore. In that context, several mineralogical analyses have been conducted on the Tulsequah ore samples and flotation products.

The Tulsequah ore body is typical of volcanogenic massive sulphide (VMS) deposits found worldwide. It contains recoverable copper, lead, zinc, gold and silver within a host that is predominantly pyrite/barite. Lead is present as galena; zinc is present as a very low iron sphalerite; and copper occurs as three main minerals:

- chalcopyrite (Cp) - $Cu\ Fe\ S_2$ (dominant)
- tennantite (Tn) – $(Cu\ Fe\ Zn\ Ag)_{12}\ As_4\ S_{13}$
- tetrahedrite (Th) – $(Cu\ Fe\ Zn\ Ag)_{12}\ Sb_4\ S_{13}$.

Tennantite and tetrahedrite are similar in terms of mineralogical appearance and flotation response. The relative proportions of all three species vary throughout the deposit. Additionally, the mineralogical work established consistency of assemblage characteristics from sample to sample, area to area, and at varying sulphide contents.

Due to the financial impact of gold revenue on project economics, there has been extensive investigation into its recovery to payable products. During previous operations, 25% to 30% of the gold was recovered by jigging prior to flotation. Some basic testwork carried out in the 1990s confirmed that around 30% was recoverable, and an opinion was expressed that, "[i]n continuous operation, the gravity concentrate should contain 60 to 80% by weight combined gold and silver."

As part of the current testwork, a full gravity recovery gold test showed that 52.7% was recoverable and this was modelled by Knelson-Consep to indicate that recovery of a little over 40% can be expected in a plant treating grinding circuit cyclone underflows.

Mineralogical work established that the gold is predominantly present as electrum with the silver content varying widely, estimated to average around 30%. The gravity concentrate is expected to contain 42% of the gold, but even with 30% silver content in the doré, only 0.5% of the silver in the feed will be contained in this product. The vast majority of the contained silver will be present in galena and tennantite and will return as leach tail to the grinding circuit. The projected metallurgical balance and concentrate qualities are shown on Tables 1-2 and 1-3.

It is noteworthy that having established the controlling mineralogical parameters, the results from actual testwork are still well within the theoretical limitations of the mineral assemblage. This gives impetus in seeking process improvements and to some extent, this is reflected in ongoing and planned testwork. In particular, there seems to be scope to improve the discrimination between chalcopyrite and tennantite/tetrahedrite and release some of the entrained sphalerite to enhance overall zinc recovery.

Table 1-2: Projected Metallurgical Balance

Product	Wt (t)	Assays					Recoveries %				
		Cu %	Pb %	Zn %	Ag g/t	Au g/t	Cu	Pb	Zn	Ag	Au
Copper Conc.	5.3	21.0	2.7	7.6	1217.5	20.8	89.0	11.8	6.0	75.0	44.0
Lead Conc.	1.3	1.0	60.0	8.4	586.4	7.6	0.8	66.2	1.6	9.0	4.0
Zinc Conc.	9.6	0.7	0.8	62.0	69.7	0.8	5.5	6.8	89.0	7.8	3.0
Pyrite Conc.	28	0.1	0.3	0.4	19.2	0.5	1.9	6.8	1.7	6.1	5.0
Tailings	55.7	0.1	0.2	0.2	2.4	0.1	2.8	8.4	1.7	1.6	2.0
Feed	100	1.3	1.2	6.7	86	2.5	100	100	100	100	100
Doré kg/100 t feed	0.155				30%	70%				0.5	42.0

Table 1-3: Concentrate Analyses

Element	Unit	Concentrate		
		Copper	Lead	Zinc
Zn	%	8.5	8.5	59.9
Cu	%	21.7	0.29	0.53
Pb	%	3	62.8	0.21
Al_2O_3	%	1.23	0.42	1.15
CaO	%	0.17	0.07	0.08
Fe_2O_3	%	31.7	8.6	3.1
K_2O	%	0.25	0.11	0.29
MgO	%	0.85	0.2	0.25
SiO_2	%	3.8	1.5	2.9
S	%	28.6	16	32.8
Na_2O	%	0.08	0.03	0.07
Ti_2O	%	0.05	0.02	0.05
As	%	1.45	0.08	0.03
MnO	ppm	129	80	297
F	ppm	194	186	163
Cl	ppm	232	134	270
Bi	ppm	7	57	0.5
Hg	ppm	36	19	171
Sb	ppm	5,647	905	102
Cd	ppm	387	354	2,434
Au	ppm	22 est *	8.3	0.9
Ag	ppm	1,339	423	59

Element	Unit	Concentrate		
		Copper	Lead	Zinc
Sr	ppm	50	38	31
U	ppm	1.4	0.8	1.6
Mo	ppm	90	19	21
Ni	ppm	3.9	6	2.4
Ba	ppm	119	147	115
Sn	ppm	5.2	0.4	0.8
Cr	ppm	25.3	11.6	28

* Gold content of the copper concentrate was assayed at 43.4 g/t, but the product was made from total without gravity removal. The recoveries to gravity and copper concentrate are approximately equal, hence the estimate of 22 ppm.

1.6 Mineral Resource Estimate

The Mineral Resource Statement presented in Table 1-4 represents the second mineral resource evaluation prepared for the Tulsequah Chief project in accordance with the Canadian Securities Administrators' NI 43-101.

The mineral resource model prepared by SRK considers 665 core boreholes drilled by Cominco, Redfern and Chieftain during the period of 1940 to 2011. The resource estimation work was completed by Dr. Gilles Arseneau, P.Geo (APEGBC # 23474) an appropriate "independent qualified person" as this term is defined in NI43-101. The effective date of the resource statement is March 15, 2012.

Table 1-4: Mineral Resource Statement* of Tulsequah Chief Deposit, BC (SRK, March 15, 2012)

Location	Category	Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Old Mine (above 5200)	Indicated	403,000	1.28	0.97	6.02	1.52	71
	Inferred						
New Mine (below 5200)	Indicated	6,113,000	1.19	1.13	6.00	2.50	88
	Inferred	204,000	0.67	0.76	4.02	1.81	62
A Extension	Indicated	247,000	0.86	0.59	2.91	1.34	44
	Inferred						
Total Indicated		**6,762,000**	**1.19**	**1.1**	**5.89**	**2.4**	**85**
Total Inferred		**204,000**	**0.67**	**0.76**	**4.02**	**1.81**	**62**

* Mineral resources are reported in relation to a conceptual mining outline. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.
** Underground mineral resources are reported at a cut-off grade of US$100. Cut-off grades are based on a price of US$1,275/oz of gold, US$21/oz for silver, US$1.10/lb for zinc and lead and US$3.25 for copper and recoveries of 81.8% for gold, 79.5 for silver, 87.8 for copper, 44.5% for lead and 88% for zinc.

Mineralized lenses were modelled by Chieftain, and audited and validated by SRK using GEMS™ (Gemcom). SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the mineralized areas and that the assaying data are sufficiently reliable to support estimating Mineral Resources. GEMS Version 6.3 was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades and tabulate mineral resources.

For the purpose of resource estimation, all assay intervals within the mineralized units were composited to 2 m and grades were capped prior to estimation. SRK decided to cap zinc at 30%, lead and copper at 10%, gold at 25 g/t and silver at 600 g/t for the resource estimate.

Mineral resources were estimated in multiple passes using inverse distance weighted to the second power interpolation method because variography did not yield sufficiently robust variograms. The first estimation pass required that at least two drill holes and three composites be available within the search ellipse to estimate a grade within a block. Where several composites were found within the search ellipse, a maximum of eight composites were used to interpolate a grade value. The second pass required that at least two composites be present within the search ellipse for grade interpolation with no restrictions on the number of drill holes. The maximum number of composites was set to 12.

Bulk density values were estimated into the resource model by inverse distance weighting to the second power. Search parameters used were the same as those used for grade interpolation.

Block model quantities and grade estimates for the Tulsequah Chief project were classified according to the Canadian Institute of Mining's (CIM) Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Dr. Gilles Arseneau, P.Geo.

SRK is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced at 20 to 30 m.

1.7 Mineral Reserve Estimate

The mineral reserves identified in Table 1-5 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

The economic viability of the project is presented in Sections 21 and 22, and confirms that the probable reserve estimates meet and comply with CIM definitions and NI 43-101 standards, including the main assumptions used in the definition of the reserves (i.e., metal prices, dilution, operating costs and recoveries).

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the mineral reserves or potential production.

Table 1-5: Mineral Reserve Estimate

Category	Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Probable	6,447,098	1.13	1.04	5.59	2.30	81.39

Underground mineral reserves are reported at a cut-off of US$125. Cut-off grades are based on a price of US$1,350/oz of gold, US$22/oz for silver, US$1.10/lb for zinc and lead and US$3.10 for copper and recoveries of 81.8% for gold, 79.5 for silver, 87.8 for copper, 44.5% for lead and 88% for zinc.

1.8 Mining

The Tulsequah deposit will be accessed via the 5400 (120 m) and 5200 (60 m) level portals. An additional portal will be driven at approximately 84 m level that will act as the exit conveyor drift from the mine. The existing 5200 and 5400 levels will be slashed to 5.0 m x 5.3 m to accommodate the trackless equipment fleet. The main mine access will be via the 5200 level and connect to the main ramp that will access the mining levels. The main ramp is 5.0 m x 5.3 m in section and inclined to 17%. The main ramp will access sublevels 30 m apart vertically.

The deposit generally dips at greater than 60 degrees and is variable in thickness from less than 3 m to over 25 m. Several mining blocks are planned to be opened simultaneously throughout the vertical extent of the deposit to give mining flexibility needed for sequencing and early access to higher grade material. The deposit is favourable to a mix of mining methods with the majority coming from longhole stoping and uppers retreat with minor contributions from cut and fill.

The deposit will support a sustainable 2,000 t/d production rate by the second year of full production. The mine development and production plan is shown in Table 1-6.

Underground mine infrastructure will include a 2,000 tonne run-of-mine (ROM) surge bin that will feed a primary crusher. Crushed ore will exit the crusher and be conveyed along the 5400 level to a fine ore bin, from where it will exit the mine via the 84 masl level portal. The paste backfill plant will also be located underground to minimize pumping requirements and optimize cement content.

Backfill is an integral part of the underground mine plan and will incorporate process plant tailings as well as mine development waste. The primary purposes of the backfill are:

- underground support and working platform in mining
- storage of potentially acid generating (PAG) waste rock and process plant sulphide tailings.

Waste rock will be scheduled so that as much PAG material will remain underground as possible. As the stoping reaches a steady state underground, development rock will preferentially be used as backfill. The backfill plan calls for all waste rock generated after production Year 2 to be stored underground.

Table 1-6: Mine Development & Production Plan

Parameter	Unit	2014 Q4	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	Totals
Total Mine Production	**t**	**-**	**50,401**	**556,697**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**6,447,098**
Daily Production Rate	t/d	-	548	1,521	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	**1,950**
Gold Grade	g/t	-	1.11	1.58	1.8	1.79	2.89	3.26	2.52	2.1	2.29	2.38	**2.3**
Silver Grade	g/t	-	44.68	65.61	70.46	80.71	101.27	105.9	85.73	74.91	69.58	77.09	**81.39**
Copper Grade	%	-	0.75	0.95	0.78	1.05	1.45	1.61	1.02	1.03	1.08	1.13	**1.13**
Lead Grade	%	-	1.00	0.96	0.92	0.98	1.13	1.14	1.22	1.00	1.01	0.98	**1.04**
Zinc grade	%	-	4.89	5.4	5.19	5.38	6.42	6.63	5.43	5.53	5.44	4.93	**5.59**
Net Smelter Return	$/t		200.89	248.88	248.03	269.95	365.11	394.42	305.24	281.52	288.15	289.02	**299.5**
Total Lateral Development	**m**	**902**	**4,339**	**4,380**	**4,266**	**4,285**	**1,100**	**112**	**-**	**644**	**293**	**-**	**20,321**
	m/d	**9.8**	**11.9**	**12**	**11.7**	**11.7**	**3**	**0.3**	**-**	**1.8**	**0.8**	**-**	**7.1**
Raise Development	m	-	324	532	427	364	190	-	-	-	-	-	**1,837**
Mined Underground Waste	t	63,263	285,955	268,489	261,152	234,477	68,640	5,564	-	28,248	13,716	-	**1,229,504**
Paste Backfill Placed	t	-	-	136,468	246,068	212,719	403,801	462,806	494,122	472,411	475,242	492,464	**3,396,101**

An insufficient volume of waste rock is available for the backfill requirement; hence, the use of pastefill has been incorporated into the mine plan. Pastefill consists of process tailings partially dewatered and mixed with cement. This material is of a consistency that can be directed to specific locations by positive displacement pumps and pipeline. The fill plant will be operated such that all tailings required for backfill will be converted to thickened slurry on surface, pumped to the underground paste plant for final dewatering using filters. Cement binder is added to produce cemented pastefill, and pumped to mined-out voids for use as fill. Tailings not required for backfill will be directed to a permanent surface tailings management facility (TMF).

1.9 Recovery Methods

The plant will accept primary crushed ore from an underground storage silo. This will be fed to a semi-autogenous grinding mill followed by two stages of ball milling to a cyclone overflow product of P_{80} 45 µm.

The cyclone underflows of the primary and secondary ball mills will be equipped with Knelson concentrators to recover gravity gold (electrum). The gravity gold concentrates will be intensively leached, electrowinned and smelted to ultimately produce a doré on site. Cyclone overflow will be sent to a sequential flotation circuit starting with copper then lead, zinc and finally pyrite.

There is no regrinding planned in the flotation circuit as the mineralogical assessment showed that below 40-50 µm particle sizing further liberation would not occur until below 5-10 µm.

The sequential flotation system is very selective at P_{80} 45 µm and although multi-stage cleaning has been designed into the plant this is conservative.

A flotation scale-up factor of 3.0 has been used for the plant design. This was felt necessary to reflect the need for effective control of pulling rates commensurate with the high selectivity.

The copper, lead and zinc concentrates will be thickened and pressure filtered in discrete circuits before storage and transport. The filtered lead concentrate will be handled in containers.

A pyrite circuit is included to remove pyrite from the final tailings.

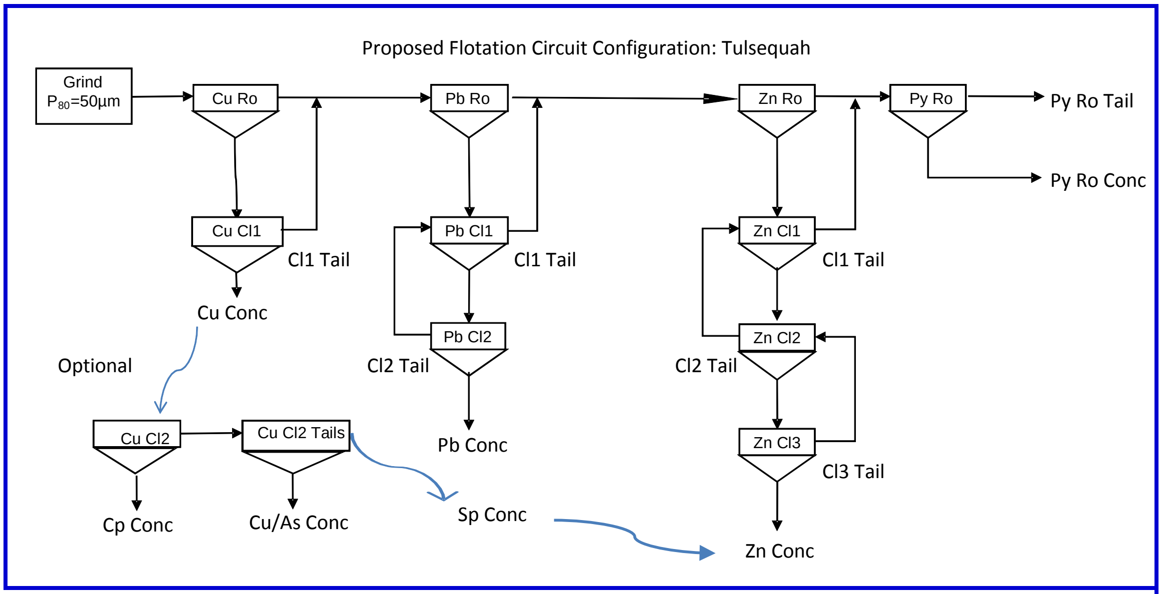
Tailings and pyrite will be separately dewatered in thickeners and pumped to the underground paste backfill plant as needed. When the paste backfill plant does not require feed material, the tailings will be pumped to the TMF and the pyrite will be pumped to the pyrite storage pond.

A limestone grinding circuit is included in the plant to accept crushed limestone and produce limestone slurry that will be added to the tailings stream to raise the pH and control the acid production potential of the tailings.

The plant will operate an on-stream analyzer integrated with a distributed digital control system.

A simplified flow diagram of the plant can be found in Figure 1-2.

Figure 1-2: Simplified Flow Diagram for the Flotation Circuit



1.10 Project Infrastructure

The main infrastructure required to support mining and processing activities are listed below:

- mine access road
- power generation facility and heat recovery
- fuel storage facilities (liquefied natural gas (LNG) and diesel)
- tailings management facility
- historical potentially acid generating (HPAG), operational potentially acid generating (OPAG), non-acid generating (NAG) and pyrite waste rock storage facilities
- limestone quarry
- effluent treatment facility
- airstrip
- accommodation complex (camp, dining, and recreational facilities)
- camp support facilities:
  - potable water system
  - sewage treatment facility
  - incinerator facility
- mine administration office and dry
- mine maintenance shop (truck shop)
- cold storage warehouse.

1.10.1 Access Road

The site access road is typical of a remote mine access road constructed through steep mountain terrain. The 128 km route will provide access from the Warm Bay road 15 km south of Atlin, BC to the mine site. The road is designed with a 5 m width with four pullouts per km to allow vehicles to pass each other. The site access road will be used for transport of construction materials and equipment to site; as well as concentrate transport, fuel and consumables supply during operations. The road will not be used for transporting personnel to and from the mine site. Access to the road will be controlled by Chieftain south of the town of Atlin for safety and environmental reasons.

1.10.2 Power Supply & Heat Recovery

Site power will be supplied by LNG/diesel fuel generators. The power plant will consist of six installed generator sets, each with a peak rating of 2,250 kW. Anticipated site-wide power load is approximately 7.5 MW, requiring four generators in operation with two as backup (n+2 configuration). Generators will normally be fueled by an LNG/diesel fuel blend (70% LNG/30% diesel) unless weather events inhibit LNG delivery for a duration exceeding LNG supply on hand (approximately seven days) for more than seven days, in which case the generators will be operated on 100% diesel. The fully enclosed generator units will be installed next to the process plant (the primary power draw).

Generator sets include high-grade heat recovery packages that will provide heating for the process plant and mine workings in the winter months. The accommodation, administration, and shop facilities will be electrically heated.

1.10.3 Fuel Storage Facility

Fuel storage on site includes five 130 m^3 LNG storage vessels and two 325,000 L capacity diesel storage tanks, each with offloading and distribution systems. LNG storage is sufficient to operate the generator units (on blended fuel) for seven days without refueling. The diesel tanks provide sufficient storage to operate the mine (generators and mobile equipment) for approximately 40 days without refueling when LNG is available, or 15 days in the event of LNG supply disruptions when the generators will be operating on 100% diesel.

1.10.4 Tailings Management Facility

The TMF is located approximately 4 km upstream (north) of the main mine facilities on the east side of Shazah Creek. The TMF will store 3 Mt of non- acid generating (NAG) tailings over the operating life of the mine. The 45 ha impoundment will be formed with a homogeneous compacted earthfill dam with a 1.0 mm linear low density polyethylene (LLDPE) geomembrane liner. The impoundment dams will be constructed prior to operations using material excavated from within the impoundment area and can be expanded later if more ore is found and processed.

The perimeter embankment will have a 6 m wide crest at El. 80.0 m (up to 14.5 m high) and will be 2.2 km long. The upstream and downstream slope angles are both 2.5H:1V and a stabilization berm will be constructed at the toe (the berm width varies based on stability requirements for the design earthquake). The dam is designed to the Canadian Dam Association Guidelines (2007) for a “High” consequence structure. Water from the TMF will be recycled back to the process plant. Storage is provided for an environmental flood and an emergency spillway is provided for dam safety. A mine access road will be routed along the toe berm on the east side of the impoundment. Riprap armouring will be placed along the toe of the stabilization berm to protect against erosion from possible flooding of Shazah Creek or Chasm Creek. On closure, the TMF will be drained, capped with a soil cover, and revegetated.

1.10.5 HPAG, OPAG & Pyrite Tailings Storage

Temporary storage of 140 kt of HPAG, 120 kt of OPAG and 75 kt of pyrite tailings are required during the project life to stage the backfilling of these materials into the underground workings. The facilities are located approximately 1 km south of the existing portals. The facilities will be formed with a cut-and-fill operation within the natural alluvial soils and lined with a 1.0 mm LLDPE liner placed over compacted natural soils. A protective soil layer will be placed over the HPAG and OPAG basins. Flows from the piles and pyrite pond will be attenuated in a storage pond within the OPAG basin and pumped to the water treatment plant. The HPAG will be progressively decommissioned during operations and upon closure, the OPAG and pyrite ponds will be drained and decommissioned. The disturbed areas will be recontoured, covered with topsoil and revegetated.

1.10.6 Limestone Quarry

An on-site limestone quarry will provide limestone for the processing plant to raise the neutralization potential (NP) of the tailings material and prevent acid generation by the tailings. The production of limestone from this quarry will be conducted by a contractor on a campaign basis. Crushed and stockpiled limestone will be hauled to the processing plant as required by mine personnel.

1.10.7 Effluent Treatment Plant

The water treatment system is comprised of two separate treatment facilities, the acid treatment plant (ATP) and the effluent treatment plant (ETP). The ATP will treat the acidic discharge from the underground workings and the drainage from the PAG waste storage facility. The ETP will treat all other mine site effluents, including the subsequent treatment of the ATP effluent, prior to it being discharged to the receiving environment through a diffuser buried below the scour depth in the Tulsequah River floodplain.

1.10.8 Airstrip

An existing airstrip located approximately 2.5 km north of the planned plant and camp facilities will be upgraded (extended) in order to accept DHC-5 (Buffalo) aircraft. The airstrip will be utilized to transport personnel and select supplies through mine construction and operations.

1.10.9 Accommodations Complex (Camp)

The camp facility will consist of a modular 216 single bed room dormitory with a kitchen and dining facility for up to 140 people at one sitting, recreation area, workshop, laundry facilities, potable water treatment system, an ambulance, fire truck, and emergency medical/first aid area, and a fire water tank and distribution system.

1.10.10 Camp Support Facilities

To support the administration complex, a potable water system will be installed which will draw fresh water from the Tulsequah River directly west of the camp and process plant facilities, or from a new water well to be determined during detailed engineering. A sewage treatment plant will be installed at the lower level of the plant site near the 5200 portal and a waste incinerator will be installed south of the plant site near the waste rock/pyrite storage facilities.

1.10.11 Mine Administration & Dry Facility

A two story modular mine administration and dry facility will be installed near the 5200 portal. The lower floor will house the mine dry as well as a small first aid and muster station to accommodate approximately 100 people. The upper floor will contain approximately 10 offices, as well as meeting and common space.

1.10.12 Shops & Warehousing

Two 400 m^2 insulated mine maintenance shops and one 400 m^2 cold storage warehouse will be installed near the 5200 portal.

1.11 Environmental Studies

1.11.1 Environmental Issues

The Tulsequah Chief project is located at a historical brownfields site with visible acidic mine drainage (AMD). Potential historic environmental liabilities include the PAG waste rock piles located on surface outside the entrances to the 5200, 5400, 5900, 6400, and 6500 level portals, as well as the AMD from the underground workings. The AMD at the Tulsequah Chief site had been subject to an Environment Canada Directive. In response, Chieftain installed and commissioned an acid water treatment plant (ATP) in late 2011. Treated effluent is discharged under a Waste Discharge Authorization issued by the BC Ministry of Environment under the *Environmental Management Act* (EMA). The operation of the treatment plant was suspended on June 23, 2012 and the plant remains on care and maintenance, in contravention of the *Fisheries Act* and the EMA permit.

A water balance model was developed to represent the proposed site-wide water management system and was run for a realistic range of operational and environmental conditions to assess the performance of the proposed system and to develop a set of procedures to be followed during operations. Overall, the water management plan represents a robust system able to meet the dynamic conditions that may be experienced during operations.

The project is expected to result in a total disturbance at end of mine life of approximately 162 ha. The existing area of disturbance at the site is approximately 106 ha. Remaining on surface at mine closure will be a TMF containing NAG tailings, a NAG waste rock storage facility and a demolition debris landfill associated with the waste rock dump.

The proposed access road to the Tulsequah Chief site from Atlin will be deactivated once mine reclamation is complete. Motor vehicle access will be restricted by establishing full fill pullback and recontouring at five identified locations. These sites were selected where fill pullback would make it very difficult for off-road vehicles to bypass thereby controlling access to parks and other sensitive areas along the road right of way.

The Tulsequah Chief project was issued a provincial Environmental Assessment Certificate M02-1 and a Canadian Environmental Assessment screening approval. The BC Environmental Assessment office considers that the project has been substantially started. Several permits related to the construction of the Tulsequah Chief project were issued to the previous owner, and have since been transferred to Chieftain.

1.11.2 Permitting

Chieftain has secured all necessary permits to commence construction at the mine site, and on October 19, 2012, received Amendment #5 to Environmental Assessment Certificate 02-01, approving the rerouting of the mine access road. All road construction permit application documentation has been submitted to the BC government for review, and the Company anticipates a decision in early course.

1.11.3 Social

The Company has undertaken an extensive community consultation program and provided numerous opportunities for stakeholders to gather information and comment on the project. A Consultation Report was prepared as part of the Environmental Assessment Amendment process and the consultation program has been deemed acceptable and approved by the provincial government.

1.11.4 First Nations

The Tulsequah Chief mine lies within the traditional lands of the Taku River Tlingit First Nation (TRTFN) and falls under the jurisdiction of the Atlin Taku Land Use Plan. The Atlin Taku Land Use Plan has been ratified by the BC government and the TRTFN has partnered with the Province in a Shared Decision-making process. The TRTFN and Chieftain signed a Memorandum of Understanding in May 2011 and have engaged in negotiations to complete an Impacts, Mitigations and Mutual Benefit Agreement (IMMBA) with the TRTFN. Chieftain will continue to seek opportunities to meet with the TRTFN to finalize the IMMBA.

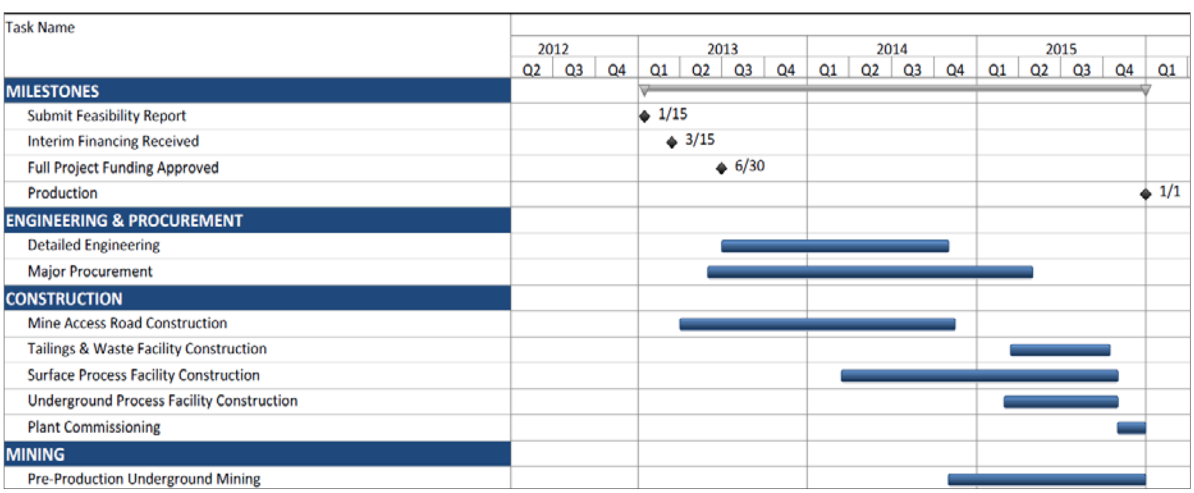
1.12 Project Execution

The development schedule is expected to take approximately 36 months following the publication of the feasibility report, assuming interim financing is established before May 1, 2013 and full project funding is received by July 2013. The project schedule is extremely sensitive to the timing of project funding due to the short annual barging season in May/June.

Equipment and construction camp marshalling to Prince Rupert will occur immediately following the receipt of interim financing with engineering and long-lead procurement activities following shortly thereafter.

Mine access road construction will begin in April on the Atlin side and in July from the site side. The access road will be constructed over two summer seasons and will be passable for construction loads by November of 2014. During winter shutdown, the site side road crew will perform the process plant/camp site earthworks.

Figure 1-3: Summary Project Schedule



Plant facility construction will begin in spring of 2014, beginning with concrete installation and subsequent building erection. The process building will be erected and cladded by December 2014. Mechanical/electrical installation will follow with mechanical completion scheduled for November 2015.

The main accommodation facility and other supporting infrastructure will be transported to site once the mine access road is completed. The main camp will be installed and operational by January 2015.

Mine personnel ramp up will occur in Q3 2014, beginning with technical services personnel. Mining equipment and operations personnel will be mobilized once the mine access road is complete.

Mine commissioning will occur in Q4 2015 with production scheduled for year end.

1.13 Capital Cost

The initial capital cost estimate is $439.5 M, as summarized in Table 1-7. Costs are expressed in Canadian dollars with no escalation (Q4 2012 dollars). The target estimate accuracy is -10%/+15%.

Table 1-7: Capital Cost Summary

Area	Initial (C$M)	Sustaining (C$M)	Total (C$M)
Site Development	7.4		7.4
Underground Mining	38.5	64.0	102.5
Underground Process Facilities	11.7		11.7
Limestone Quarry	0.2		0.2
Processing Plant	63.1		63.1
Tailings and Waste Rock Management	15.5		15.5
On-Site Infrastructure	61.5		61.5
Mine Access Road (directs)	54.2		54.2
Project Indirects	91.6		91.6
Engineering and EPCM	31.4		31.4
Owner's Costs	17.3		17.3
Pre-contingency Total	**392.4**	**64.0**	**456.4**
Contingency (12%)	47.1		47.1
Total	**439.5**	**64.0**	**503.5**

* All cost data are presented in Q4 2012 dollars.

1.13.1 Reclamation/Closure & Salvage Costs

Reclamation, closure and salvage costs are listed in Table 1-7.

Table 1-7: Reclamation/Closure & Salvage Costs

Cost	C$M
Reclamation/Closure	13.8
Salvage	-7.6

* All cost data are presented in Q4 2012 dollars.

1.13.2 Basis of Capital Estimates

The capital cost estimates were prepared using first principles, applying direct project experience and avoiding the use of general industry factors. The estimate is based on feasibility level engineering, quantity estimates, supplier/contractor quotations for equipment and materials, as well as estimated labour rates and productivity factors from the area.

The initial capital estimate includes all preproduction underground mining activities (Y-1 and Y-2) and is based on self-performed mining (Owner forces) with contract Alimak raise mining. No leasing contracts or used equipment supply for underground equipment have been considered in this estimate.

The initial capital estimate is based on the execution plans described in this study. Some infrastructure and facilities are already available on site and therefore not added as additional capital. Sunk costs and Owner's reserve were not considered in the initial capital estimate.

The sustaining capital estimate is based on required capital waste development, mining equipment acquisition, and mining infrastructure installations as defined by the mine plan.

The closure/reclamation estimate is based on preliminary scope determined through a design report issued by Gartner Lee.

1.14 Operating Cost

The unit operating cost is estimated at a total of $125.96/t processed and is summarized in Table 1-8.

Table 1-8: Operating Costs Summary

Area	Unit Operating Cost (C$/t processed)
Mining	30.06
Processing	23.02
Power Generation	22.47
G&A	22.58
Concentrate Transport	27.83
Total	**125.96**

* All cost data are presented in Q4 2012 dollars.

The following list summarizes key project assumptions used to develop the operating cost estimate:

- Tulsequah's mine site will be accessible by a new all-season, 128 km long, class 5 logging access road.
- Mining operations will be performed by Owner forces utilizing Owner purchased equipment.
- All electrical power will be generated on site using diesel/LNG generators with a long-term delivered price for LNG and diesel of $19.50/GJ and $1.32/L diesel, respectively.
- The process plant will process 2,000 t/d (730,000 t/a) of ore and produce approximately 98,000 wet tonnes per year of concentrate.
- Concentrate production will be trucked to a transfer point in Atlin, BC and then trucked using highway trucks to Skagway, Alaska. An existing terminal facility at the port of Skagway will be improved to handle the concentrate production, and all the mine supplies.
- Tailings will be disposed of in a lined, conventional tailings dam.
- Site personnel will reside in a 216 bed camp located within walking distance of the mine portal and processing facilities.
- The mine will utilize a peak workforce of approximately 267 people (including all contract labour).

1.15 Economic Analysis

All operating scenarios were modeled to estimate the value that each could potentially realize. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to be more indicative of true investment value. Sensitivity analyses were performed for variation in metal prices, grades, recoveries, operating costs, and capital costs to determine their relative importance as project value drivers. The economic effects on development scenarios of changes in metal prices were assessed. The economic analysis presented does not include financial securities posted by Chieftain for the Tulsequah project for the purposes of permitting.

This technical report contains forward-looking information regarding projected mine production rates and forecast of resulting cash flows as part of this study. The grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

1.15.1 Metal Price Cases

One metal price case was modeled and used in the economic analysis. This case is described in relation to two other cases in Table 1-9.

Table 1-9: Metal Price Cases

Case	Description
Base Case	Metal prices and exchange rate at three-year trailing average as at October 31, 2012
Case 2	Metal prices and exchange rate at two-year average as at October 31, 2012
Case 3	Long-term metal prices published by Consensus Economic and long-term exchange rate based on five major Canadian banks

Metal prices chosen for the Case 1 (Base Case) were the three-year trailing averages as at October 2012. This choice is accepted by the BC Securities Commission and standard with the SEC of the United States. A three-year average foreign exchange rate of C$1.01:US$1.00 as at October 31, 2012 was used.

The reserve amounts used in the economic analysis are outlined in Section 1.7 of the Executive Summary and Section 15 of the report.

1.15.2 Zinc, Copper, Lead, Gold, Silver Production

Recovered metals for all three scenarios calculated by year are shown in Table 1-10.

Table 1-10: Recovered Zinc, Copper, Lead, Gold, Silver

LOM Payable Metal (Pre-Streaming)	Unit	Value
Gold	Au koz	404
Silver	Ag koz	11,972
Zinc	Zn Mlbs	601.5
Copper	Cu Mlbs	135.5
Lead	Pb Mlbs	93.0
Average Annual Concentrate Production	dmt	98,889
Zinc Concentrate	dmt	517,686
Copper Concentrate	dmt	307,324
Lead Concentrate	dmt	73,989
Total Concentrate Production	dmt	898,999

1.15.3 Financial Parameters

The financial parameters used in the economic analysis are shown in Table 1-11.

Table 1-11: Financial Parameters used in Economic Analysis

Parameter	Units	Base Case	Case 2	Case 3
Gold Price	US$/oz	1,455.00	1,592.00	1,338.00
Silver Price	US$/oz	28.00	33.00	22.00
Zinc Price	US$/lb	0.97	0.96	1.09
Copper Price	US$/lb	3.66	3.85	2.95
Lead Price	US$/lb	1.01	1.03	1.02
Average F/X Rate	C$:US$	1.01	1.00	1.06
Zinc Concentrate Treatment Charge	$/dmt concentrate	145.00	145.00	145.00
Copper Concentrate Treatment Charge	$/dmt concentrate	125.00	125.00	125.00
Lead Concentrate Treatment Charge	$/dmt concentrate	100.00	100.00	100.00
Silver Refining Charge in Copper Concentrate	US$/payable oz	1.50	1.50	1.50
Silver Refining Charge in Lead Concentrate	US$/payable oz	1.50	1.50	1.50
Gold Refining Charge in Copper Concentrate	US$/payable oz	25.00	25.00	25.00
Gold Refining Charge in Lead Concentrate	US$/payable oz	25.00	25.00	25.00
Gold Refining Charge in doré	US$/payable oz	6.00	6.00	6.00
Silver Refining Charge in doré	US$/payable oz	1.50	1.50	1.50
Transport to Port	US$/wmt	80.00	80.00	80.00
Port Charges – Zinc and Copper Concentrate	US$/wmt	14.02	14.02	14.02
Port Charges – Lead Concentrate	US$/wmt	17.77	17.77	17.77
Ocean Freight	US$/wmt	65.00	65.00	65.00
Building Capital at Port	US$/wmt	23.65	23.65	23.65

A discount rate of 8% was used to value the project on a pre-tax and after-tax basis. Discount rates ranging from 0% to 8% were applied to examine the project financial performance under more rigorous economic conditions.

1.15.4 Streaming Contract

Chieftain has entered into a gold and silver purchase transaction with Royal Gold, Inc. (Royal Gold) to sell a portion of the precious metals stream expected to be produced at the Tulsequah Chief mine. As part of this transaction, Chieftain received an up-front payment of US$10 M at closing and will receive an additional US$50 M for the project build (upon certain conditions being met) that will be pro-rated during the development of the project. This advance payment will allow Royal Gold, to purchase production from the Tulsequah Chief mine as follows:

- 12.5% of payable gold at US$450/oz (up to the first 48 k oz); 7.5% of payable gold at US$500/oz thereafter
- 22.5% of payable silver at US$5.00/oz (up to the first 2.8 Moz); 9.75% at US$7.50/oz thereafter.

This contract has been taken into account in the economic evaluation of the project.

1.15.5 Taxes

The project has been evaluated on an after-tax basis to reflect a more indicative value of the project. Both BC Provincial and Federal tax rates were applied to the project.

The BC Mineral tax is comprised of two tiers:

- The Tier 1 Tax is 2% of net current proceeds defined as (the current year's gross revenue less operating costs). Operating costs are all current operating costs, but do not include expenses due to capital investment such as preproduction exploration and development expenses. If the mine has an operating loss, no net current proceeds tax (Tier 1 Tax) is payable.
- After the company's investment and a reasonable return on investment have been recovered, the company must pay the Tier 2 Tax of 13% of adjusted net revenue, essentially the net current proceeds from Tier 1 Tax computations from the mine. The Tier 1 Tax is deducted from the Tier 2 Tax owed, so the maximum tax does not exceed 13%. Any previous Tier 1 Tax paid is deductible from the Tier 2 Tax owed. It can be carried forward indefinitely.

Federal Corporate Tax:

- Federal Corporate income tax rate of 15% and a blended BC and Ontario Provincial Income Tax rate were used to calculate income tax amounting to 25%.

The tax calculations performed produce indicative results of the value of the project on an after-tax basis. The following assumptions were made in calculating the taxes payable for the project:

- Mineral Property Tax Pools – Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with appropriate opening balances to calculate income taxes.
- Federal Investment Tax Credits – Appropriate opening balances were used to calculate the Federal Investment Tax Credits for the project with respect to the preproduction capital costs of the project.
- Capital Cost Allowance (CCA) – Capital cost specific CCA rates were applied to and used to calculate the appropriate amount of CCA the Company can claim during the life of the project.
- Streaming Revenues – Streaming revenues were adjusted according to income tax regulations to appropriately determine the taxable income for the project.
- Corporate Expenses – To provide a more accurate indicative value of the project on an after-tax basis, taxable income was adjusted to include corporate overhead expenses incurred over the life of mine (LOM). These costs were calculated by Chieftain and amounted to $168.1 M. Since Chieftain is primarily a one-asset company, this assumption was deemed reasonable for the purposes of calculating the taxable income.

The tax analysis completed amount to a LOM taxes payable of $119.5 M. The after-tax values are determined solely for project valuation purposes.

1.15.6 Financial Performance

Pre-tax financial performance is summarized in Table 1-12 to provide a point of comparison with similar projects and is not intended to represent a measure of absolute economic value. Pre-tax financials reflect total project value and so do not include either corporate tax.

Table 1-12: Summary of Pre-Tax Financial Performance

Category	LOM Base Case C$M	Case C$M	Case 3 C$M
Net revenue (net smelter return + streaming revenues)	1,713.0	1,805.4	1,659.8
Operating costs (excluding transport costs)	632.6	632.6	632.6
Cash flow from operations	1,080.4	1,172.7	1,027.2
Capital (incl. sustaining capital, contingency, reclamation, salvage)	509.8	509.8	509.8
Net profit	570.7	663.0	517.5
Pre-Tax IRR (%)	16.5	18.5	15.4
Pre-Tax $NPV_{8\%}$	192.7	246.4	162.9
Payback Period	4.3	4.1	4.4

After-tax financial performance is summarized in Table 1-13.

Table 1-13: Summary of After-Tax Financial Performance

Category	LOM Base Case C$M	Case 2 C$M	Case 3 C$M
Net revenue (net smelter return + streaming revenues)	1,713.0	1,805.4	1,659.8
Operating costs (excluding transport costs)	632.6	632.6	632.6
Cash flow from operations	1,080.4	1,172.7	1,027.2
Capital (incl. sustaining capital, contingency, reclamation, salvage)	509.8	509.8	509.8
Net profit	570.7	663.0	517.5
After-Tax IRR (%)	14.7	16.5	13.8
After-Tax $NPV_{8\%}$	138.7	177.1	117.6
Payback Period	4.3	4.1	4.5

1.15.7 Sensitivity Analysis

A sensitivity analysis was conducted on pre-tax and after-tax project net present values (NPV) for individual parameters including metal prices, operating costs, and capital costs. The results are shown in Tables 1-14 and 1-15. The Base Case is most sensitive to variations in metal price and less sensitive to changes in operating costs. Differences in the cases presented are solely based on differences in metal prices throughout the project (-15% to +15% sensitivity factors were used for capital cost, operating costs, metal prices and grade).

Table 1-14: Sensitivity Analysis on Project Pre-Tax NPV 8%

Case	Variable	Pre-Tax $NPV_{8\%}$ C$M		
		-15% Variance	0% Variance	+15% Variance
Base Case	Capital Costs	260.0	192.7	125.4
	Operating Costs	249.5	192.7	135.9
	Metal Prices	22.1	192.7	359.8
Case 2	Capital Costs	313.7	246.4	179.1
	Operating Costs	303.2	246.4	189.6
	Metal Prices	67.8	246.4	421.5
Case 3	Capital Costs	230.2	162.9	95.6
	Operating Costs	219.7	162.9	106.1
	Metal Prices	-1.6	162.9	322.0

Table 1-15: Sensitivity Analysis on Project After-Tax NPV 8%

Case	Variable	After-Tax $NPV_{8\%}$ C$M		
		-15% Variance	0% Variance	+15% Variance
Base Case	Capital Costs	206.0	138.7	71.4
	Operating Costs	179.8	138.7	97.5
	Metal Prices	11.0	138.7	257.2
Case 2	Capital Costs	244.4	177.1	109.8
	Operating Costs	217.6	177.1	136.0
	Metal Prices	47.5	177.1	300.5
Case 3	Capital Costs	184.9	117.6	50.3
	Operating Costs	158.8	117.6	75.8
	Metal Prices	-8.1	117.6	231.1

The Base Case was evaluated at different discount rates to determine the effect on project NPV. Project NPV declined as the discount rate increased. Tables 1-16 and 1-17 demonstrate the summary of the discount rate sensitivity results on all three cases evaluated.

Table 1-16: Pre-Tax NPV Discount Rate Sensitivity Analysis in C$M

Discount Rate	Base Case	Case 2	Case 3
0%	570.7	663.0	517.5
6.5%	244.2	303.3	211.1
7.0%	226.3	283.5	194.3
7.5%	209.1	264.5	178.3
8.0%	192.7	246.4	162.9

Table 1-17: After-Tax NPV Discount Rate Sensitivity Analysis in C$M

Discount Rate	Base Case	Case 2	Case 3
0%	451.1	511.1	416.6
6.5%	181.9	223.5	158.8
7.0%	166.8	207.4	144.5
7.5%	152.4	191.9	130.8
8.0%	138.7	177.1	117.6

1.16 Interpretations & Conclusions

The feasibility study represents an economically viable, technically credible, and environmentally sound mine development plan for the Tulsequah Chief project.

The project is economically viable, generating operating cash flow of $1,080.4 M and an after-tax cash flow of $451.1 M over a nine-year mine life. This results in an after-tax IRR of 14.7% and a $138.7 M NPV at 8%.

It is recommended that the Tulsequah Chief project be advanced for development.

2 Introduction & Terms of Reference

2.1 Basis of Technical Report

This Technical Report was compiled by JDS for Chieftain to summarize the results of the feasibility study. This report was prepared following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101F1.

2.2 Scope of Work

This report is the work carried out by several consulting companies, none of which is associated or affiliated with Chieftain. The scope of work for each company is listed below.

2.2.1 JDS Energy & Mining Inc.

- compile a technical report that includes the data and information provided by other consulting companies
- select mining equipment
- estimate capital and operating costs for mining
- summarize capital and operating costs
- prepare a financial model and conduct an economic evaluation including sensitivity and project risk analysis
- make recommendations to improve value, reduce risks and move the project toward construction
- estimate power requirements
- identify proper site plant facilities and other ancillary facilities
- estimate all initial and sustaining capital expenditures requirements and operating costs for processing
- estimate all initial and sustaining capital expenditures requirements and operating costs for waste storage, tailings disposal and water storage.

2.2.2 SRK Consulting (Canada) Inc.

- project setting, history and geology description
- mineral resource estimate.

2.2.3 Ken Sangster and Associates Ltd.

- implement and supervise metallurgical a testing program
- establish recovery values based on metallurgical testing results
- design process plant to realize the predicted recoveries.

2.2.4 Marsland Environmental Associates Ltd.

- review environmental and other permit requirements
- summarize environmental results and concerns.

2.2.5 Klohn Crippen Berger Ltd.

- design waste management facilities including TMF, historical potential acid generating (HPAG), operating potentially acid generating (OPAG) and pyrite tailings storage facilities.

2.2.6 David West Consulting

- assess the mining rock geomechanics of the project.

2.2.7 Kovit Engineering Limited

- assess the paste backfill testing done on the project to date
- preliminary paste backfill plant and distribution system.

2.3 Qualifications & Responsibilities

Qualified persons are listed in Table 2-1. Qualified Person certificates are provided in Attachment 1 at the end of this technical report.

Table 2-1: Qualified Person Responsibilities

Author	Company	Report Section(s) of Responsibility
Mr. Gordon E. Doerksen, P.Eng.	JDS	1,2,3,19,21,22,24,25,26,27
Mr. Michael E. Makarenko, P.Eng.	JDS	15,16, excluding 16.3 and 16.8
Mr. Robert L. Matter, P.E.	JDS	18,excluding 18.11 and 18.12
Mr. Gilles Arseneau, Ph.D., P.Geo	SRK	4,5,6,7,8,9,10,11,12,14,23
Mr. Kenneth J. Sangster, C.Eng.	Sangster	13,17
Mr. Robert Marsland, P.Eng.	MEA	20
Mr. Harvey N. McLeod, P.Eng.	KCB	18.11, 18.12
Mr. Dave West, P.Eng.	Dave West	16.3
Mr. Frank Palkovits, P.Eng.	Kovit	16.8

2.4 Site Visits

- Mike Makarenko visited the project site from November 5-6, 2012.
- Rob Matter has not visited the site.
- Gord Doerksen visited the project on March 20-21, 2011.
- Gilles Arseneau inspected the project during May 18-19, 2006; September 13-14, 2006; and October 25-26, 2011.
- Ken Sangster visited the project on May 3-6, 2011 and Sept 13-16, 2011.
- Rob Marsland was last on site Nov 6-9, 2012. He also visited the site Sept 11-14, June 6 and May 14-17, 2012.
- David West visited the site on March 14-15, 2012.
- Harvey McLeod has not visited the site.
- Frank Palkovits has not visited site.

2.5 Currency

Unless otherwise specified, all costs in this report are presented in Canadian Dollars (C$).

2.6 Units of Measure, Calculations & Abbreviations

All units in this report are based on the International System of Units (SI), except industry standard units, such as troy ounces for the mass of precious metals and pounds for the mass of base metals.

A list of main abbreviations and terms used throughout this report is presented in Table 2-2.

This report may include technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a slight margin of error. Where these occur, JDS does not consider them to be material.

Table 2-2: Units of Measure & Abbreviations

Units of Measure

Above mean sea level	amsl
Ampere	A
Annum (year)	a
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per second	ft^3/s or cfs
Cubic foot per minute	ft^3/min
Cubic foot	ft^3
Cubic metre per hour	m^3/h
Cubic metre per second	m^3/sec
Cubic metre	m^3
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	dwt
Degree	°
Degrees Celsius	°C
Diameter	ø
Dry metric tonne	dmt
Foot	ft
Gallon	gal
Gallons per minute	gal/min
Gigajoule	GJ
Gram per centimeter cube	g/cm^3
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Kilo (thousand)	k
Kilogram	kg
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilovolt	kV
Kilovolt-ampere	kVA
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Kilowatt	kW
Kilowatt-hour	kWh
Litre	L
Litres per minute	L/m
Mega ounces	Moz
Mega pounds	Mlbs
Megapascal	Mpa

Megavolt-ampere	MVA
Megawatt	MW
Meter per annum	m/a
Meter per day	m/d
Metre	m
Metres above sea level	masl
Metres per second	m/s
Metric ton (tonne)	t
Micrometre (micron)	µm
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million tonnes	Mt
Million	M
Month	mo
Parts per billion	ppb
Parts per million	ppm
Percent	%
Pounds per square inch	psi
Square foot	ft^2
Square kilometer	km^2
Square meter	m^2
Thousand ounces	koz
Thousand tonnes	kt
Thousandths of an inch (thickness)	mil
Tonne (1,000 kg)	t
Tonnes per cubic meter	t/m^3
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Volt	V
Wet metric tonne	wmt
Year (annum)	a

Abbreviations & Acronyms

Acid base accounting	ABA
Acid mine drainage	AMD
Acid potential	AP
Acid rock drainage	ARD
Acid treatment plant	ATP
Ammonium nitrate/fuel oil	ANFO
British Columbia Geological Survey	BCGS
Canada Pension Plan	CPP
Canadian certified reference materials project	CCRMP
Canadian Dam Association	CDA
Canadian development expense	CDE
Canadian exploration expense	CEE
Canadian Institute of Mining	CIM
Capital costs	CAPEX
Capital cost allowance	CCA
Cartesian coordinates, also easting, northing and elevation	X,Y,Z
Chalcopyrite	Cp
Copper equivalent	CuEq

Copper facies	CUF
Critical path method	CPM
Cut-off grade	COG
Diameter	Ø
Dissolved oxygen	DO
Effluent treatment plant	ETP
Electronic distance measuring	EDM
Elevation above sea level	Elev
Emergency Preparedness Plan	EPP
Employment insurance	EI
Energy x-ray analyzer	EDX
Environmental Management Act	EMA
Environmental-Social Impact Assessment	ESIA
Exchange traded fund	ETF
Feasibility Study	FS
Financial quarter (1, 2, 3 or 4)	Q1, Q2, Q3 or Q4
Flotation	FLOT
Fresh air raise	FAR
Front- end engineering design	FEED
Front-end loader	FEL
High-density polyethylene	HDPE
Historically potentially acid generating	HPAG
Horizontal to vertical	H:V
Impacts, Mitigations and Mutual Benefit Agreement	IMMBA
Inflow design flood	IDF
In-situ rock mass rating	ISRMR
Interim water treatment plant	IWTP
Internal rate of return	IRR
International Systems of Units	SI
Inter-ramp angles	IRA
JDS Energy & Mining Inc.	JDS
Klohn Crippen Berger Ltd.	KCB
Land use plan	LUP
Less than trailer-load	LTL
Life of mine	LOM
Linear low density polyethylene	LLDPE
Liquefied natural gas	LNG
Load-haul-dump	LHD
Locked cycle test	LC01
Longhole	LH
Marsland Environmental Associates	MEA
Maximum design earthquake	MDE
Mechanized cut and fill	MCF
Metal leaching/acid rock drainage	ML/ARD
Methyl isobutyl carbinol	MIBC
Mineral deposits research unit	MDRU
Minimum mining width	MMW
Motor control center	MCC
National Instrument 43-101	NI 43-101
Nearest neighbor	NN
Net present value	NPV
Net smelter return	NSR
Neutralization potential	NP
Non-acid-generating	NAG
Non-Owner supplied	NOS

Non-potentially acid generating	non-PAG
North, South, East, West	N,S,E,W
Operating costs	OPEX
Operational potentially acid generating	OPAG
Operations Maintenance and Surveillance Manual	OMS
Original Equipment Manufacturers	OEM
Particle size distribution	PSD
Potassium amyl xanthate	PAX
Potentially acid generating	PAG
Prefeasibility Study	PFS
Preliminary Economic Assessment	PEA
Probable maximum flood	PMF
Process flow diagram	PFD
Project management	PM
Project procedures manual	PPM
Proportional integral derivative	PID
Protected areas	PA
Pyrite facies	PYF
Quality assurance/quality control	QA/QC
Quality Management System	QMS
Return air raise	RAR
Reverse air blast	RAB
Reverse circulation	RC
Rock quality designation	RQD
Run of mine	ROM
Semi-autogenous grinding	SAG
Sewage treatment plant	STP
Sodium metabisulphite	SMBS
Sodium metaphosphate	SMP
Special used permit	SUP
Specific gravity	S.G.
Standard penetration test	SPT
Standard reference material	SRM
Static cone penetration test	CPT
Tailings management facility	TMF
Taku River Tlingit First Nation	TRTFN
Tennanite	Tn
Tetrahedrite	Th
Universal Transverse Mercator	UTM
Variable frequency drive	VFD
Volcanogenic massive sulphide	VMS
Waste water treatment plant	WWTP
Water balance model	WBM
Weight percentage	wt%
Work breakdown structure	WBS
Zinc facies	ZNF

3 Reliance on Other Experts

Preparation of this report is based upon public and private information provided by Chieftain and information provided in various previous technical reports listed in Section 27 of this report.

TetraTech provided engineering support for the FS, particularly in the area of plant design and site facilities. Responsibility for TetraTech's work has been taken by the appropriate Qualified Persons.

The authors have carried out due diligence reviews of the information provided to them by Chieftain and others for preparation of this report. The authors are satisfied that the information was accurate at the time of writing and that the interpretations and opinions expressed are reasonable and based on current understanding of mining and processing techniques and costs, economics, mineralization processes and the host geologic setting. The authors have made reasonable efforts to verify the accuracy of the data relied on in this report. Section 20.9 was provided by Joanne Thompson of Chieftain. Rob Marsland of MEA reviewed this section and assumed responsibility for its content.

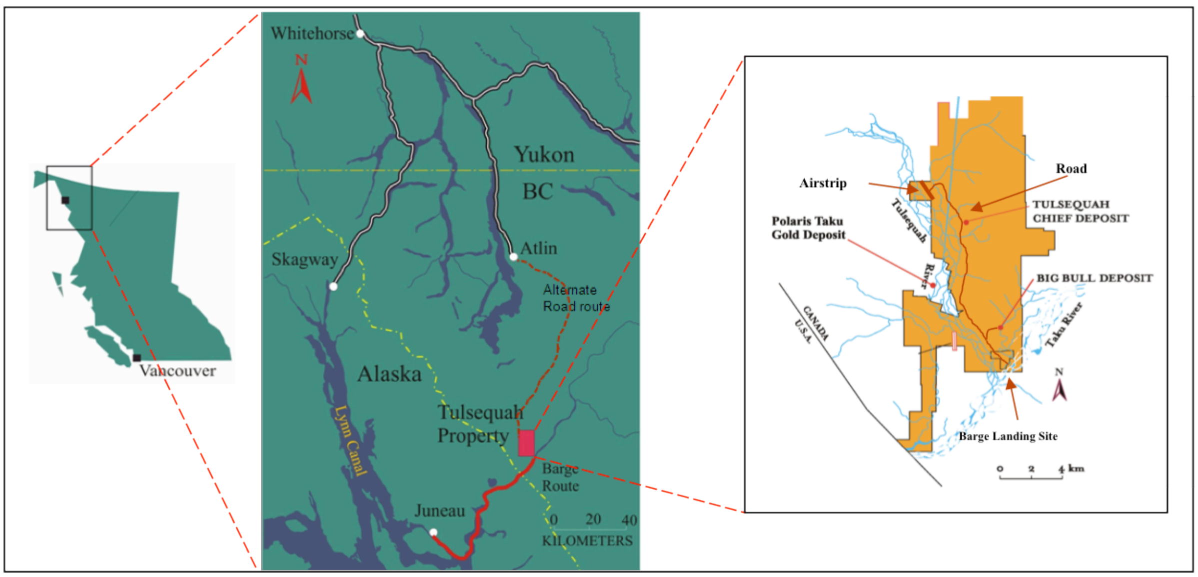
The results and opinions expressed in this report are conditional on the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein. The authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to the authors subsequent to the date of this report.

Neither JDS nor the authors of this technical report are qualified to provide extensive comment on legal issues associated with the property. As such, portions of Section 4 (mineral tenures and licenses, title and interest in the Tulsequah Chieftain property, royalties, back-in rights, payments or other agreements and encumbrances to which the property is subject) are descriptive in nature and are provided exclusive of a legal opinion.

4 Property Description & Location

Chieftain's property is located at 58°43'N and 133°35'W on the Tulsequah River in northwestern BC, as shown on Figure 4-1. The property is located 97 km south of the town of Atlin, BC (59°35'N, 133°40'W), which is the nearest Canadian community. Juneau (58°18'N, 134°24' W), the capital of Alaska, is situated 64 km southwest of the property. The property is accessible by air from both Atlin and Juneau, and by water during high-water periods from Juneau. The exploration base camp is situated on the east bank of the Tulsequah River at an elevation of 55 masl.

Figure 4-1: Property Location Map



Source Chieftain 2011

4.1 Mineral Tenure

The Tulsequah Chief property comprises 14 cell mineral claims (Table 4-1) and 25 crown granted mineral claims (Table 4-2 and Figure 4-2) totalling 140.8 km^2. All mineral claims are in good standing until 2022, while Crown grants are maintained through annual tax payments due on July 2 of each year, and are in good standing through July 3, 2012. They will remain in good standing thereafter provided annual tax payments are made. The mineral claims have not been surveyed but all crown grants have been surveyed.

Mining operations in the early 1950s by a prior owner of Tulsequah have left a residual acid mine drainage (ARD) problem resulting from oxidation of in-mine sulphides and acidic waters

Table 4-1: Chieftain Tulsequah Chief Cell Mineral Claims

Tenure No.	Area (ha.)	Expiry Date
513806	1241.297	December 31, 2022
513807	1242.293	December 31, 2022
513809	1393.208	December 31, 2022
513812	622.526	December 31, 2022
513813	806.766	December 31, 2022
513814	1160.494	December 31, 2022
513815	1310.797	December 31, 2022
513818	1615.841	December 31, 2022
513819	841.076	December 31, 2022
513820	1094.340	December 31, 2022
513821	842.324	December 31, 2022
513828	1331.763	December 31, 2022
590422	419.996	December 31, 2022
1011222	151.48	July 16, 2013

Table 4-2: Tulsequah Chief Crown Grants

Property Area	Record No.	Units	Area (ha.)	Expiry Date
Crown Grants 1				
River Fr.	5669	1	7.99	July 3, 2012
Tulsequah Bonanza	5668	1	20.9	July 3, 2012
Tulsequah Bald Eagle	5676	1	14.16	July 3, 2012
Tulsequah Chief	5670	1	20.9	July 3, 2012
Tulsequah Elva Fr.	5679	1	9.7	July 3, 2012
Big Bull Crown Grants 1				
Big Bull	6303	1	20.65	July 3, 2012
Bull No. 1	6304	1	16.95	July 3, 2012
Bull No. 5	6306	1	14.57	July 3, 2012
Bull No. 6	6305	1	17.22	July 3, 2012
Hugh	6308	1	20.71	July 3, 2012
Jean	6307	1	17.02	July 3, 2012
Banker Crown Grants 1				
Vega No. 1	6155	1	20.9	July 3, 2012
Vega No. 2	6156	1	17.62	July 3, 2012
Vega No. 3	6157	1	18.97	July 3, 2012
Vega No. 4	6158	1	19.85	July 3, 2012
Vega No. 5	6159	1	14.94	July 3, 2012
Janet W. No. 1	6160	1	18.95	July 3, 2012
Janet W. No. 2	6161	1	18.75	July 3, 2012
Janet W. No. 3	6162	1	16.6	July 3, 2012
Janet W. No. 4	6163	1	20.76	July 3, 2012
Janet W. No. 5	6164	1	18.2	July 3, 2012
Janet W. No. 6	6165	1	19.02	July 3, 2012
Janet W. No. 7	6166	1	18.78	July 3, 2012
Janet W. No. 8	6167	1	17.98	July 3, 2012
Joker	6169	1	16.6	July 3, 2012

1. Maintained through annual tax payments due July 2 of each year. All Crown grants are overlapped by the mineral clams except for 15.465 ha of the Big Bull Crown Grants.

Figure 4-2: Tulsequah Chief Mineral Claims

carrying dissolved metals draining into the Tulsequah River. Previous remediation efforts by Redfern moderated the discharge, but did not achieve the levels required by BC and Canada environmental protection statutes. In May 2004, Environment Canada issued a directive to Redcorp requiring Redcorp to install a water treatment facility for the treatment of acidic mine waters from historic operations to be operational by June 30, 2005. The insolvency of Redfern and its parent, Redcorp, in 2009 has resulted in the removal of assets from site that were part of the planned remediation works and the degradation of some of the remaining infrastructure. Fines for noncompliance could be as much as $100,000 per day and senior management could be subject to jail terms.

4.2 Underlying Agreements

In January 2010, Chieftain negotiated a Purchase Agreement with the Receiver and the Trustee in bankruptcy of Redcorp and Redfern to purchase the 13 mineral claims, 25 crown-granted claims and four fee-simple lots comprising the Tulsequah project plus some miscellaneous equipment assets including a water treatment plant. That agreement was subsequently amended to include agreements reached with the holders of registered lien claims on the property assets subject of the purchase. On September 22, 2010, the purchase was approved by the British Columbia Supreme Court and a Vesting Order issued to Chieftain granting full unencumbered ownership of the Tulsequah property, free of any liens or debts. Title to all of the real property assets and the mineral claims were transferred to Chieftain on September 29, 2010.

Chieftain holds a 100% interest in both the mineral claims and the crown grants. There are no back-in rights of royalties on the Tulsequah Chief property.

4.3 Permits & Authorization

All pending permits are listed in Table 20-2 of Section 20 of this report.

4.4 Environmental Considerations

Chieftain has commenced to rebuild the site infrastructure and capacity for support of renewed remediation works over the first year of ownership and also acquired and transported the water treatment facility to the mine site. Chieftain achieved operational status of the interim water treatment plant (IWTP) at the Tulsequah Chief Mine site in the second half of 2011.

4.5 Mining Rights in British Columbia

Under the *BC Mineral Tenure Act*, Chieftain can maintain the located mineral claims in good standing by filing assessment work for $200 per unit per year. Crown granted claims are maintained through the payment of annual taxes. Crown granted claims at the Tulsequah Chief mine have been legally surveyed.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

The Tulsequah Chief property is located on the east side of the Tulsequah Valley, in the flood plain of the Tulsequah River near its junction with the Taku River. Topographic elevations on the property range from 50 m at river level to over 1,800 m at the top of Mount Eaton. The property is located 16 km upstream of the US-Canadian border and 64 km northeast of Juneau, Alaska.

Presently, the only access to the mine site is by air. Access is easiest by fixed-wing aircraft or by helicopter from Atlin or Juneau. Three airstrips are serviceable at, or in close proximity to, the Tulsequah Chief property. A gravel strip is located northwest of the confluence of the Taku and Tulsequah Rivers, which will accommodate a DC3 or Caribou aircraft but is subject to flooding two or more times each summer. An airstrip located at the Polaris-Taku mine site, which is located 4.7 km south-southwest of the camp, is less flood-prone and allows fixed wing access but has a difficult approach. The airstrip is suitable for aircraft up to Shorts Sky Van in size; however, travel from these airstrips to other parts of the map area requires a helicopter. In 2008, a 1,050 m gravel airstrip was constructed west of Shazah Creek on the east side of the Tulsequah River. Aircraft up to Buffalo size have utilized this strip, which is also connected by roads to the Tulsequah Mine site and south to the Taku River. Helicopters are intermittently based in the Tulsequah Valley, but otherwise must be chartered from Atlin or Juneau.

Shallow-draft boat access is available to the confluence of the Tulsequah and Taku rivers; however, the Tulsequah River is not easily navigated due to high and variable flows, debris hazards, and shallow areas. Hydrographic assessments determined that the Taku River broadens to extremely shallow water in its lower reach before the Taku Glacier. Channel locations within this area vary and would require more or less continuous dredging during the shipping season to maintain an open channel. The period available to conventional barging varies from year to year, ranging from less than three months to as much as six months; however, riverboat access from Juneau is possible for most of the early summer months.

A few short road segments were built during development and production years of the Tulsequah Chief and Polaris-Taku mines, but all are washed out and overgrown to some extent and none are linked to the provincial road network.

5.2 Local Resources & Infrastructure

The property is remote and currently only accessible by air or barge service. Local infrastructure is limited. Grid electric power is not available at or near the mine site. Water is available from streams adjacent to the mine site, from the Tulsequah River, and from the Tulsequah River aquifer via sandwells.

Mining personnel can be recruited from Atlin, Whitehorse (Yukon), or more distant centres, and flown to the mine site on a rotating shift basis.

A potential area suitable for tailings storage has been identified along the Shazah Creek close to its confluence with the Tulsequah River.

Waste rock disposal areas have been identified one km south of the mine portals near Rogers Creek on the east side of the Tulsequah River.

A potential site for the processing plant is on the area immediately adjacent to the 5200 level portal and the 5400 level portal.

5.3 Climate

Situated in the inland area of the north coast of BC, the climate at Tulsequah Chief is characterized by high precipitation and relatively moderate winter temperatures due to the influence of the Pacific Ocean. Atlin, BC and Juneau, Alaska, the closest towns to the property, provide the most representative climate data for the Tulsequah Chief area. At the lowest level of the property, at river level, snow cover typically lasts from mid-November to early May.

Vegetation ranges from dense coastal forest at the lowest elevations, to bare rock and ice at the higher elevations. Dense, mature coastal forest with thick undergrowth covers approximately 60% of the property, with roughly 5% outcrop located within these forested areas. Large, covered areas are restricted by ice cover, river bottoms and swamp, which collectively amount to about 30% of the area. Approximately 15% of the present property area is concealed by two major icefields: Mount Eaton and Manville. Fieldwork is generally hampered by steep topography, snow and ice cover and poor weather.

5.4 Physiography

In the ranges between Stewart and Mount Foster, where the Tulsequah Chief is located, a very high percentage of the area is under a cover of glacial ice. The Taku Icefield, a very large icefield that extends southward from Skagway to the Taku River, and the Tulsequah Glacier, which flows southward to the head of Tulsequah River, both play an important role in the physiography of the region (Holland, 1976). The Tulsequah and Taku River valleys display typical glacial morphology, with broad flat floodplains, each several kilometres wide, and steep valley walls. The property area lies mainly north of the steep-sided Taku River. The gentler and drier Stikine Plateau uplands flank the area to the east.

The Tulsequah River, which originates 15 km north of the property at the toe of the Tulsequah glacier, is a braided stream occupying a valley comprised of glacio-fluvial debris with little vegetative cover (Figure 5-1).

Figure 5-1: Typical Landscape in the Project Area (looking to the north over the airstrip)

6 History

Mining exploration has occurred since the early 1800s in the Tulsequah and Taku Valleys; however, the first official record of mining and prospecting in the district was in 1923 when George A. Clothier, resident engineer for the northwest district of BC, first visited the area. Earlier that year, W. Kirkham of Juneau had staked the Tulsequah Chief after locating high-grade barite, pyrite, sphalerite, galena, and chalcopyrite mineralization outcropping in a gully at about 500 masl. The Big Bull portion of the Tulsequah Chief property was then optioned by the Alaska Juneau Gold Mining Company (Juneau Gold), which drove an adit on the Big Bull occurrence in an unsuccessful search for ore and then abandoned its operations.

In 1928, a syndicate represented by W.A. Eaton and Dan J. Williams found impressive widths of mixed sulphides; and in 1929, the United Eastern Mining Company optioned the property and carried on efficient and aggressive development. In May of 1929, V. Manville discovered the Big Bull massive sulphide deposit, on which the Juneau Gold acquired a working option. The later discovery of the Potlatch (Sparling), Banker, Ericksen-Ashby, and the Whitewater (Polaris Taku) deposits contributed further to the favourable publicity given to the area.

In 1946, Cominco Ltd. (Cominco) acquired the Tulsequah Chief and Big Bull deposits, and exploration and preproduction work began shortly after in 1947. By 1951, Cominco's two properties, Big Bull and Tulsequah Chief, were mined successfully at an average production rate of 482 t/d (530 tons). Total production was 935,536 tonnes (575,463 tonnes from the Tulsequah Chief mine and 360,073 tonnes from the Big Bull deposit). Average grade of ore was 1.59% Cu, 1.54% Pb, 7.0% Zn, 3.84 g/t Au, and 126.5 g/t Ag. The mines produced 14,765 tons Cu, 11,439 tons Pb, 54,910 tons Zn, 95,340 oz Au, and 3,329,938 oz Ag at a recovery of about 88% Cu, 94% Pb, 87% Zn, 77% Au, and 89% Ag.

Low metal prices in 1957 forced the suspension of mining activity at both of Cominco's mines. Cominco never reopened the mines, and caretakers lived at the site until the mill equipment was dismantled and sold in the late 1970s. At shutdown, ore reserves at the Tulsequah Chief Mine were estimated at 707,616 tonnes grading 1.3% Cu, 1.6% Pb, 8.0% Zn, 2.40 g/t Au, and 116.5 g/t Ag. Cominco geologists estimated these reserves in 1957. They are based on detailed underground drilling and sampling. The estimates were prepared before the implementation of NI 43-101 and, as such; do not conform to the NI 43-101 standards. The historical estimates do not use mineral resource and mineral reserve categories that are in accordance with NI 43-101. The historical estimates are believed to be reliable as they were based on historical plans at the time the mine was in operation. The historical estimates should not be relied upon.

In 1981, Redfern commenced a reconnaissance exploration joint venture with Comaplex Resources Corp. (Comaplex) in northwestern BC, which ultimately resulted in the staking of a block of claims surrounding the residual claims held by Cominco over the Tulsequah Chief mine. Geological mapping (1:2500) was completed in 1981, and the property was flown by Dighem and

Input EM/Mag in 1982. Redfern then recognized that the deposit had the geological characteristics of a VMS deposit rather than the replacement/shear-hosted affinity previously ascribed to the deposits.

This recognition meant that there was likely to be more ore at the Tulsequah Chief property than previously identified which resulted in Cominco staking additional claims to expand their holdings around the known deposits. Redfern acquired its partner's interest in their JV Tulsequah Chief claims and initiated discussions with Cominco concerning joint exploration. Eventually, an agreement was signed whereby Redfern could acquire a 40% interest in the amalgamated claims by funding the first $3 M of renewed exploration.

Work started in 1987 with surface diamond drill holes and progressed to drilling from the rehabilitated underground workings in 1988. By 1989, Redfern had earned its interest and the subsequent ongoing exploration was jointly funded. Extensive exploration programs continued each year on this basis until 1991. This work ultimately included an extension of the historic underground workings in 1989, 1990, and 2004 to develop new drill platforms.

In 1992, Redfern negotiated and exercised an option to purchase Cominco's interest in the property. Redfern, as sole owner, proceeded with a comprehensive work program in 1993. The large program included an initial evaluation of the stratigraphy between the Tulsequah Chief and Big Bull, as well as diamond drilling of the Big Bull property, which eventually led towards feasibility assessment and permitting decisions.

In 1993, Redfern received a positive prefeasibility study in 1994, they initiated full feasibility studies and an application was made to obtain a Mine Development Certificate under the prevailing provincial and federal environmental assessment regulations. Due to delays associated with permitting and subsequent litigation, including a legal challenge launched by the Taku River Tlingit First Nation, technical geological work during the period between 1994 and 2003 was limited to the collection of a bulk sample from the 5200 level.

Technical work resumed in 2003, after a Project Approval Certificate was granted by the Government of British Columbia, and an amended screening level environmental assessment was authorized by the federal government in July of 2005.

The 2003 program focused on the search for new deposits at the same stratigraphic level and within the same hydrothermal system as the Tulsequah Chief deposit. This program successfully discovered a new mineralized zone stratigraphically above the Tulsequah Chief, which remains open along strike to the west.

In 2006, Redcorp commissioned Wardrop Engineering to carry out a feasibility study for the Tulsequah Chief deposit (McVey, 2007). Wardrop estimated that the Tulsequah Chief deposit had probable mineral reserves of 5,378,788 tonnes grading 6.33% Zn, 1.40 % Cu, 1.20 % Pb, 2.59 g/t Au and 186 g/t Ag. Mineral reserve tonnages and grades were derived by performing detailed mine planning based on an orebody represented by a geological block model. In all, twelve solids were developed from the drill data. The reserve was derived from a mineral resource determined at an NSR cut-off of US$94/tonne of ore. Subsequent detailed mine planning indicated that the orebody could be economically mined at an NSR cut-off of US$71/tonne of ore. The NSR was based to the following metal prices, gold US$550/oz, silver US$8.95/oz, copper US$1.85/lb lead US$0.42/lb and zinc US$0.92/lb. Mineral reserves were assumed to be sufficient to support mining operation for eight years at an annual production rate of 2,000 t/d. The study concluded that the Tulsequah Chief could be developed with an initial capital cost of $201 M and that the project had a pre-tax NPV of $160 M and an IRR of 30% based on an 8% discount rate. The Wardrop mineral reserves estimates were prepared in accordance with NI 43-101 and used categories for mineral reserves as stipulated in NI 43-101. The estimates are believed to be reliable.

SRK has not done sufficient work to classify the mineral reserves as current mineral reserves and Chieftain is not treating the historical mineral reserves as current mineral reserves.

Subsequent to the completion of the feasibility study and technical report on the Tulsequah Chief, completed and filed by Redcorp in 2007, Redfern the 100% owned subsidiary of Redcorp, undertook a comprehensive mine permitting and development program at the Tulsequah property. This work was suspended by Redcorp in December 2008 on a temporary basis and later extended into an indefinite shutdown in February 2009, followed by Redcorp's filing for creditor protection under CCAA in March 2009. Attempts to restructure Redcorp's debt or obtain a project partner were unsuccessful and in late May 2009, the Court appointed a Receiver over the assets of Redcorp and Redfern.

Prior to the shutdown, Redfern had secured a number of key permits for the development, including Mineral Exploration Code permits for initial access roads from a barge landing on the Taku River to the Tulsequah Chief mine site and construction of a new airstrip on the east side of the Tulsequah River. A *Mines Act* permit was obtained to convert these facilities for eventual mine production purposes and to allow construction of roads connecting the new airstrip to the Tulsequah Chief site and to the barge landing. An amendment to the *Mines Act* permit further allowed construction of waste storage pads and preliminary mill and plant site foundation preparations (partially completed). Other permits acquired by Redfern included a License to cut from the BC Ministry of Forests, a construction discharge permit from the BC Ministry of Environment and a number of stream crossing and bridge authorizations from Fisheries and Oceans Canada, Transport Canada and BC Ministry of Environment.

No additional resource drilling was conducted by Redfern in the time and no mine development works were initiated.

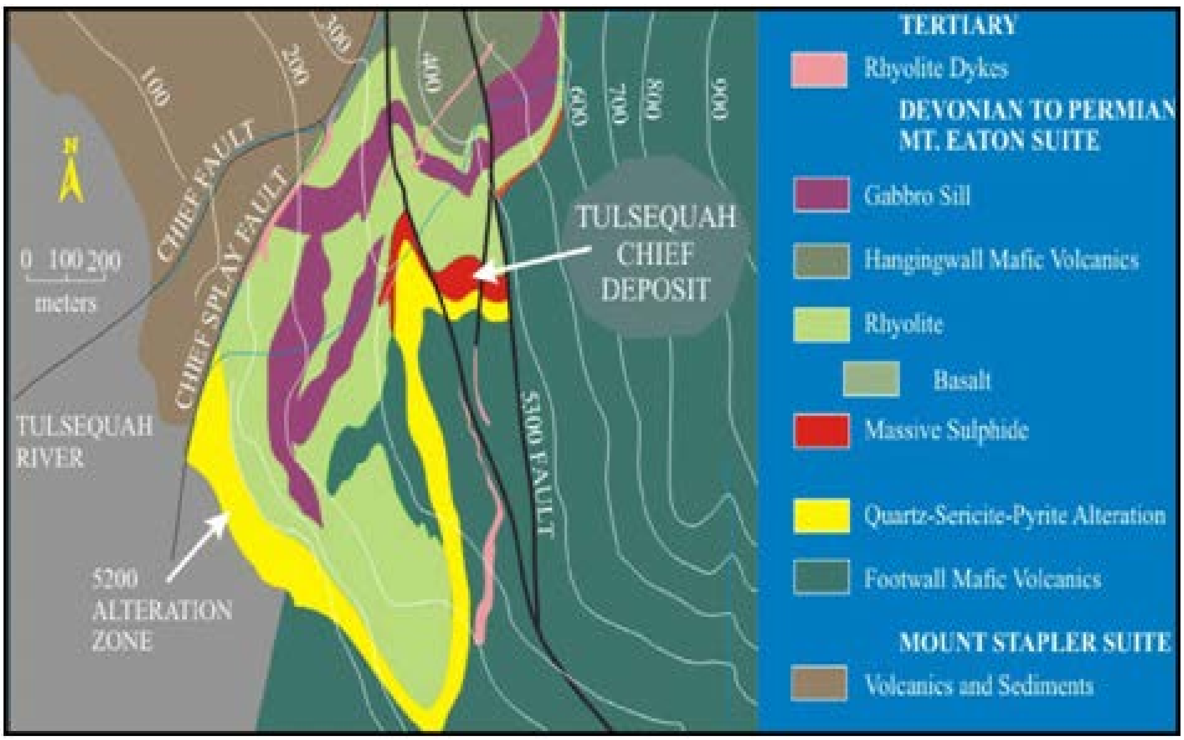
In January 2010, Chieftain negotiated a Purchase Agreement with the Receiver and the Trustee in the bankruptcy of Redcorp and Redfern to purchase the 13 mineral claims, 25 crown-granted claims and four fee-simple lots comprising the Tulsequah project plus some miscellaneous equipment assets including a water treatment plant. That agreement was subsequently amended to include agreements reached with the holders of registered lien claims on the property assets subject of the purchase. On September 22, 2010, the purchase was approved by the British Columbia Supreme Court and a Vesting Order issued to Chieftain granting full unencumbered ownership of the Tulsequah property, free of any liens or debts. Title to all of the real property assets and the mineral claims were transferred to Chieftain on September 29, 2010.

7 Geological Setting & Mineralization

7.1 Regional Geology

Regionally, the Tulsequah Chief area is characterized by fault juxtaposition and deformation of several Palaeozoic to Mesozoic and older tectono-stratigraphic terranes. Subsequent intrusions by Jurassic to Cretaceous aged Coast plutons, and unconformable burial by Tertiary Sloko volcanic rocks, contributes further to a significant amount of deformation and complexity in this region (Figure 7-1).

Figure 7-1: Tulsequah Chief Deposit Area Geology



Source Chieftain 2011

The Llewelyn fault, the dominant structural feature of the Tulsequah Chief area, separates the higher-grade metamorphic rocks of the Palaeozoic and older ages on the west, from weakly metamorphosed Palaeozoic and Mesozoic rocks to the east. On the western flank of the fault, three suites are recognized based based on lithological associations and degree of deformation.

From west to east, and corresponding with decreasing metamorphic grade and degree of deformation and variation from predominately basinal to predominately arc character, they are:

- Whitewater Suite (amphibolite-grade quartz-rich metamorphic sequence of sedimentary origin)
- Boundary Ranges suite (schists of volcanic and sedimentary origin)
- Mount Stapler suite (a low-grade metamorphic package that shares characteristics of both the Whitewater and Boundary Range suites and locally can be demonstrated to be gradational into both) (Mihalynuk et al., 1994).

East of the fault, Palaeozoic rocks of the Stikine assemblage, a low-grade unit of volcanic arc rocks, hosts the Tulsequah Chief and Big Bull sulphide deposits.

7.2 Property Geology

The Tulsequah Chief deposit is dominantly underlain by rocks of the Devono-Mississippian to Permian-aged Mount Eaton group, which is a low metamorphic grade, island arc volcanic assemblage contained within the Stikine Terrane of northwest BC (Mihalynuk et al, 1994). These rocks are situated east of the Chief (Llewelyn) fault, and are predominately located north of the Taku River, and east of the Tulsequah River.

The enclosing stratigraphy of the Mount Eaton Block, which hosts the Tulsequah Chief and Big Bull VMS deposits, has been well defined based on surface and underground development mapping, biochronology, lithogeochemistry, isotopic age determinations, as well as from extensive surface and underground drilling. This work, completed by the British Columbia Geological Survey (BCGS) (Mihalynuk et al., 1994), Mineral Deposits Research Unit (MDRU) (Sherlock et al., 1994) and by Redfern, has subdivided the stratigraphy into three divisions. The Lower division is dominated by Devonian to early Mississippian age bimodal volcanic units, which include the Mine series felsic rocks hosting the Tulsequah Chief and Big Bull deposits. The Middle Division, Mississippian to Pennsylvanian in age, is composed primarily of pyroxene bearing mafic breccias and agglomerates with locally extensive accumulations of mafic ash tuffs and volcanic sediments. Polymictic debris flows and/or conglomerates define a transition zone from the Middle to Upper Divisions. The Pennsylvanian to Permian Upper Division rocks are dominated by volcanic derived and clastic sediments with lesser mafic flows. Near the top of the Upper Division, distinctive bioclastic rudite and intercalated chert, shales and occasional sulphidic exhalite occur. Late Tertiary Sloko rhyolite and mafic dykes commonly intrude the Palaeozoic units.

The Mount Eaton block is structurally dominated by the north trending, eastward verging Mount Eaton anticline, which plunges moderately north and dips steeply west. On the western limb of this anticline, numerous parasitic, upright to overturned, folds (F1), occur ranging from open to nearly isoclinal in form. Excluding the case in extremely appressed folds, penetrative fabric is weak or poorly developed. The first phase of folding (F1) is refolded by a second, east-west folding phase (F2) that is irregularly expressed across the property and locally produces a crosscutting cleavage (S2). In general, the F2 folds are upright and open. F1 folds are not significantly reoriented by the F2 second phase of folding; however, the F1 phase of folding does exhibit variable plunge attitudes. Generally, F1 fold axes plunge to the north in the northern region of the property, with southern trending plunges more common in the southern region. In the Tulsequah Chief Mine area, folds are open and plunge at 55° to 60° to the north with steep westerly dipping axial planes.

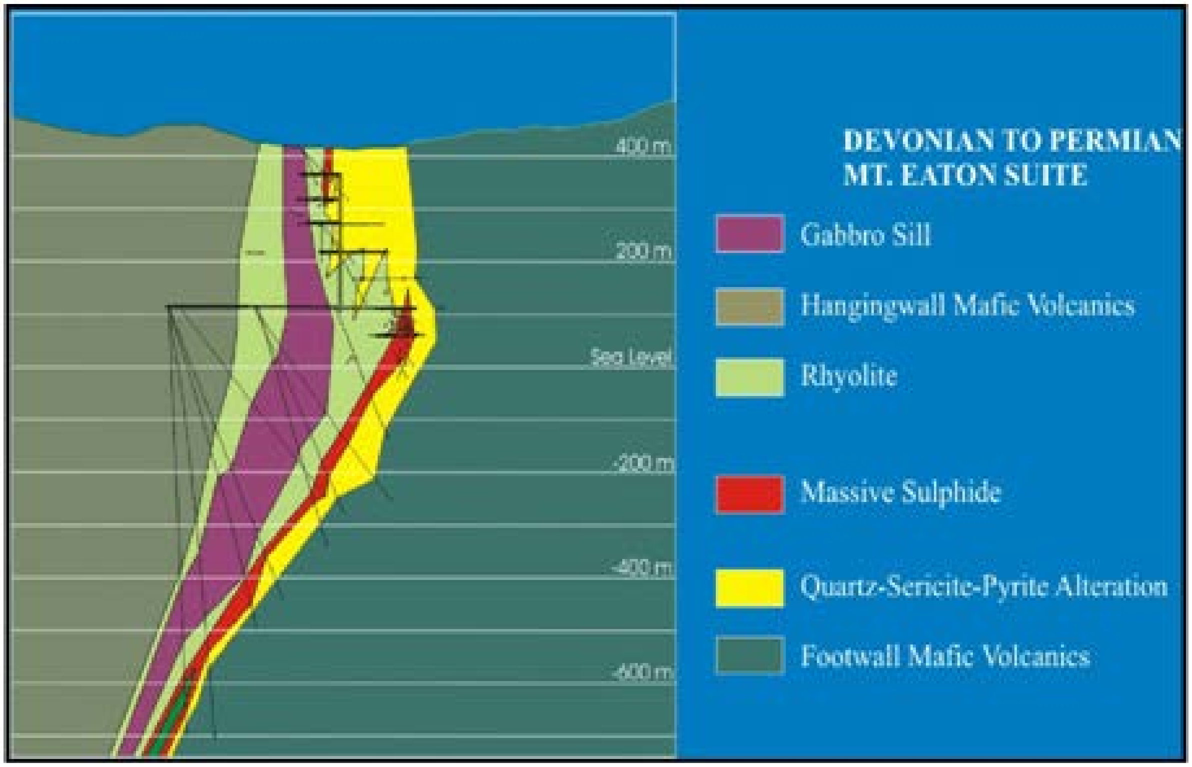
North to northwest-trending high-angle faults with complex displacement histories are common within the Tulsequah Chief region, with the largest and most significant being the Llewelyn fault. Displacement appears to be small on these faults, and most faults are marked by topographic depressions in the form of steep-sided gullies and ravines. The north trending faults are commonly intruded by late Tertiary Sloko rhyolite and mafic dykes. Thrust faults occur within the Mount Eaton block, and offset is considered relatively minor.

7.3 Deposit Geology

The Tulsequah Chief deposit consists of numerous stacked sulphide lenses developed within the basal stratigraphy or a rhyolite-rich sequence of volcanic flows and fragmental units (Figure 7-2). These felsic volcanics rest above a thick assemblage of mafic volcanics (primarily basalt, and basaltic andesite). Above the assemblage of rhyolitic volcanics, a mafic dominated sequence of basalt flows, breccias and sills, overlays the unit. Within the mine area, a thick diorite/gabbro sill, which is geochemically identical to the upper mafic volcanic units, intrudes the rhyolite above the sulphide deposits. Basaltic dykes recognized to be feeders to the thick sill, cut through the sequence. Late stage Sloko dykes of Tertiary age are associated with faults cutting all of the Mine sequence rocks.

A synclinal structure, termed the H syncline, is the host to the thickest section (approximately 30 m) of the sulphide deposit. The thinner areas of the deposit extend into the limbs of this structure and into an anticline to the west (F anticline). Two prominent faults are sub-parallel to the axial plane of the fold within the H syncline. These faults, 4400E and 5300E, may represent focal points of renewed movement on older basin-bounding growth faults at the time of sulphide deposit deposition. Within the fold limb east of the 5300E fault, the G lens is interpreted to be a fault offset of the main H lens within the main H syncline structure.

Figure 7-2: Schematic Vertical Section through Tulsequah Deposit



Source Chieftain 2011.

7.4 Mineralization

Mineralization consists of massive lenses of pyrite and chalcopyrite, and semi-massive sphalerite, galena and pyrite. Accessory ore minerals include tetrahedrite-tennantite and rare native gold. Gangue consists of barite (averaging approximately 6%), chert, gypsum, anhydrite and carbonate near the top of the lens, and carbonate quartz, chlorite and sericite with silica altered volcaniclastic rocks near the base of the lens. Visually, the sulphides can be divided into three distinct sulphide facies: copper facies (CUF), zinc facies (ZNF), and pyrite facies (PYF). CUF mineralization is characterized by massive to banded pyrite and chalcopyrite with minor sphalerite and galena. ZNF mineralization consists primarily of sphalerite and galena in barytic gangue, with much less pyrite and chalcopyrite. PYF mineralization consists of massive pyrite with little to no base metal sulphides. These ore types may occur within a single lens, typically with sharp boundaries between them. Despite the occurrences of several distinct ore types (based on the relative abundance of different sulphide minerals) no clear geographical zonation pattern has emerged.

8 Deposit Types

The Paleozoic-aged Tulsequah Chief and Big Bull VMS deposits can be classified as bimodal-felsic VMS deposits as described by Galley et al (2007). These deposits are associated with marine volcanism; commonly during a period of more felsic volcanism in an andesite (or basalt) dominated succession; locally associated with fine-grained marine sediments; also associated with faults or prominent fractures. The deposits commonly form one or more lenses of massive pyrite, sphalerite, galena and chalcopyrite commonly within felsic volcanic rocks in a calcalkaline bimodal arc succession. The lenses are zones with a Cu-rich base and a Pb-Zn-rich top; low-grade stockwork zones commonly underlie lenses and barite or chert layers may overlie them. Deposits form concordant massive to banded sulphide lens which are typically metres to tens of metres thick and tens to hundreds of metres in horizontal dimension.

The Myra Falls VMS deposits are believed to share quite a similar geological setting and age to those found in the immediate Tulsequah regions of northwestern BC. In the case of the Tulsequah Chief and Big Bull deposits, sub-seafloor emplacement in permeable rhyolitic hyaloclastites of precious and base metal-bearing massive sulphides likely account for the bulk of the VMS mineralization. These rhyolitic hyaloclastite host rocks can be viewed as carapace breccias developed either upon and/or immediately adjacent to thick linear rhyolitic flow ridges formed along syn-rift and/or transform growth faults within an extensional volcanic environment developed along the western margin of ancestral North America during the Devonian–Mississippian period. Significant base and precious metal VMS discharge sites are normally associated with thick footwall alteration zones and are often localized at rift and nearly orthogonal transform fault intersections marked by higher heat flow and permeability.

9 Exploration

In 2011, Chieftain carried out a detailed drilling program focused at upgrading some of the inferred mineral resource to the indicated category. In total, 10 surface holes and 50 underground diamond drill holes totalling 22,630 m were completed. Overall, the drilling program was successful in upgrading some of the inferred resources to the indicated category.

10 Drilling

The first diamond drill campaign carried out at the Tulsequah Chief property was in the early 1940s. Drilling programs were ongoing from then until the mine closed in 1957. The property remained inactive until 1987 when a small drilling program, five holes totalling 3,500 m, was carried out. During the period from 1987 to 2005, a total of 80,843 m was drilled in 167 holes (Table 10-1). These holes generally range in length from 134 m to 1,000 m. There are 807 holes in the Tulsequah database, 665 were used to generate the resource estimate. The other 142 holes were exploration drill holes drilled along strike of the resource area, but not within the deposit area.

Table 10-1: Summary of Drilling Campaigns at Tulsequah Chief

	Underground		Surface		
Year	**No. holes**	**(m)**	**No. holes**	**(m)**	**Total (m)**
1940-1957	515	27047	20	3138	30185
1987-1989	21	7935	8	4012	11947
1990-1994	51	24812	10	3512	28324
2003-2004	75	39503	2	1069	40572
2006-2007	10	2802	35	11006	13808
2011	50	18625	10	4005	22630
Total	**722**	**120724**	**85**	**26742**	**147466**

Collar locations for all the surface holes and all underground holes drilled since 1987 were surveyed in Universal Transverse Mercator (UTM) coordinates relative to established mine survey stations with a total station electronic distance measuring (EDM) system. Down hole surveys were done using the Maxi Bore system for holes drilled after 1994. Light Log system was used for the 1990 to 1994 holes and Sperry Sun instrument was used on holes drilled between 1987 and 1989. Holes surveyed with the Maxi Bore and Light Log were also surveyed by Sperry Sun or EZ Shot as a backup. Drill core was moved by diesel locomotive for the underground holes and by helicopter for surface holes, to the Tulsequah Chief camp where it was logged. For all core, RQD was measured, and geological logging captured lithological, alteration and structural information. Data was entered in to GEMS, which utilizes a Microsoft Access database and allows for 3D visualization of drill holes. All drill core drilled since 1993 has been photographed prior to splitting. Core is cross-piled and racked at the Tulsequah Chief site.

Core logging procedures were reviewed at site in 2003 and 2004 by Independent Qualified Persons; the author observed core logging procedures for the 2006 drilling program during the field visit in May, and in September 2006 and October 2011. The drill core was found to be well handled and maintained. Data collection was competently done with the logging information recorded on logging sheets and transferred in electronic format every night. Core recovery in the mineralized units was excellent, usually between 95% and 100%. Overall, the Redfern and Chieftain drill programs and data capture were performed in a competent manner.

Overall, the drilling over the main Tulsequah Chief deposit area is at a nominal 30 m spacing. SRK is of the opinion that this drilling density is appropriate for the estimation of mineral resources for this type of mineralization.

11 Sample Preparation, Analyses & Security

Drill core samples were collected in areas of mineralization or alteration as determined by the geologist logging the core. The core was marked with grease markers and sample numbers are inserted in the box and recorded on the logging sheet. Altered zones containing low levels of lead-zinc mineralization or pyritic mineralization were also sampled, as weak mineralization can be important to the overall geological interpretation and precious metals values can be significant in areas with little base-metal mineralization. Sample lengths were typically 1 m to 1.5 m, with all samples honouring lithological boundaries. All drill cores were geologically logged prior to the collection of samples. The majority of samples were cut with a diamond saw, although some of the 1987 and 1988 core was split with a manual core splitter. Half of the core was placed in labelled polyethylene sample bags for analysis with the other half returned to the core box. Core recoveries were generally good and the samples collected were representative of the mineralization present in drill core.

11.1 Sample Preparation & Analyses

Sampling was done by Redfern for the 2004 to 2007 drilling campaigns and by Chieftain staff for the 2011 drilling following the procedures implemented by Redfern.

Core samples were collected by sawing the core along its length and collecting one half of the core for assay. Sample bags were sealed with flagging tape, placed into rice bags which were sealed with tie straps, and transported by helicopter or fixed-wing aircraft to Atlin and shipped by bonded carrier to Whitehorse and then via surface transport to Eco Tech Laboratory Ltd. (Eco Tech) situated in Kamloops.

Eco Tech is registered for ISO 9001:2008 by KIWA International (TGA-ZM-13-96-00) for the “provision of assay, geochemical and environmental analytical services.” Eco Tech also Participates in the annual Canadian Certified Reference Materials Project (CCRMP) and Geostats Pty bi-annual round robin testing programs. The laboratory operates an extensive quality control/quality assurance program, which covers all stages of the analytical process from sample preparation through to sample digestion and instrumental finish and reporting.

At Eco Tech, samples were prepared using a standard rock preparation procedure (drying, weighing, crushing, splitting and pulverization).

Samples were first catalogued and logged into the sample-tracking database. The samples are transferred into a drying oven and dried. Rock samples are crushed on a Terminator jaw crusher to -10 mesh ensuring that 70% passes through a Tyler 10 mesh screen. Every 35 samples a re-split is taken using a riffle splitter to be tested to ensure the homogeneity of the crushed material. A 250 gram subsample of the crushed material is pulverized on a ring mill pulverizer ensuring that 95% passes through a -150 mesh screen. The sub sample is rolled, homogenized and bagged in a pre-numbered bag. A barren gravel blank is prepared before each job in the sample prep to be analyzed for trace contamination along with the processed samples.

Gold was assayed by fire assays. The following procedures were provided by Eco Tech:

- A 30 g sample size is fire assayed along with certified reference materials using appropriate fluxes.
- The flux used is a pre-mix that is purchased from Anachemia. It contains Cookson Granular Litharge and is free of gold and silver. Flux weight per fusion is 150 g.
- Purified silver nitrate or inquarts for the necessary silver addition is used for inquartation. The resultant doré bead is parted and then digested with nitric acid followed by hydrochloric acid solutions and then analyzed on an atomic absorption instrument (Perkin Elmer/Thermo S-Series Atomic absorption (AA) instrument. Gold detection limit on AA is 0.03-100 g/t.
- Any gold samples over 100g/t will be run using a gravimetric analysis protocol.
- Appropriate certified reference material and repeat/re-split quality control (QC) samples accompany the samples on the data sheet for quality control assessment.

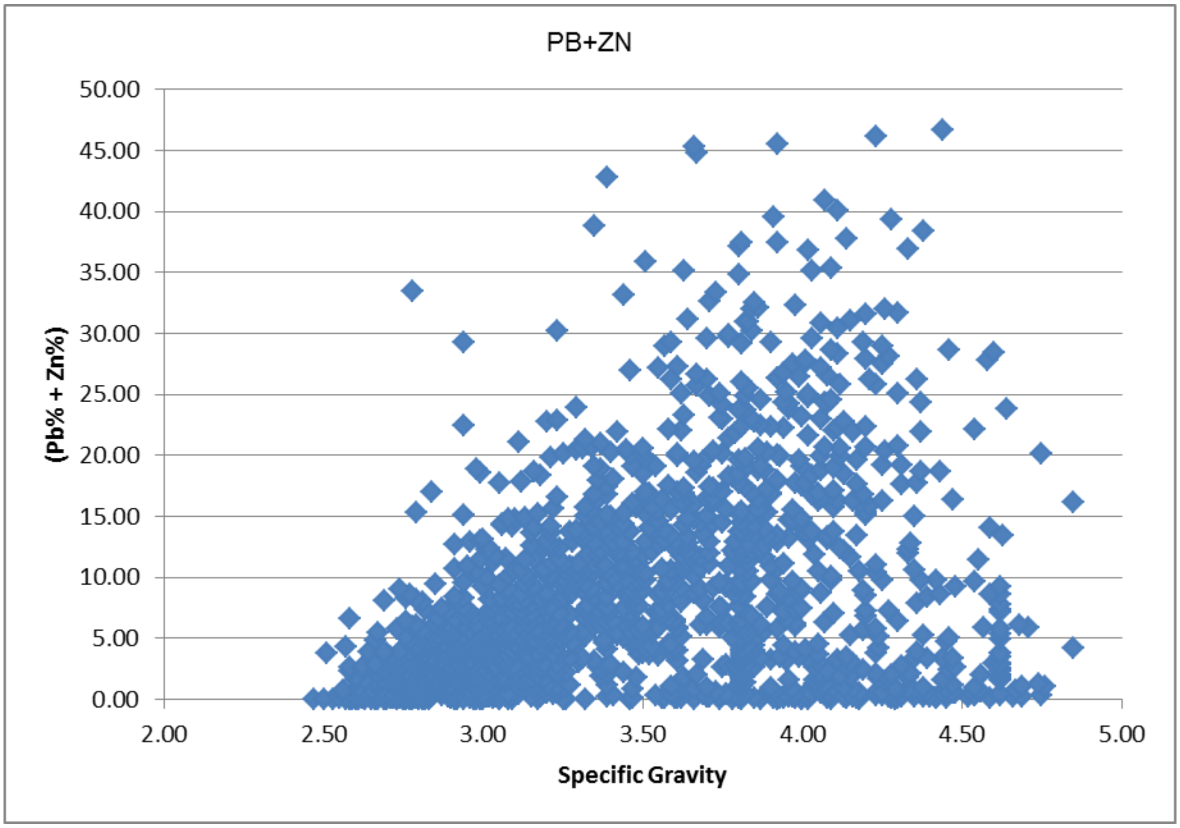
Copper, lead and zinc contents were determined using an aqua regia digestion and inductively coupled plasma atomic emission spectroscopy (ICP-AES; ME-ICP61) on a 0.5 gram subsample. The sample is digested with a 3:1:2 ($HCl:HN0_3:H_20$) solution in a water bath at 95°C. The sample is then diluted to 10 ml with water. All solutions used during the digestion process contain beryllium, which acts as an internal standard for the ICP run. The sample is analysed on a Thermo IRIS Intrepid II XSP ICP unit. Certified reference material is used to check the performance of the machine and to ensure that proper digestion occurred in the wet lab. QC samples are run along with the client samples to ensure no machine drift occurred or instrumentation issues occurred during the run procedure. Repeat samples (every batch of 10 or less) and re-splits (every batch of 35 or less) are also run to ensure proper weighing and digestion occurred. Results are collated by computer and are printed along with accompanying quality control data (repeats, re-splits, and standards). Any of the base metal elements (Ag, Cu, Pb, Zn) that are over limit (>1.0%) are run as an ore grade assay.

11.2 Specific Gravity Data

Bulk density determinations were carried out on all samples shipped for assays between 2004 and 2011. In total 6,358 specific gravity determinations have been carried out on drill core. Of the total specific gravity determinations, 1,776 samples were from the mineralized intervals. The average of all mineralized specific gravity measurement is 3.44, which is slightly lower than the 3.5 that was used when the mine was in operation in 1957.

While there is a broad correlation between grade (Pb + Zn) and specific gravity, there are significant high specific gravity values associated with little or no Pb-Zn values (Figure 11-1). For this reason, SRK decided not to weight the assays with the specific gravity for grade interpolation.

Figure 11-1: Plot of Specific Gravity Values against (Pb + Zn) in Percent



A total of 1,180 mineralized samples, mainly representing samples from within the existing mine workings had no specific gravity data. These samples were assigned a specific gravity of 3.5 if the combined lead plus zinc content was greater than 2% (which was the specific gravity used by Cominco during mine operation); or a specific gravity of 2.7 if the Pb + Zn value was less or equal to 2.0% (the average specific gravity of unmineralized waste).

11.3 Quality Assurance & Quality Control (QA/QC) Programs

Redfern had established an extensive quality control program consisting of sample blanks, standards, and duplicates to ensure the quality of the assay data. Control samples accounted for approximately 10% of all samples collected and assayed. All samples were collected by Redfern employees, sample preparation and analyses were carried out by independent laboratories. The same procedures were followed by Chieftain in 2011.

The control samples were inserted into the sample sequence by selecting ten random numbers between one and one hundred. Three of the random numbers correspond to sample blanks, three to duplicate samples and four to sample standards (including two high-grade and two low-grade standards). For every 100 samples, the last two digits in the sample number correspond to the type of control sample inserted. International Metallurgical and Environmental Inc. of Kelowna, BC (IME) supplied two base metal standards, one high-grade standard and one low-grade standard. These were made from material left over from a bulk sample collected from the Tulsequah Chief deposit in 1996. No variance data were provided for the base metal standards. Early in the 2004 program, the standard material left over from the 2003 drilling program was used. When that was exhausted, a new 2004 standard was obtained. Gold standards were supplied by WCM Mineral Ltd of Burnaby, BC. A standard sample was made up of one packet of the base metal standard and one packet of the gold standard. Blanks consisted of sawn sections of drill core from a barren quartz-feldspar porphyry dyke that is commonly cut by drill holes. In the case of duplicates, one-half of the original core was submitted for analysis; the remaining half was split in half again and submitted as a duplicate.

No systematic quality control was carried out prior to the 2003 drill campaign other than the standard procedures offered by the assay laboratory carrying out the assays.

11.4 SRK Comments

In the opinion of SRK the sampling preparation, security and analytical procedures used by Redfern and Chieftain are consistent with generally accepted industry best practices and are therefore adequate.

12 Data Verification

12.1 Verifications by Redfern & Chieftain

A total of 1,750 samples were collected during the 2004 drilling and 278 samples were collected from the 2006 drilling program. In addition to the samples collected for assaying for the 2004-06 drilling programs, 56 blank samples, 61 duplicates samples and 79 standard samples were inserted in the shipments sent to the lab for analysis.

The 2011 QA/QC program conducted by Chieftain involved the insertion of 10% standards, blanks and duplicates into the sample shipment stream. At the conclusion of the program, 5% of the sample pulps were submitted to ACME Labs for third-party checks. These results were monitored in real time with the lab requested to reanalyze and explain discrepancies.

Twenty sample results out of the 3210 samples submitted were investigated. The majority of these were related to procedural errors caused by the change in principal laboratory halfway through the program.

12.1.1 Blank Material

The blank sample results are acceptable and pass the QC check. A small number are above the background levels; most of these results had high-grade samples prepared immediately prior indicating a slight contamination at the laboratory crushing. However, the variation from the background is less than 1% and well below ore grade values, which is acceptable for resource estimation.

12.1.2 Standard Reference Material (SRM)

The results of the standard reference material pass the QC. All the low-grade Au samples are within two standard deviations (SD) of the expected SRM value. All the high-grade Au standards analyzed at Eco Tech are above 2SD but within 3SD of the expected SRM value. All the base metal low-grade standards pass the QC. The base metal high-grade standards pass the QC, with one result outstanding.

12.1.3 Field Duplicates

Field duplicates test the precision of the analysis. Field duplicates were prepared by selecting ¼ core split of the original sample. The precision was evaluated graphically by plotting the original and duplicate assays on 1:1 plots. The graphical results are acceptable; a few results show more variability generally at the lower end of the grade range. For gold duplicates, an average co-efficient of variation is less than 25. For base metals field duplicates, an average coefficient of variation is less than 15. Both the Assay and ICP results pass the QC, with the ICP results slightly less precise.

12.1.4 Check Assays

A random 5% of the 2004 sample population was submitted to ACME Analytical in Vancouver for check assay. ACME Analytical is an ISO certified laboratory, SRK is unaware of the certification that ACME held in 2004. The assay techniques differ slightly in that ACME used a 29.2 g subsample and fire assays for Au and Ag, while Eco Tech used a 30 g subsample and fire assays for Au only. Fire Assay Au (+Ag) and Induced Coupled Plasma (ICP) were completed first, with any sample that returned values of >1 g/t Au, >30 g/t Ag, or >10,000 ppm for Cu, Pb, or Zinc being wet assayed and subjected to a specific gravity determination. For the entire population (n=80), ACME's gold assays were 21% higher than Eco Techs'. For higher-grade gold samples (>1 g/t Au), ACME Fire Assay Au values were 29% higher (n=28). The reason for the higher gold grades obtained in the check samples is unclear; however, the presence of coarse gold may be part of the explanation or possibly ACME results are biased on the high side. Given that the standards were not included in the batch of samples sent to ACME, it is not possible to determine the cause of the discrepancy between the two laboratories.

The results of the 2011 ACME Lab checks are better. Only one gold sample displays poor correlation, sample 4458 (original: 2.36 Au g/t; ACME: 13.9 Au g/t); the three subsequent Au assays in this hole are 12.9; 8.7; and 57.5 Au g/t, this discrepancy can be clearly be identified as an expression of the gold nugget effect. Because the vast majority of the samples have good correlation, the Eco Tech and ALS labs are considered to be performing at a satisfactory standard.

12.2 Metallic Screen Assays

Three entire massive sulphide intervals were metallic screened as part of the 2004 program; TCU04104, TCU04106, and TCU04109, plus one selected sample from hole TCU04113, from 84 samples (~4.8% of the samples cut in the 2004 program). All of these holes were in the H zone. The holes were selected to be a high-grade hole (TCU04109), a low-grade hole (TCU04104) and an average hole (TCU04106). The single sample from TCU04113 had visible gold. A simple average of metallic/fire assay gives an increase of 13% on the gold grade using metallic screen assay. Percentile plots were made to establish how the change in gold grade manifested itself in the population.

The results show that for assays <6 g/t Au, screened metallic assays consistently returned a higher value than fire assay, and above 6 g/t Au fire assay returns a higher value than screened metallic assays. This is likely due to the size of the subsample used in the different assay techniques. The screened metallic assays use a larger subsample and is likely more representative of the true grade.

12.3 Verifications by SRK

The author has reviewed the sampling procedures and sample intervals for the Tulsequah Chief 2004 to 2011 drilling and concluded that the sampling quality and methodologies utilized were appropriate for this type of deposit. Data collection was competently done with the logging information recorded on logging sheets and transferred into electronic format every night. Core recovery in the mineralized units was excellent, usually between 95% and 100%. Overall, the Redfern and Chieftain drill programs and data capture were performed in a competent manner.

The samples collected are representative of the mineralization and no apparent biases were observed in the sampling protocols or the samples collected.

As a test of assay data integrity, the data used to estimate the Tulsequah Chief mineral resource were verified with a random comparison of 20% of the database records against the original electronic assay certificates. Only three discrepancies were found and corrected. Collar coordinates from drill logs were checked against the database entries. No discrepancies were observed. The author concluded that the assay and survey database is sufficiently free of error to be adequate for resource and reserve estimation of the Tulsequah Chief deposit.

During the site visits, the author did not collect verification samples as samples of the mineralization had been taken by previous independent Qualified Persons and samples of the mineralization were taken for metallurgical testing. The author did examine several drill intersections and verified that base metal mineralization was present in drill core. The author also visited the underground workings on the 5400 and 6500 level and examined underground drill setups as well as broken mineralization in-situ in several stopes.

12.4 SRK Comments

The author has analyzed the assay results of the duplicates, blanks and standards and has concluded that the QA/QC program implemented by Redfern and Chieftain was adequate and that the assay database is sufficiently accurate and precise for resource estimation.

13 Mineral Processing & Metallurgical Testing

13.1 Introduction

The metallurgical test program for the Tulsequah ores was initiated with the objective of determining a treatment route that would result in saleable concentrates at economic recoveries.

Production records from 1953, as well as work conducted by Beattie in 1993 through 1995, IMEL in 1996 and others (Wardrop, March 2007), showed that the ore could be effectively processed. However, attempts to replicate some of the work from the early 1990s did not achieve the high levels of concentrate grades and recoveries reported in that work. The lack of consistency between the various historical figures contributed to Chieftain's decision to conduct its own sampling and test program. It should be noted that at that time arsenic was not considered an issue (Watters Consulting Pty Ltd, September 2011).

Consequently, a laboratory in Australia, ALS AMMTEC at Burnie in Tasmania was chosen to do the work. This choice was made based on their proven expertise, their close association with the operations at Rosebery Tasmania, which is an analogous deposit to Tulsequah Chief, as well as their ability to carry out test programs relatively quickly when compared to other laboratories.

Extensive mineralogy was conducted as a precursor to establish the likely optimum for liberation and therefore grinding protocols.

13.2 Historical Context

13.2.1 Previous Operations

Ore was extracted and treated from the upper levels of the Tulsequah Chief orebody in the 1950s and 1960s. The ore from the nearby Big Bull mine was also treated in the same facility. The processing facilities were located on the western side of the Tulsequah River and connected to the mine by a bridge.

The metallurgical results obtained were only moderate, but in retrospect did show that the minerals were responsive to selective flotation (Table 13-1).

Table 13-1: Metallurgical Test Results Jan –Nov 1953 (CMM Bulletin, Sept 1954)

Product	Grade	Recovery
Gravity conc.	38.8 oz/t Au	7.5%Au
Cu conc.	17.6%Cu, 3.5%Pb, 12.7%Zn	76.2%Cu
Pb conc.	44.4%Pb, 7.9%Cu, 9.7%Zn	69.8%Pb
Zn conc.	55.8%Zn, 0.7%Cu, 0.8%Pb	73.3% Zn

Gold recovery to Gravity plus copper and lead concentrates was reported as 81.3%.

13.2.2 Previous Testwork

Four distinct phases of testwork have taken place.

13.2.2.1 Period of 1992-1993

The samples used were blended quarter cores from the 1991/92 drilling program, and composited to represent the average grade of the deposit. Two further composites were also made up with low and high copper values. The metallurgical testwork was conducted by Beattie Consulting Ltd in March 1992.

The tests demonstrated that the ore responded well to selective sequential flotation in conjunction with some gravity separation of gold.

A system was developed to recover the lead minerals followed by copper and then zinc. The results showed a significant improvement over the reported results from operation and were based on the average ore grade samples.

Table 13-2: Test Results, Beattie Consulting Ltd, March 1992

Product	Grade	Recovery
Gravity conc.	38.8 oz/t Au	7.5%Au
Cu conc.	17.6%Cu, 3.5%Pb, 12.7%Zn	76.2%Cu
Pb conc.	44.4%Pb, 7.9%Cu, 9.7%Zn	69.8%Pb
Zn conc.	55.8%Zn, 0.7%Cu, 0.8%Pb	73.3% Zn

Gold recovery (i.e., gravity+Cu+Pb concentrates) reported as 83.8%. Silver recovery (i.e., gravity+Cu+Pb concentrates) reported as 77.2%

It should be noted that as the lead minerals were floated first, the bulk of the gold reported to that concentrate indicating that the gold was present as free particles. It is also worthy of note that the best results were obtained at the finer grind of 80% minus 60 µm. Some semi-quantitative mineralogy was conducted post testing and displayed much the same particle characteristics that have been determined in the current (2011/2012) program.

13.2.2.2 Period of 1994-1995

The samples for metallurgical testwork were obtained from quartered core from the 1994/95 drilling program. Three composites were made: a base composite at typical ore reserve grades, as well as a high lead and low lead composite. Brenda Process Technology conducted this work and focused on producing a copper lead bulk concentrate (the rationale behind this strategy is not explained and seems to be at odds with the Beattie work, which indicates good liberation and selective mineral recovery).

Brenda reported that separation of the bulk into two separate concentrates was not successful.

The results were summarized as presented in Table 13-3.

Table 13-3: Test Results, Brenda Process Technology, 1994-1995

Product	Grade	Recovery
Gravity conc.	38.8 oz/t Au	7.5%Au
Cu conc.	17.6%Cu, 3.5%Pb, 12.7%Zn	76.2%Cu
Pb conc.	44.4%Pb, 7.9%Cu, 9.7%Zn	69.8%Pb
Zn conc.	55.8%Zn, 0.7%Cu, 0.8%Pb	73.3% Zn

Gold recovery to gravity+Cu/Pb concentrates reported as 80.5%. Silver recovery to gravity+Cu/Pb concentrates reported as 73.7%.

A bond work index was reported as 12.6 kWh/t on the base composite at a grind size of P_{80} = 52 µm.

No explanation is noted for not following up the Beattie work and although the headline recoveries are good, the reported selectivity is very poor.

There is scant reference to mineralogy and the choice of a relatively coarse primary grind meant that many composite particles were recovered which necessitated the use of stronger, less selective, reagents.

13.2.2.3 Year 1996

Six 250 kg samples were collected from the 5200 portal level and composited based on predicted grades of the deposit.

The composite was divided between three different testing laboratories, IME, Hazen and G&T for independent testing.

IME

Eighteen open cycle and three locked cycle tests were performed.

The basic flowsheet produced a bulk Cu/Pb concentrate followed by zinc and pyrite flotation. LC10, the best reported locked cycle test, gave the results shown in Table 13-4.

Table 13-4: Test Results, IME, 1996

Product	Grade	Recovery
Gravity	7,869 g/t Au	32.4% Au
Cu/Pb conc.	21.0% Cu, 17.0% Pb, 7,8% Zn	91.3,Cu, 95.4% Pb
Zn conc.	56.5% Zn, 0.28% Cu , 0.28% Pb	80.6% Zn

Gold recovery to gravity+Cu/Pb concentrates reported as 81.0%. Silver recovery to gravity+Cu/Pb concentrates reported as 84.2%. Arsenic reported in Cu/Pb concentrates as 1.2% Antimony reported in Cu/Pb concentrates as 0.5%. Mercury reported in zinc concentrates as 200 ppm. Bond work index reported as 10.8kWh/t at P_{80} of 59 µm.

Further work was conducted on copper lead separation. Five tests were reported, as shown in Table 13-5.

Table 13-5: Test Results, Copper & Lead Separation

	Copper Concentrate		Lead Concentrate	
Test	**Grade**	**Recovery**	**Grade**	**Recovery**
185	29.6%Cu, 7.7%Pb	74.9%Cu	52.8%Pb, 8.3%Cu	46.2%Pb
187	29.3%Cu, 8.3%Pb	87.1%Cu	57.1%Pb, 3.0%Cu	50.7%Pb
189	26.7%Cu, 11.6%Pb	83.7%Cu	69.0%Pb, 0.7%Cu	30.9%Pb
190	28.1%Cu, 6.5%Pb	76.8%Cu	64.0%Pb, 0.7%Cu	46.2%Pb
191	29.9%Cu, 10.5%Pb	79.6%Cu	78.1%Pb, 1.0%Cu	38.9%Pb

The above results illustrate the difficulty in separating already collected mineral even with good liberation.

G&T

The laboratory conducted mineralogical work, which accords well with the current work. Of note is the detection of a small quantity of chalcocite indicating that the sample had suffered weathering type alteration.

A sequential flotation regime was chosen, recognizing the difficulties that would be expected in separating the copper and lead minerals. The reported results are shown in Table 13-6.

Table 13-6: Test Results, G&T

Product	Grade	Recovery
Gravity	59 g/t	36.0%Au
Cu conc.	27.8%Cu, 4.1%Pb, 7.9%Zn	78.0%Cu
Pb conc.	49.4%Pb, 8.2%Cu, 8.3%Zn	59.0%Pb
Zn conc.	57.6%Zn, 1.3%Cu, 1.3%Pb	84.0%Zn

Gold recovery to gravity+Cu+Pb concentrates reported as 68.0%. Silver recovery to gravity+ Cu+Pb concentrates reported as 64.0%. Arsenic reported in Cu concentrates as 1.5%. Antimony reported in Cu concentrates as 0.5%. Mercury reported in Zn concentrates as 75 ppm.

Hazen

Work conducted by AR MacPherson for Hazen on grindability gave the results shown in Table 13-7.

Table 13-7: Test Results, Hazen

Index	Result
SAG work index	7.7 kWh/t
Bond rod work index	9.3 kWh/t
Bond ball work index 100 µm P_{80}	9.8-10.2 kWh/t
Bond impact work index	7.7 kWh/t
Bond abrasion index	0.0461

13.2.2.4 Year 2006

Two composites were prepared. One was made from the quartered core as well as assay rejects from the 2006 drilling program for metallurgical testing. The second was a larger scale sample generated from three samples collected from the 5400 portal level for the purposes of dense media separation testing.

Flotation

Flotation tests using the bulk copper/lead flowsheet were conducted on quartered core and it is stated that these produced similar results to that reported in 1995/96, but the tests on assay rejects could not reproduce these results. Possible oxidation or other contamination was suggested as an explanation for this discrepancy.

DMS

Two sets of tests were conducted, one on footwall and hangingwall material and one on ore grade material.

Tests showed that between 46% and 79% of the hangingwall and footwall material could be rejected as sub economic, but only at a relatively high specific gravity split of 2.8. DMS plants generally operate at a specific gravity of 2.6. A split of 2.8 is possible, but is at the limit of effectiveness for ferrosilicon suspensions and beyond the limit of magnetite slurries.

It is now clear from current mineralogy that the presence of significant barite in the gangue matrix, that low density pre-concentration is not possible. The test on the ore grade material, which included some drill core, showed that at a specific gravity 2.8 only some 10% could be rejected at a contained metal loss of 1% to 2%. At a specific gravity of 2.9, this doubled to approximately 20% with a metal loss of 2.3% to 5.5%. The relatively low weight rejected cannot justify the capital and operating costs of a plant.

Other

The 2007 technical report also included a table of projected metallurgical performance, which assumed enhanced performance on the bulk Cu/Pb flowsheet. This is shown in Table 13-8 for reference only.

Table 13-8: Projected Performance

Product	Grade	Recovery
Gravity	6552g/t Au	28.0%Au
Cu conc.	22.5%Cu, 9.7%Pb, 7.9%Zn	88.5%Cu
Pb conc.	53.0%Pb, 3.1%Cu, 15.0%Zn	44.3%Pb
Zn conc.	59.0%Zn ,0.25%Cu, 0.21%Pb	87.4%Zn

Gold recovery: gravity +Cu+Pb concentrates predicted as 82.0%. Silver recovery: gravity+Cu+Pb concentrates predicted as 76.9%.

It is important to note that historical testwork has almost no consideration (other than Hazen's work) of the crushing and grinding properties of the ore. The laboratories appear to have used conventional grind establishment techniques to produce the target P_{80} for the flotation work. As such, there has been little analysis of alternate beneficiation options.

13.3 Current Metallurgical Sampling

13.3.1 Introduction

The first consignment of core samples was received at the Burnie laboratory in July 2011. The handling and preparation of samples for flotation testing is detailed in the report by Watters Consulting Pty Ltd (August 2011); the process is summarized below:

- Metallurgical test holes were planned and oriented within the orebody lenses to provide fresh drill core sufficient for the proposed work program. Targets were selected within a GEMS block model of the orebody, and the model data used to predict grades and intercepts.
- On completion of the drilling, the cores from the mineralized zones were split with a core saw and half of the core bagged and set aside for the proposed metallurgical test program pending completion of assays.
- The remaining half core was split (quartered core) and sent to the assay lab for assays and specific gravity determinations.
- The assay results were compared with the expected model results and good correlation was obtained.
- The core intervals were then selected to be representative of both ore types and of grade. Very high- and very low-grade samples were not included. Immediate zone wall-rock samples were included as being representative of typical dilution.
- The metallurgical shipment was then compiled on an individual sample basis and prepared for shipping to the lab with recommended compositing instructions.

The second and third consignments of samples, from the lower ore zones were selected and processed in the same manner (Chieftain Inc. Brett Armstrong, May 2011 and August 2011).

13.3.2 Flotation Testwork Samples

The three suites of drill core samples that were sent to the Burnie laboratory for testwork can be summarized as shown in Table 13-9.

Table 13-9: Sample Description

Chieftain Sample	Description	Composite Number	New name	Predicted Assays %Cu	%Pb	%Zn	ppm As	Locked Cycle Tests Conducted	Locked Cycle Tests Proposed
1	Upper Zone	1	Upper Zone	1.35	1.20	6.91	595	LC01, LC02	LC05
	Half core total	2							
		3							
2	Lower Zone	4	Lower Zone: High Arsenic	1.15	1.32	6.03	840	LC03	LC04
	Half core total	(Hi Arsenic: lower ore body average)							
3	Lower Zone	5	Lower Zone: Low Arsenic	0.97	1.32	5.64	482		LC06
	Quarter core	(Low Arsenic – excluding							
		DDH 1170)							

The first sample was composed of drill cores from upper areas of the main lenses and is representative of around one third of the ore resource.

The second sample came from the lower areas of the main lenses. When it was realized that unexpectedly high arsenic levels were sourced from one drill hole, a second suite (third sample) was prepared so that the offending intercepts might be excluded (and included at a later stage) to simulate an intermediate level of arsenic in the composites to be tested.

It was during this process, that discrepancies between the arsenic assays supplied and those obtained at Burnie were found. Check assays were carried out in Canada, but repeated flotation tests and head assays pointed to an underestimation of the arsenic content which seems to be due to the presence of a very pronounced nugget effect in the arsenic distribution through the samples (using original assays) is shown in Chieftain (2012). It is worth noting that of all metal assays, arsenic is one of the most difficult to replicate. Consideration will be required to better map the high arsenic hotspots as elimination or campaigning of these areas will pay considerable benefits in controlling copper concentrate quality.

13.3.3 Physical Parameter Testwork Samples

Tests were conducted for grindability, abrasion and compressive strength properties.

These were accommodated as follows:

- abrasion (ALS Burnie Research Laboratories, September 2011) – composite 1, flotation test sample

- grindability (ALS Burnie Research Laboratories, August 2011) – composite 1, flotation test sample
- unconfined compressive strength (ALS AMMTEC, Malaga, December 2011) – four whole core samples (see Table 13-10).

Table 13-10: UCS Samples

Sample Identity
TCU 11192 (120.60 – 120.70 m)
TCU 11194 (114.15 – 114.25 m)
TCU 11195 (112.06 – 112.06 m)
TCU 11198 (124.60 – 124.70 m)

13.3.4 Mineralogical Samples

Successful metallurgical testwork depends on a robust understanding of the mineral assemblages in the ore. In that context, several mineralogical analyses have been conducted on Tulsequah ore samples and flotation products. These can be found in MODA Pty Ltd Reports 1-6 from July 2011 to December 2011 and are listed as shown in Table 13-11.

Table 13-11: Mineralogical Samples

MODA report No.	Sample	Examination	Date
1	1	Mineragraphy	July 2011
2	1	Mineralogy	August 2011
3	2	Mineralogy	November 2011
4	2	Mineragraphy	December 2011
5	Float Products Tennantite	Mineralogy	December 2011
6	1 and 4 Tennantite	Microprobe	December 2011

13.4 Ore Characterization

13.4.1 Mineralogy

When sample 1 was received, a consultant mineralogist (Gary MacArthur, MODA) inspected the cores and identified five general ore types based on the host rock as follows:

- massive sulphide barite
- massive sulphide coarse sphalerite
- massive sulphide recrystallized pyrite
- massive sulphide banded
- mixed low-grade alteration.

Two samples of each were selected for thin section examination. These were prepared as polished sections with the remainder of each core specimen returned to the original sample. Examination of these samples was completed and reports issued (MODA Pty Ltd, Reports 1 and 2, July and August 2011).

The reports need to be fully considered to gain an adequate understanding of the mineralogy. The key points are:

- The examination of spot samples reflects the various ore type classifications within the block model.
- The major portion of the valuable minerals occurs as coarse grains or in simple associations with other minerals. Such grains appear to be easily liberated during grinding to a P_{80} of 40 to 50 µm.
- Much of the remaining occurrences are as very fine grains, evenly dispersed within the host matrix and should not be considered recoverable. Effective liberation would require a grind of something less than 20 µm, which is impractical given the proportion of the ore within this categorization.
- Nothing was observed that was at odds with earlier mineralogical work.

This preliminary review of the ores suggested that good metallurgical separation should be possible. It should be noted that areas of lower sulphide content displayed no difference in general characteristics-dilutive gangue material should not influence performance to any material extent.

When sample 2 was received, the same process was applied as per sample 1 and the following differences noted (MODA Pty Ltd, Reports 3 and 4, November and December 2011):

- The sphalerite showed less association with galena.
- There is a lower galena content.
- The galena is less associated with sphalerite and more associated with gangue.
- There is a higher chalcopyrite content.
- Chalcopyrite is less liberated and more associated with sphalerite.
- Tennantite is more liberated and less associated with gangue.

It was unnecessary to carry out any mineralogical or mineragraphic examination of sample 3, as it is duplicate core to sample 2.

With the flotation test program aiming for the extraction of arsenic into a separate concentrate (see Section 5.4 below), the tennantite mineral was targeted for detailed analysis to identify the species and to assess variation in mineral content. Two flotation products, copper cleaner 2 and lead float gold scavenger concentrate, were examined MODA Pty Ltd, Report 5 (December 2011). This showed that the tennantite:

- In the copper cleaner concentrate is well liberated with minor sphalerite binaries.
- In the gold rougher is less liberated with pyrite and sphalerite binaries.
- Both samples are remarkably uniform in colour, suggesting a uniform composition.

Testwork analyses had indicated that metal ratios between arsenic and other metals were not consistent through the floats, which contradict point 3 above, so it was decided to subject a number of tennantite grains from each product for microprobe analysis at the University of Tasmania (MODA Pty Ltd, Report 6, December).

There was no discernible difference between the grains from each product, but the probe analyses showed that the tennantite is the sole source of arsenic and a significant source of silver in the products, and that there is considerable variation in the minor metal composition of individual grains, NB, As:Sb ratios.

Table 13-12: Sample 1 Tennantite Analyses

Grain	%Cd	%S	%Zn	%Cu	%Fe	%Ag	%As	%Sb	%As:%Sb
Tn1	0.06	27.6	7.1	40.8	1.3	1.15	14.8	7.17	2.10
Tn2	0.09	27.4	7.0	40.0	1.5	1.27	12.7	9.97	1.30
Tn3	0.07	27.6	6.9	40.5	1.4	1.39	13.6	8.71	1.60
Tn4	0.03	27.8	4.6	40.7	3.6	0.74	13.9	8.61	1.61
Tn5	0.01	27.9	4.1	40.5	4.1	0.42	12.8	10.25	1.25
Tn6	0.04	26.1	6.7	37.6	1.2	2.09	5.2	21.2	0.25
Tn7	0.04	27.3	4.6	39.6	3.2	2.15	12.1	11.04	1.10
Tn8	0.06	26.0	6.4	36.0	2.0	3.69	5.6	20.29	0.28
Tn9	0.06	27.7	4.7	40.2	3.5	0.90	12.6	10.33	1.22
Tn10	0.05	28.3	8.3	43.1	0.2	0.02	18.2	1.79	10.17
Mean	**0.05**	**27.4**	**6.0**	**39.9**	**2.2**	**1.38**	**12.1**	**10.94**	**1.10**

Table 13-13: Sample 2 Tennantite Analyses

Grain	%Cd	%S	%Zn	%Cu	%Fe	%Ag	%As	%Sb	%As:%Sb
Tn1	0.01	28.6	5.2	43.1	3.2	0.13	19.2	0.63	30.5
Tn2	0.04	28.8	5.3	42.8	3.1	0.09	19.1	0.80	23.9
Tn3	0.04	27.9	5.5	41.6	2.6	0.75	14.8	6.76	2.2
Tn4	0.04	28.7	5.4	43.0	3.0	0.08	19.0	0.80	23.8
Tn5	0.02	28.7	4.4	42.1	3.8	0.88	18.3	1.88	9.7
Tn6	0.07	27.6	6.2	40.5	2.1	1.62	14.0	7.97	1.8
Tn7	0.03	28.7	5.2	42.9	3.2	0.07	19.1	0.82	23.3
Tn8	0.04	27.9	5.2	40.8	2.9	1.45	14.4	7.30	2.0
Tn9	0.05	28.7	4.8	43.2	3.6	0.09	19.0	0.60	31.1
Tn10	0.03	28.6	5.1	43.2	3.3	0.13	18.7	1.10	17.0
Mean	**0.04**	**28.4**	**5.2**	**42.3**	**3.1**	**0.53**	**17.6**	**2.87**	**6.1**

Microprobe analysis is not a precise analytical tool, but the figures serve to demonstrate the wide range of minor elements within the tennantite mineral.

General comments that can be made are that tennantite in the deeper half of the orebody is lower in silver, higher in arsenic and considerably lower in antimony.

13.4.2 Whole Ore Multi-Element Analysis – Sample 1

Table 13-14: Sample 1 Partial Analysis

Cu	Pb	Zn	Fe	SiO_2	Ag	As	Sb	S	C	CO_3	Au	Au	SG
XRF	XRF	XRF	XRF	XRF	AAS	AAS	AAS	Leco	Leco	Leco	Fire	Fire	
%	%	%	%	%	ppm	ppm	ppm	%	%	%	ppm	ppm	kg/l
1.28	1.11	6.27	12.5	23.4	84	702	262	18.7	0.4	1.05	2.67	2.5	3.74

An EDTA leach of the feed sample showed that 3.7% of the lead is oxidized and lead ions in solution may activate sphalerite in the copper float. Sodium metaphosphate (SMP) has been added to flotation feed as a precautionary measure to negate this effect through precipitation of any lead ions in solution.

13.5 Batch Bench Tests

Concurrent with the mineralogical examination, the remaining 59 samples from the first consignment were individually crushed to -3 mm by passing through a laboratory rolls crusher at successively finer settings to avoid the production of excessive fines (if attempted in one pass).

Initial testwork on the Tulsequah ores indicated that it was feasible to produce copper, lead, zinc and pyrite concentrates. This had been achieved with a simple flotation regime, but with market indications that arsenic levels in the copper concentrate could not exceed 0.5%, considerable effort was put into the production of both chalcopyrite (low arsenic) and tennantite (high arsenic) concentrates.

Table 13-15: Head Assay Comparisons Sample 1

	Au g/t	Ag g/t	Cu %	Pb %	Zn %	As ppm
BRL Heads (avg. 16)	2.3	89	1.3	1.05	6.71	757
BRL Composite	2.6	84	1.28	1.11	6.27	702
Predicted (Core)	2.9	99	1.35	1.2	6.91	595

Due to results achieved, as well as market considerations, the batch test program has evolved into a program in two phases. The first phase isolates the arsenic-bearing minerals into a separate product, while the second phase, based on robust performance and simple economics, focuses on maximizing copper recovery to the copper concentrate.

13.5.1 First Phase

The first sample was made up of all the ore types and has a ratio 3:1 or 75% Cp: 25% (Tn+Th). Given the inherent variability between samples, a bulk flotation strategy was rejected on the basis that it was necessary to separate the two copper minerals in the first instance. Tailoring the conditions at the outset to maximize the notional recovery of copper was seen as overkill on the chalcopyrite. It would also make it very difficult to prevent the lead mineral, galena, from floating as well.

13.5.1.1 Copper Flotation

With the tennantite and tetrahedrite minerals grading 30% to 40% Cu, they are an important contributor to the overall copper recovery and important sources of silver; however, they do bring high levels of arsenic and antimony. A weak collector float for chalcopyrite and galena does risk poorer recovery of tennantite and tetrahedrite, bearing in mind that a portion does float within the optimal parameters for chalcopyrite.

Based on the mineralogical evidence, the best approach to the test program was seen as:

- optimize conditions for chalcopyrite; permit tennantite and tetrahedrite to float to the extent possible without harming the chalcopyrite response
- apply low reagent doses of weak, more selective collector for both chalcopyrite and galena floats
- incorporate a separate tennantite, tetrahedrite float with a stronger collector to produce a separate copper concentrate via a refloat of the copper cleaner concentrate
- activate the sphalerite to conduct the zinc float.

The intention was to avoid the situation of chasing a variable compromise due to variable mineral ratios in the ore and to prevent a buildup of middlings as circulating loads.

13.5.1.2 Lead Flotation

Lead is entirely present as galena, but is the least liberated of the main minerals and suffers from micro lockings mainly with gangue and sphalerite. The fine composite nature of the galena is further support for retaining a sequential float. Mineralogical examination indicated that further lead particle liberation would only be achieved at a grind of $P_{80} \approx 20$ µm, which was considered both impractical and cost negative.

The best approach was to deal with the 'acceptably liberated' component as selectively as possible to the exclusion of the tennantite and tetrahedrite minerals. Concentrate grades were expected to be acceptable, but recovery would be lower than for either copper or zinc.

13.5.1.3 Zinc Flotation

Zinc is present mainly as a very low iron sphalerite but with minor amounts in the tennantite and tetrahedrite minerals. The sphalerite is the best liberated of the main metal minerals but as this is a volcanogenic deposit, much of the periodic table is at play and the concentrate does have an elevated mercury level at around 200 ppm. Fortunately, given that it is not possible to control, this is not excessive and all likely purchasers have mercury removal facilities in their acid plants – but will charge a small penalty.

The batch test program for sample 1 is summarized in ALS Burnie Report T0662-1 (ALS AMMTEC, Burnie, October 2011). That work established a set of conditions that were then applied to a locked cycle test (LC01) which produced the following results:

Grind size.......... P_{80} 50 µm; no regrinds
pH.......... Condition neutral; copper 9.5; lead 9.5; zinc 10.0; pyrite 5.5.
Copper roughing.......... SMBS, A9810 collector
Copper cleaning.......... two stages with SMBS, $ZnSO_4$

Lead roughing..lime, cyanide, $ZnSO_4$, A3418A collector
Lead cleaning...two stages, lime, cyanide, $ZnSO_4$, conditioning
Zinc roughing .... lime, $CuSO_4$, A7021 collector
Pyrite roughing.. PAX
Frother..MIBC in all stages

Table 13-16: Locked Cycle LC01 Metallurgical Statement

	Copper Concentrate		Lead Concentrate		Zinc Concentrate		Pyrite Concentrate	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
%Cu	**25.5**	**82.2**	0.4	0.3	1.33	9.8	0.33	7.1
%Pb	3.29	13.2	**61.0**	**65.9**	1.22	11.2	0.34	9.2
%Zn	7.07	4.6	5.76	1	**61.1**	**91**	0.69	3
%Fe	25.6	9.2	8.45	0.8	2.52	2.1	33.46	81.1
ppm Ag	1,068	53.1	555	7.4	163	18.5	57	19.1
ppm As	5,346	34.8	1,296	2.3	1462	21.6	637	28.6
%S	33.1	7.8	19.7	1.3	29.2	15.7	38.36	61
ppm Au	23	55.5	13	8.5	2.72	15	1.07	17.1
Wt %	4.57		1.23		10.4		1.07	

While good grades and recoveries were achieved for copper, lead and zinc, predicting gold deportment is difficult at bench scale. Although it was possible at Burnie to recover over 30% of the gold into a gravity concentrate, the subsequent flotation test was poor due to oxidation of the ground slurry during the time between the gravity gold test and the flotation test. It was then decided to engage a third party, Consep Pty Ltd, a manufacturer of Knelson gravity concentrators to carry out standard gravity recoverable gold tests to simulate the likely deportment of gold in the proposed circuit. This work is discussed under separate heading below.

Good separation of copper, lead and zinc was achieved in LC01. The problematic issue is that the copper is contained in both chalcopyrite as well as tennantite and that mineral contains virtually all of the arsenic and much of the silver in the ore (K Sangster, September 2011).

The basic process was modified to give a sub-split of chalcopyrite and tennantite.

A partial separation was shown to be possible whereby, at median arsenic head grades, a copper concentrate containing less than 0.5% As could be produced.

Consequently, a second copper concentrate containing high arsenic and silver levels is produced. The relative proportion of each of these products is determined by the chalcopyrite tennantite ratio in the ore being treated at any point in time.

13.5.2 Second Phase

At this point, the test objective was revised with the aim of maximizing copper recovery to copper cleaner 1, which even at around 20% grade and 0.8% As was calculated to be the better value approach. It was still intended to attempt a separation of this product, but solely on a batch basis. Given that such a split had been shown to be possible, it was decided to carry out the work for future reference.

Various permutations of reagents and circuit configurations were attempted with considerable success, so that with the robust nature of the lead, zinc and pyrite floats established, focus centred on the copper flotation section.

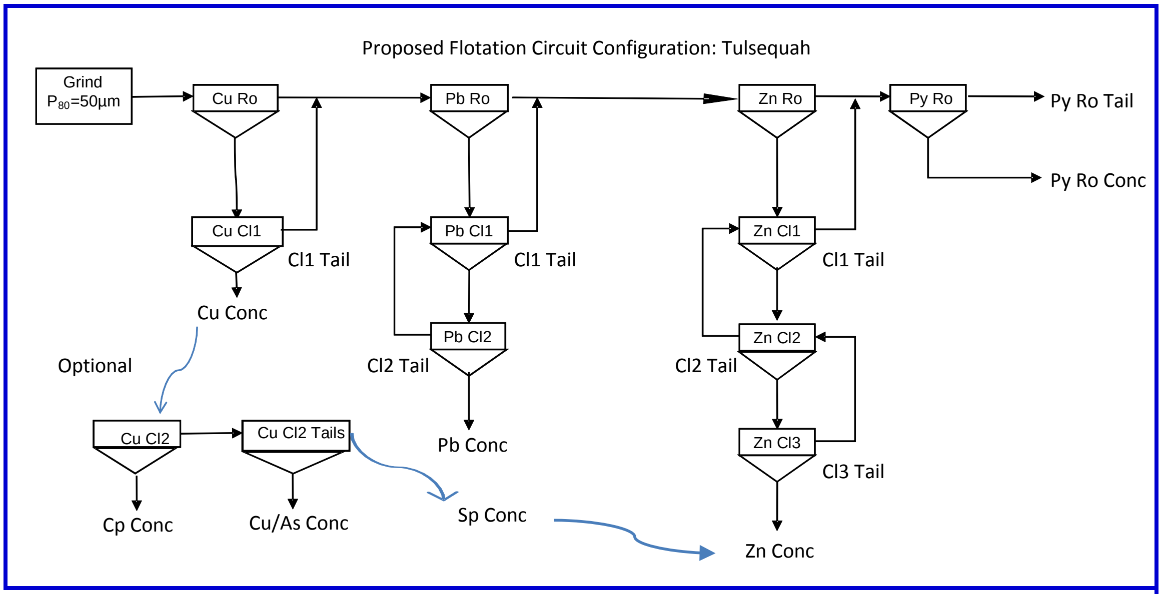
It was decided to adopt a single copper concentrate strategy with the aim of producing a copper cleaner concentrate containing a minimum of lead and zinc at maximum copper recovery (Consep Pty Ltd, February 2012; K Sangster, January 2012). This decision was taken because the ratio of tennantite to chalcopyrite varies throughout the orebody; the flotation split between the two minerals is sensitive to flotation conditions; and the copper and silver metal payment loss through the production of a lower grade concentrate is significant.

The copper concentrate was then to be tested to establish an efficient separation between the chalcopyrite and tennantite. As per the original circuit, copper cleaner 1 tails would be directed to lead flotation feed, but no other product from the copper circuit would be recirculated.

The probable circuit, with the two copper concentrates option included, is shown in Figure 13-1. To that end, three series of locked cycle tests, LC04, LC05, LC06, have been completed at Burnie and provide statistical support data for the expected production. The three samples appear to have a good spread of arsenic levels (i.e., 600 ppm, 800 ppm and 1,300 ppm), so it will not be necessary to include an artificially constructed sample with an intermediate level of arsenic.

In support of this work, two additional tests were conducted at ALS AMMTEC, Perth, whereby feed was crushed to P_{80} of 425 µm, passed through a Knelson to recover gold, with the tailing being further reduced to a P_{80} of 106 µm and the test repeated. The tailings were then reground to a P_{80} of 50 µm and floated as per standard conditions. The ground tailings were split and the flotation test repeated after storage for seven days to assess the flotation effects of sample weathering. The tests demonstrated poor selectivity exhibited little difference between the fresh and aged sample, adding further weight to the sample oxidation hypothesis (ALS AMMTEC, Malaga, April 2012).

Figure 13-1: Proposed Flotation Circuit



13.5.3 Discussion

Test results indicate that successful separation of the chalcopyrite and tennantite is feasible with the most likely mechanism capitalizing on the different flotation rates of the two minerals. There is enough data to suggest that in a locked cycle environment where sufficiently large circulating loads can be established, consistent mass pulls at enhanced selectivity can be achieved.

Test data also show that there is free sphalerite reporting to the copper cleaner tailings streams and that such misplaced sphalerite can be effectively recovered. Such a product would contain around 40% zinc and would be added directly to the final zinc concentrate. Because of the high grade of the concentrate (i.e., > 60% zinc) and the low mass of ≈ 40% product, the payment for contained zinc in the combined product will remain at the maximum 85%. A 40% zinc concentrate if sold separately would attract a payment of (40-8)/40 or 80%. The stream will be small and may add only around 1% mass, but with free carry at that point in the circuit is well worth pursuing.

Towards the end of the test program, consideration was given to an alkaline pyrite float with the objective of removing the need to add sulphuric acid as a pH modifier. This had already been shown possible in the latter locked cycle tests, but further work was required to ensure that the targets for sulphide sulphur in rougher tailings were met. Total sulphur content of the tailings is misleading due to the presence of variable quantities of barites in the gangue minerals. Testwork has now confirmed that the use of sulphuric acid can be eliminated. This will also have the effect of reducing the neutralization load on the water treatment facilities.

13.6 Gold Recovery

13.6.1 Introduction

Due to the financial impact of gold revenue on project economics, there has been extensive investigation into its recovery to payable products. During previous operations, 25% to 30% of the gold was recovered by jigging prior to flotation. Testwork carried out in the 1990s and summarized by Beattie (Beattie Consulting Ltd, March 1993) confirmed that around 30% was recoverable; however, an opinion was expressed that "[i]n continuous operation, the gravity concentrate should contain 60 to 80% by weight combined gold and silver." No data have been discovered to support this claim, and later testwork has reported that a recovery of a little over 40% can be expected (AMTEL Ltd University of West Ontario, April 2011). It should be noted that a recommendation was made in that report to consider centrifugal concentrators to recover gold after grinding, which is the strategy that has now been adopted.

13.6.2 Current Program

Preliminary testing confirmed that around 30% of the gold was recoverable through gravity means (ALS Burnie tests # 22, #32 Mozeley separator) and alternatively, that around 20% would report to a prefloat concentrate. However, unacceptable losses of copper, lead and zinc to the prefloat product made this option uneconomic.

Diagnostic each testing of the lead cleaner tails has shown that 96% of the gold reporting to that product is cyanide soluble and a bottle roll test of the final tailings gave a gold recovery of around 50% indicating that without gravity concentration before flotation, some free gold is reporting to the tails streams.

Flotation testing of all composites, without prior gravity concentration, but with a gold scavenger float after lead float, shows consistently, that ≈ 90% of the gold reports to payable products (i.e., copper, lead and gold scavenger concentrates).

One aspect of earlier testwork that was not repeated in the recent program is the finding that after gravity recovery of gold and the use of cyanide as a lead depressant in the copper circuit, only minimal gold reports to the copper concentrate. In the current program, gold recovery to the copper concentrate seemed to be largely unaffected with the gravity gold recovered being sourced from all products.

13.6.3 Consep (Metcon) Testwork & Modelling

A 15 kg sample of composite No. 4 was tested at Metcon in Sydney to determine the gravity recoverable gold in the ore. Given the robustness of the testwork and subsequent analysis, these results have produced the defining parameters for both the design and performance of the gravity plant to be included in the grinding circuit. While the testwork calculated the maximum recovery of gravity recovery gold at 53.7%, the simulations indicated that a gold recovery of 41.3% was the most likely outcome, given the size of Knelsons to be used and the proportion of cyclone underflow to be processed (Consep Pty Ltd, February 2012; Brendan Mahony Consep Pty Ltd, March 2012).

13.6.4 Flotation Response after Gravity Gold Recovery

The assessment of gold deportment through a five-stage sequential float with and without gravity recovery in the grinding stage has proved difficult to manage given the small weight of products being handled (i.e., as little as 20 g in many cases). As such, conclusions that are drawn should be considered as approximations rather than definitive design parameters. However, overall gold recovery to payable products is not in doubt. As stated above, testwork has shown repeatedly that gold recovered to those products is of the order of 90% (ALS AMMTEC, Burnie, February 2012 and ALS AMMTEC, Malaga, April 2012).

One difficulty is, given that we have no definitive flotation testwork following 41% gold recovery as prescribed by Consep, there are no corresponding figures for gold recovery to concentrates. There are, however, enough bench test results to draw a reasonable approximation for a metallurgical balance.

Table 13-17: Gold Recovery

Test #	29 – No Gravity	32 – Inc. Gravity	Est @ 40% Au Gravity Recovery
Gravity Concentrate		33.4	41.3
Copper Concentrate	67	55.4	47.5
Lead Concentrate	13.8	0.5	0.5
Gold Scavenger	6.6	1.8	1.8
Subtotal	**87.4**	**91.1**	**91.1**
Zinc Concentrate	1.4	2.6	2.6
Zinc Rougher Tails	11.2	6.3	6.3

As an example, if tests 29 (no gravity) and 32 (including gravity) are compared, it appears that:

- As there is little remaining gold in the lead concentrate and the gold scavenger, it is likely that the additional 8% gold reporting to the gravity concentrate (as determined by Consep) will come mostly from the copper concentrate.
- It would be still prudent to configure the circuit for a gold scavenger as a trap for any gold that might get through the copper/lead float for whatever reason.

The robustness of the gold recovery from the Tulsequah ore is further reinforced by results from the first AMMTEC test, which shows a gold recovery to Knelson (panned) concentrate of 31% followed by a gold recovery to copper concentrate of 51%, which confirms prior work.

13.6.5 Silver Deportment

The gold is almost entirely present as electrum with the silver content varying widely but thought to average around 30%. The gravity concentrate that is fed to the leach feed is expected to contain 42% of the gold, but even with 30% silver content in the doré, only 0.5% of the silver in the feed will be contained in this product. The vast majority of the contained silver will be present in galena and tennantite, which will not be leached and will return as leach tail to the grinding circuit.

13.6.6 Sample Oxidation

There is considerable evidence that the ground samples are subject to oxidation of varying levels when in slurry form. For example, test # 32, did demonstrate poorer zinc selectivity, which indicates possible copper ion activation from oxidation. Some tests at ALS Burnie exhibited the same problem, as did the early tests at ALS AMMTEC. It is apparent that the process of putting sample through a Knelson concentrator at reducing sizes coupled with the volumes of water in play is sufficient to cause oxidation even without long delays before flotation. Planned tests at AMMTEC are designed to explore this aspect of the testwork. It is important to note that this exercise is designed to resolve testwork anomalies only. As such, oxidation has not been reported in operating plants nor is it anticipated.

13.7 Ore Physical Parameter Tests & Other Tests

13.7.1 Specific Gravity

The specific gravity of all intercepts was measured in Canada giving results of 3.38 (sample 1: 59 samples ranging 2.5 to 4.5) and 3.20 (sample 2: 102 samples ranging 2.5 to 4.5). The composite sample at Burnie was measured at 3.74. (Chieftain Inc. Brett Armstrong, May and August 2011; ALS AMMTEC, Burnie, October 2011). As the Burnie figure is from only one measurement, an average specific gravity of 3.3 derived from the Canadian figures is the more accurate.

13.7.2 Unconfined Compressive Strength

UCS measurements were conducted on four core samples supplied by Chieftain with results shown in Table 13-18.

Table 13-18: UCS Testwork Results

Sample Identity	UCS (MPa)	Failure Mode	Strength Description
TCU11192 (120.60 – 120.70 m)	35.1	Shear	Medium Strong
TCU11194 (114.15 – 114.25 m)	48.9	Columnar	Medium Strong
TCU 11195 (112.06 – 112.16 m)	74.6	Columnar	Strong
TCU11198 (124.60 – 124.70 m)	43.2	Columnar	Medium Strong

13.7.3 Abrasion Index

The Bond Abrasion Index, Ai, was calculated at 0.0743 (ALS Burnie Research Laboratories, September 2011).

13.7.4 Ball Mill Work Index

The Bond Work Index was calculated at 12.9 kWh/dry tonne (ALS Burnie Research Laboratories, August 2012).

13.7.5 Settling Rates

Settling tests were conducted on two samples of final tailings as well as one sample each of copper and zinc concentrates (ALS AMMTEC, Burnie, September 2011). The rapid settling and clear supernatant liquors indicate that the anticipated underflow densities in the design flowsheet should be easily obtainable. Given the un-reground nature of the concentrates, it has still been assumed that air flush pressure filtering will be used despite clear indications that vacuum filtration may well be adequate. This will also ensure adequate flexibility in dealing with transportable moisture constraints.

13.8 Testwork Interpretation & Performance

13.8.1 Introduction

The Tulsequah orebody is typical of VMS deposits found worldwide. It contains recoverable copper, lead, zinc, gold and silver within a host that is predominantly pyrite/barite. Lead is present as galena, zinc is present as a very low iron sphalerite and the copper occurs as three main minerals:

- chalcopyrite (Cp) – $Cu\ Fe\ S_2$ (dominant)
- tennantite (Tn) – $(Cu\ Fe\ Zn\ Ag)_{12}\ As_4\ S_{13}$
- tetrahedrite Th – $(Cu\ Fe\ Zn\ Ag)_{12}\ Sb_4\ S_{13}$. This explains why some zinc will always report to the copper concentrates.

Tn and Th are similar in terms of mineralogical appearance and flotation response. The relative proportions of all three species vary throughout the deposit.

13.8.2 Selection of Processing Method

VMS ores are universally treated by flotation, but a gravity circuit is often included if gold recovery is warranted. Given that the Tulsequah ore was successfully treated by flotation in the 1950s and test programs in the 1990s all utilized flotation, this was the obvious route to take. With evidence of free gold in the ore, serious consideration was given to the inclusion of gravity gold recovery prior to flotation. Again, this circuit is widely used for this type of ore.

13.8.3 Comminution Circuit

As mentioned above, there appears to have been an assumption made in past testwork, that the ore would be reduced to flotation feed size through conventional closed circuit crushing, followed by rod/ball mill reduction. While this would be effective, the unique Tulsequah site considerations may be better suited to SAG milling followed by two-stage ball milling. The ore is remarkably similar in physical characteristics to the VMS Hellyer deposit in northern Tasmania, which utilized SAG milling to good effect.

This option would seem to have the following advantages:

- a substantially smaller footprint and consequent development capital costs
- simpler operational requirements, especially with many fewer drives
- shorter construction time
- possible standardization of drive motors
- the avoidance of difficulties in handling conductive and flammable dusts.

Potential disadvantages are:

- marginally higher power consumption
- lag time in sourcing an appropriately sized SAG mill
- unanticipated results from prescribed SAG mill testwork.

13.8.4 Confirmation of Processing Method

Initial flotation tests at Burnie gave very encouraging results, so that by test #16, even allowing for some more adventurous options in some tests, a set of flotation parameters that achieved acceptable concentrate grades and recoveries was achieved. As described above, work then focused on arsenic rejection into a separate copper concentrate; however, while operationally feasible, it has been calculated to be less economical than producing a single lower grade product with maximum copper, gold and silver recoveries, but at the expense of a higher arsenic level.

The two product route remains as an option if price and marketing conditions dictate such a route. Ongoing testwork to establish the separation is planned.

13.8.5 Flotation Performance Predictions

Mineralogical analysis was used to predict maximum possible copper and zinc flotation performance. The graphs above indicate a maximum possible copper concentrate of 35% Cu at a recovery of 85% to 90% and a zinc performance of 64% Zn at over 90% recovery. It is likely that the copper recovery will approach 90% once in operation, but the grade will be only a little over 20% due to the strategy of maximizing copper recovery. Testwork indicates that zinc concentrate will contain >60% Zn, at a recovery of around 90%.

This correlation is typical of sound flotation performance.

Figure 13-2: Composite 1 Copper Cumulative Yield Curves

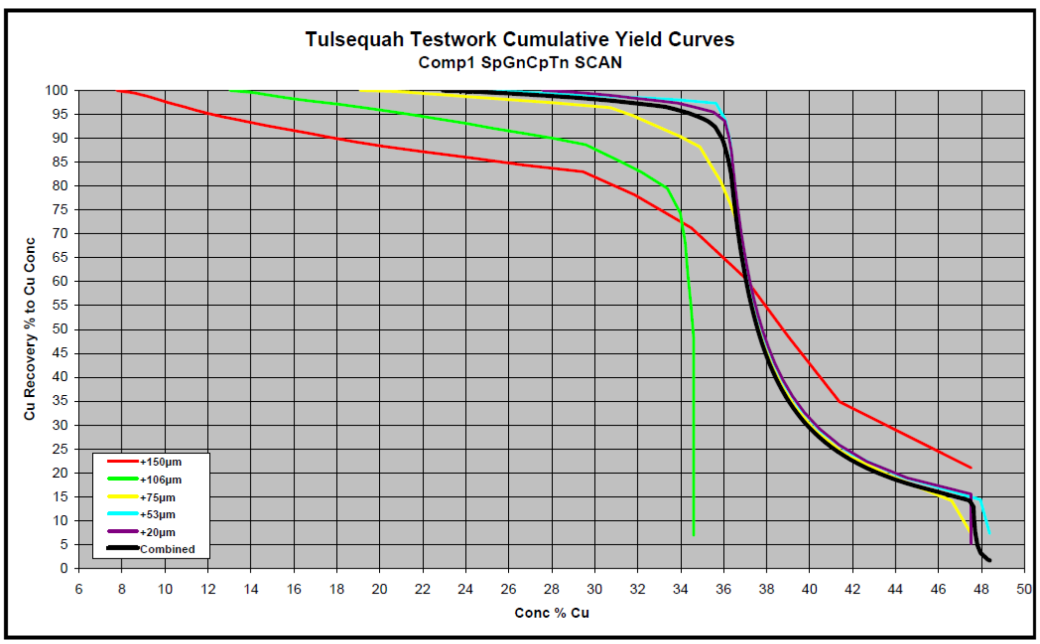
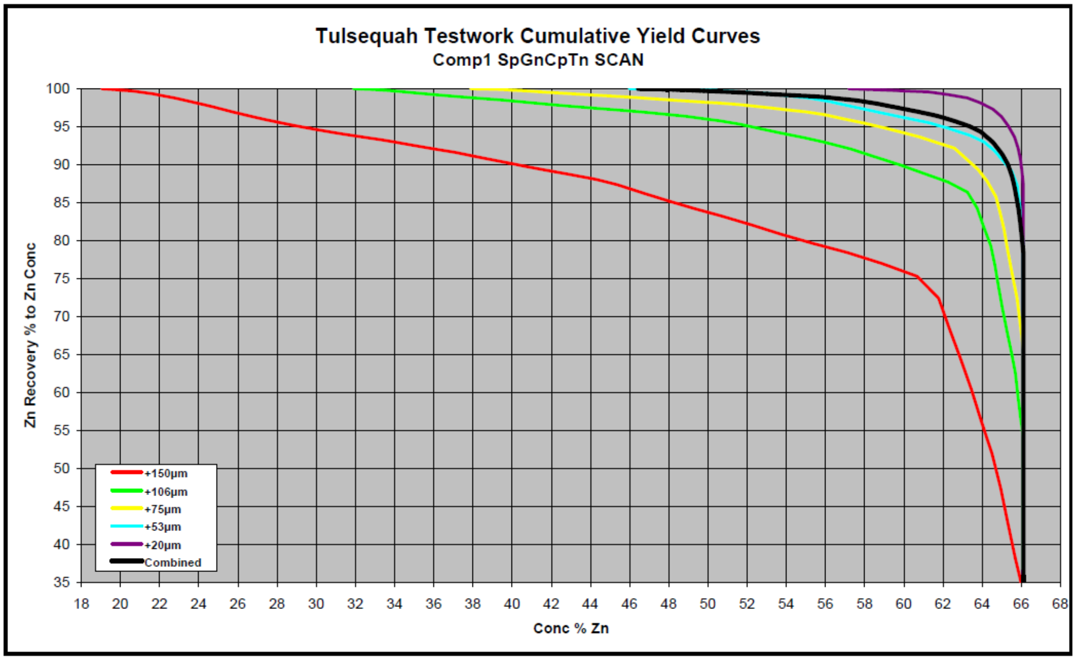


Figure 13-3: Composite 1 Zinc Cumulative Yield Curves



13.8.6 Concentration Performance Predictions

The plots shown below have been used to predict global recoveries to concentrates for the Tulsequah orebody. In each case, a line of best fit has been generated through the locked cycle tests and the grade recovery curves for test #16 have been added for comparison. Locked cycle tests 01 and 02 were considered developmental; test 03 was subject to a lengthy delay between cycles three and four, which, as we now know, would have resulted in oxidation resulting in a loss of selectivity. There was also the risk that if attempts were made to reduce the activation, that tennantite could have been depressed as far as the gold scavenger float. Locked cycle tests 04, 05, 06 are considered to be the most accurate representation of grades and recoveries produced in the testwork. The individual cycles were demonstrably consistent so it is considered valid to extrapolate the line of best fit to appropriate target results as shown in the metallurgical statement. It should be noted that in all cases, the selected points are conservative with respect to recovery. Given that the samples are sensitive to oxidation over time the latter tests on older samples the effect was relatively small and further reinforces the slight degree of conservatism adopted.

Figure 13-4: Copper Grade vs. Recovery – Locked Cycle Tests

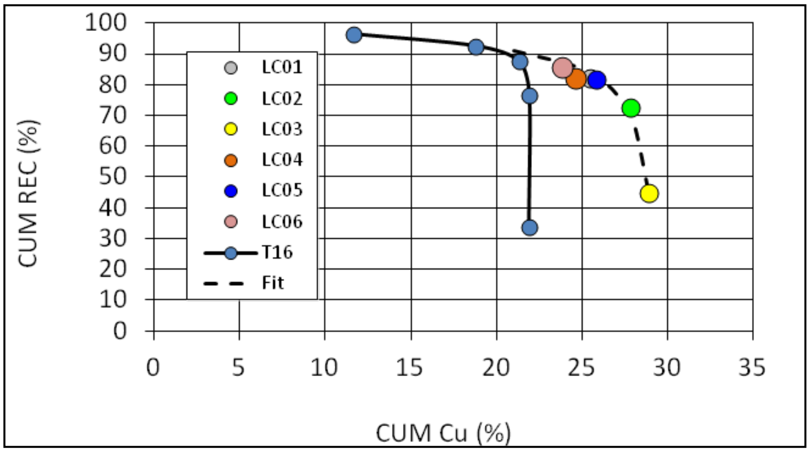


Figure 13-5: Zinc Grade vs. Recovery – Locked Cycle Tests

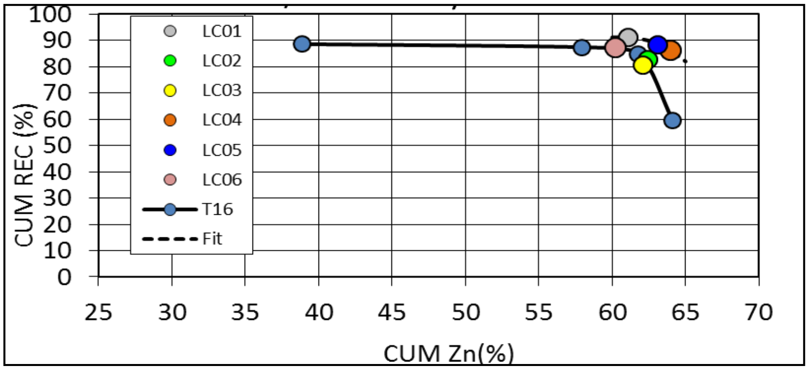


Figure 13-6: Lead Grade vs. Recovery – Locked Cycle Tests

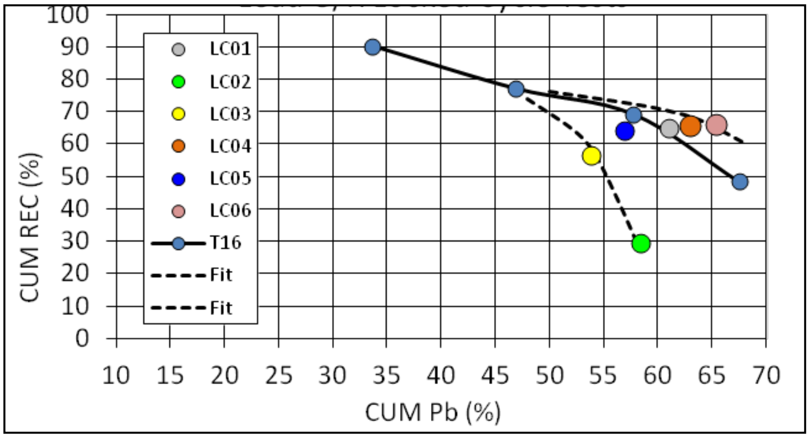


Table 13-19: Metallurgical Statement

Product	Wt (t)	Assays					Recoveries %				
		Cu %	Pb %	Zn %	Ag g/t	Au g/t	Cu	Pb	Zn	Ag	Au
Copper Conc.	5.3	21.0	2.7	7.6	1217.5	20.8	89.0	11.8	6.0	75.0	44.0
Lead Conc.	1.3	1.0	60.0	8.4	586.4	7.6	0.8	66.2	1.6	9.0	4.0
Zinc Conc.	9.6	0.7	0.8	62.0	69.7	0.8	5.5	6.8	89.0	7.8	3.0
Pyrite Conc.	28	0.1	0.3	0.4	19.2	0.5	1.9	6.8	1.7	6.1	5.0
Tailings	55.7	0.1	0.2	0.2	2.4	0.1	2.8	8.4	1.7	1.6	2.0
Feed	100	1.3	1.2	6.7	86	2.5	100	100	100	100	100
Doré Kg/100t feed	0.155				30%	70%				0.5	42.0

** Brendan Mahony Consep Pty Ltd, March 2012 Gold Recovery Forecast

13.8.7 Product Quality Predictions

Table 13-20: Typical Concentrate Analyses

Element	Unit	Concentrate		
		Copper	Lead	Zinc
Zn	%	8.5	8.5	59.9
Cu	%	21.7	0.29	0.53
Pb	%	3	62.8	0.21
Al_2O_3	%	1.23	0.42	1.15
CaO	%	0.17	0.07	0.08
Fe_2O_3	%	31.7	8.6	3.1
K_2O	%	0.25	0.11	0.29
MgO	%	0.85	0.2	0.25
SiO_2	%	3.8	1.5	2.9
S	%	28.6	16	32.8
Na_2O	%	0.08	0.03	0.07
Ti_2O	%	0.05	0.02	0.05
As	%	1.45	0.08	0.03
MnO	ppm	129	80	297
F	ppm	194	186	163
Cl	ppm	232	134	270
Bi	ppm	7	57	0.5
Hg	ppm	36	19	171
Sb	ppm	5,647	905	102
Cd	ppm	387	354	2,434

Element	Unit	Concentrate		
		Copper	Lead	Zinc
Au	ppm	22 est *	8.3	0.9
Ag	ppm	1,339	423	59
Sr	ppm	50	38	31
U	ppm	1.4	0.8	1.6
Mo	ppm	90	19	21
Ni	ppm	3.9	6	2.4
Ba	ppm	119	147	115
Sn	ppm	5.2	0.4	0.8
Cr	ppm	25.3	11.6	28

* Gold content of the copper concentrate was assayed at 43.4 g/t, but the product was made from total without gravity removal. The recoveries to gravity and copper concentrate are approximately equal, hence the estimate of 22 g/t.
* Average of products from locked cycle tests LC04, LC05, LC06 refers ALS AMMTEC, Burnie, August 2012 for complete analyses.

13.8.8 Forecast Reagent Consumptions

Table 13-21: Reagent Consumption

Reagent	Dose Rate g/t ore	Manufacturer
Lime	1,130	Generic
Sodium metabisulphite(SMBS)	2,000	Generic
AMG 9810	8	Sigma
Sodium cyanide (flotation)	330	Generic
Zinc sulphate	500	Generic
Aerophine 3418A	10	Cytec
Potassium amyl xanthate (PAX)	70	Generic
MIBC	160	Cytec

The figures shown are best estimates at this point. It is likely that consumption of zinc sulphate will increase. The rate will be confirmed with current testwork.

Cyanide will be used in gold recovery, but that amount is still to be precisely determined; however, it is unlikely to be an additional amount as the cyanide rich bleed off from the gold recovery circuit will be used as make up for the flotation circuit.

13.9 Testwork Summary

The work described above has been shown to be robust and provides a set of operating parameters on which an economic assessment of the treatment of the Tulsequah orebody can be made. It has been successful in establishing a treatment route for the production of marketable copper, lead and zinc concentrates as well as gold doré.

It is particularly reassuring that the later testwork was almost as good as the earlier work, although it must have been impaired slightly by oxidation.

Indeed it appears that as well as inadequate liberation, varying amounts of alteration ranging from partial oxidation to flotation surface tarnishing was governing previous testing, particularly the 1996-2006 work.

The general consistency of the major mineralogical variables is reflected in the similarity of the bulk of the tests conducted.

The predicted performance levels are well within the release curves for liberated particles indicating that further improvement is possible.

There is also a need to do further work to measure physical characteristics necessary for the design study.

13.10 Future Testwork

There are three main areas of planned metallurgical work.

13.10.1 Copper minerals separation

The aim is to use the knowledge gained in trying to separate chalcopyrite and tennantite to improve the quality of the copper concentrate. Also, the removal of the misreporting sphalerite which is known to be well liberated but is recovered by both minor Cu ion activation and by mass split is an important issue. At present, a consequence of maximizing the recovery of tennantite for both copper and silver values is that it makes it difficult to suppress this effect, which similarly allows some pyrite to intrude into the same products. It appears that a protocol can be devised to accomplish this to benefit copper concentrate grade and zinc recovery. Given the high degree of liberation of the sphalerite (MODA Pty Ltd, Burnie, Report #3, November 2011) any sphalerite removed will be of sufficiently high grade to be directed to the final zinc concentrate. This will not only increase zinc recovery and metal revenue, but also reduce the penalty metal levels in the copper concentrate.

13.10.2 Gold Leaching

The refinement of the leaching circuit for the gravity concentrates needs to be completed. This includes optimization of the desired weight and grade of the gravity concentrate and gets a fix on the recovery of the non-electrum silver. The gravity concentrate will contain some galena and tennantite, both of which are relatively insoluble in cyanide solution and will be returned to the grinding circuit. As such, the non-electrum silver contained in the tennantite and galena should be recovered to the respective concentrates, but this needs confirmation. The exact levels of cyanide consumption are also required—although it probably has little impact on the total cyanide usage.

13.10.3 Physical Properties

From the point of view of better operational understanding, it is necessary to look at treating a larger sample. The resultant larger subsamples will assist in the above exercises and generate sufficient tailings for cemented backfill tests. Such a test will be scaled to provide enough zinc concentrate (i.e., minimum 4.5 kg) to conduct statutory TML tests for shipping purposes. The same sample will also be used to establish process design criteria, such as bulk density and angle of repose. The issue of representivity is a major one; mining a bulk sample close to the adits is not ideal as clearly demonstrated in previous work. Such material will have suffered some alteration (as well as being a single point sample) as the use of older core. Obtaining fresh material requires planning and is probably best done with a large diameter drill to drill multiple holes over a larger area.

14 Mineral Resource Estimates

14.1 Introduction

The Mineral Resource Statement presented in this report is the second mineral resource evaluation prepared for the Tulsequah Chief project in accordance with the Canadian Securities Administrators' NI 43-101 guidelines.

The mineral resource model prepared by SRK considers 665 core boreholes drilled by Cominco, Redfern and Chieftain during the period of 1940 to 2011. The resource estimation work was completed by Dr. Gilles Arseneau, P.Geo (APEGBC # 23474), who is an appropriate "independent qualified person" as defined in NI 43-101. The effective date of the resource statement is March 15, 2012.

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

The database used to estimate the Tulsequah Chief project mineral resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for VMS mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

Most of the 2004-2011 data were verified by SRK. The older historical data could not be verified against original assay certificates as these no longer exist. SRK carried out a review of about 20% of the assay data and noted only three minor errors. Mineralized lenses were modelled by Chieftain and audited and validated by SRK using GEMs. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the mineralized areas and that the assaying data are sufficiently reliable to support estimating mineral resources.

Gemcom GEMs Version 6.3 was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades and tabulate mineral resources.

14.2 Resource Estimation Procedures

The resource evaluation methodology involved the following procedures:

- database compilation and verification
- construction of wireframe models for the boundaries of the massive sulphide mineralization
- definition of resource domains
- data conditioning (compositing and capping) for geostatistical analysis and variography
- block modelling and grade interpolation
- resource classification and validation
- assessment of “reasonable prospects for economic extraction” and selection of appropriate cut-off grades
- preparation of the Mineral Resource Statement.

14.3 Resource Database

The assay database for Tulsequah comprises 17,329 samples, 3,075 of which are contained within the mineralized units and used to estimate the mineral resource.

Raw assay data distributions were examined by visualizing histograms and cumulative probability plots. Basic statistical data such as mean, standard deviation, mode and skewness were tabulated for all assay data within the mineralized zones (Table 14-1).

The assay data were also analyzed for each lens separately prior to compositing to identify any possible zoning or unusual assay distribution. The deposit at Tulsequah Chief is comprised of 17 discreet mineralized lenses, four G lenses termed G1 to G4, ten H lenses termed H1 to H10 and three A Extension lenses, termed AEX_U, AEX_M and AEX_L. For coding the block model, the lenses were assigned a corresponding integer code as indicated in (Table 14-2).

In order to determine if the material left within the mine workings at closure had different statistical characteristics, all mineralized material above the 5200 level was assigned a temporary lens code of “MW.” From the statistical analysis of temporary lens MW, it was determined that the character of the MW lens was not significantly different from the other mineralized lenses, so the samples from lens MW were reclassified to their appropriate lenses before block modeling, either lens H3 or H4.

Figure 14-1to Figure 14-5 are box and whisker plots of the G and A lenses for Cu, Pb, Zn, Au and Ag while Figure 14-6 to Figure 14-10 show the same data for the H lenses. The box plot displays maximum and minimum values, the box is around the 75th and 25th percentile, the median is indicated with the line across the box and mean is indicated by the black dot.

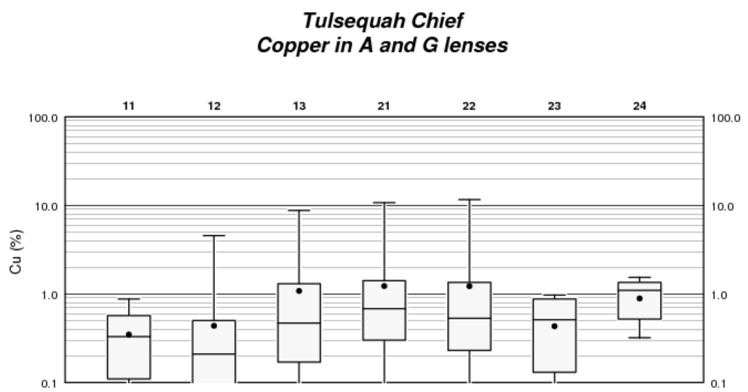
Table 14-1: Descriptive Statistics of Assay Data within the Mineralized Zones

Statistical Parameters	Cu%	Pb%	Zn%	Au (g/t)	Ag (g/t)
Valid cases	3075	3075	3075	3075	3075
Mean	1.31	1.2	6.54	2.23	84.97
Std. error of mean	0.03	0.03	0.13	0.06	2.03
Variance	3.02	3.48	52.11	11.26	12,631.54
Std. Deviation	1.74	1.87	7.22	3.36	112.39
Variation Coefficient	1.32	1.55	1.1	1.5	1.32
rel. V. coefficient (%)	2.39	2.8	1.99	2.71	2.39
Skew	3.14	4.26	1.5	7	4.23
Minimum	0	0	0	0	0
Maximum	16.7	34.3	42.7	59.9	1,519.05
1st percentile	0	0	0	0	0.2
5th percentile	0.01	0	0.01	0	1.2
10th percentile	0.04	0	0.05	0.08	5.72
25th percentile	0.3	0.04	0.91	0.55	20.57
Median	0.8	0.51	4	1.37	54
75th percentile	1.6	1.6	9.9	2.74	104
90th percentile	3.07	3.35	17	4.8	188.96
95th percentile	4.8	4.65	21.22	7.1	268.85
99th percentile	8.92	8.3	30.52	14.85	573.46

Table 14-2: Mineralized Lenses & Corresponding Block Model Codes

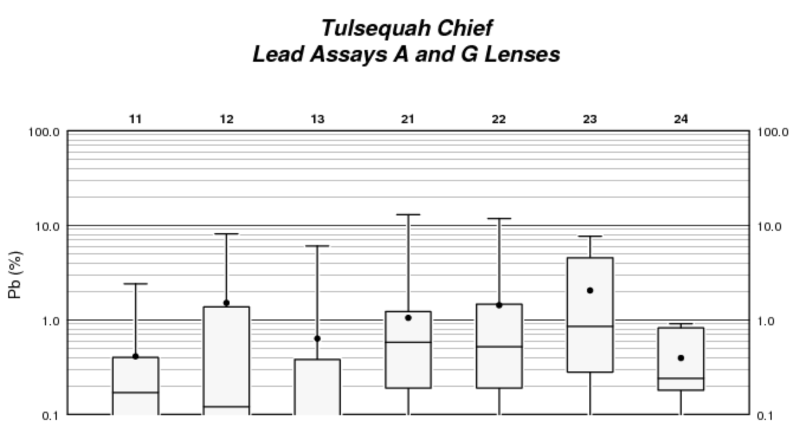
Lens Name	Block Model Code
AEX_L	11
AEX_M	12
AEX_U	13
G1	21
G2	22
G3	23
G4	24
MW	30
H1	31
H2	32
H3	33
H4	34
H5	35
H6	36
H7	37
H8	38
G9	39
H10	40

Figure 14-1: Box & Whisker Plot for Copper in A & G Lenses



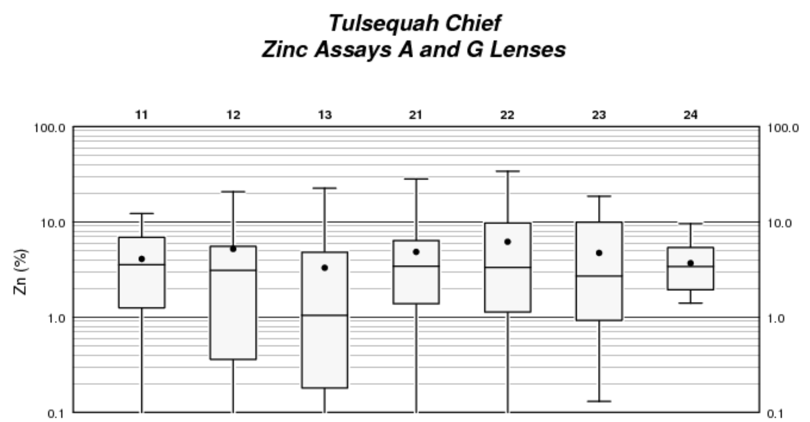
	11	12	13	21	22	23	24	
Number of data	24	26	114	307	71	13	8	Number of data
Mean	0.348	0.437	1.082	1.228	1.225	0.432	0.889	Mean
Maximum	0.88	4.55	8.76	10.72	11.6	0.97	1.54	Maximum
Upper quartile	0.57	0.5	1.3	1.41	1.35	0.88	1.35	Upper quartile
Median	0.33	0.21	0.47	0.68	0.53	0.51	1.1	Median
Lower quartile	0.11	0.08	0.17	0.3	0.23	0.13	0.52	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.01	0.04	0.32	Minimum
Variance	0.068	0.76	2.495	2.736	3.87	0.123	0.174	Variance
Std. Dev.	0.261	0.872	1.58	1.654	1.967	0.35	0.418	Std. Dev.
CV	0.749	1.997	1.459	1.347	1.606	0.81	0.47	CV

Figure 14-2: Box & Whisker Plot for Lead in A & G Lenses



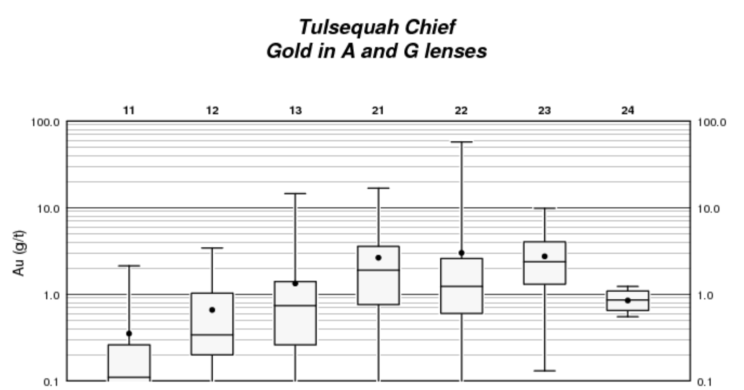
	11	12	13	21	22	23	24	
Number of data	24	26	114	307	71	13	8	Number of data
Mean	0.411	1.51	0.633	1.05	1.427	2.045	0.395	Mean
Maximum	2.41	8.1	6.05	12.9	11.8	7.68	0.91	Maximum
Upper quartile	0.4	1.37	0.38	1.22	1.46	4.53	0.82	Upper quartile
Median	0.17	0.12	0.05	0.58	0.52	0.85	0.24	Median
Lower quartile	0.01	0.01	0.01	0.19	0.19	0.28	0.18	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.01	0.03	0.07	Minimum
Variance	0.472	6.437	1.896	2.197	5.801	6.419	0.089	Variance
Std. Dev.	0.687	2.537	1.377	1.482	2.409	2.534	0.299	Std. Dev.
CV	1.671	1.681	2.176	1.412	1.688	1.239	0.757	CV

Figure 14-3: Box & Whisker Plot for Zinc in A & G Lenses



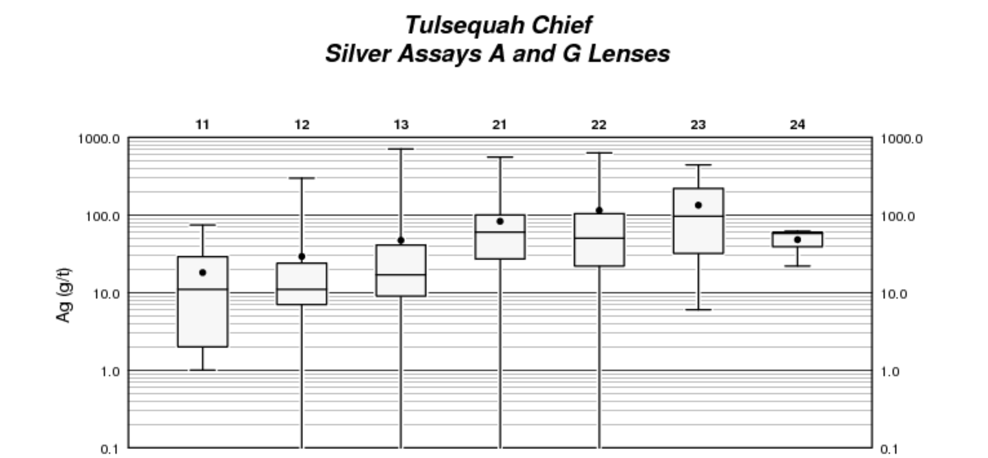
	11	12	13	21	22	23	24	
Number of data	24	26	114	307	71	13	8	Number of data
Mean	4.076	5.187	3.291	4.836	6.175	4.714	3.681	Mean
Maximum	12.2	20.7	22.5	28.1	34.0	18.6	9.58	Maximum
Upper quartile	6.85	5.52	4.8	6.36	9.7	9.89	5.37	Upper quartile
Median	3.55	3.1	1.04	3.42	3.31	2.7	3.4	Median
Lower quartile	1.25	0.36	0.18	1.38	1.13	0.92	1.93	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.02	0.13	1.4	Minimum
Variance	11.284	41.426	24.767	24.523	52.997	30.57	6.697	Variance
Std. Dev.	3.359	6.436	4.977	4.952	7.28	5.529	2.588	Std. Dev.
CV	0.824	1.241	1.512	1.024	1.179	1.173	0.703	CV

Figure 14-4: Box & Whisker Plot for Gold in A & G Lenses



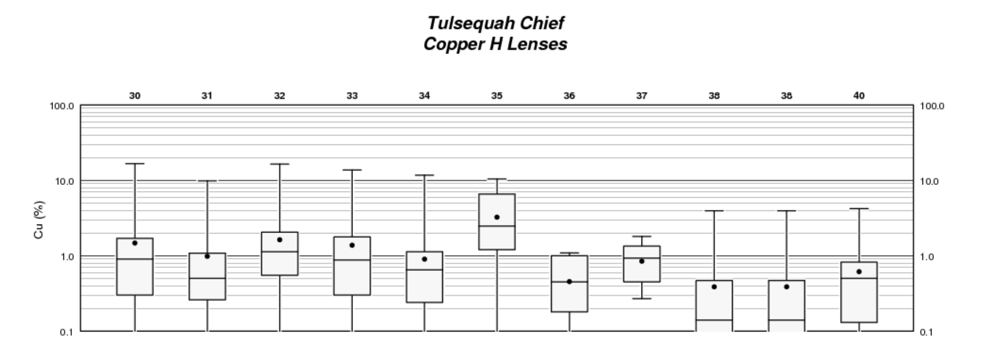
	11	12	13	21	22	23	24	
Number of data	24	26	114	307	71	13	8	Number of data
Mean	0.351	0.659	1.332	2.639	3.001	2.738	0.846	Mean
Maximum	2.13	3.43	14.5	16.8	57.5	9.81	1.23	Maximum
Upper quartile	0.26	1.03	1.4	3.56	2.58	4.02	1.09	Upper quartile
Median	0.11	0.34	0.74	1.89	1.23	2.37	0.86	Median
Lower quartile	0.04	0.2	0.26	0.76	0.6	1.3	0.65	Lower quartile
Minimum	0.03	0.0	0.0	0.0	0.03	0.13	0.55	Minimum
Variance	0.347	0.588	3.815	7.809	49.577	5.887	0.049	Variance
Std. Dev.	0.589	0.767	1.953	2.795	7.041	2.426	0.22	Std. Dev.
CV	1.68	1.164	1.467	1.059	2.346	0.886	0.26	CV

Figure 14-5: Box & Whisker Plot for Silver in A & G Lenses



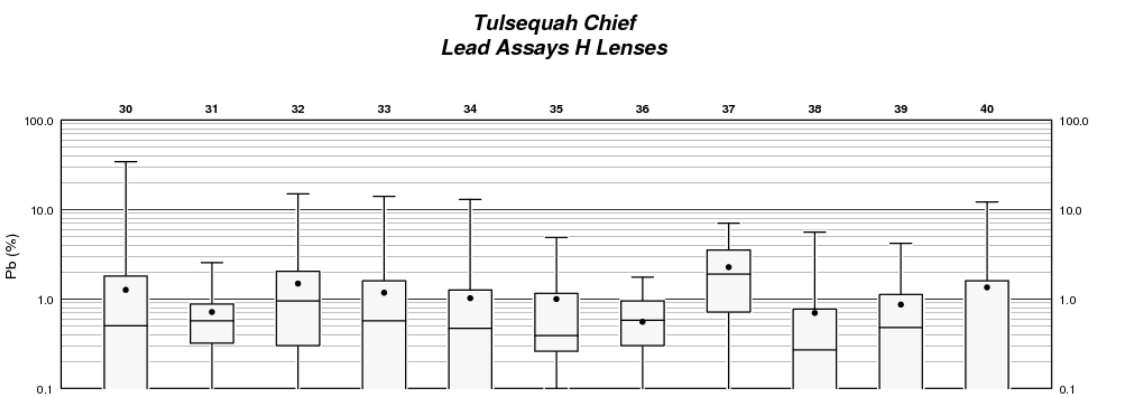
	11	12	13	21	22	23	24	
Number of data	24	26	114	307	71	13	8	Number of data
Mean	18.083	29.154	47.096	82.808	114.535	133.538	48.0	Mean
Maximum	74.0	298.0	710.0	555.0	631.0	442.0	62.0	Maximum
Upper quartile	29.0	24.0	41.0	100.0	104.0	220.0	60.0	Upper quartile
Median	11.0	11.0	17.0	60.0	50.0	96.0	58.0	Median
Lower quartile	2.0	7.0	9.0	27.0	22.0	32.0	39.0	Lower quartile
Minimum	1.0	0.0	0.0	0.0	0.0	6.0	22.0	Minimum
Variance	451.16	3293.13	9402.16	7574.34	25865.5	20490.7	183.75	Variance
Std. Dev.	21.241	57.386	96.965	87.031	160.828	143.146	13.555	Std. Dev.
CV	1.175	1.968	2.059	1.051	1.404	1.072	0.282	CV

Figure 14-6: Box & Whisker Plot for Copper in H Lenses



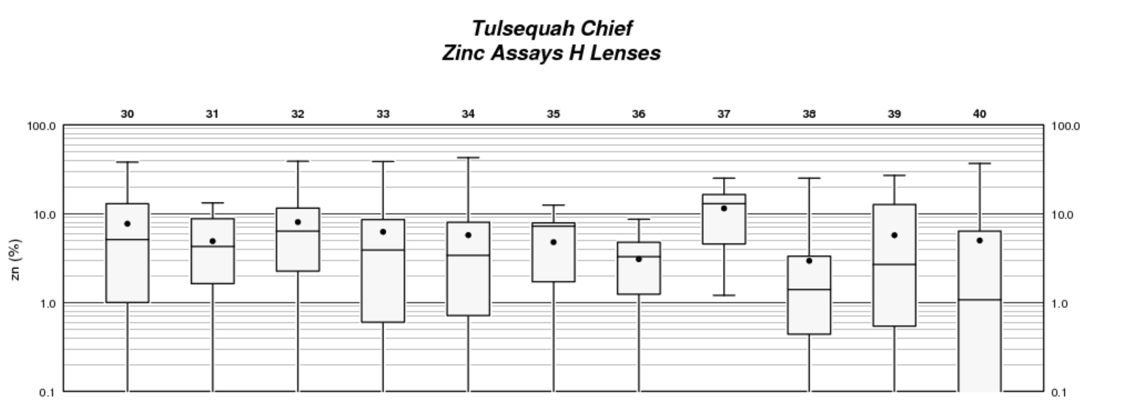
	30	31	32	33	34	35	36	37	38	38	40	
Number of data	946	34	477	619	244	11	11	19	59	59	79	Number of data
Mean	1.474	0.983	1.628	1.378	0.901	3.243	0.454	0.845	0.387	0.387	0.615	Mean
Maximum	16.7	9.85	16.4	13.7	11.7	10.4	1.09	1.8	3.96	3.96	4.2	Maximum
Upper quartile	1.7	1.08	2.05	1.78	1.13	6.6	1.0	1.34	0.47	0.47	0.82	Upper quartile
Median	0.9	0.5	1.13	0.88	0.65	2.47	0.45	0.93	0.14	0.14	0.5	Median
Lower quartile	0.3	0.26	0.55	0.3	0.24	1.2	0.18	0.45	0.04	0.04	0.13	Lower quartile
Minimum	0.0	0.01	0.0	0.0	0.0	0.07	0.0	0.27	0.0	0.0	0.0	Minimum
Variance	3.854	3.089	3.11	2.906	1.331	10.433	0.147	0.246	0.397	0.397	0.413	Variance
Std. Dev.	1.963	1.758	1.764	1.705	1.154	3.23	0.383	0.496	0.63	0.63	0.643	Std. Dev.
CV	1.332	1.789	1.083	1.237	1.281	0.996	0.845	0.587	1.63	1.63	1.046	CV

Figure 14-7: Box & Whisker Plot for Lead in H Lenses



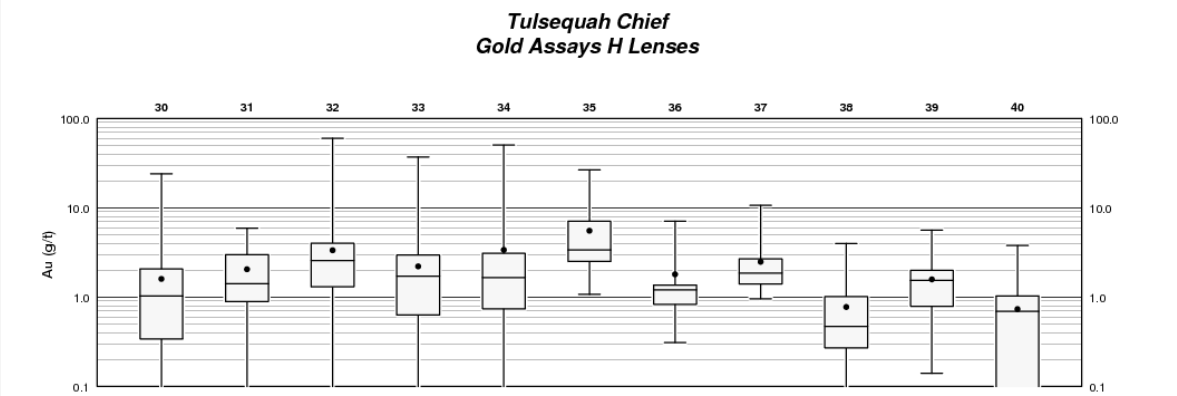
Lens	30	31	32	33	34	35	36	37	38	39	40
Number of data	946	34	477	619	244	11	11	19	59	13	79
Mean	1.263	0.714	1.485	1.174	1.019	0.997	0.555	2.265	0.697	0.865	1.345
Maximum	34.3	2.54	14.9	14.0	12.95	4.85	1.75	7.0	5.57	4.17	12.09
Upper quartile	1.8	0.88	2.03	1.6	1.26	1.15	0.95	3.53	0.77	1.12	1.6
Median	0.5	0.57	0.95	0.57	0.47	0.39	0.58	1.89	0.27	0.48	0.01
Lower quartile	0.0	0.32	0.3	0.02	0.04	0.26	0.3	0.71	0.02	0.07	0.0
Minimum	0.0	0.01	0.0	0.0	0.0	0.1	0.0	0.08	0.0	0.02	0.0
Variance	4.445	0.363	3.002	3.031	2.781	1.984	0.235	3.781	1.415	1.704	6.844
Std. Dev.	2.108	0.603	1.733	1.741	1.668	1.408	0.484	1.944	1.189	1.305	2.616
CV	1.67	0.843	1.167	1.483	1.636	1.412	0.874	0.859	1.707	1.508	1.945

Figure 14-8: Box & Whisker Plot for Zinc in H Lenses



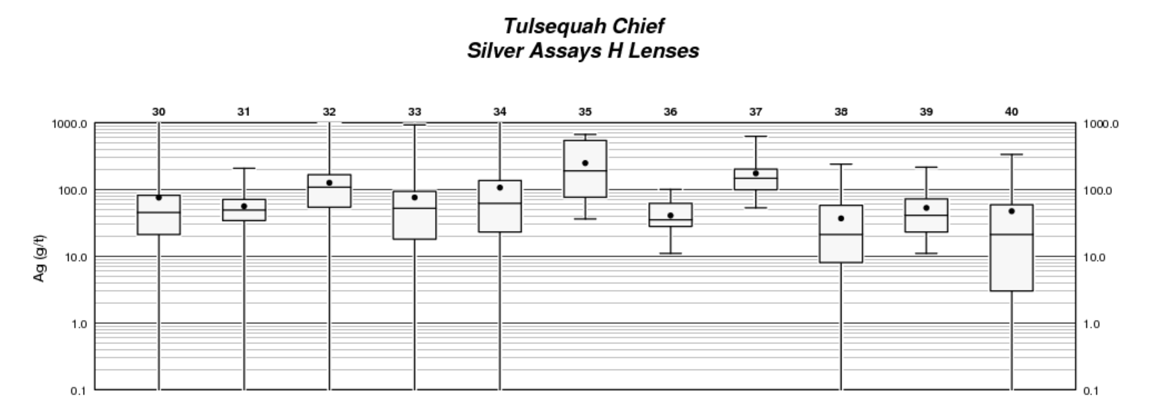
Lens	30	31	32	33	34	35	36	37	38	39	40
Number of data	946	34	477	619	244	11	11	19	59	13	79
Mean	7.698	4.902	8.039	6.245	5.732	4.777	3.072	11.467	2.946	5.718	4.985
Maximum	38.2	13.25	39.0	38.5	42.7	12.5	8.65	25.09	25.1	26.97	36.7
Upper quartile	12.9	8.78	11.56	8.6	8.0	7.9	4.75	16.43	3.32	12.7	6.37
Median	5.1	4.26	6.35	3.9	3.4	7.22	3.28	12.9	1.4	2.67	1.07
Lower quartile	1.0	1.63	2.25	0.6	0.71	1.71	1.24	4.56	0.44	0.54	0.01
Minimum	0.0	0.07	0.0	0.0	0.0	0.01	0.02	1.2	0.01	0.05	0.0
Variance	61.824	15.915	52.912	52.125	50.942	15.736	6.948	44.196	22.794	64.24	68.291
Std. Dev.	7.863	3.989	7.274	7.22	7.137	3.967	2.636	6.648	4.774	8.015	8.264
CV	1.021	0.814	0.905	1.156	1.245	0.83	0.858	0.58	1.621	1.402	1.658

Figure 14-9: Box & Whisker Plot for Gold in H Lenses



	30	31	32	33	34	35	36	37	38	39	40	
Number of data	946	34	477	619	244	11	11	19	59	13	79	Number of data
Mean	1.598	2.049	3.343	2.208	3.374	5.535	1.799	2.484	0.774	1.578	0.734	Mean
Maximum	24.0	5.9	59.9	37.03	50.27	26.5	7.07	10.62	3.99	5.62	3.77	Maximum
Upper quartile	2.06	2.98	4.0	2.95	3.08	7.06	1.36	2.67	1.01	1.99	1.03	Upper quartile
Median	1.03	1.41	2.56	1.71	1.65	3.36	1.2	1.85	0.47	1.54	0.69	Median
Lower quartile	0.34	0.89	1.3	0.63	0.74	2.5	0.83	1.4	0.27	0.79	0.06	Lower quartile
Minimum	0.0	0.03	0.0	0.0	0.0	1.07	0.31	0.96	0.02	0.14	0.0	Minimum
Variance	4.343	2.524	15.862	6.99	37.797	47.612	3.748	4.99	0.707	1.838	0.568	Variance
Std. Dev.	2.084	1.589	3.983	2.644	6.148	6.9	1.936	2.234	0.841	1.356	0.754	Std. Dev.
CV	1.304	0.775	1.191	1.198	1.822	1.247	1.076	0.899	1.087	0.859	1.027	CV

Figure 14-10: Box & Whisker Plot for Silver in H Lenses



	30	31	32	33	34	35	36	37	38	39	40	
Number of data	946	34	477	619	244	11	11	19	59	13	79	Number of data
Mean	75.428	55.941	125.573	75.715	106.435	247.909	40.636	174.0	36.746	52.846	47.165	Mean
Maximum	1519.0	207.0	1040.0	929.0	1100.0	668.0	100.0	621.0	238.0	215.0	333.0	Maximum
Upper quartile	82.0	71.0	166.0	93.0	136.0	540.0	62.0	201.0	58.0	72.0	59.0	Upper quartile
Median	45.0	49.0	108.0	52.0	62.0	190.0	35.0	147.0	21.0	41.0	21.0	Median
Lower quartile	21.0	34.0	54.0	18.0	23.0	76.0	28.0	99.0	8.0	23.0	3.0	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	36.0	11.0	53.0	0.0	11.0	0.0	Minimum
Variance	13516.2	1595.88	12104.9	9209.07	21446.9	52434.3	654.777	17123.1	1976.16	2843.05	4625.18	Variance
Std. Dev.	116.259	39.948	110.022	95.964	146.448	228.985	25.589	130.855	44.454	53.32	68.009	Std. Dev.
CV	1.541	0.714	0.876	1.267	1.376	0.924	0.63	0.752	1.21	1.009	1.442	CV

Zone 30 (zone MW) represents all the drill samples found within the mine workings. These samples were reclassified to their appropriate zone for modeling, either 33 or 34. Zone 35 appears to demonstrate a higher copper and silver concentration and zone 37 appears to have higher zinc content than all the other zones. However, these slight differences are attributed to a small sample population comprising zones 35 and 37.

14.4 Grade Correlation

Pb-Zn, Zn-Cu, Pb-Ag, Au-Ag, Au-Cu, and Cu-Ag relationships were evaluated; results are summarized in Table 14-3. Usually in VMS deposits, Pb-Zn, Pb-Ag, and Au-Ag display good positive correlations. At Tulsequah Chief, Pb and Zn correlate well together though the relationship becomes poor at higher grades, which is usually expected. Ag and Pb correlate reasonably well, but correlation coefficients are not as high as would normally be expected if all the silver were associated with the galena. Zn-Cu relationships at Tulsequah Chief show typical mixed weak correlations and independent trends. This probably reflects the typical zoning patterns often associated with VMS deposits.

Table 14-3: Assay Correlation Coefficient Matrix for Tulsequah Deposit

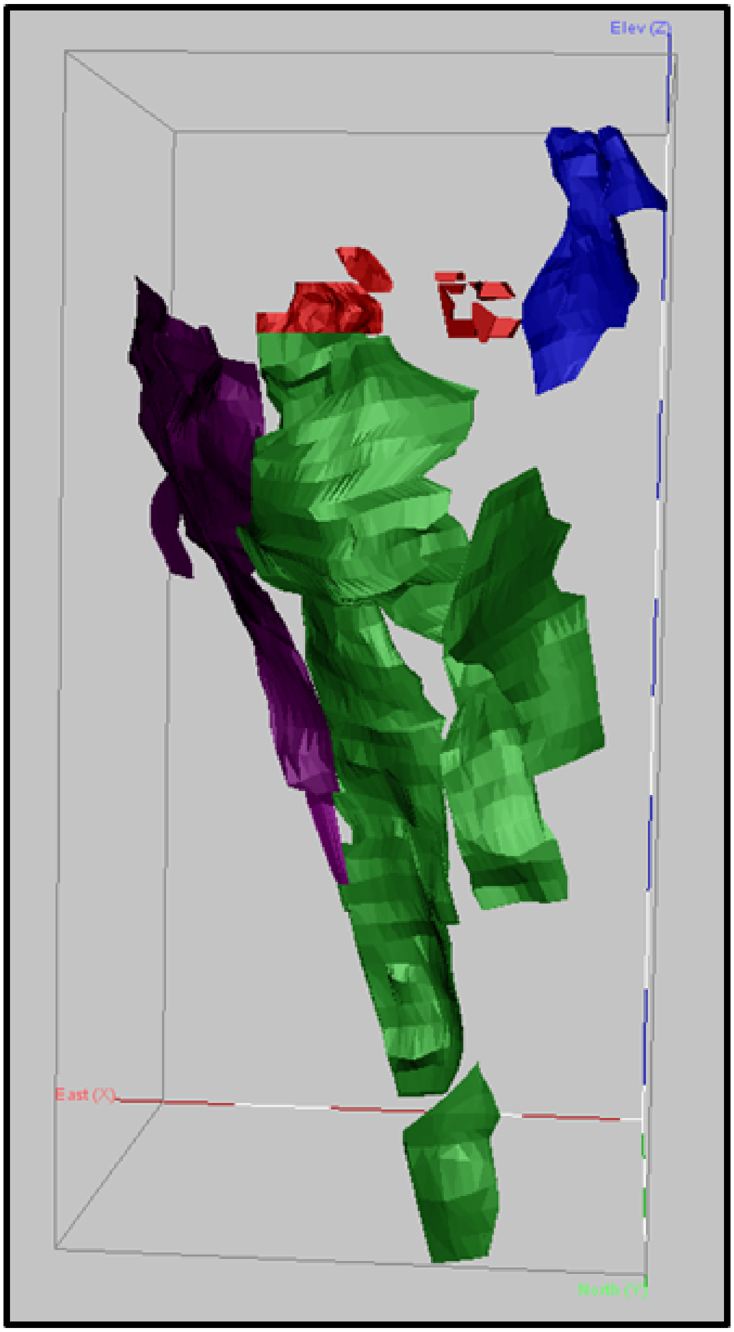
Metal	Zn	Cu	Pb	Au	Ag
Zn	1.000	0.111	0.593	0.196	0.316
Cu	0.114	1.000	-0.009	0.239	0.373
Pb	0.593	-0.009	1.000	0.240	0.420
Au	0.196	0.239	0.240	1.000	0.554
Ag	0.316	0.373	0.420	0.544	1.000

14.5 Solid Body Modeling

The Tulsequah Chief deposit comprises seventeen distinct semi-massive to massive sulphide lenses termed G1, G2, G3 and G4, H1to H10 and AEX_U, AEX_M, and AEX_L. The H series are separated from the G lenses by the 5300 fault. The G Zone lenses lie east of the fault and trend northerly with steep westerly dips. The H Zone lenses lie between the 5300 and 4400 faults and are distributed around and along the plunge line of an upright fold having a 55° to 65° plunge to the northwest (~325 degrees azimuth). The A-Extension Zone consists of three sub-parallel zones simply named AEX_L, AEX_U and AEX_23. The zones occur west of the 4400E fault and may represent the faulted extension of the H series lenses within the Tulsequah Chief mine (Figure 14-11).

All zones were modelled in oblique section perpendicular to the lens and validated on plan views. The solid models were prepared by Chieftain and verified by SRK.

Figure 14-11: 3D View Looking South of the Mineralized Lenses



Note: G lenses (G1 to G4) are in magenta, H lenses (H 1 to H10) are in green. A lenses (AEX_U, AEX_M and AEX_L are in blue) and lenses within the old mine (MW) are in red. Markers on the axes are 100 m in length.

Individual sulphide lenses were determined by relative position to one of four mineralized stratigraphic intervals. Contacts between lenses are sharp. Most of the mineralization resides in the G1, H2, H3, and H4 lenses.

Sulphide metal zoning is present but complex. In the three larger H lenses, distinct copper-rich regions are present that generally occupy the thicker portions of the lens.

Most of these Cu-rich areas, though, also contain significant zinc mineralization.

The solid models were used to code the drill hole data and block model cells. The individual lenses were reviewed to determine appropriate estimation or grade interpolation parameters.

14.6 Compositing

All assay data were composited to a fixed length prior to estimation. SRK evaluated the assay lengths to determine an optimum composite length. Less than eight percent of the samples are longer than 2 m and all of these were from the old drill holes drilled by Cominco in the mid-1950s. The mean of all the sample lengths is 1.22 m. For the purpose of resource estimation, all assay intervals within the mineralized units were composited to 2 m. The assays were composited into 2 m downhole composites. The compositing honoured the lens zone by breaking the composites on the lens code values. The block model was coded with respective lens code prior to estimation. Any composite with length less than 1 m after compositing was added to the previous composite length and the composites were recalculated before estimation. Twelve composites that were less than 1 m could not be linked to previous composites, because the mineralized zone consisted of a single composite value; these composites remained in the database and were used during estimation. The statistical properties of the composited metal data by lens are summarized in Table 14-4.

Table 14-4: Statistical Data for 2 m Capped Composites by Lens

Element	Lens	Mean	Q25	Q50	Q75	Max	No. Of Comps
Cu	AEX_U	0.36	0.28	0.37	0.5	0.82	15
	AEX_M	0.2	0	0	0.21	2.7	38
	AEX_L	0.65	0	0.26	0.83	6.42	95
	G1	1.09	0.29	0.65	1.45	8.83	153
	G2	1.07	0.25	0.51	1.4	7.14	33
	G3	0.41	0.1	0.51	0.63	0.96	8
	G4	0.75	0.65	1.21	1.21	1.21	3
	H1	0.69	0.22	0.42	0.72	4.91	19
	H2	1.55	0.59	1.16	2.09	7.12	259
	H3 & MW	1	0.06	0.6	1.28	10	1326
	H4 & MW	0.72	0	0.25	0.88	9.79	319
	H5	3.09	1.33	3.57	5.56	6.98	7
	H6	0.44	0.2	0.54	0.66	1.01	7
	H7	0.54	0	0.44	1.12	1.8	14
	H8	0.29	0.02	0.1	0.42	2.59	40

Element	Lens	Mean	Q25	Q50	Q75	Max	No. Of Comps
	H9	0.98	0.83	0.92	2.08	2.08	6
	H10	0.58	0.19	0.55	0.8	2.46	57
Pb	AEX_U	0.37	0.02	0.2	0.4	2.01	15
	AEX_M	0.96	0	0	0.63	7.97	38
	AEX_L	0.37	0	0.02	0.15	5.76	95
	G1	0.88	0.27	0.58	1.12	4.73	153
	G2	1.1	0.27	0.52	1.7	7.95	33
	G3	1.38	0.22	0.68	3.96	4.56	8
	G4	0.54	0.52	0.82	0.82	0.82	3
	H1	0.62	0.38	0.53	0.7	1.57	19
	H2	1.34	0.44	1.07	1.88	7.35	259
	H3 & MW	0.95	0	0.41	1.37	8.4	1326
	H4 & MW	0.6	0	0.02	0.83	5.55	319
	H5	1.07	0.27	1.11	1.46	3.45	7
	H6	0.49	0.3	0.52	0.81	1.23	7
	H7	1.21	0	1.79	2.23	2.54	14
	H8	0.46	0.02	0.15	0.49	3.71	40
	H9	0.6	0.37	0.49	1.75	1.75	6
	H10	1.07	0	0.08	1.8	6.41	57
Zn	AEX_U	4.35	3.08	4.01	6.85	9.79	15
	AEX_M	2.97	0	0	3.33	20.61	38
	AEX_L	1.93	0	0.41	2.89	16.32	95
	G1	4.25	1.82	3.48	5.37	24.7	153
	G2	5.48	1.86	3.56	6.63	27	33
	G3	3.34	0.74	2.4	8.43	9.35	8
	G4	3.32	3.4	3.52	3.52	3.52	3
	H1	4.17	1.86	3.74	5.26	11.48	19
	H2	7.48	3.19	6.62	11.14	30	259
	H3 & 30	5.72	0.22	3.45	9.28	30	1326
	H4 & 30	3.1	0	0.84	4.71	30	319
	H5	5.55	4.85	7.6	7.76	10.94	7
	H6	2.68	1.86	2.37	3.56	7.05	7
	H7	6.32	0	3.66	13.34	17.51	14
	H8	2.05	0.03	1.05	3.15	19.23	40
	H9	3.84	2.89	5.16	8.02	8.02	6
	H10	3.85	0.01	1.76	5.84	21.64	57
Au	AEX_U	0.36	0.07	0.14	0.26	2.04	15
	AEX_M	0.37	0	0.03	0.36	3.4	38
	AEX_L	0.82	0	0.34	1.11	7.45	95
	G1	2.41	0.88	1.71	3.21	10.83	153
	G2	2.27	0.67	1.28	2.06	17.38	33
	G3	2.19	1.25	1.5	3.95	6.61	8
	G4	0.96	0.96	1.09	1.09	1.09	3
	H1	1.85	1.07	1.96	2.95	5.18	19
	H2	3.05	1.52	2.51	4.03	14.77	259
	H3 & MW	1.35	0.09	0.89	1.84	20.69	1326
	H4 & MW	1.72	0	0.71	2.28	24.88	319
	H5	4.63	3.07	5.71	7.06	9.27	7
	H6	2.02	0.92	1.26	2.44	7.07	7
	H7	1.58	0	1.7	2.63	5.44	14
	H8	0.56	0.16	0.38	0.75	2.77	40

Element	Lens	Mean	Q25	Q50	Q75	Max	No. Of Comps
	H9	1.29	1.1	1.28	2.9	2.9	6
	H10	0.71	0.16	0.56	1.06	2.54	57
Ag	AEX_U	18.4	5.13	14.4	37.98	56.85	15
	AEX_M	17.53	0	0.2	14.28	296.14	38
	AEX_L	29.84	0	9.28	25.93	560.38	95
	G1	73.59	29.61	53.49	98.55	365.61	153
	G2	96.25	21.96	46.76	117.77	537.81	33
	G3	93.42	39.64	53.17	240.32	289.2	8
	G4	50.8	58.09	60	60	60	3
	H1	49.04	36.85	50.35	67.07	134.05	19
	H2	116.41	58.17	106.05	157.12	443.38	259
	H3 & MW	52.49	6.89	35.92	69.18	600	1326
	H4 & MW	62.95	0	29.73	98.32	597	319
	H5	238.68	142	293.25	410.04	572.7	7
	H6	38.22	32.27	33.54	62	74.3	7
	H7	97.19	0	135.96	170.33	239.46	14
	H8	27.46	3.21	11.57	36.71	139.05	40
	H9	41.97	35.61	38.13	97.54	97.54	6
	H10	42.56	3.26	20.57	53.7	224.51	57

14.7 Evaluation of Outliers

Block grade estimates may be unduly affected by high-grade outliers. Therefore, assay data were evaluated for high-grade outliers and capped to values determined based on decile and probability plot analyses.

Generally, the distributions do not indicate a problem with extreme grades; however, a few outliers do exist and SRK decided to cap the assay data prior to compositing. Capping levels are summarized in Table 14-5.

SRK did estimate the model using uncapped values to compare the influence of capping on the total resource numbers. The difference between capped and uncapped estimates was negligible, less than 2% differences between the two estimates.

Table 14-5: Capping Levels

Metal	Cap Level	No Cap	CoV Uncapped	CoV Capped	Metal Loss (%)
Zn	30%	38	1.10	1.08	0.9
Cu	10%	17	1.33	1.28	0.9
Pb	10%	17	1.55	1.43	1.7
Au	25 g/t	8	1.52	1.31	2.1
Ag	600 g/t	22	1.30	1.19	2.0

14.8 Statistical Analysis & Variography

Paucity of data per lens combined with complex metal zonation patterns precluded detailed variographic analysis by lens. Attempts were made to establish robust variograms but results were mixed at best. Variography was useful in determining maximum ranges but these were similar to the long axes of the mineralized bodies, which was to be expected. The patterns of anisotropy demonstrated by the various variograms mimicked the general attitudes of the H lenses: northeast trending with a moderately steep dip or plunge to the northwest. Ranges were 100 m along strike, 80 m down dip or plunge, and 15 m across the dip or plunge. For this reason, SRK decided that an ID2 interpolation with searches oriented parallel to the long axes of the mineralized lenses was probably better than using ordinary kriging with poor or inconsistent variograms.

14.9 Block Model & Grade Estimation

Assay grades were interpolated by inverse distance weighting to the second power (ID2) for copper, zinc, lead, gold and silver values. The interpolation was carried out in two separate passes and separate search ellipses were used for H and G lenses. Table 14-6 summarizes the search parameters used to interpolate the block model.

Table 14-6: Search Ellipse Parameters

Zone	Estimator	Search Pass	Search Type	Rotation			Search Ellipse Size			Number of Composites		Max per DDH
				Z	Y	Z	X (m)	Y (m)	Z (m)	Min	Max.	
G	ID2	1	Ellipse	-35	-50	-30	70	60	40	3	8	2
H8	ID2	1	Ellipse	-40	-50	50	40	60	20	3	8	2
H1-7, 9&10	ID2	1	Ellipse	-80	-60	0	60	45	30	3	8	2
All Zones	ID2	2	Sphere	0	0	0	120	120	120	2	12	No restriction
A Extn	ID2	1	Ellipse	-35	-50	-30	70	60	40	3	8	2

The first pass required that at least two drill holes and three composites be available within the search ellipse to estimate a grade within a block. Where several composites were found within the search ellipse, a maximum of eight composites were used to interpolate a grade value. The second pass required that at least two composites be present within the search ellipse for grade interpolation with no restrictions on the number of drill holes. The maximum number of composites was set to 12.

Bulk density values were estimated into the resource model by inverse distance weighting to the second power. Search parameters used were the same as those used for grade interpolation. A maximum of eight and minimum of three composites were used for the interpolation. In the event a block was not estimated, a default density value was assigned equal to 2.70 g/cc. In upper portions of H3 and H4 lenses, above the 5200 level, for drill holes that contained no density measurements, a default value equal to 3.50 was assigned to the mineralized blocks containing greater than 2% combined (Pb + Zn). This value was derived from the average of the bulk sample measurements of the mineralized units taken from the 2004, 2006 and 2011 drilling.

14.10 Model Validation & Sensitivity

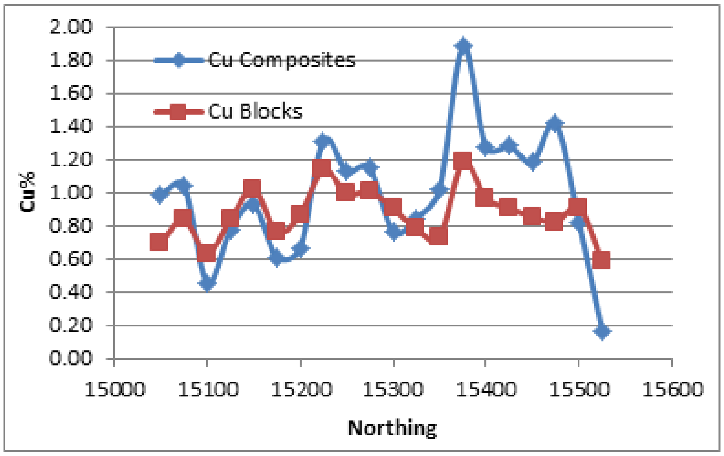
14.10.1 Visual Inspection

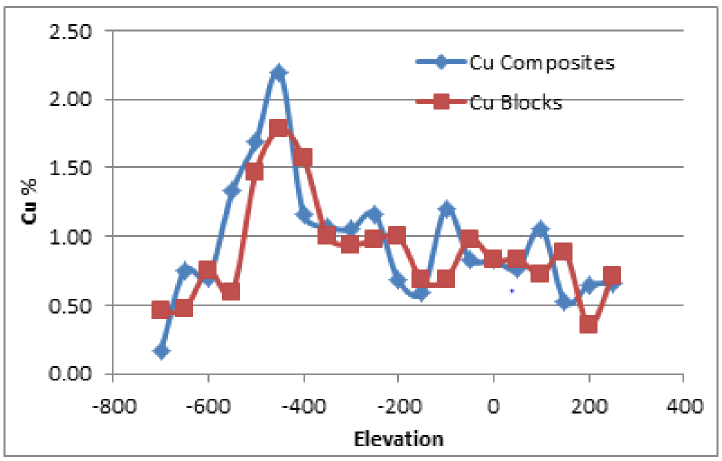
SRK completed a detailed visual validation of the Tulsequah Chief block model. The model was checked for proper coding of drill hole intervals and block model cells, in both section and plan. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values.

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparison with average block estimates along east-west, north-south, and horizontal swaths.

Figure 14-12 to Figure 14-16 show the swath plots in the mineralized domain. The average composite grades and the average estimated block grades are quite similar in all directions. There are some indications that the block estimates at some locations are slightly higher and the zinc and copper block grades appear to be lower than the composite grades in the northern swath between 15,400 and 15,500, this is attributed to the paucity of data in this area of the model.

Figure 14-12: Swatch Plots of Copper Composites & Copper Block Grades





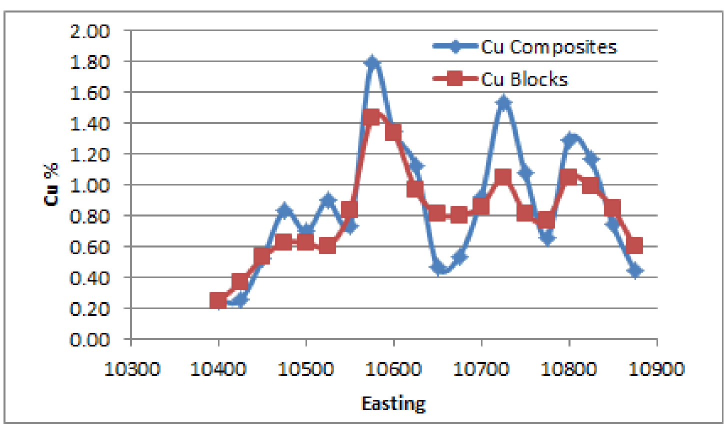


Figure 14-13: Swatch Plots of Lead Composites & Lead Block Grades

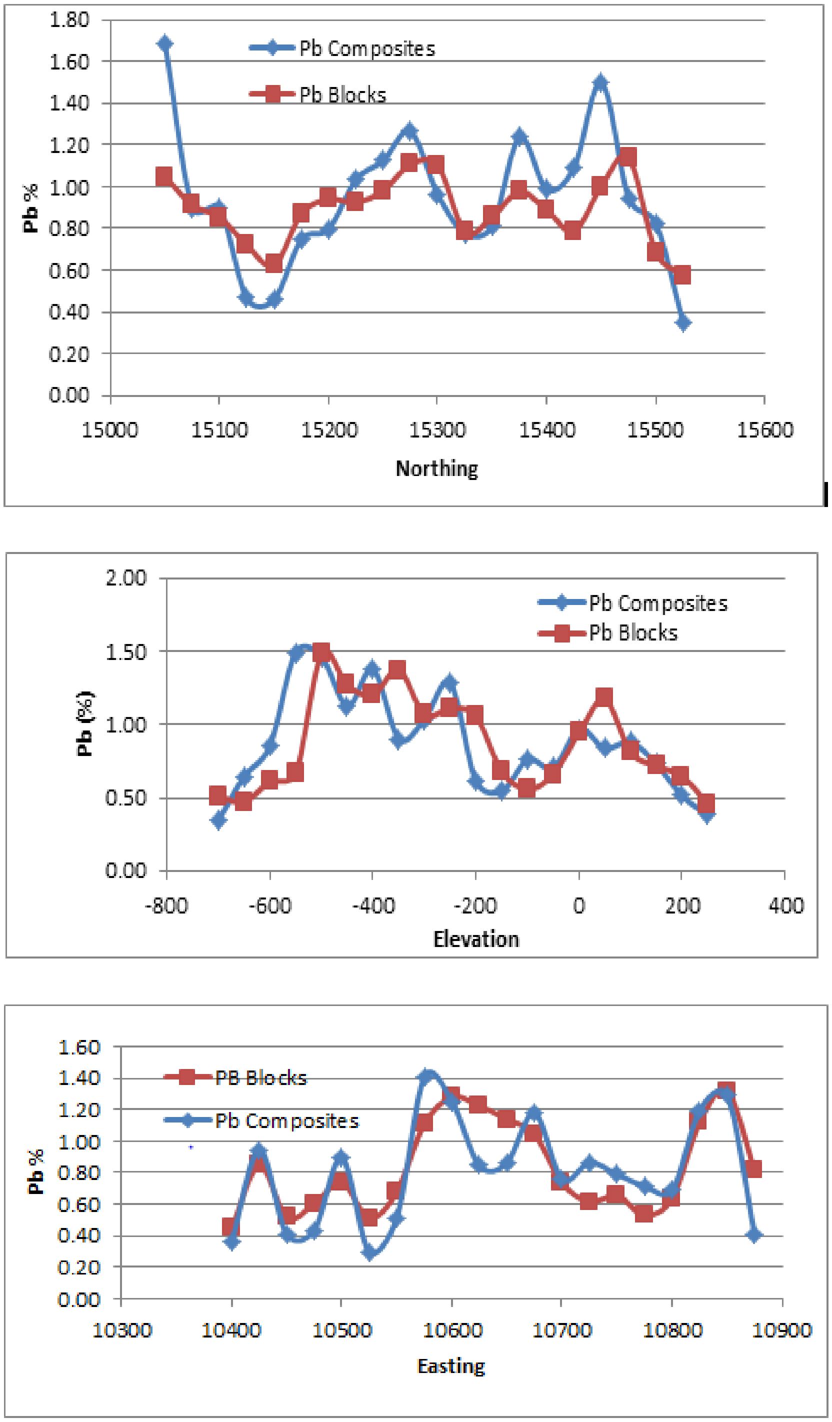
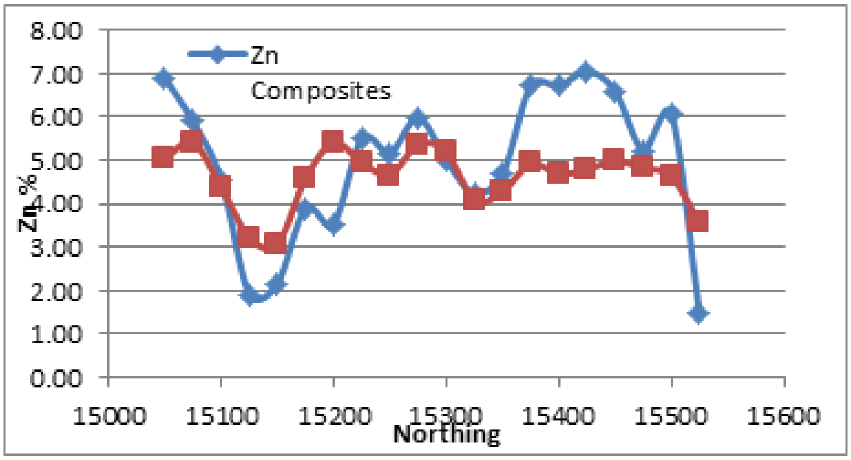
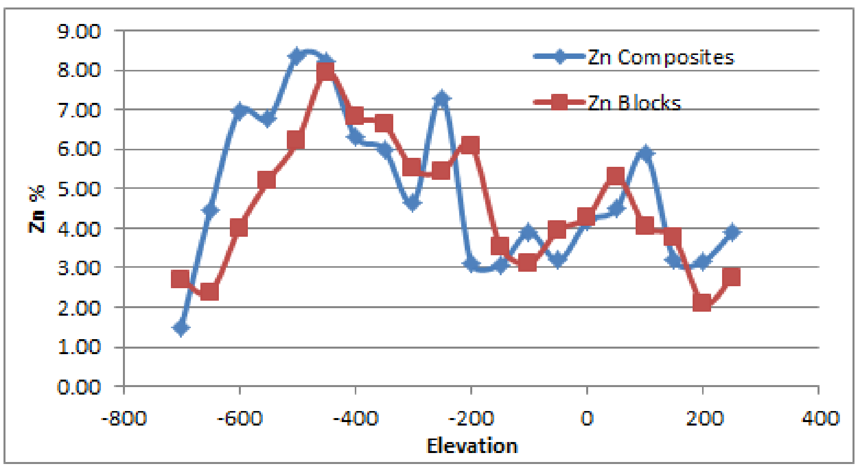


Figure 14-14: Swatch Plots of Zinc Composites & Zinc Block Grades





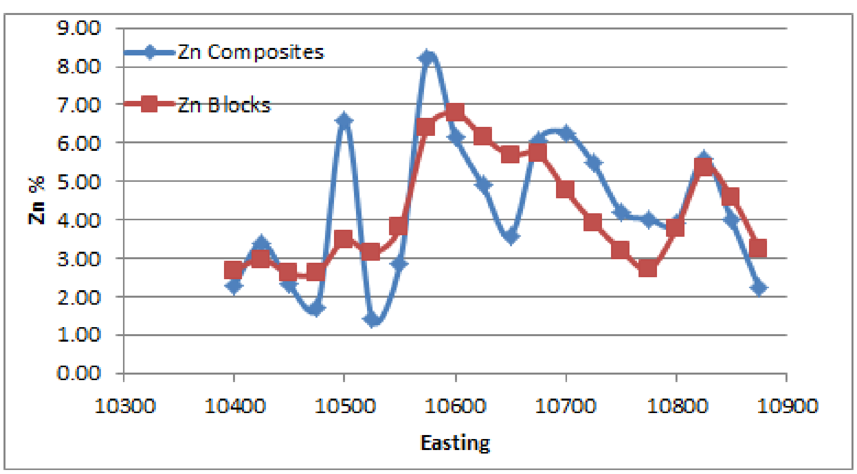
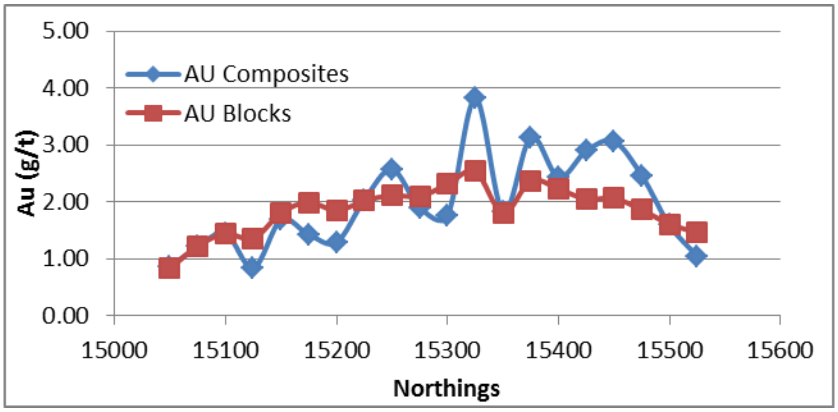
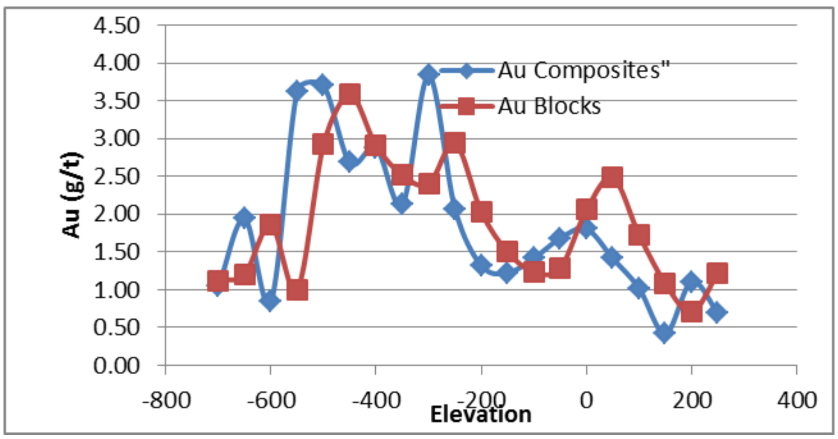


Figure 14-15: Swatch Plots of Gold Composites & Gold Block Grades





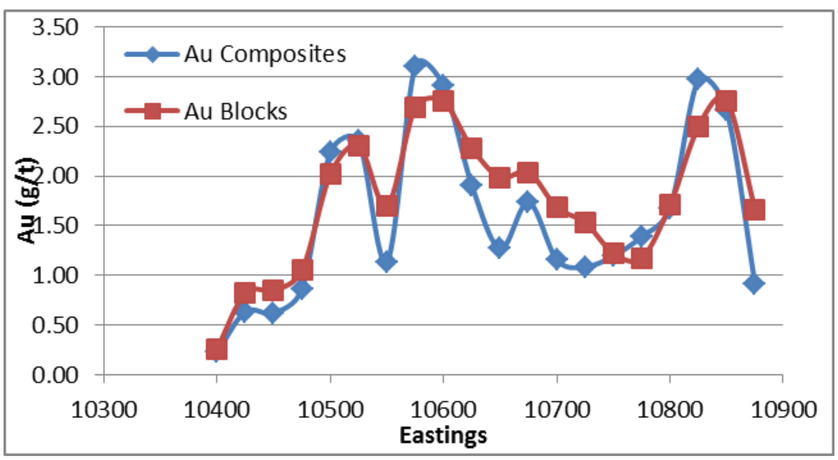
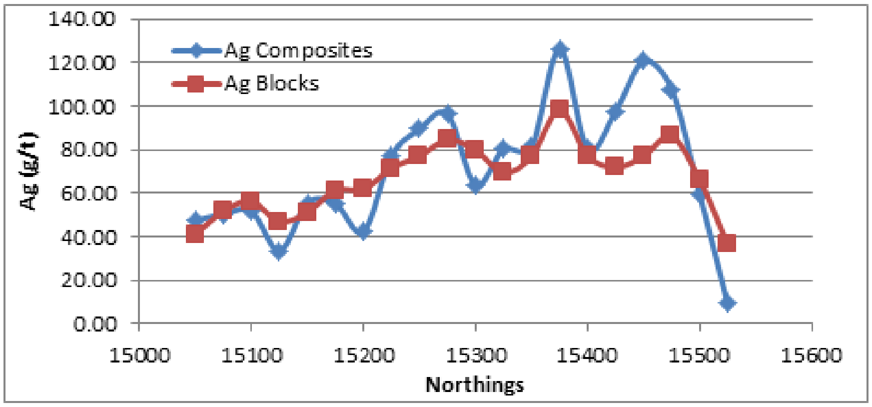
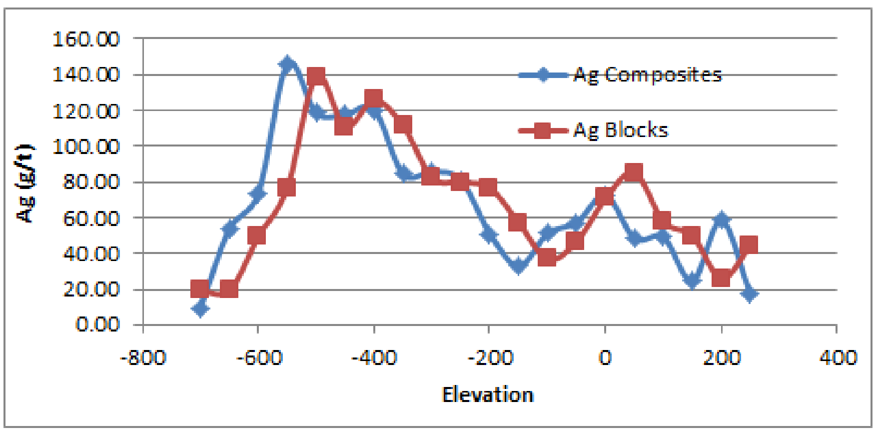
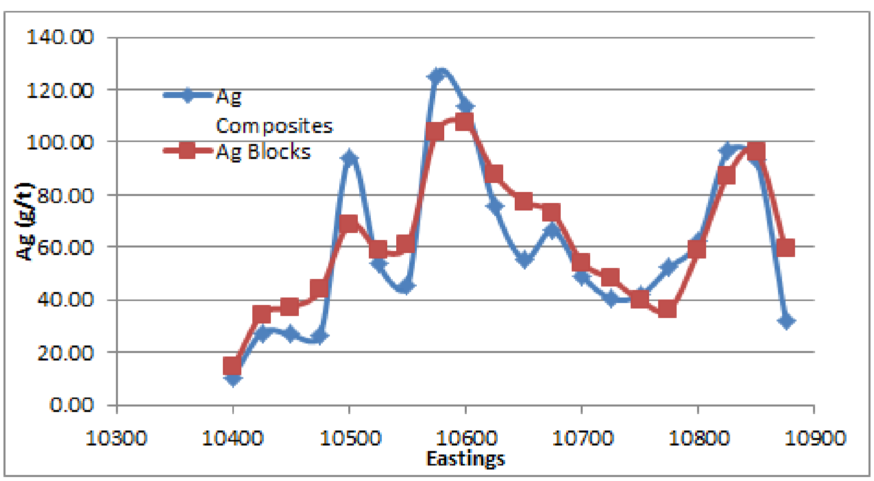


Figure 14-16: Swatch Plots of Silver Composites & Silver Block Grades







14.10.2 Model Check for Bias

The model was checked for global bias by comparing the ID^2 model results with a separate estimate prepared using the nearest neighbour (NN) method of estimation.

The nearest-neighbour method of estimation essentially de-clusters the data and produces an estimate of the average value. When compared at a 0.0 cut-off, the NN method offers a good basis for checking the performance of different estimation methods.

The NN estimate returned similar values to the ID^2 model when compared at a 0.0 cut-off and even when compared at the US$100 Eq cut-off the two models returned similar overall values (Table 14-7). The NN grades are marginally higher than for the ID^2 estimate but the tonnes are slightly lower which yields a very similar overall total metal content, about 3% less metal is reported in the NN model than in the ID^2 model at the US$100 Eq cut-off.

Table 14-7: Percent Difference between NN & ID^2 Results at US$100Eq Cut-off

	$(NN-ID^2)/NN$ in %	Metal Grade Differences in % $(NN-ID^2)/NN$				
Class	**tonnes**	**Zn**	**Cu**	**Pb**	**Au**	**Ag**
Indicated	-7.6	4.6	6.7	5.9	5.8	5.7
Inferred	-15.1	-16	-10	-17	-33	-21
Indicated + Inferred	-7.8	4.4	6.5	5.5	5.3	5.3

14.11 Mineral Resource Classification

Block model quantities and grade estimates for the Tulsequah Chief project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Dr. Gilles Arseneau, P.Geo (APEGBC, # 23474), an independent qualified person.

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

SRK is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced at 20 to 30 m.

Generally, for mineralization exhibiting good geological continuity investigated at an adequate spacing with reliable sampling information accurately located, SRK considers that blocks estimated during the first estimation run can be classified in the Indicated category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005). For those blocks, SRK considers that the level of confidence is sufficient to allow appropriate application of technical and economic parameters to support mine planning and to allow evaluation of the economic viability of the deposit. Those blocks can be appropriately classified as Indicated.

Conversely, blocks estimated during the second pass are best appropriately classified in the Inferred category because the confidence in the estimate is insufficient to allow for the meaningful application of technical and economic parameters or to enable an evaluation of economic viability.

14.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

> "[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

The "reasonable prospects of economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considered that all portions of the Tulsequah Chief deposit are amenable for underground mining.

The block model tonnes and grade estimates were reviewed to determine the portions of the Tulsequah Chief deposit having "reasonable prospects for economic extraction" from an underground mine, based on parameters summarized in Tables 14-8 and 14-9.

Table 14-8: US$Eq Calculation Data

Metal	Price (US$)	Recovery (%)
Au	1,275/oz	81.8
Ag	21.00/oz	79.5
Cu	3.25/lb	87.8
Pb	1.1/lb	44.5
Zn	1.1/lb	88

Table 14-9: Conceptual Assumptions Considered for Underground Resource Reporting

Parameter	Value	Unit
Exchange rate	1:01	US$/$CND
Mining costs	$32.00	US$/t mined
Process cost	$18.00	US$/t of feed
General and Administrative	$10.00	US$/t of feed
Power	$20.00	US$/t of feed
Transport	$20.00	US$/t of feed
Total Costs	$100.00	US$/t of feed
Assumed mining rate	2,000	t/d

SRK considers that the blocks that had total dollar values above US$100.00 satisfied the "reasonable prospects for economic extraction" and could be reported as a mineral resource as summarized in Table 14-10.

Table 14-10: Mineral Resource Statement of Tulsequah Chief Deposit (SRK, March 15, 2012)

Location	Category	Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Old Mine (above 5200)	Indicated	403,000	1.28	0.97	6.02	1.52	71
	Inferred						
New Mine (below 5200)	Indicated	6,113,000	1.19	1.13	6.00	2.50	88
	Inferred	204,000	0.67	0.76	4.02	1.81	62
A Extension	Indicated	247,000	0.86	0.59	2.91	1.34	44
	Inferred						
Total Indicated		**6,762,000**	**1.19**	**1.1**	**5.89**	**2.4**	**85**
Total Inferred		**204,000**	**0.67**	**0.76**	**4.02**	**1.81**	**62**

*Mineral resources are reported in relation to a conceptual mining outline. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate. **Underground mineral resources are reported at a cut-off grade of US$100. Cut-off grades are based on a price of US$1,275/oz of gold, US$21/oz for silver, US$1.10/lb for zinc and lead and US$3.25 for copper and recoveries of 81.8% for gold, 79.5 for silver, 87.8 for copper, 44.5% for lead and 88% for zinc.

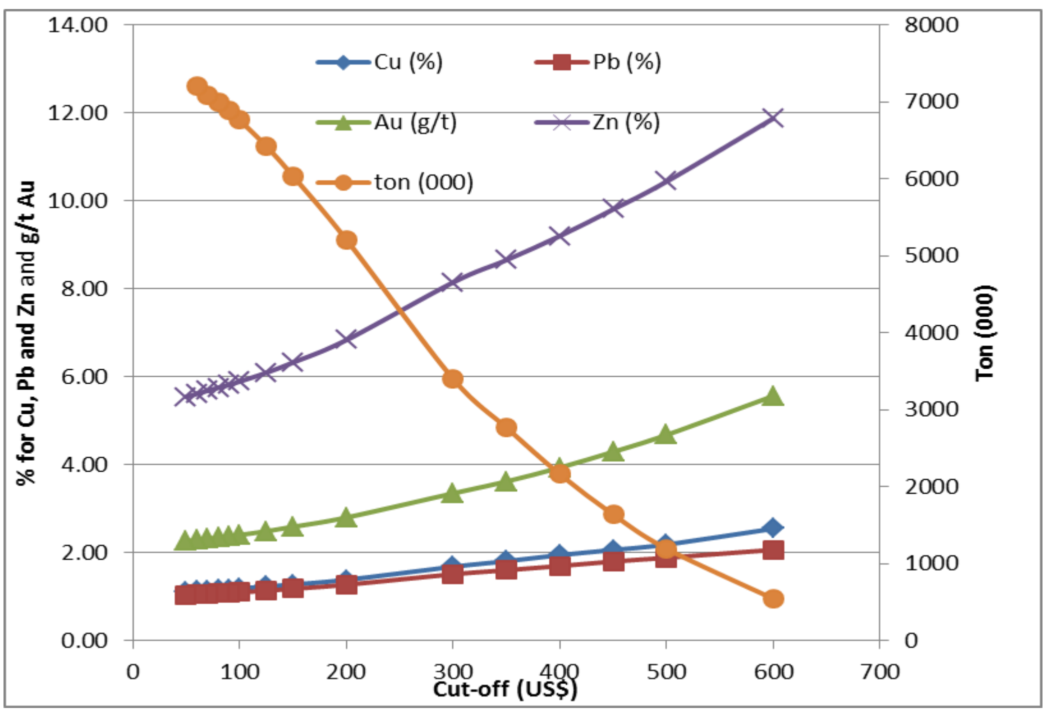
14.13 Grade Sensitivity Analysis

The mineral resources of the Tulsequah Chief project are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the Indicated mineral resource and grade estimates are presented in Table 14-11 at different cut-off grades. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. Figure 14-17 presents this sensitivity as grade tonnage curves for the indicated mineral resource.

Table 14-11: Indicated class Block Model Quantities & Grade Estimates*, Tulsequah Deposit, at Various Cut-off Grades

Cut-off (US$)	Tonnage	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
>150	6,036,550	1.28	1.18	6.32	2.59	92
>140	6,188,731	1.26	1.16	6.23	2.55	91
>130	6,344,480	1.24	1.14	6.14	2.51	89
>120	6,497,743	1.22	1.13	6.05	2.47	88
>110	6,635,112	1.20	1.11	5.97	2.43	87
>100	6,762,000	1.19	1.10	5.89	2.40	85
>90	6,888,018	1.17	1.08	5.81	2.37	84
>80	6,988,856	1.16	1.07	5.75	2.34	83
>70	7,091,335	1.15	1.06	5.68	2.32	82
>60	7,202,461	1.13	1.05	5.61	2.29	81
>50	7,310,177	1.12	1.03	5.54	2.26	80
>40	7,410,730	1.11	1.02	5.48	2.24	79

Figure 14-17: Grade Tonnage Curves for the Indicated Mineral Resources



15 Mineral Reserve Estimate

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

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

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

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

A “Proven Mineral Reserve” is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A “Probable Mineral Reserve” is the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource, demonstrated by at least a Preliminary Feasibility Study. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.1 Cut-off Criteria

Similar to the resource cut-off calculations in Section 14.11, equivalent mining reserve values were calculated from block model tonnes and grades to define an economic cut-off grade (COG) to determine the mineable portions of the Tulsequah Chief deposit. The parameters used for the calculation were based on the data shown in Tables 15-1 and 15-2.

Table 15-1: Equivalent Value Calculation Parameters

Metal	Price (US$)	Recovery (%)
Au	1,350/oz	81.8
Ag	22.00/oz	79.5
Cu	3.10/lb	87.8
Pb	1.10/lb	44.5
Zn	1.10/lb	88.0

Table 15-2: Mining Planning Assumptions for Reserve Reporting

Parameter	Value	Unit
Exchange rate	1:01	US$/C$
Mining costs	$30.00	US$/t mined
Process cost	$23.00	US$/t of feed
General and Administrative	$22.50	US$/t of feed
Power	$22.50	US$/t of feed
Transport	$27.00	US$/t of feed
Total Costs	**$125.00**	**US$/t of feed**
Assumed mining rate	2,000	t/d

Mineable blocks were defined based on COG values greater than US$125.00/t and a minimum mining width (MMW) of 3 m.

AMC Consultants (AMC) created 3D wireframes as described below from their "Tulsequah Chief Project Feasibility Review" report dated December 2012.

> "Vertical outlines were digitized for each lens in sections every 5m, 6m or 7m along the strike of the lens where both the COG and MMW requirements were met.
>
> Each of the outlines was then offset 2.5m, 3m or 3.5m along strike to generate an overall mining shape which nominally followed the ore lens."

JDS imported the AMC stope shapes into Maptek Systems Inc. Vulcan™ software where each shape was reviewed and approved for stope design and mine planning.

15.2 Dilution

The wall dilution (waste that falls into the stope from outside the geometry of the stope shape) was calculated by assuming 0.70 m for longhole stopes and 0.50 m for cut and fill and ore development drifts for the entire length and height of the stope and/or drift. Both the quality condition of the walls and longhole drilling deviation are considered as key to minimizing wall dilution. The dilution is within the sulphide envelope and was assumed to carry the grades shown in Table 15-3.

Table 15-3: Dilution Grade Values

Metal	Dilution Grade
Au	0.124 g/t
Ag	3.319 g/t
Cu	0.038%
Pb	0.040%
Zn	0.197%

Dilution from pastefill is another source of dilution as some paste from adjacent stopes is expected to fall into the stope being mined or inadvertently scraped off the stope floors during mucking. Fill dilution of 0.15 m has been assumed for the entire length and width of the stope.

15.3 Ore Recovery

Mining ore recovery is a function of mineralized material left behind due to operational constraints typical in the mining process.

The longhole mining method is largely dependent on accuracy of longhole drilling and explosive detonation over 30 m distances to properly fracture the ore. Where holes deviate from the ore limits, some material will remain hung up and may never report to the floor for recovery.

Lesser factors considered to affect recoveries in longhole mining include ragged mucking floors and limited visibility for remote mucking.

Secondary stopes recognize higher recoveries due to improved probability of blasted mineralization making its way to the stope floor for mucking.

A mining recovery of 95% was assigned based on industry norms as well as JDS operational experience for remote mucking stopes of similar size and dip.

Cut-and-fill mining and ore sublevel drift development typically results in higher recoveries then bulk mining methods. Of highest import, the method ensures no fragmented material will be overlooked due to blind-spots or hang-up as often occurs in remote mucking scenarios.

Ore recovery of 100% has been assumed for cut-and-fill stoping and ore sublevel development.

15.4 Mineral Reserve Estimate

The mining stope and sublevel designs with dilution and ore recovery factors applied determined the mineral reserve estimate shown in Table 15-4.

Table 15-4: Mineral Reserve Estimate

Category	Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Probable	6,447,098	1.13	1.04	5.59	2.30	81.39

Underground mineral reserves are reported at a cut-off of US$125. Cut-off grades are based on a price of US$1,350/oz of gold, US$22/oz for silver, US$1.10/lb for zinc and lead and US$3.10 for copper and recoveries of 81.8% for gold, 79.5 for silver, 87.8 for copper, 44.5% for lead and 88% for zinc.

The mineral reserves identified in Table 15-3 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

The economic viability of the project is presented in Sections 21 and 22. The probable reserve estimates meet and comply with CIM definitions and NI 43-101 standards, including the main assumptions used in the definition of the reserves (i.e., metal prices, dilution, operating cost and recoveries).

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the mineral reserves or potential production.

16 Mining Methods

16.1 Introduction

The mine design and planning for Tulsequah Chief is based on the resource model completed by SRK dated March 15, 2012, as detailed in Section 14 of this report. The mine design and plan considers indicated mineral resources of the Tulsequah Chief deposit only. Inferred resources have been excluded from mine planning for this study. Inferred mineral resources are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to Measured and Indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

16.2 Mine Planning Criteria

Mine planning criteria are listed below:

- Preproduction period is approximately 15 months, with development ore mined and stockpiled in Q4 2015 for processing during commissioning in Q4 2015 and ramping to production in 2016.
- Full mine production is achieved in Q3 2016.
- Underground mining and maintenance carried out by Owner.
- Contract Alimak raise mining will be utilized.
- Conventional, trackless diesel-electric mining equipment will be utilized.
- Mined voids will be filled with pastefill and mine development waste.

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

Table 16-1: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	365
Shifts per Day	Shifts	2
Hours per Shift	Hour	10
Work Rotation	Two weeks in/one week out	2x1
Nominal Ore Mining Rate	t/d	2,000
Annual Ore Mining Rate	t/a	730,000
Ore Density	t/m^3	variable, from block model
Waste Density	t/m^3	2.70
Swell Factor		1.35

Cut-off grade, dilution and ore recovery criteria have been defined previously in Sections 15.1 to 15.3 of this report.

16.3 Geotechnical Criteria

The report on the geotechnical requirements for the Tulsequah Chief project (Dave West, 2012) draws on previous work by the following:

- B+L Rock Group Consulting Engineers (1995)
- Wardrop Engineering (2007)
- TetraTech (2012).

These data have been combined with level plans and sections showing the proposed mine openings; measured physical properties of the mine rocks: estimates of the rock mass quality from drill cores; an assessment of the in-situ stress to assess the long-term stability. The analysis has used the widely accepted Mathews/Potvin Stability Graph approach to calculate the stope dimensions and sill pillar dimensions were determined using the procedure established by Carter. The conclusions and recommendations are summarized below.

The following structures were identified at Tulsequah Chief Mine (Dip/Dip Direction).

1. Hangingwall Volcanics (Figure 16-1):

- Joint Set 1 – 86°/323°
- Joint Set 2 - 88°/273°
- Joint Set 3 - 67°/291°
- Joint Set 4 - 54°/108°
- Joint Set 5 - 46°/335°.

2. Felsic Volcanics and Massive Sulphides (Figure16- 2):

- Joint Set 1 (major) - 43°/115°
- Joint Set 2 (major) - 22°/206°
- Joint Set 3 (minor) - 64°/244°
- Joint Set 4 (minor) - 74°/336°.

Figure 16-1: Tulsequah Chief Mine – Hangingwall Volcanics

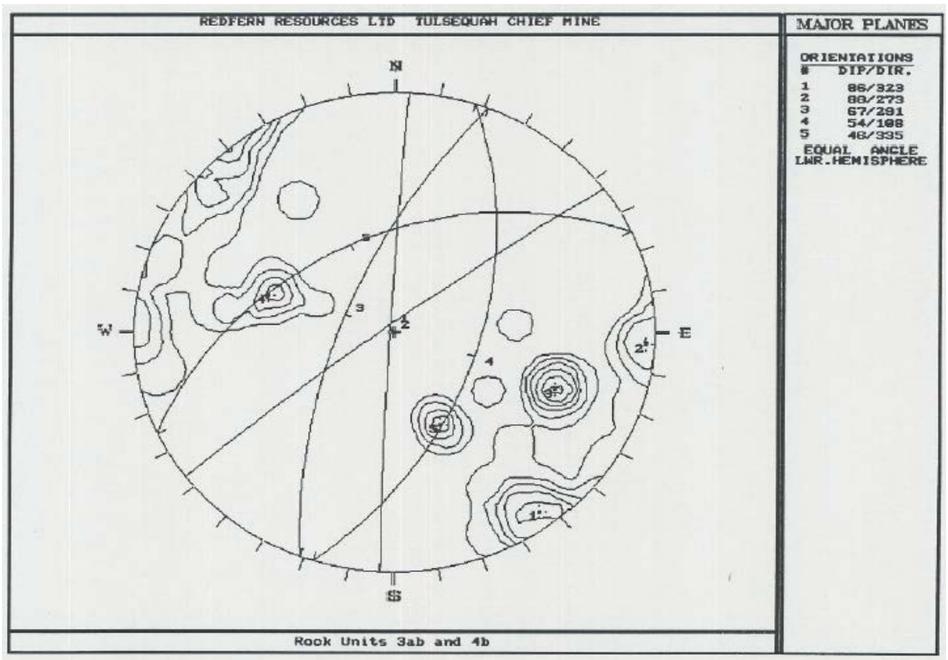
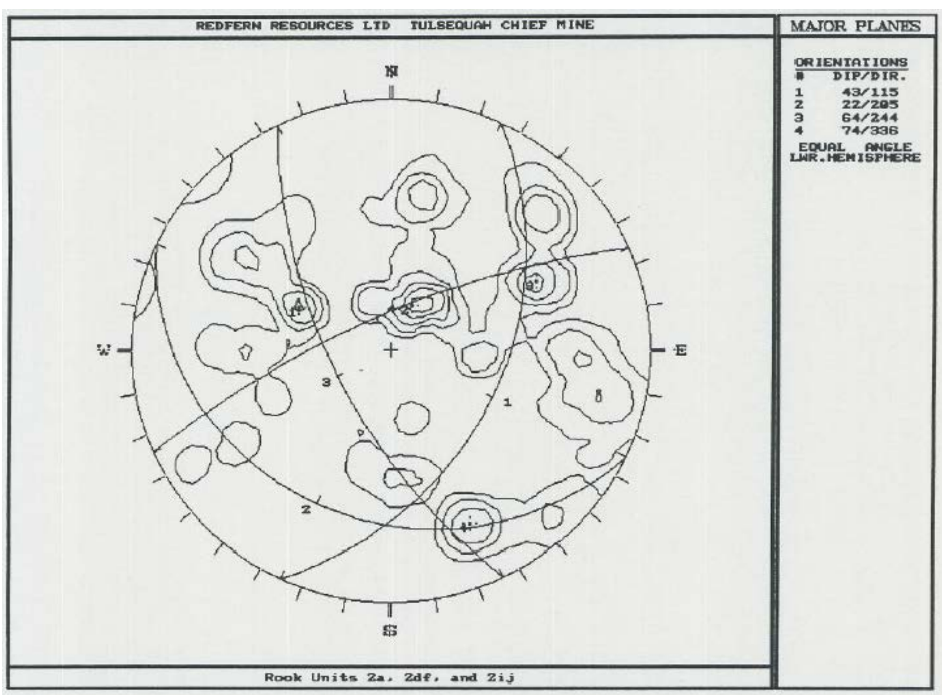


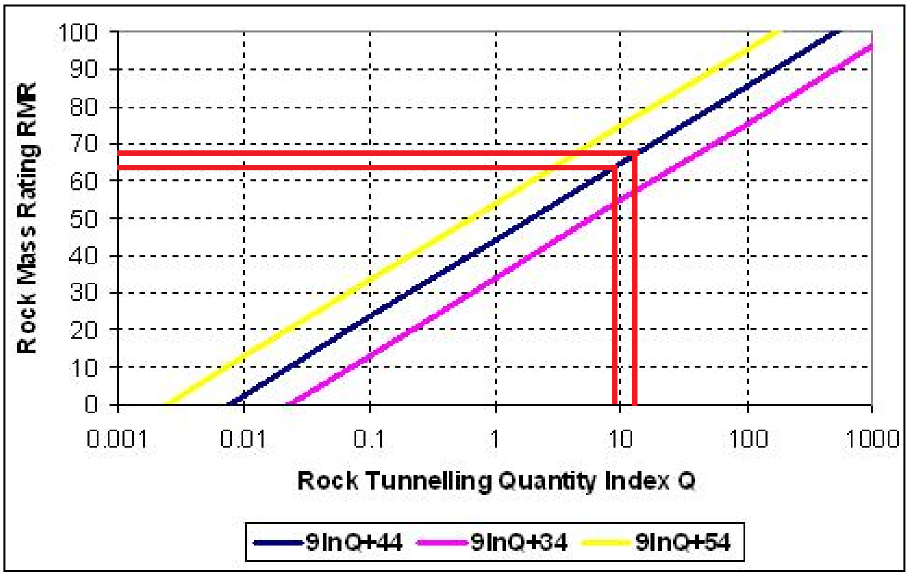
Figure 16-2: Tulsequah Chief Mine – Felsic Volcanics & Massive Sulphides



Observations indicate that all joint sets are not present in any given location. In general, there are two joint sets present in the hangingwall and footwall at any time and random jointing to one joint set present in the ore. All of the observed discontinuity surfaces were rough and planar with moderate staining. The average joint separation is generally <1 mm, and the average joint spacing is between 200 to 600 mm.

The overall ground conditions are described as "good." Rock mass classifications give a conservative or lower bound estimate of 26 for Q' and an RMR of 75. This represents ground conditions that can be described as "good." These values of Q and RMR correlate well and indicate that the data collection procedure did not contain any inherent bias (Figure 16-3).

Figure 16-3: Comparison of Q & RMR

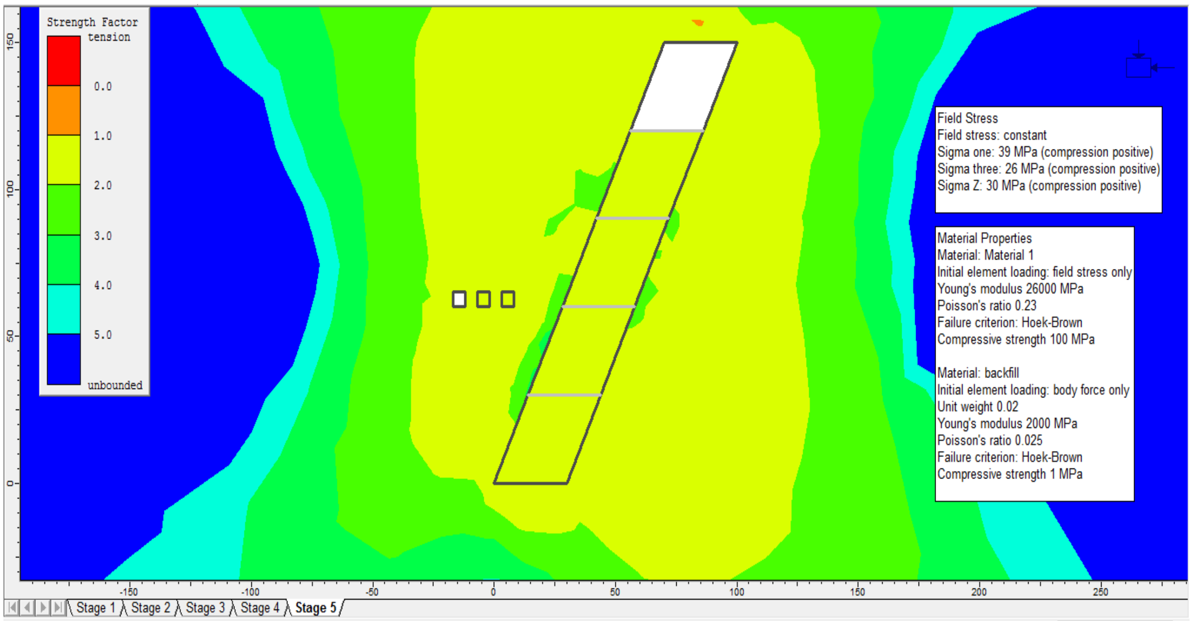


The intact rock strengths were estimated from point load testing. The tests and calculations were performed using the procedure suggested by the International Society for Rock Mechanics (ISRM). All the rock types display high compressive strengths in the order of 100 MPa.

The maximum principal stress is assumed to be orientated horizontally, trending approximately east-west and parallel to the trend of the ore body. The minimum principal stress is assumed to be orientated horizontally with a north-south trend, and the intermediate principal stress is assumed to be vertical. The maximum and intermediate principal stresses are assumed to be 2.0 times and 1.5 times the vertical stress. These values are in general agreement with the published measurements in northern BC.

No adverse structures were identified and no mining-induced stress problems are anticipated. No problems are anticipated for a ramp location 30 m from the ore body (Figure 16-4).

Figure 16-4: Simplified Phase2 Model to Determine the Ramp Location



Transverse stopes will be limited to a width of 20 m to minimize additional support requirements (Table 16-2 and 16-3). The analysis is considered to represent an adequate level of accuracy at the feasibility study and preliminary design stage. Flexibility exists in the mining method where a reduction in the longitudinal stope length can be made during the production planning stage when more site-specific details will be known. Supplementary support using cable bolts will only be required locally. A cost/benefit analysis of supplementary cable bolt support versus pendant pillar support should be examined further at the detailed design phase.

The proposed 5 m wide cut-and-fill panels are stable.

Table 16-2: Summary of Stability Graph Output for Transverse Longhole Stopes

Stope width [m]	**5**		**10**		**20**		**30**		**40**	
Stope length [m]	**30**		**30**		**30**		**30**		**30**	
Primary & Secondary Stopes	**1°**	**2°**	**1°**	**2°**	**1°**	**2°**	**1°**	**2°**	**1°**	**2°**
Hangingwall										
Hydraulic Radius [m]	7.5	14.6	7.5	14.6	7.5	14.6	7.5	14.6	7.5	14.6
Stability number, N'	44.8	64	38.13	54.7	28.94	42	23.56	34.5	20.31	30
Condition	Stable	Stable	Stable	T	Stable	T^s	Stable	T^s	Stable	T^s
Transverse stopes, 30 m high x 30 m long x different widths, hangingwalls are all stable without cable bolts.										
T^s = plots within transition zone, with cable bolts required										
Footwall										
Hydraulic Radius [m]	7.5	14.6	7.5	14.6	7.5	14.6	7.5	14.6	7.5	14.6
Stability number, N'	26.7	64.6	22.71	64.6	17.24	55.2	12.03	46	12.09	40.6
Condition	Stable	T	Stable	T	Stable	T	Stable	T^s	T^s	T^s
Transverse stopes, 30 m high x 30 m long x different widths, footwalls are all stable without cable bolts.										
T^s = plots within transition zone, with cable bolts required										
Back										
Hydraulic Radius [m]	2.14	2.49	3.75	4.95	6	9,81	7.5	14.59	8.57	19.27
Stability number, N'	9.95	8.41	9.95	8.41	9.95	8.41	9.95	8.41	9.95	8.41
Condition	Stable	Stable	Stable	Stable	T	T^s	T	**Fail**	T^s	**Fail**
Transverse stopes, 30 m high x 30 m long x 30 m wide, backs are stable without cable bolts.										
Cable bolt support is required for 40 m wide stopes.										
Conclusion: Limit transverse stope width to 20 m.										
T^s = plots within transition zone, with cable bolts required										
End walls										
Hydraulic Radius [m]	17.3	17.64	17.3	23.28	26.13	36.83	9.95	52.5	53.98	68.73
Stability number, N'	2.14	2.14	3.75	3.75	6	6	7.5	7.5	8.57	8.57
Condition	Stable	Stable	Stable	Stable	Stable	Stable	Stable	Stable	Stable	Stable
Transverse stopes: 30 m high x 30 m long x different widths, all end walls are stable without cable bolts.										

Table 16-3: Summary of Stability Graph Output for Longitudinal Longhole Stopes

Stope width [m]	**5**	**10**	**20**	**30**	**40**
Stope length [m]	**30**	**30**	**30**	**30**	**30**
Primary & Secondary stopes	**1°**	**1°**	**1°**	**1°**	**1°**
Hangingwall					
Hydraulic Radius [m]	7.5	7.5	7.5	7.5	7.5
Stability number, N'	69.08	69.08	69.08	69.08	69.08
Condition	Stable	Stable	Stable	Stable	Stable
Longitudinal stopes, 30 m high x 30 m long x different widths, are all stable without cable bolts.					
T^s = plots within transition zone, with cable bolts required					
Footwall					
Hydraulic Radius [m]	7.5	7.5	7.5	7.5	7.5
Stability number, N'	41.14	41.14	41.14	41.04	41.04
Condition	Stable	Stable	Stable	Stable	Stable
Longitudinal stopes, 30 m high x 30 m long x different widths, are all stable without cable bolts.					
T^s = plots within transition zone, with cable bolts required					
Back					
Hydraulic Radius [m]	2.14	3.75	6	7.5	8.57
Stability number, N'	6.88	6.88	6.88	6.88	6.88
Condition	Stable	Stable	T	T	**T^s**
Longitudinal stopes, 30 m high x 30 m long x different widths, are stable without cable bolts. Cable bolt support is required for 40 m stope widths.					
Conclusion: Limit longitudinal stope width to 20 m.					
T^s = plots within transition zone, with cable bolts required					
End walls					
Hydraulic Radius [m]	17.3	17.3	17.3	17.3	17.3
Stability number, N'	2.14	3.75	6	7.5	8.57
Condition	Stable	Stable	Stable	Stable	T
Longitudinal stopes: end walls are stable without cable bolts for all stope widths.					

The ground support requirements for the mine infrastructure excavations were examined using the StopeSoft software. The minimum support requirements are shown in Table 16-4.

A 15 m thick sill pillar provides adequate stability, and implies that 50% of the planned 30 m thick sill pillars can be recovered towards the end of the mine life.

Approximately 450 kPa is required for free-standing height of paste backfill, containing 4 wt% of binder. A minimum of 1 to 2 wt% binder will be required to prevent liquefaction.

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Table 16-4: Minimum Support Requirements for Infrastructure Excavations

Excavation	Width (m)	Height (m)	Support System	Bolt length & spacing			
				Back		Walls	
				Length (m)	Spacing (m x m)	Length (m)	Spacing (m x m)
Ramp	5.5	5.5	1) 2.4 m long, 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	1.8	1.2 x 1.5
Crusher Station	10.6		1) 4,0 m long single strand 15 mm dia. cable bolts. 2) 2.4 m long 16 mm dia. mechanical rock bolts and wire mesh. 3) 50 mm of shotcrete applied to the back and walls.	4.0	2.0 x 3.0	2.4	1.2 x 1.5
Coarse Ore Bin	9.2		1) 4,0 m long single strand 15 mm dia. cable bolts. 2) 2.4 m long 16 mm dia. mechanical rock bolts and wire mesh.			4.0 2.4	2.0 x 3.0 1.3 x 3.0
Workshop	12.2	7.7	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh. 2) 50 mm of shotcrete applied to the back and walls.	2.4	1.5 x 1.5	1.8	1.5 x 1.5
Switchroom	6.4	5.8	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh. 2) 50 mm of shotcrete applied to the back and walls.	[2.4]	1.5 x 1.5	1.8	1.5 x 1.5
Haulage	5	5.3	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	1.8	1.2 x 1.5
Cross-cut	4.2	4.2	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	1.8	1.2 x 1.5
Fuel Bay	5	4.3	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	1.8	1.2 x 1.5
Sump	5.7	10	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5		
Powder Magazine	8.1	5	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.5 x 1.5	1.8	1.2 x 1.5
Pump Station	4.5	4	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	2.4	1.2 x 1.5
Refuge Chamber	5	4.7	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	2.4	1.2 x 1.5

A 15% cost contingency has been included for supplementary support in areas that may contain worse than anticipated ground conditions. The following additional work will be required during the detailed design and implementation phase when more site-specific details are known:

- A hydrology study is required to determine the ground water inflow.
- The support requirement for multiple cut-and-fill panels.
- Trade-off studies on supplementary cable bolting versus temporary, permanent or artificial pillars (i.e., shotcrete posts).

16.4 Mining Methods

Two mining methods are proposed for the Tulsequah Chief deposit, sublevel longhole (LH) stoping and mechanized cut and fill (MCF). A combination of paste backfill and development waste rock fill will be used in the mining sequence. MCF will be utilized in the shallow dipping areas (less than 55°) while LH is proposed for areas where the dip is greater than 55°.

Approximately 90% of the total mining resource will be mined with LH stoping (including ore sublevels) and the remaining 10% with MCF stopes. The majority of the stopes will be mined longitudinally (along strike) with both methods. Wider portions will be mined with transverse primary/secondary stoping.

16.4.1 Sublevel Longhole Stoping

Longhole stoping provides high productivity at low mining costs from a small number of working faces. All stopes will be filled with a mixture of pastefill and/or development waste.

Geotechnical design have led to stope sizes of 30 m along, with mineralization widths up to 20 m wide and sublevel to sublevel intervals of 30 m. Stope extraction sequencing should be from the centre outwards with the lower stopes leading the stopes above.

Where mineralized zone widths are greater than 20 m, multiple transverse stopes (each with up to 20 m transverse width and length along strike of 30 m) will be mined in primary-secondary mining sequence. After the primary stopes are mined, they will be filled with paste backfill of adequate strength to allow exposure of a 30 m high x 20 m wide fill wall within the secondary stopes that will be mined alongside. Two lifts of primary stopes will be mined before the first secondary stopes are started to allow the drilling drifts to be reused as mucking drifts for the next sublevel above.

LH stopes will be developed by driving a 5 m wide (or up to mineralization thickness for the narrow zones) by 4 m high access drift central to the stope. The central drive will be slashed to the full stope width where the mineralized zones are wider than the initial drive.

A slot raise will be developed in the corner of the stope by longhole drilling and short stage blasting from the bottom up using drop-raise blasting techniques. The slot raise will be enlarged to form a slot across the full width of the stope.

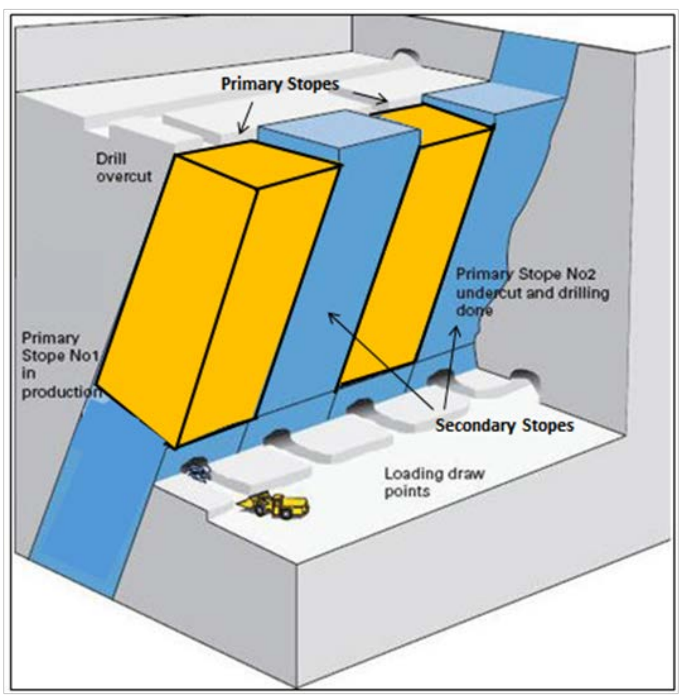
Vertical rings of drill holes will be blasted into the open stope and mineralized material will be mucked from the bottom of the stope by load-haul-dump (LHD) with remote control.

The sublevel mining sequence in the ore lenses will be from the bottom up where possible to avoid leaving sill pillars. When mining cannot begin at the bottom of an ore body, the bottom of the first mined stope will be filled with higher strength backfill to facilitate underhand mining for the stope below. After the stopes at the bottom sublevel in a mining block is mined out, it will be backfilled to form the mucking level for the stope above. This sequence will ensure availability of multiple stopes on different sublevels.

No rib pillars were planned as the mining will likely be a combination of end slicing longitudinal and primary-secondary transverse long-hole stoping. The stoping sequence with the paste backfilling will allow 100% extraction in the LH stoping blocks.

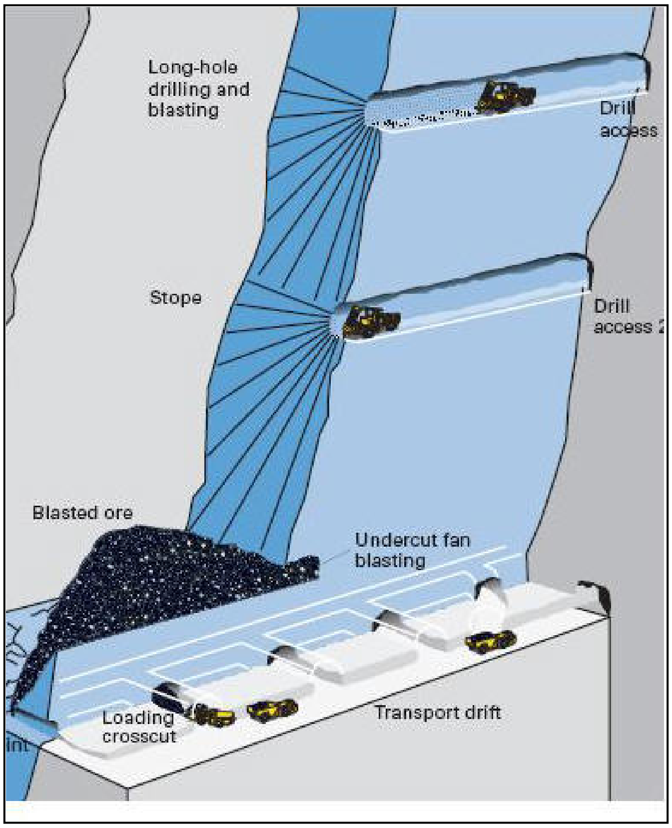
Illustrative, generic sublevel stoping diagrams are shown in Figures 16-5 and 16-6.

Figure 16-5: Primary/Secondary Longhole Stoping



Source: Atlas Copco. Modified by JDS.

Figure 16-6: Sublevel Longhole Mining Method



Source: Atlas Copco

16.4.2 Mechanized Cut & Fill

Mechanized cut-and-fill mining will be utilized in shallower dipping areas of the deposit. MCF is a lower productivity, higher cost mining method than LH stoping, but provides highly selective mining with minimal dilution. Stopes can be sized with irregular backs and walls to match the ore boundaries.

Each MCF mining block is accessed by an 18% to 20% access ramp and mined in 4 m high lifts (MCF stopes). Stopes are developed on the lowest level first, and each subsequent stope or 4 m lift is developed above the depleted and backfilled stope.

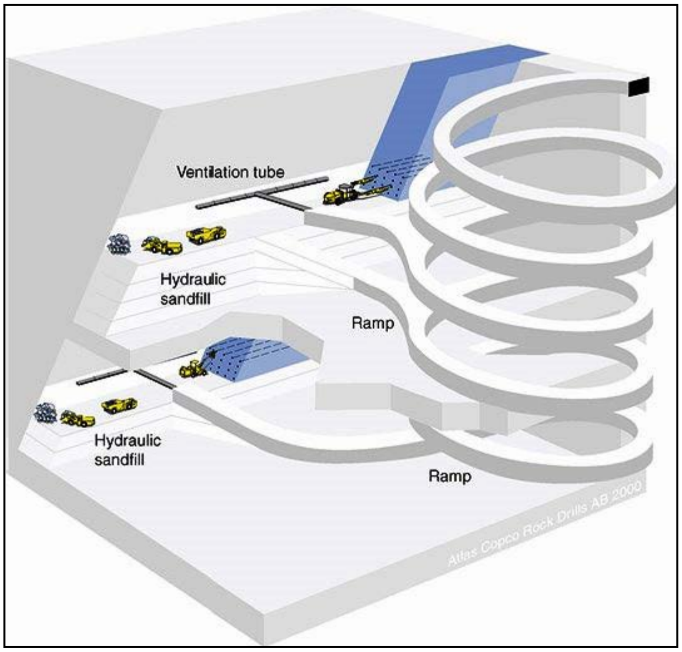
Stops that average 4 m high x 5 m wide have been assumed for production and cost estimation.

A two-boom electric hydraulic drill will drill 4 m long rounds on a standard development heading pattern. The drilled holes will be charged with high explosives primers and ANFO and initiated with non-electric caps. After blasting, the heading will be washed and scaled and then bolted with a mechanized bolter as required.

The broken ore will then be mucked with LHDs into trucks and hauled to surface. The completed 4 m high stope is then filled with pastefill and/or development waste. The next 4 m lift will then commence on top of the hardened fill of the previous lift.

A generic illustration of MCF mining is shown in Figure 16-7.

Figure 16-7: Cut & Fill Mining



Source: Atlas Copco

16.5 Mine Design

The existing adits at 5200 (60 m) level and 5400 (120 m) level will be used as the primary access to the mine for all personnel, mine services, equipment and supplies. The adits will be slashed to 5.3 m high x 5 m wide accommodate diesel trackless equipment and satisfy ventilation requirements.

Access to the various mining levels will be provided by a spiral ramp located in the hangingwall of the deposit. This location was selected because of the predominantly non-acid-generating (NAG) nature of the hangingwall stratigraphy, as compared to the potentially acid-generating (PAG) footwall. The ramp dimensions are 5.3 m high x 5.0 m wide.

Mining levels (4.6 m x 4.6 m) will be located at 30 m vertical intervals. Truck loading will be done on each mining level to minimize LHD haulage distances. The deepest mining level, -690 m, will be located 750 m below the 60 m level. Three-dimensional views of the mine design are shown in Figures 16-8 and 16-9.

Figure 16-8: Mine Design with Existing Workings

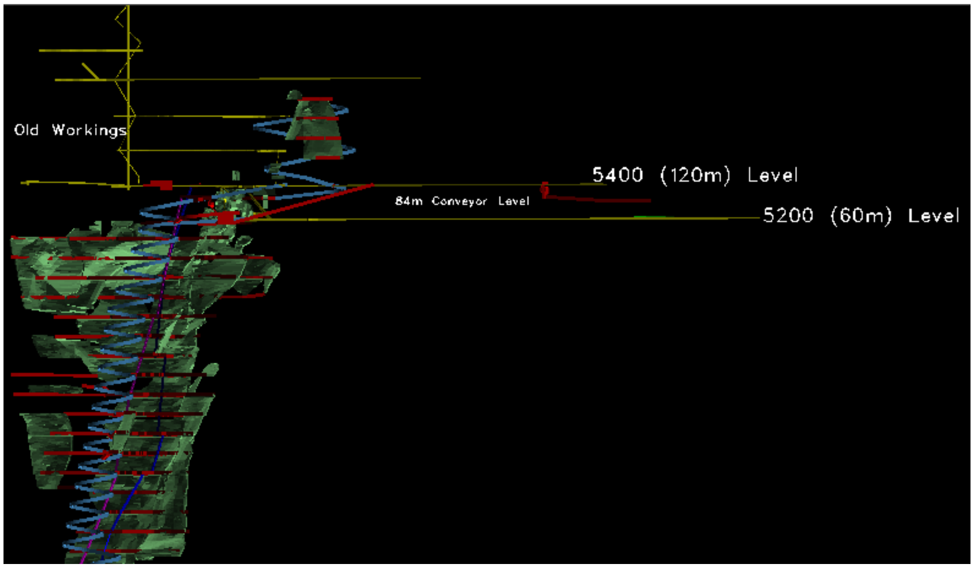
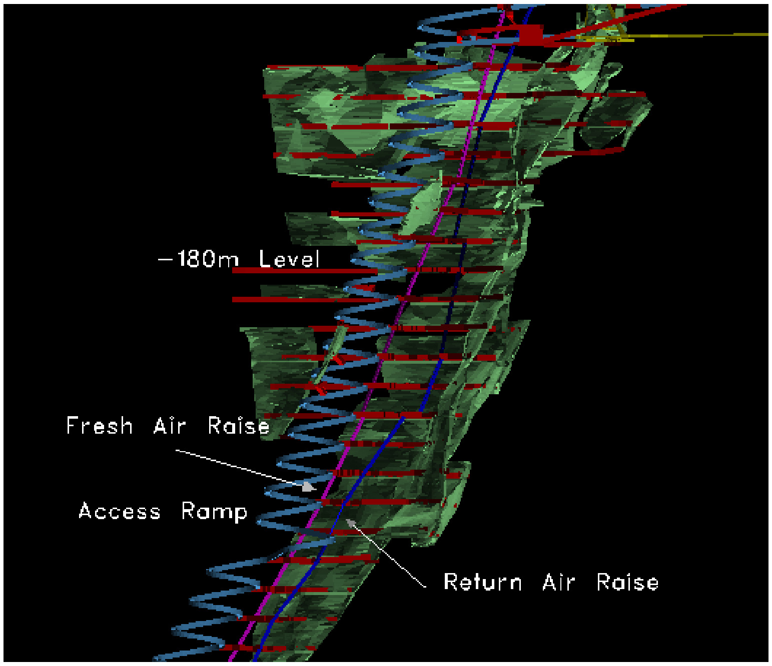


Figure 16-9: 3D Mine Design Looking East



Remuck stations will be driven every 150 m to facilitate the clean-up of the heading and expedite the blast-muck cycle while advancing the face.

Stope access drifts will be driven from the truck loading area near the main ramp towards the stopes in the hangingwall. In certain areas, the stope access drifts will intersect the hangingwall stope to permit access into any footwall stopes. The stope access drifts will be driven 4.6 m wide x 4.6 m high and follow the hangingwall contact.

Ventilation access drifts are driven on each level to ensure fresh and exhaust air raise connections to the stoping levels. The cross-cuts are approximately 4 m wide x 4 m high.

Remucks are excavated on the main ramp to help speed up the development mucking cycle. A maximum of 150 m separates the remucks, which are typically driven 4 m wide x 5 m high x 15 m long.

Water collection sumps are located on every level. Sumps have been sized at 3.5 m high x 4 m wide.

Main sumps are planned for 240 m vertical intervals on the level. A main sump on 60 m level will discharge the water to the surface collection pond.

There are storage areas for both detonators and explosives underground. These will be placed every third level.

Electric power centres will be located on each level in drifts 5.3 m high x 5 m wide.

Refuge stations will be every third level with the first located on the 60 m (5200) level. Portable refuge stations will also be moved and located as required throughout the mine.

There is no plan to develop drifts dedicated entirely to diamond drilling. Any definition diamond drilling will likely be carried out from the main ramp or the truck loadout zone.

Fresh and exhaust air raises will be driven in parallel with the main ramp, reducing the need for vent ducting and air velocities in the main ramp.

Raises with a cross-sectional area of approximately 9 m^2 are constructed for fresh air and secondary egress from the mine. The raises are driven conventionally from Alimak raise climbers using hand held drills. The raises are sequenced in a leapfrog pattern to enable the fresh air to be carried in the direction of the ramp progression.

Typical layouts for levels -30 m, -210 m, -390 m and -590 m are shown on Figures 16-10 to 16-13.

Figure 16-10: Typical Level Layout -30 m

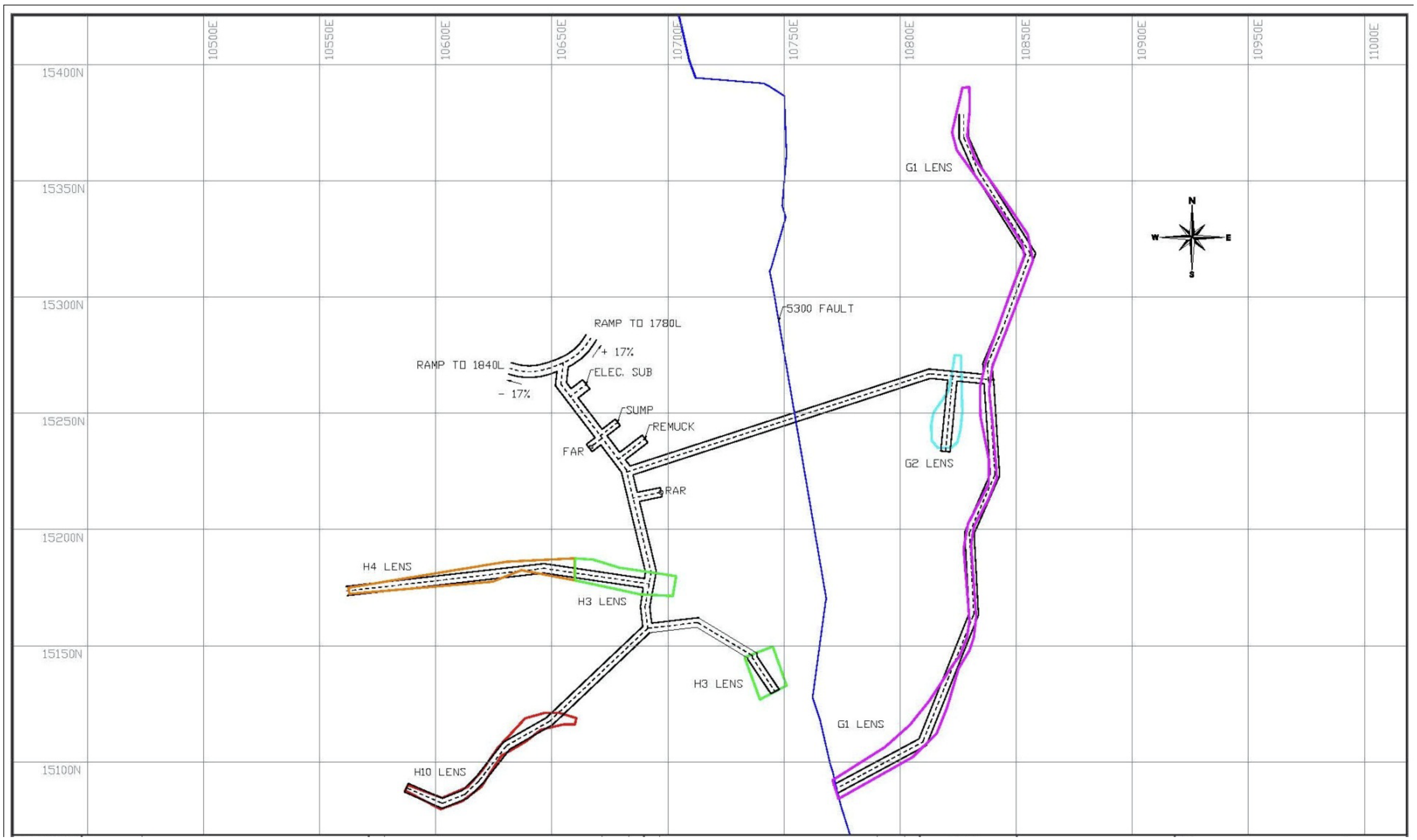


Figure 16-11: Typical Level Layout -210 m

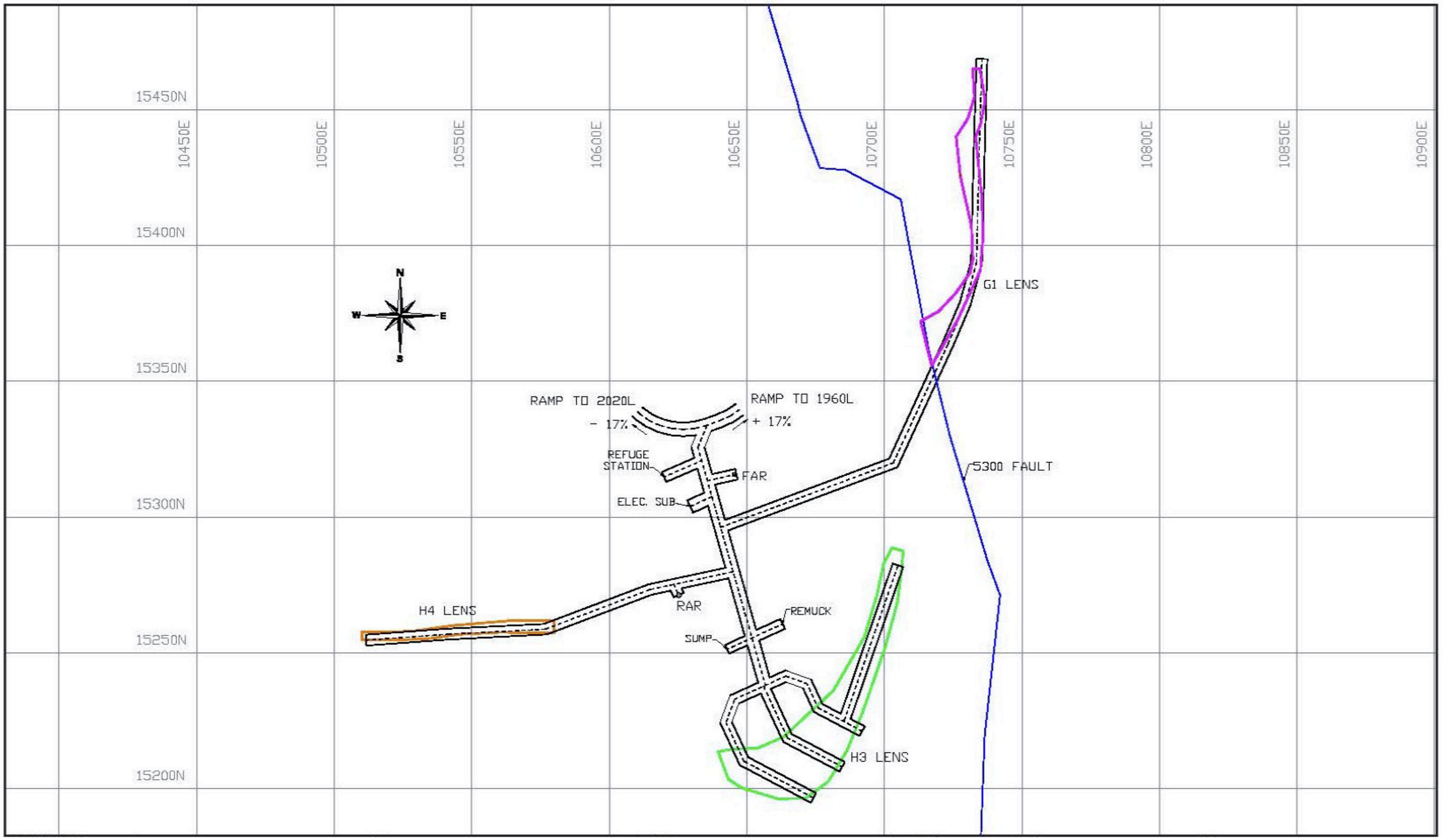


Figure 16-12: Typical Level Layout -390 m

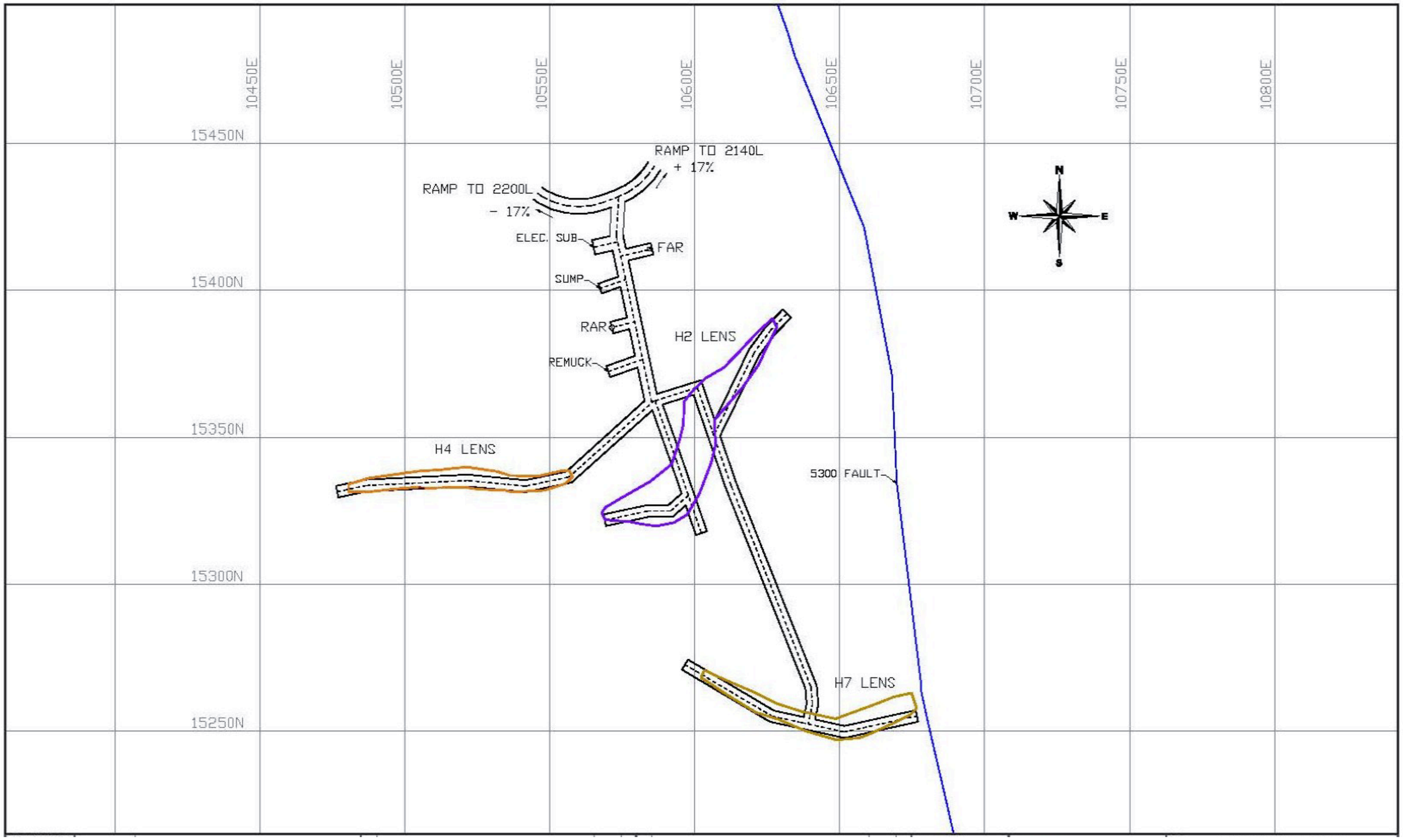
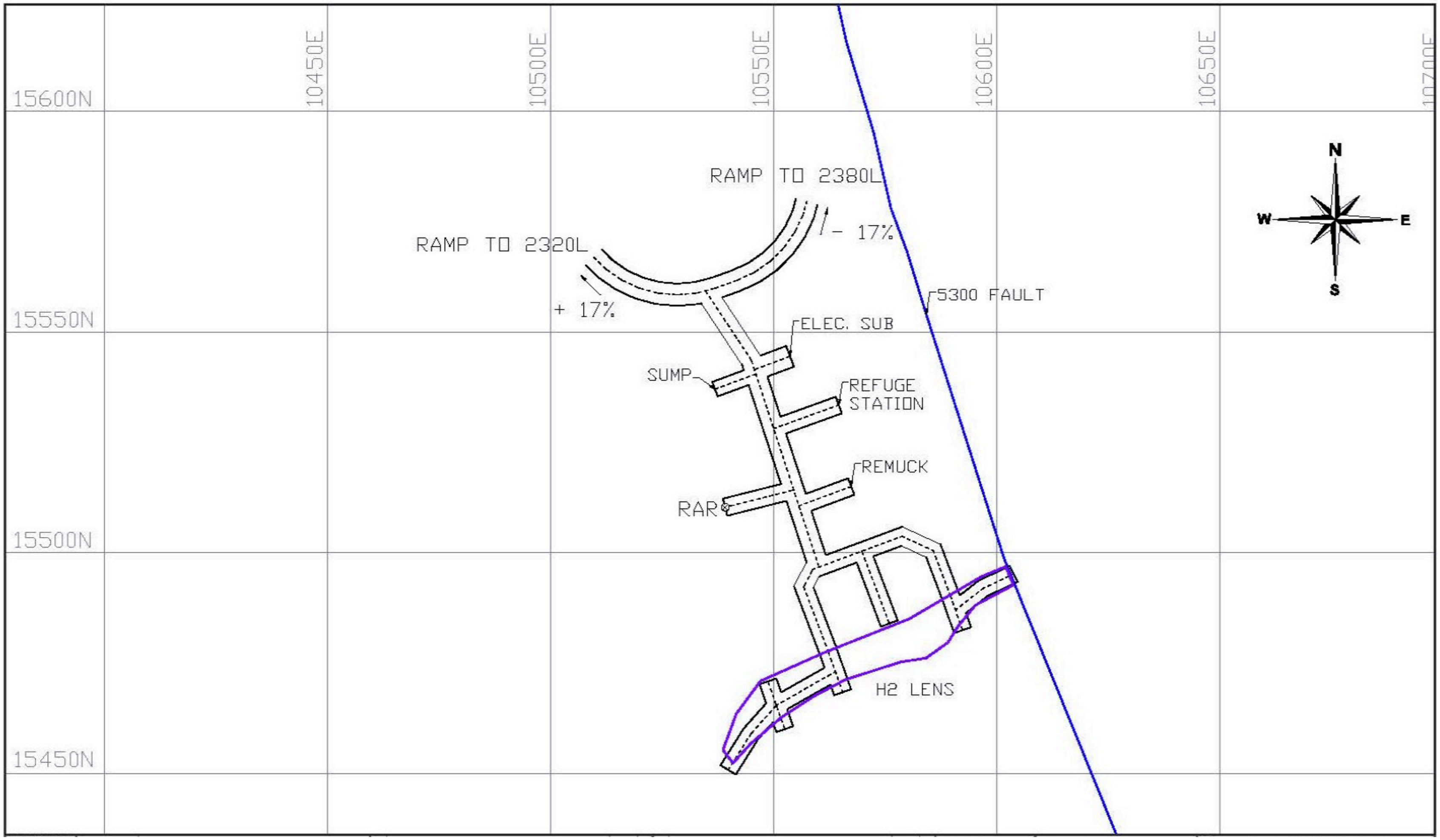


Figure 16-13: Typical Level Layout -570 m



16.6 Mine Ventilation

The design basis of the ventilation system at Tulsequah Chief underground operation was to adequately dilute exhaust gases produced by underground diesel equipment. Air volume was calculated on a factor of 100 ft^3/min per installed horsepower of diesel engine power. The horsepower rating of each piece of underground equipment was determined, and then utilization factors representing the diesel equipment in use at any time were applied to estimate the amount of air required. Ventilation losses were included at 20% of the total ventilation requirements. Table 16-5 lists the air requirements for full production with the total of 446,400 ft^3/min (210 m^3/s) air volume required.

Table 16-5: Diesel Equipment Ventilation Requirements

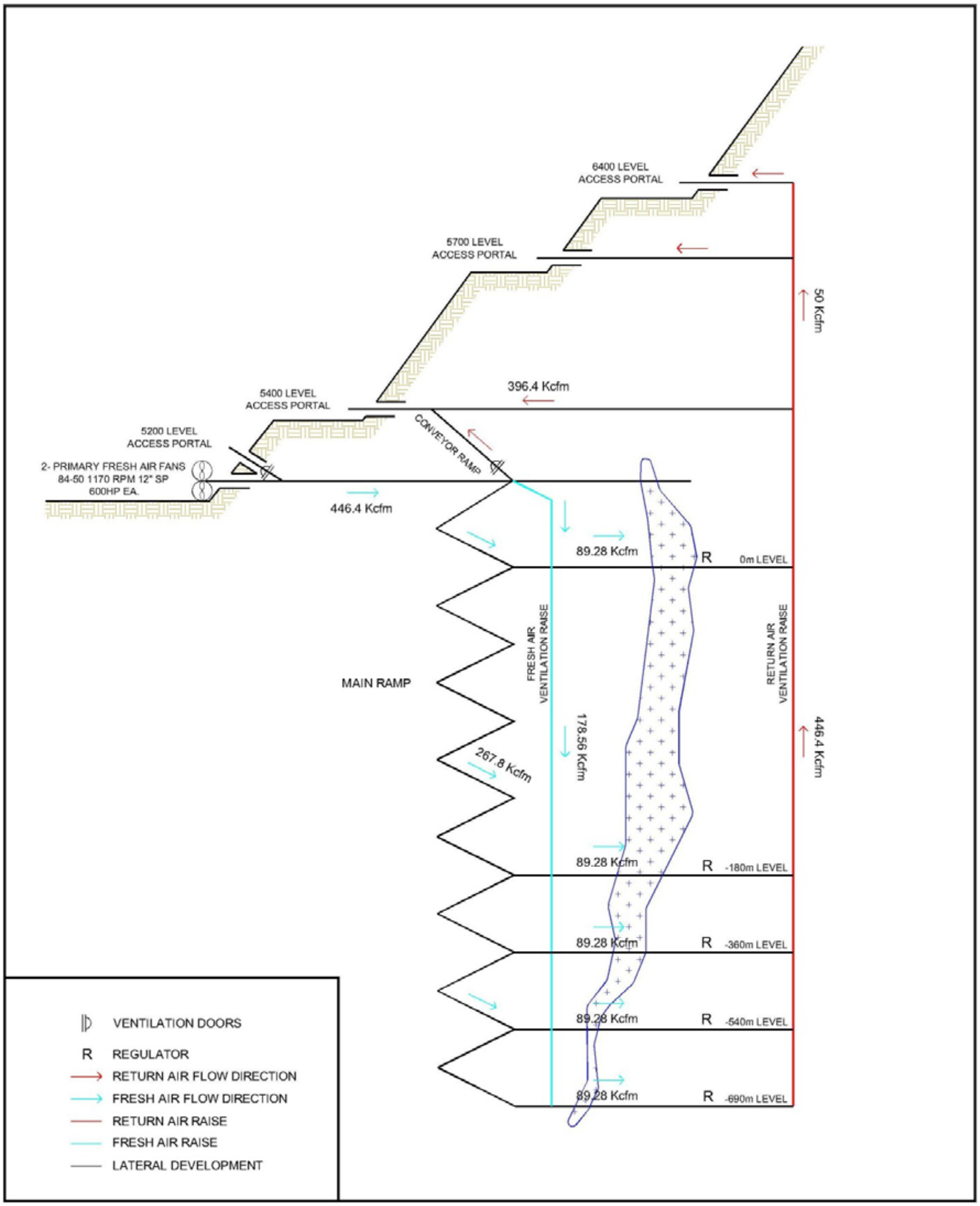
Equipment	Units	Total (hp)	Availability (%)	Utilization (%)	Utilized (hp)	Air Volume (ft^3/min)
Jumbo	2	296	85	50	125	12,500
LH Drill	2	198	85	50	84	8,400
Mechanized Bolter	2	198	85	50	84	8,400
40 t Truck	5	2,515	85	90	1,920	192,000
7.0 m^3 LHD	3	975	85	90	745	74,500
3.5 m^3 LHD	1	201	85	50	85	8,500
Grader	1	110	90	60	60	6,000
Scissor Lift	2	294	90	50	79	7,900
ANFO Loader	1	147	90	50	40	4,000
Fuel/Lube Truck	1	147	90	50	66	6,600
Utility Truck	1	147	90	40	53	5,300
Supervisors/Mechanics Vehicles	5	636	90	40	228	22,900
Crusher						15,000
Losses (20%)						74,400
Total						**446,400**

The primary ventilation system is required during mine development and uses two parallel axial vane fans as the prime mover. The fans will be installed at the 60 m (5200) level portal and push air into the mine. Fresh air will be directed down the haulage ramp and fresh air raise (FAR) from the 5200 level adit. The primary ventilation system is shown in Figure 16-10 (above).

Fresh air will enter the mine through the 5200 (60 m) level portal, flow down the ramp and fresh air raise (FAR) system into the production and development faces, and will exhaust up the return air raise (RAR) system and then out of the mine via the 5400 level adit and the existing old development. Airflow in the ramp will be controlled by ventilation regulators and doors placed appropriately for the mining taking place at any time.

The ventilation schematic shown in Figure 16-14 depicts five RAR levels in use for illustrative purposes. There will be fresh and return air connects on every level of the mine.

Figure 16-14: Ventilation System Schematic



Air movement to the stopes will be controlled using the auxiliary ventilation fans. Ventilation regulators, doors, and bulkheads will also be used to control airflow in the mine.

Ventilation will be distributed to the working areas via auxiliary ventilation fans and ducting. The development headings have been sized to accommodate the large ducting to reduce head losses.

16.6.1 Ventilation Fan Selection

Two 448 kW (600 hp) main intake fans operating at 3 kPa (12 inches) static pressure will be required when the mine is operating at its maximum depth. The two fans will operate in parallel and will be located inside the portal of 5200 level adit.

Both of these adjustable pitch vane-axial fans will combine to deliver the required 446,400 ft^3/min (210 m^3/s) air volume required at the maximum mining depth. Fans have been sized to overcome any resistance of the intake assembly and fresh air heaters.

Auxiliary ventilation comprises 56 kW (75 hp) fans with 1.22 m (48 inches) diameter flexible ducting for production and level development and, 90 kW (120 hp) fans with 1.37 m (54 inches) flexible ducting for ramp development.

16.6.2 Mine Air Heating

Heating of the intake air will be required during the winter months to prevent water freezing underground and to provide acceptable conditions for underground workers and equipment. Mine air will be heated to +2°C by a utilizing waste heat from the LNG/Diesel generators. The waste heat will be piped to the fresh air heaters located at the 5200 level portal. A parallel portal and drift will house the waste heater infrastructure. The air in the heater drift will be heated and blended with the cold air entering through the 5200 level adit, allowing vehicles and personnel to enter the mine with the use of double air lock doors.

16.6.3 Emergency Stench System

A stench gas system utilizing Ethyl Mercaptan will be installed on 5200 level portal at the fresh air fans and compressed air system and may be triggered as appropriate to alert underground personnel in the event of an emergency. Airflow velocities will permit all personnel to be alerted of the emergency within an acceptable period. Once underground operators smell the stench they will immediately take refuge in appropriately outfitted lunchroom/refuge stations. If required and after confirmation that all personnel are secured in refuge stations, ventilation can be adjusted via variable frequency drives (VFD) or remote access.

16.7 Underground Mine Services

16.7.1 Mine Power

The primary underground feed voltage at the Chieftain mine will be 13.8 kV from surface through a 4/0 15 kV Teck cable.

The major electrical power consumption in the mine will be from the following:

- mine ventilation
- underground crushing plant and conveyor system
- underground paste plant
- underground dewatering
- underground mobile equipment
- compressed air
- mine lighting
- refuge stations.

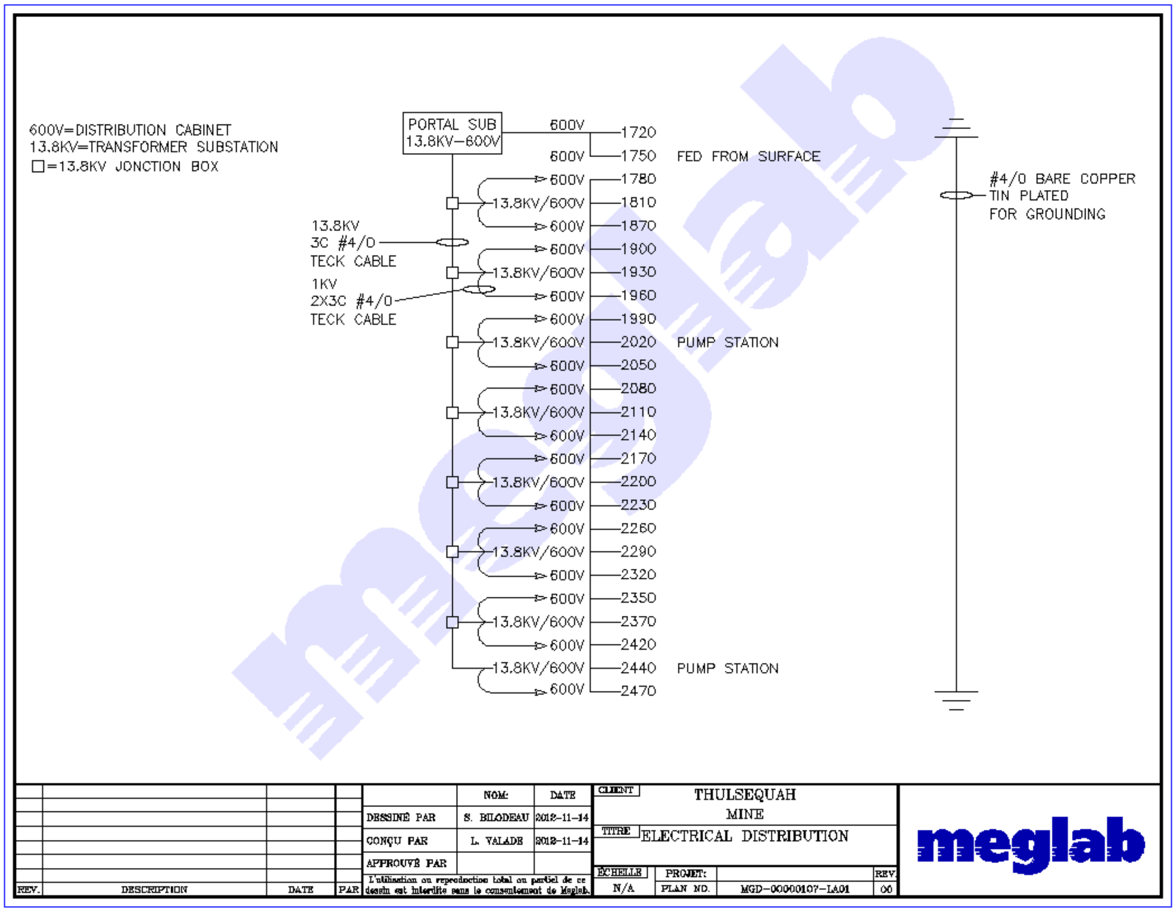
The underground crusher and conveyor systems will be fed from surface from the process plant feeder. Seven main 13.8 kV, 1 MVA substations used to supply energy to 26 levels and sublevels. Working in nine levels simultaneously will require three substations and nine substation distribution cabinets. Pump levels will be fed at all times from a transformer substation. Five 1 MVA substations will be required. A simplified electrical schematic is shown in Figure 16-15.

This figure shows the portal substation feeding the first levels, portal fans and pumps. It also holds 13.8 kV switches ready to feed the underground transformer subs at 13.8 kV. The 13.8 kV substations feed power at 600 V for all underground services such as fans, pumps and mobile equipment.

Key power statistics are summarized below:

- average underground power load – 1,575 kW
- annual average consumption – 14,826,117 kWh.

Figure 16-15: Mine Power Distribution



16.7.2 Communications

A leaky feeder communication system will connect mine and surface operations. Telephones will be located at key infrastructure locations such as the crusher, paste backfill plant, electrical substations, refuge stations, and main sump. Key personnel (such as mobile mechanics, crew leaders and shift supervisors) and mobile equipment operators (such as loader, truck and utility vehicle operators) will be supplied with an underground radio for contact with the leaky feeder network.

16.7.3 Compressed Air

Compressed air will be used for stoper, jackleg and sinker drilling, secondary pumping, ANFO loading, and blasthole cleaning. The underground mine will have a dedicated compressed air system, consisting of two 500 L/s compressors providing 1,000 L/s. Compressed air will be delivered underground in a 150 mm diameter pipe via the 5200 level adit and main ramp, and 100 mm pipes in the sublevel development and stopes.

The underground mobile drilling equipment such as jumbos, production drills and ANFO loaders will be equipped with their own compressors. Two portable compressors will be used to satisfy compressed air consumption for miscellaneous underground operations. The underground crusher and paste backfill plants will have their own compressors and distribution systems.

16.7.4 Mine Water Supply

Mine supply water from the process freshwater tank will be distributed to the underground levels via 100 mm (4 inches) diameter pipelines. Further distribution to work headings will be via 50 mm (2 inches) diameter water lines. Pressure reducers will be located along the main ramp. Process water will be retrieved from the dewatering stream following de-sedimentation. Existing diamond drill holes making water will also be considered to augment the mine supply water.

16.7.5 Mine Dewatering

Based on an estimate provided by Rock Group Consulting Engineers in their report entitled “Geomechanics Assessment for Mine Design” (December, 1995), an average water inflow of 30 L/s (475 gpm) can be expected, which was used for the mine dewatering design basis. Dewatering will expand in nine stages as the main ramp advances to the bottom of the known reserves on -690 m level. The mine dewatering system involves two main pump stations, seven secondary pump stations and 27 level sumps.

The secondary pump station on 5200 (60 m) level will collect water from the upper workings and from the -210 m level main pump station. This 60 m level secondary pump station will feed all the underground water to the treatment plant on surface.

The deepest main pump station will be on -480 m level, and will automatically pump water up to the -210 m pump station. Secondary pump stations will be established on every third level provided a main pump station does not already exist. Secondary pump stations will be stage connected, as required, to other secondary sumps and ultimately to main pump stations. Small collection sumps on each level will drain through screened 100 mm diameter drill holes to the sumps and pump stations on levels below. The final secondary pump station will be located on -665 m level to service the mine bottom.

16.7.6 Explosives & Detonator Storage

Explosives will be stored underground in permanent magazines, while detonation supplies (NONEL, electrical caps, detonating cords, etc.) will be stored in a separate magazine. Underground powder and cap magazines will be prepared on 60 m (5200 level). Day boxes will be used as temporary storage for daily explosive consumption.

A mixture of ANFO will be used as the major explosive for mine development and stoping. Packaged emulsion will be used as a primer and for loading lifter holes in the development headings and for wet longholes. Smooth blasting techniques may be used as required main access development headings, with the use of trim powder for loading the perimeter holes.

During the preproduction period, blasting in the development headings will be done at any time during the shift when the face is loaded and ready for blast. All personnel underground will be required to be in a designated Safe Work Area during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of the shift.

16.7.7 Fuel Storage & Distribution

Two 75,000 L fuel tanks will be located near the 5200 level portal to provide fuel for the underground mobile equipment fleet.

Portable fueling stations, or fuel-lube-Sats, will be used for the underground mobile fleet. There will be two units available, one for fuel and one for lubes located near the active levels. The fuel-lube-Sats will house a lubrication/oil dispenser in addition to fuel. Fuel-lube-Sats come complete with emergency spill catchment, automatic roll-down doors and fire suppression per local code and regulations.

16.7.8 Central Blasting

Central blasting used at the Tulsequah Chief mine allows the operation to initiate blasts remotely from a safe control point on the surface. Digital central blast systems have been sourced from the major explosives suppliers. These systems are extremely safe and contain redundancy coding that prevents accidental initiations. These systems will work through the leaky feeder mine communications system.

16.7.9 Mobile Equipment Maintenance

Mobile underground equipment will be maintained in the surface maintenance located 5200 portal. A mechanics truck will be used to perform emergency repairs underground.

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

A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

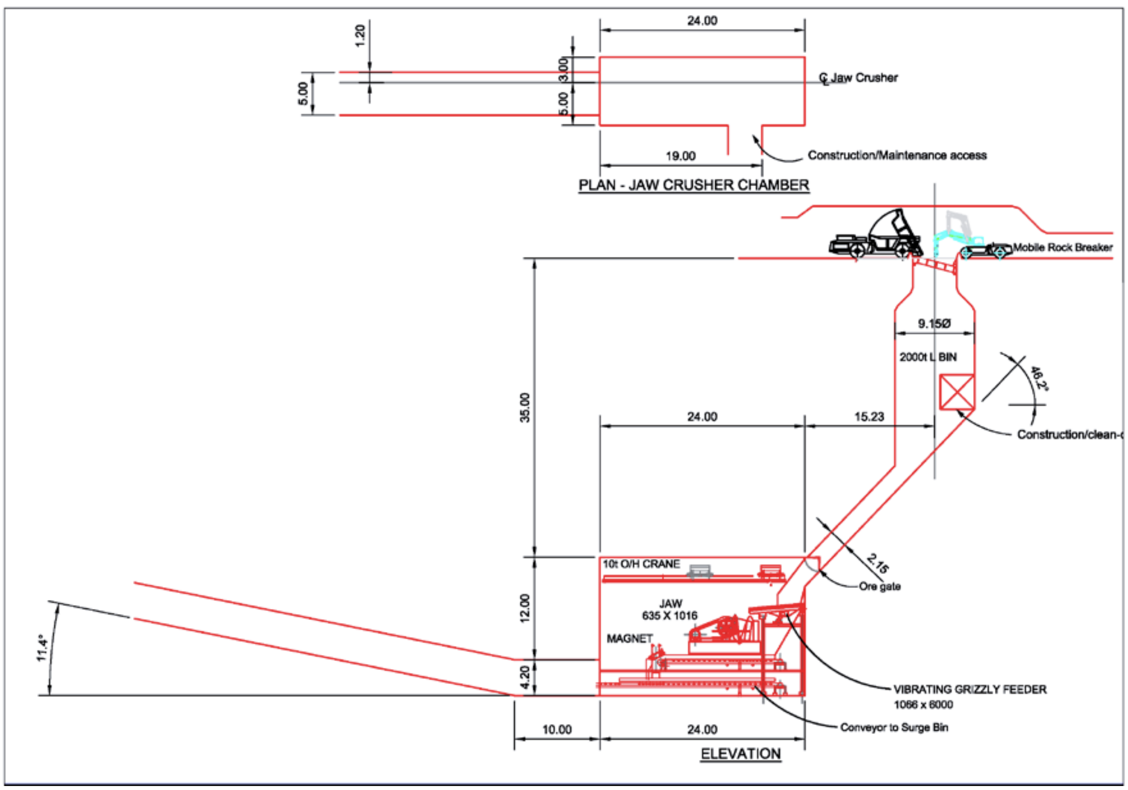
The equipment operators will provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

16.7.10 Underground Crushing

The lack of surface area near the mill and ROM ore moisture content led to the decision to place the primary crusher and fine ore bin underground. The primary jaw crusher will be located on the 5200 (60 m) level and will be fed ROM ore through a feed raise and 2,000 tonne capacity surge bin. Crushed, fine ore will be conveyed up to the 5400 (120 m) level where it reaches the fine ore bin. Ore from the bin is then conveyed to process plant via the new 84 m level portal.

The crusher station is shown in Figure 16-16.

Figure 16-16: Underground Crusher



16.7.11 Mine Safety

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

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fuelling stations, and other strategic areas. Every vehicle will carry at least one fire extinguisher of adequate size. It is recommended that underground heavy equipment be equipped with automatic fire suppression systems.

The 5200 (60 m) level adit and main access decline will provide primary access to the underground workings. The 5400 (120 m) level adit and ventilation raise with dedicated manway will provide the secondary exit in case of emergency. The manway will be equipped with ladders and platforms.

16.8 Paste Backfill

An important driver for the Tulsequah Chief project is reduction of the environmental impact from waste disposal, largely consisting of processed tailings. Chieftain's approach is to include a pyrite flotation circuit and to provide a de-pyritized tailings (final tailings) for surface disposal, mixed with a quantity of limestone for buffering capacity.

Pyrite concentrate (pyrite tailings) along with a balance of final tailings to meet backfilling and productivity needs will be blended and mixed with binder to produce a high-pyrite paste backfill.

The backfill system will be located underground near the top centre of the planned production zones. This location was selected to reduce the capital cost of paste pumps and redundancy, as well as to reduce operating costs (from binder, paste pump power and maintenance). The overall philosophy is to pump thickened slurry from surface to the underground paste plant, where vacuum filtration, cement addition, mixing and paste pumping/distribution will be carried out.

It is assumed that the Tulsequah Chief Mine will operate at an annual production of 730,000 t/a. The pyrite concentrate tailings production rate will be approximately 26 t/h, while the production rate for full tailings will be 44 t/h. The nominal tailings production rate will be 70 t/h; however, use of stored pyrite concentrate tailings increases the tailings feed rate to the paste plant to 90 t/h, assuming four days backfilling and three days of storage.

The design criteria indicate that backfilling will be conducted during 60% of the operating time of the mill, and at that time, 100% of the tailings generated by the mill coupled with the stored pyrite concentrate tailings will be used for backfill.

Over 365 days, the average daily backfill requirement is 1,377 t/d, while the paste plant production rate is approximately 2,160 t/d based on 90 t/h dry solids. The plant will be required to operate approximately 4.4 d/wk on average.

The paste backfill system includes a surface component within the processing plant and an underground component (see Figure 16-17). The following major components are situated on surface:

- pyrite concentrate thickener underflow slurry agitated storage tank with three-day retention
- final tailings thickener underflow slurry agitated storage tank (two to four hours of retention)
- combined pyrite concentrate/final tailings pump box
- surface bulk cement storage systems and blower
- building enclosure
- centrifugal pumps (six total)
- hopper and conveyor to feed reclaimed pyrite concentrate into pyrite concentrate slurry tank
- tie-in to medium voltage power, gland water, instrument air, HVAC, building.

Transport of tailings to the remote paste backfill plant option will require an HDPE pipeline, given the thickener underflow dilution described above. The tailings will be received in the remote paste backfill plant site within the agitated tank will further ensure full mixing of the pyrite concentrate and final tailings slurry to account for any internal redistribution of the tailings during pipeline transport.

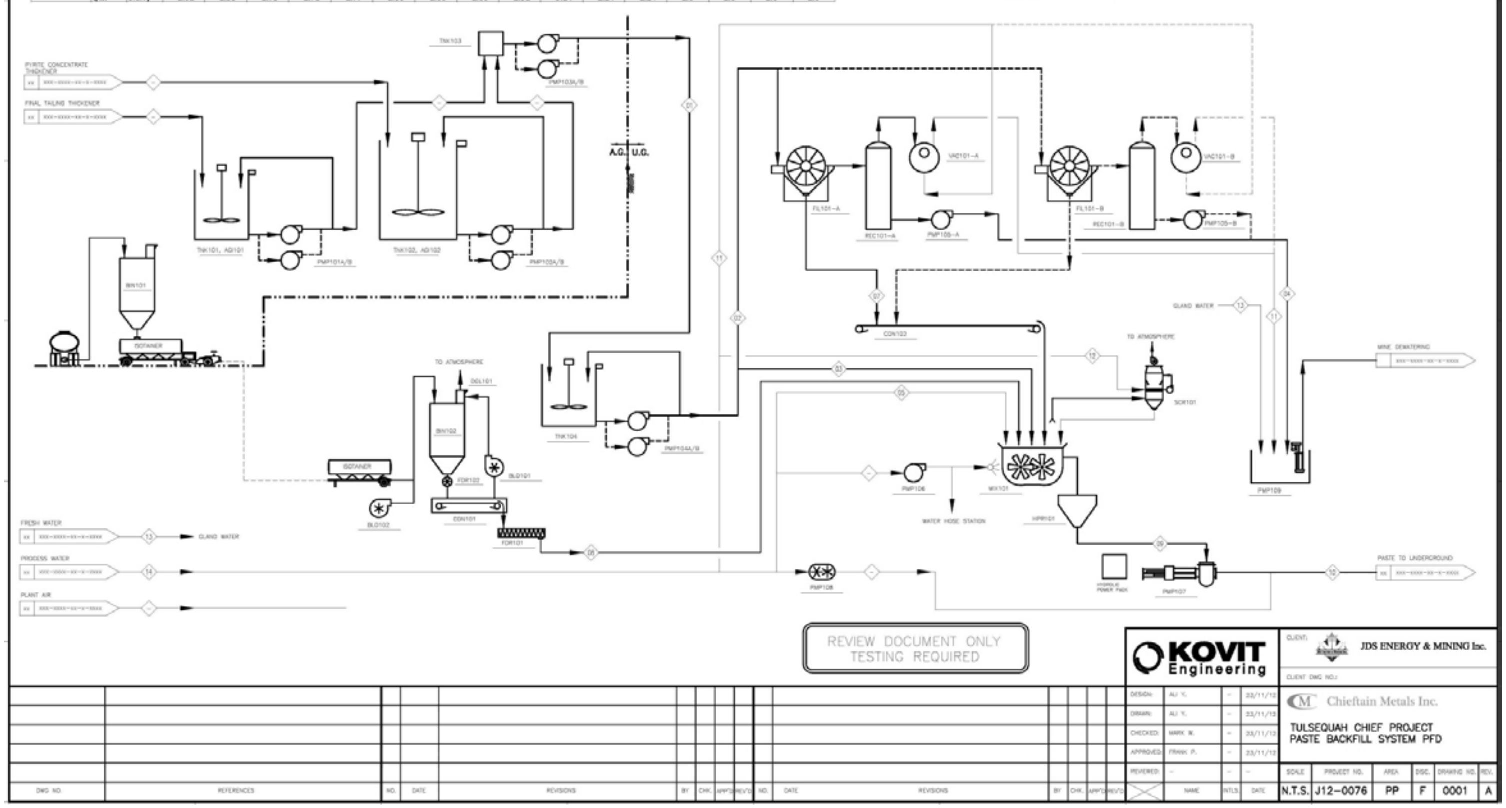
The paste backfill plant is proposed to have a single bank of disc vacuum filters. Redundancy has not been considered for these units, as utilization (filling) time is 60%, leaving 40% or approximately three of seven days for maintenance.

Binder will be delivered to the primary surface storage silos and delivered by modified bulk cement carrier (trailer or truck mounted) to the underground day silo.

Figure 16-17: Paste Backfill System Flowsheet

THE MASS BALANCE TABLE REPRESENTS ONE OPERATING CONDITION

	Line No.				01	02A/B	03	04A/B	05	06	07A/B	08	09	10	11A/B	12	13	14
		Description	Final Tailings Discharge	Pyrite Concentrate	Combined Tailings	Feed To Filter	Mixer Dilution	Filtrate Water	Dilution Water	Clean Up Pump	Filter Cake	Binder	Mixer Discharge	Paste to UG	Seal Water	Dust Water	Gland Water	Mine Water
Mass	tph	Solids	43.9	43.5	87.4	67.6	19.7	0.00	0.00	0.00	67.6	2.70	90.1	90.1	0.00	0.00	0.00	0.00
	tph	Water	29.3	23.4	52.7	40.6	12.1	24.7	0.0	130.4	15.9	0.0	28.4	28.4	16.0	8.0	1.0	25.0
	tph	Slurry	73.2	66.9	140.0	108.2	31.8	24.7	0.00	130.4	83.5	2.70	118.5	118.5	16.0	8.0	1.0	25.0
Percent Solids	%		60%	65%	62%	62%	62%	0%	0%	0%	81.0%	100%	76%	76%	0.0%	0.0%	0.0%	0.0%
Flow	m³/hr	Solids	16.3	10.9	27.1	20.2	5.9	0.0	0.0	0.0	20.2	0.9	26.9	26.9	0.0	0.0	0.0	0.0
	m³/hr	Water	29.3	23.4	52.7	40.6	12.1	24.7	0.0	130.4	15.9	0.0	28.4	28.4	16.0	8.0	1.0	25.0
	m³/hr	Slurry	45.5	34.3	79.8	60.8	18.0	24.7	0.0	130.4	36.1	0.9	55.4	55.4	16.0	8.0	1.0	25.0
S.G.	t/m³	Solids	2.70	4.00	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.14	3.35	3.35	2.70	2.70	2.70	2.70
	t/m³	Water	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
	t/m³	Slurry	1.61	1.95	1.75	1.78	1.77	1.00	1.00	1.00	2.31	3.14	2.14	2.14	1.0	1.0	1.0	1.0



Major components of the underground paste backfill plant will include the following:

- agitated receiving slurry (pyrite concentrate, final tailings) filter feed tank complete with pumps
- underground day-silo cement storage and metering system
- process water tank and pumps
- one disc filter presses x eight discs in each x 10 ft, 6 inches diameter with vacuum pumps and ancillary equipment
- filter cake conveyor and weigh belt
- continuous high-intensity paste mixer complete with washing system
- paste pump complete with hydraulic power pack
- high-pressure paste pipeline flush pump.

The tailings will be received in an agitated tank to allow for continuous operation for two hours independently of the surface storage system. Tailings will be pumped into the filter and a slip-stream will be by-passed into the mixer to produce desired paste. Final top-up water for slump adjustment will be sourced from the process water removed the filtering process.

Binder (cement and or other binder mix) will be added at between 2 wt% and 10 wt% of solids, depending on the backfill recipe requirement.

Paste will be mixed in a high-intensity shear mixer and discharged into a hopper for distribution into a gravity fed system or into the paste pump for distribution to other areas of the mine. Slump will be managed to minimize binder consumption though this will be balanced with paste pump operation and maintenance.

Key aspects of the design are based on a range of assumptions, which will need to be confirmed through future testing, consisting of:

- rheology to determine range of blends (variability in ratio of pyrite and non-pyrite tailings)
- strength and cost relationship with one or more binders, recognizing there is potential degradation of strength due to sulphide and cement reaction
- process dewatering (thickening and filtration) testing to determine the equipment sizing and applicability for paste preparation and capital cost.

The underground paste plant general arrangement is shown in Figures 16-18 and 16-19.

Figure 16-18: Underground Paste Plant Plan View

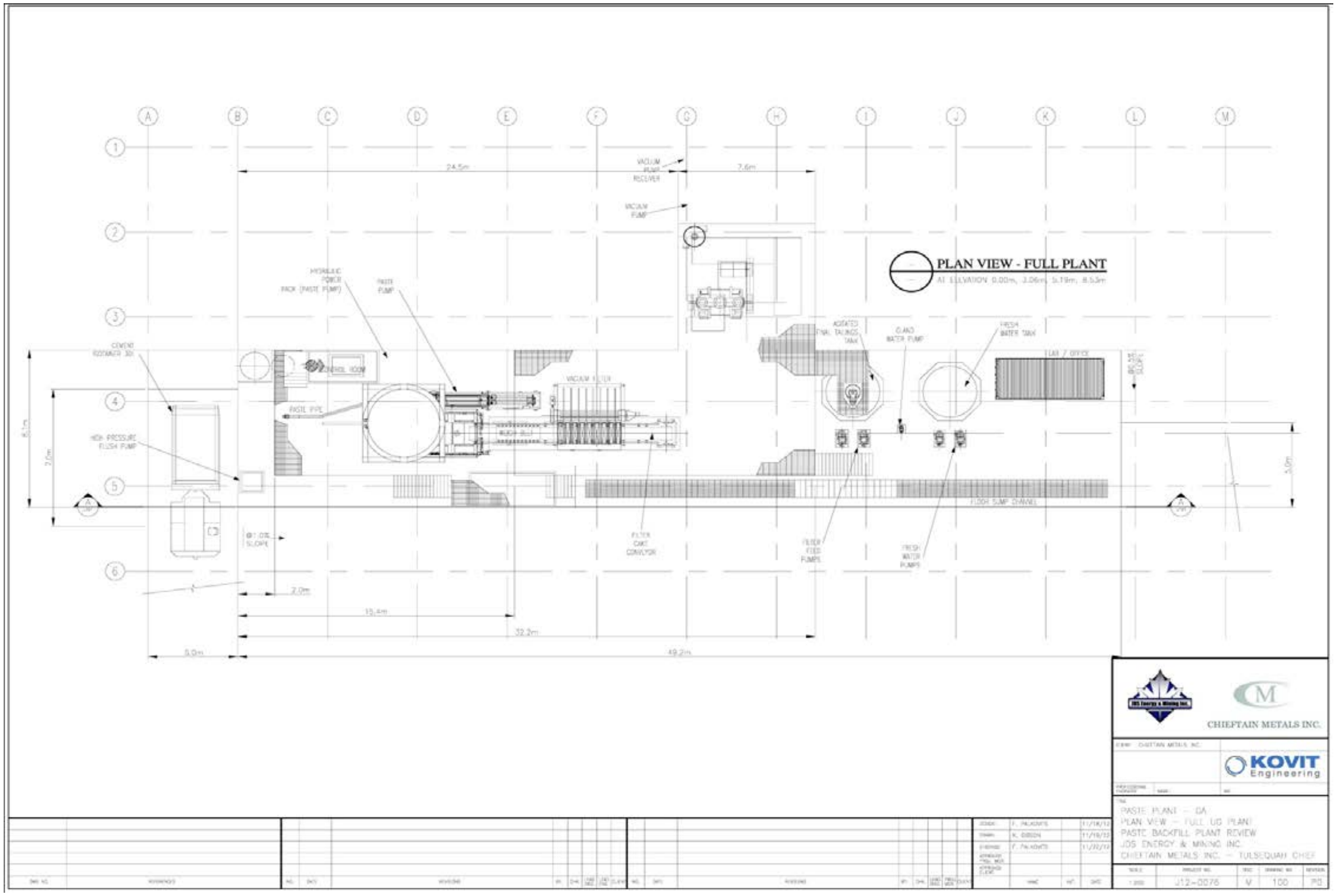
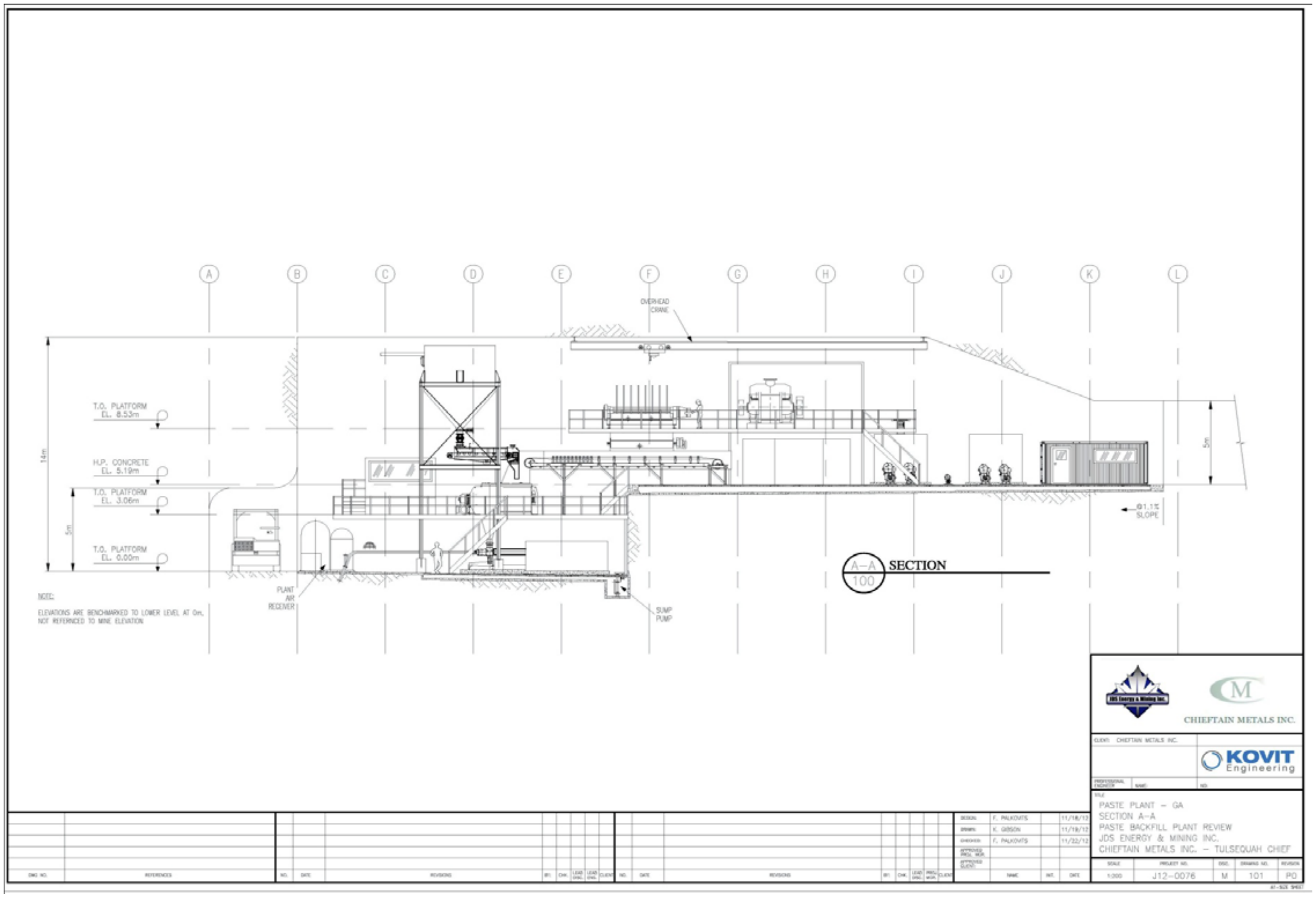


Figure 16-19: Underground Paste Plant Section A-A



16.9 Mine Equipment

The selection of underground mining equipment is based on mine plan requirements, mining methods, operating drift and stope dimensions. No work was undertaken in this feasibility study to evaluate alternates or new technology. Since the life of mine plan is less than 10 years, it is assumed that all mobile equipment will be new to avoid major refurbishment expenditures.

Two boom diesel/electric jumbos will be used for lateral development and MCF stoping, while production drilling will be completed by a diesel/electric LH drill. Mucking will be carried out with 7 m^3 LHDs with remote operating capabilities (used for development and stope mucking). Waste and ore will be hauled in 40 tonne trucks.

The underground equipment fleet is summarized in Table 16-6.

Table 16-6: Mine Equipment Summary

Equipment Type	Quantity
Two Boom Jumbo	2
Single Boom Jumbo	1
Production Drill	2
7 m^3 LHD with Remote	3
3 m^3 LHD with Remote	1
40 tonne Truck	5
Mechanized Bolter	2
Fuel/ Lube Truck	1
Grader	1
Deck and Boom Truck	1
Scissor Lift	2
ANFO Loader	1
Supervisor Vehicles	5
Mechanic Vehicles	3

16.10 Mine Personnel

The mine will operate on two 10-hour shifts, 365 days per year with three mining and maintenance crews. Two crews will be on site at any one time, one on dayshift and one on nightshift, with the other crew off site on break. The majority of the mining and maintenance personnel will work a two-week-on, one-week-off (2x1) rotation, while technical staff and management will work a four-day-on, three-day-off (4x3) schedule.

Ten-hour shifts exceed the hours allowed underground by regulation and a variance will be required from the BC Labour Board. Given the nature and location of the mine, and referencing other northern BC operations where similar variances have been given, it is expected that this variance will be granted.

The underground mine personnel requirement peaks at 123 personnel during full production, with 83 on site at one time. Excluded from this total are personnel required to operate the processing, underground crusher, site services and site general administration as well as mining contractors.

Mining personnel requirements are summarized in Tables 16-7 to 16-10.

Table 16-7: Mine Operations Personnel

Position	Quantity	Schedule	Hourly/Salary
Mine Superintendent	1	4x3	Salary
Mine Captain	1	4x3	Salary
Mine Shift Supervisors	5	2x1	Hourly
Production Drillers	6	2x1	Hourly
Jumbo and LH Drillers	6	2x1	Hourly
LHD Operators	9	2x1	Hourly
Truck Drivers	12	2x1	Hourly
Blasters	6	2x1	Hourly
Services	6	2x1	Hourly
Ground Support	9	2x1	Hourly
General and Backfill Labourers	12	2x1	Hourly
Paste Plant Operators	3	2x1	Hourly
Mine Operations Total	**76**		

Table 16-8: Mine Maintenance Personnel

Position	Quantity	Schedule	Hourly/Salary
Maintenance Superintendent	1	4x3	Salary
Maintenance Shift Supervisors	1	4x3	Salary
Maintenance Foreman	1	4x3	Salary
Maintenance Planner	1	4x3	Salary
Mechanics and Welders	18	2x1	Hourly
Electrician	6	2x1	Hourly
Bit and Lamp Man	4	2x2	Hourly
Mine Maintenance Total	**32**		

Table 16-9: Technical Services Personnel

Position	Quantity	Schedule	Hourly/Salary
Chief Mine Engineer	1	4x3	Salary
Senior Mine Engineer	1	4x3	Salary
Mine Engineer	1	4x3	Salary
Ground Control Engineer	1	4x3	Salary
Senior Mine Technician	2	2x2	Salary
Surveyor/ Mine Technician	2	2x2	Salary
Chief Geologist	1	4x3	Salary
Mine Geologists	2	2x2	Salary
Geotechnical Technician/Sampler	4	2x2	Salary
Technical Services Total	**15**		

Table 16-10: Total Mine Personnel Summary

Position	Quantity
Mine Operations	76
Mine Maintenance	32
Technical Services	15
Grand Total Mine Personnel	**123**

16.11 Mine Production Plan

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

- mining inventory tonnage and grade
- geometry of the mineralized zones
- amount of required development
- stopes productivities
- sequence of mining and stope availability.

The underground mine production rate of 2,000 t/d is considered appropriate due to the high degree of mechanization and potential high productivities of selected stoping methods. Based on the presence of several mineralized zones and ability to have production from different sublevels, JDS considers the underground production rate to be achievable.

The underground mine life is estimated at nine years in addition to the 15 months of preproduction.

16.11.1 Mine Development

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

Preproduction development is scheduled to:

- Minimize amount of development prior to production.
- Provide access for trackless equipment.
- Provide ventilation and emergency egress.
- Establish underground crushing and paste backfill infrastructure.
- Install mining services.

Two Owner development crews will start working at 5200 (60 m) level, 5400 (120 m) level and the new 84 m conveyor portals. Vertical raise development will be done with contract mining crews. The combined Owner and contract mining crews will:

- Slash 5200 level portal and adit to 5.0 m x 5.3 m size.
- Develop a 5200 level ventilation bypass for ventilation and mine air heating equipment.
- Develop underground infrastructure on 5200 level.
- Develop the main decline from 60 m level to -60 m level.
- Develop the main incline from 60 m level to 180 m level.
- Excavate crusher and paste backfill plant chambers.
- Excavate surge and fine ore bins and associated raises.
- Enlarge the 5400 (120 m) level portal 5400 level adit to 5.0 m x 5.0 m size.
- Provide sublevel lateral development on the levels between 5200 and -60 m.
- Develop fresh and return air raises between -30 m and 110 m levels.

The development schedule was planned based on estimated cycle times for jumbo and raise development, and benchmarked against best practices of North American mining companies and contractors. The underground mine will be nearly fully accessible by ramp at Year 4 of mine production.

Total underground capital and sustaining lateral waste development is 20,321 m and averages 1,942 m/a or 5.7 m/d over the 10-year project life. Annual waste development is shown in Figure 16-20.

Total ore sublevel development is 12,388 m and averages 1,239 m/a or 3.1 m/d over the nine-year ore production period. Annual ore development is shown in Figure 16-21.

Figure 16-20: Annual Waste Development

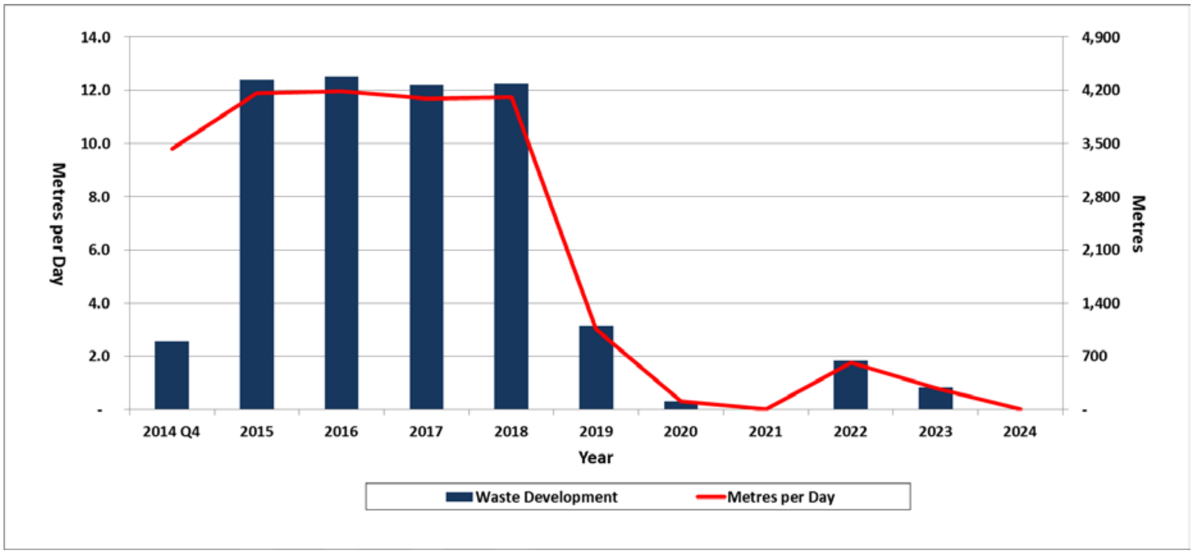
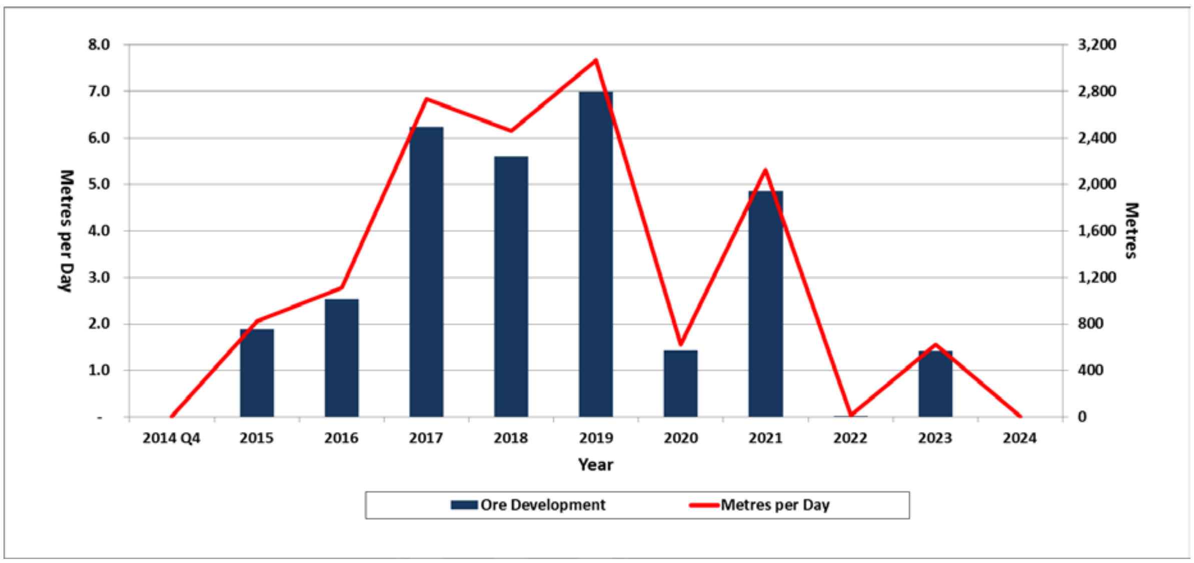
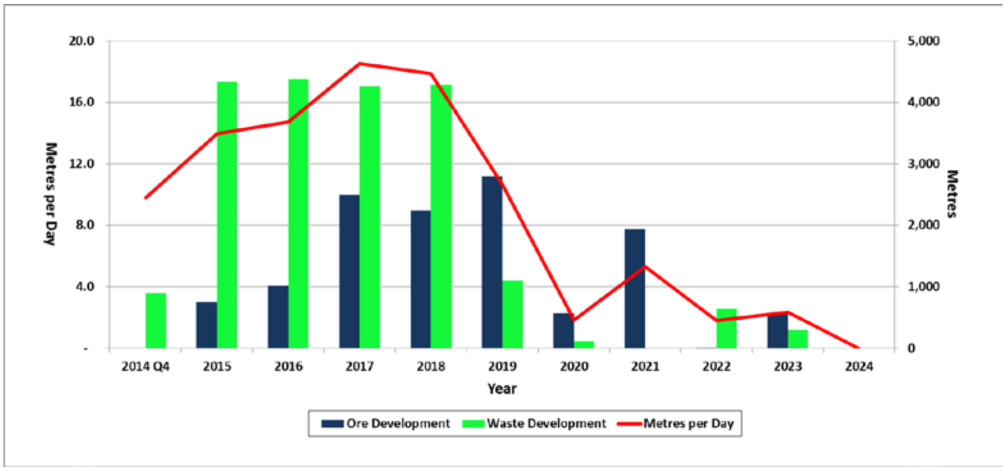


Figure 16-21: Annual Ore Development



Total ore and waste development is 32,709 m and averages 3,181 m/a or 8.8 m/d over the mine life. Annual total ore and waste development is shown in Figure 16-22.

Figure 16-22: Annual Total Development



16.11.2 Mine Production

The criteria used for scheduling underground mine production at the Tulsequah Chief mine were as follows:

- Target the mining blocks with higher grade rock in the early stages of mine life to improve project economics.
- Where possible, maintain a minimum zinc grade of greater than 5%.
- An average annual mill feed production rate of 730,000 t/a was scheduled, including ore from development and stopes.
- The mine will operate two 10-hour shifts per day, 365 days per year.
- Provide enough production faces to support a daily mine production rate of 2,000 t/d.

Mine production will commence from the stopes above -100 m level targeting the higher-grade mineralized zones while production from deeper higher-grade zones is deferred until the required development is completed in Year 4.

The stope cycle times and productivities were estimated from the first principles. It will require three production stopes working at any time to meet daily production requirements of 2,000 t/d. Initial ore development mined in Q4 2015 is stockpiled on surface for processing in 2016.

The average mined grades for the nine-year mine life are 1.13% copper, 5.59% zinc, 1.04% lead, 81.4 g/t silver and 2.30 g/t gold. Annual production by ore source and metal grades is shown in Figures 16-23 to 16-25.

The annual mine production schedule is provided in Table 16-11 and shows annual summaries of ore tonnage mined by deposit, ore grades and development quantities.

Figure 16-23: Annual Development & Stoping Production

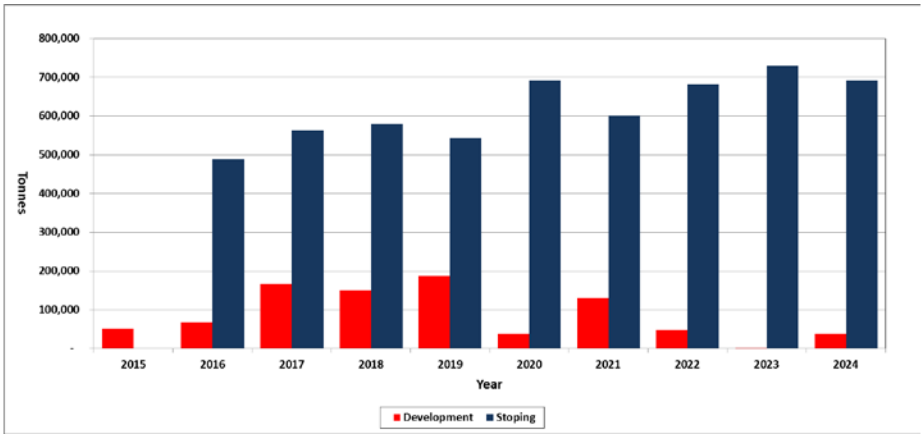


Figure 16-24: Annual Zinc, Copper & Lead Grades

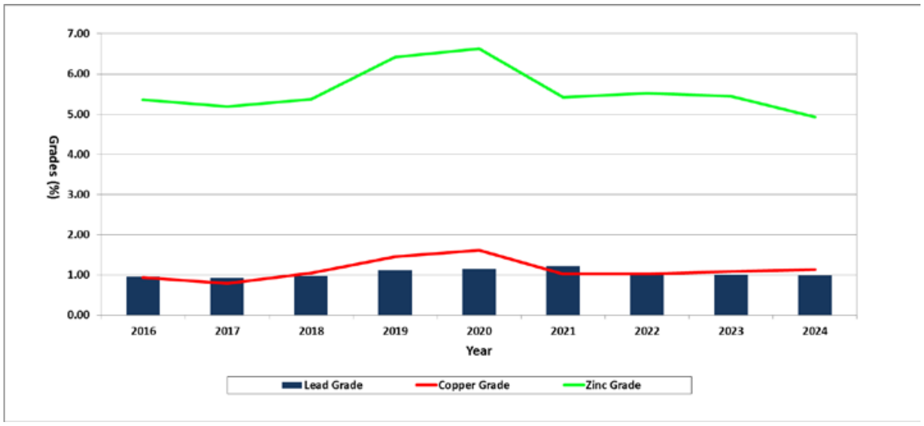


Figure 16-25: Annual Gold & Silver Grades

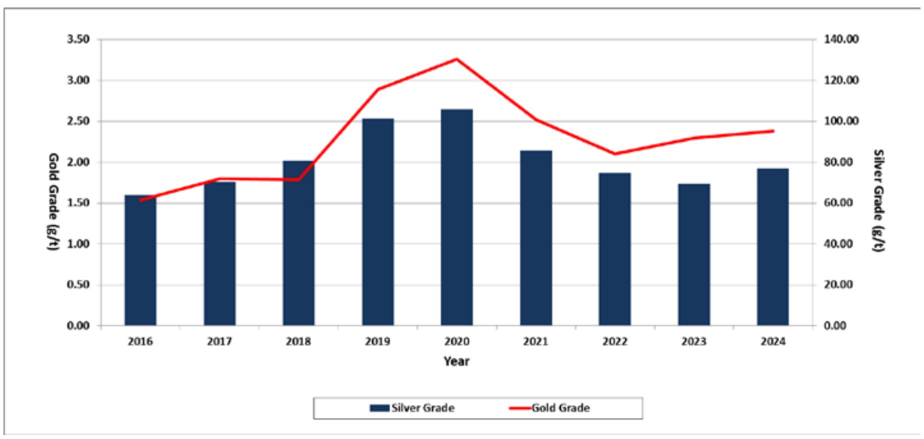


Table 16-11: Annual Mine Production & Development Schedule

Parameter	Unit	2014 Q4	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	Totals
Total Mine Production	**tonnes**	**-**	**50,401**	**556,697**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**730,000**	**6,447,098**
Daily Production Rate	t/d	-	548	1,521	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	**1,950**
Gold Grade	g/t	-	1.11	1.58	1.8	1.79	2.89	3.26	2.52	2.1	2.29	2.38	**2.3**
Silver Grade	g/t	-	44.68	65.61	70.46	80.71	101.27	105.9	85.73	74.91	69.58	77.09	**81.39**
Copper Grade	%	-	0.75	0.95	0.78	1.05	1.45	1.61	1.02	1.03	1.08	1.13	**1.13**
Lead Grade	%	-	1.00	0.96	0.92	0.98	1.13	1.14	1.22	1.00	1.01	0.98	**1.04**
Zinc grade	%	-	4.89	5.4	5.19	5.38	6.42	6.63	5.43	5.53	5.44	4.93	**5.59**
Equivalent Value	$/t		200.89	248.88	248.03	269.95	365.11	394.42	305.24	281.52	288.15	289.02	**299.5**
Total Waste Development	**m**	**902**	**4,339**	**4,380**	**4,266**	**4,285**	**1,100**	**112**	**-**	**644**	**293**	**-**	**20,321**
	m/d	**9.8**	**11.9**	**12**	**11.7**	**11.7**	**3**	**0.3**	**-**	**1.8**	**0.8**	**-**	**5.7**
Raise Development	m	-	324	532	427	364	190	-	-	-	-	-	**1,837**
Mined Underground Waste	t	63,263	285,955	268,489	261,152	234,477	68,640	5,564	-	28,248	13,716	-	**1,229,504**
Paste Backfill Placed	t	-	-	136,468	246,068	212,719	403,801	462,806	494,122	472,411	475,242	492,464	**3,396,101**

17 Recovery Methods

17.1 General

The plant will accept primary crushed ore from an underground storage silo. This will be fed to a SAG mill followed by two stages of ball milling to a cyclone overflow product of P_{80} 45 µm.

The cyclone underflows of the primary and secondary ball mills will be equipped with Knelson concentrators to recover gravity gold (electrum). The gravity gold concentrates will be intensively leached to ultimately produce doré.

The cyclone overflows will be treated in a sequential flotation circuit starting with copper then lead, zinc, and finally pyrite.

There is no regrinding in the flotation circuit as the mineralogical assessment showed that below the 40-50 µm particle sizing, further enhancement of liberation will not occur until below 5-10 µm.

The sequential system is very selective (with adequate liberation) and although multi-stage cleaning has been designed into the plant, this is conservative.

A flotation scale-up factor of 3.0 has been used for the plant design. This was necessary to reflect the need for effective control of pulling rates commensurate with the high selectivity.

The copper, lead and zinc concentrates will be thickened and pressure filtered in discrete circuits before storage and transport. The filtered lead concentrate will be handled in containers.

The pyrite circuit will produce a pyrite concentrate that will be fed to mine backfill as needed. The excess will be stored in a dedicated storage area.

A simplified flow diagram can be found on Figure 17-1. Detailed flow diagrams have been developed, but are not included in this report. Mill facility general arrangement plan and sections are presented on Figures 17-2 and 17-3.

Figure 17-1: Simplified Process Flow Diagram

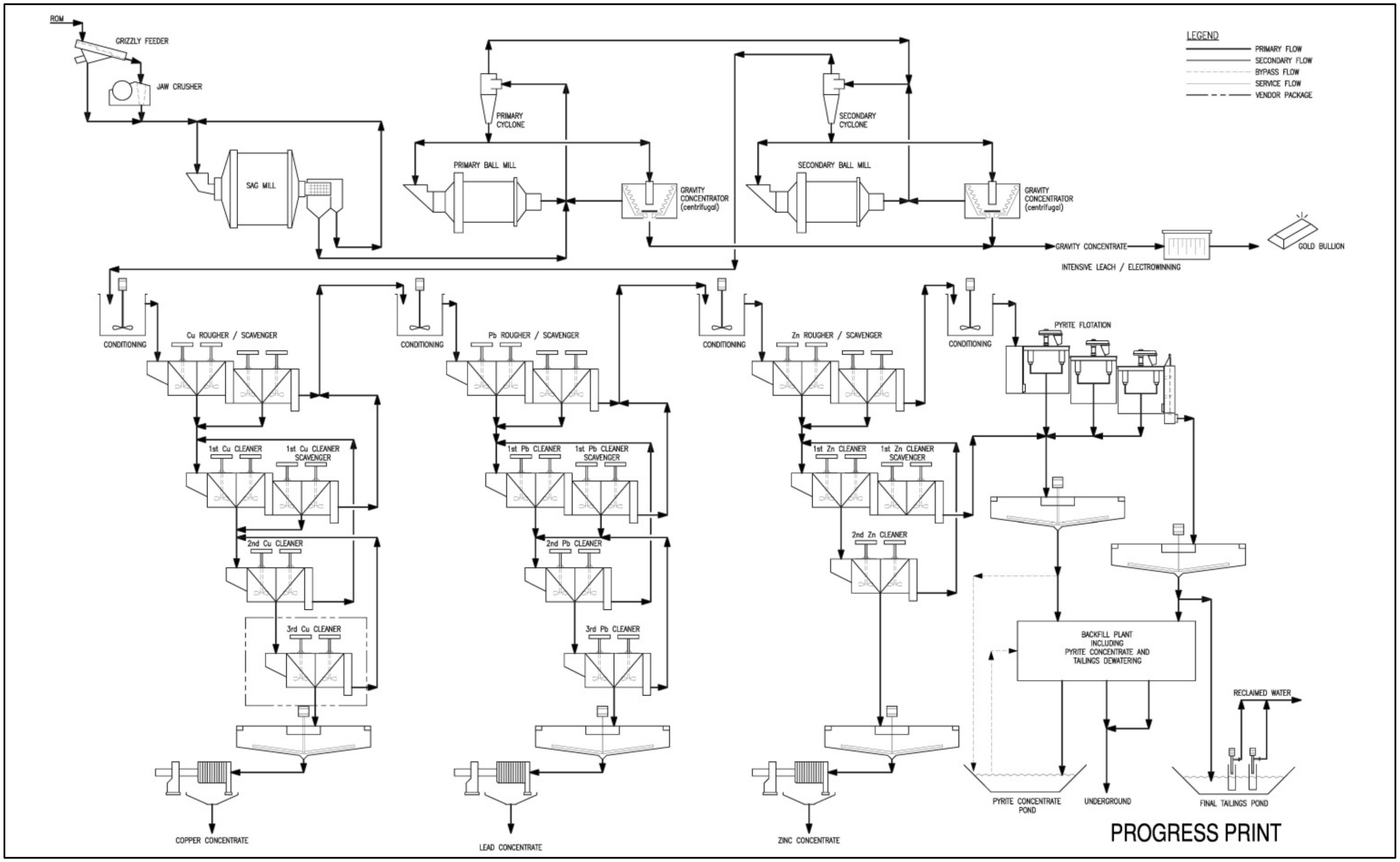


Figure 17-2: Mill Facility General Arrangement Plan

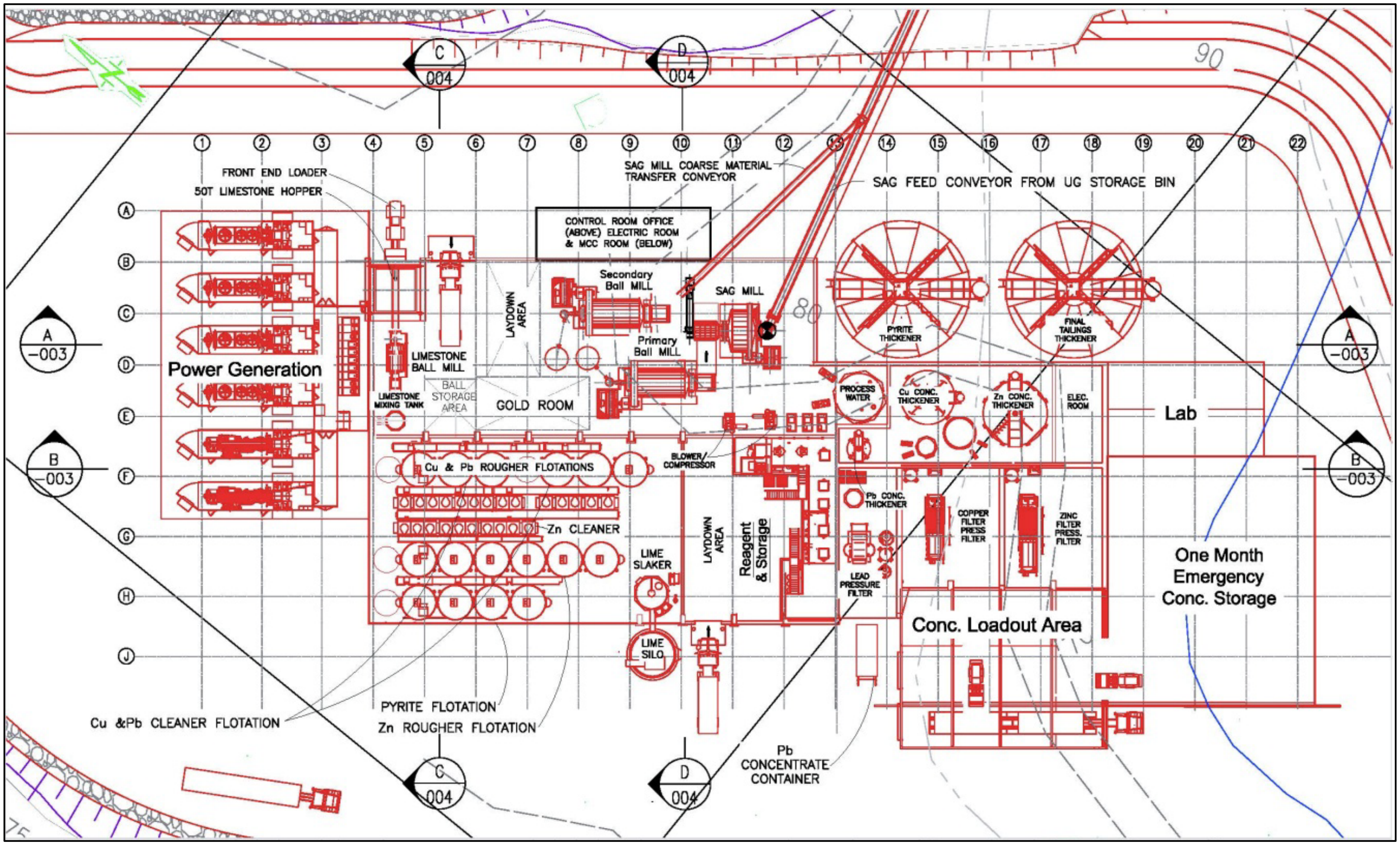
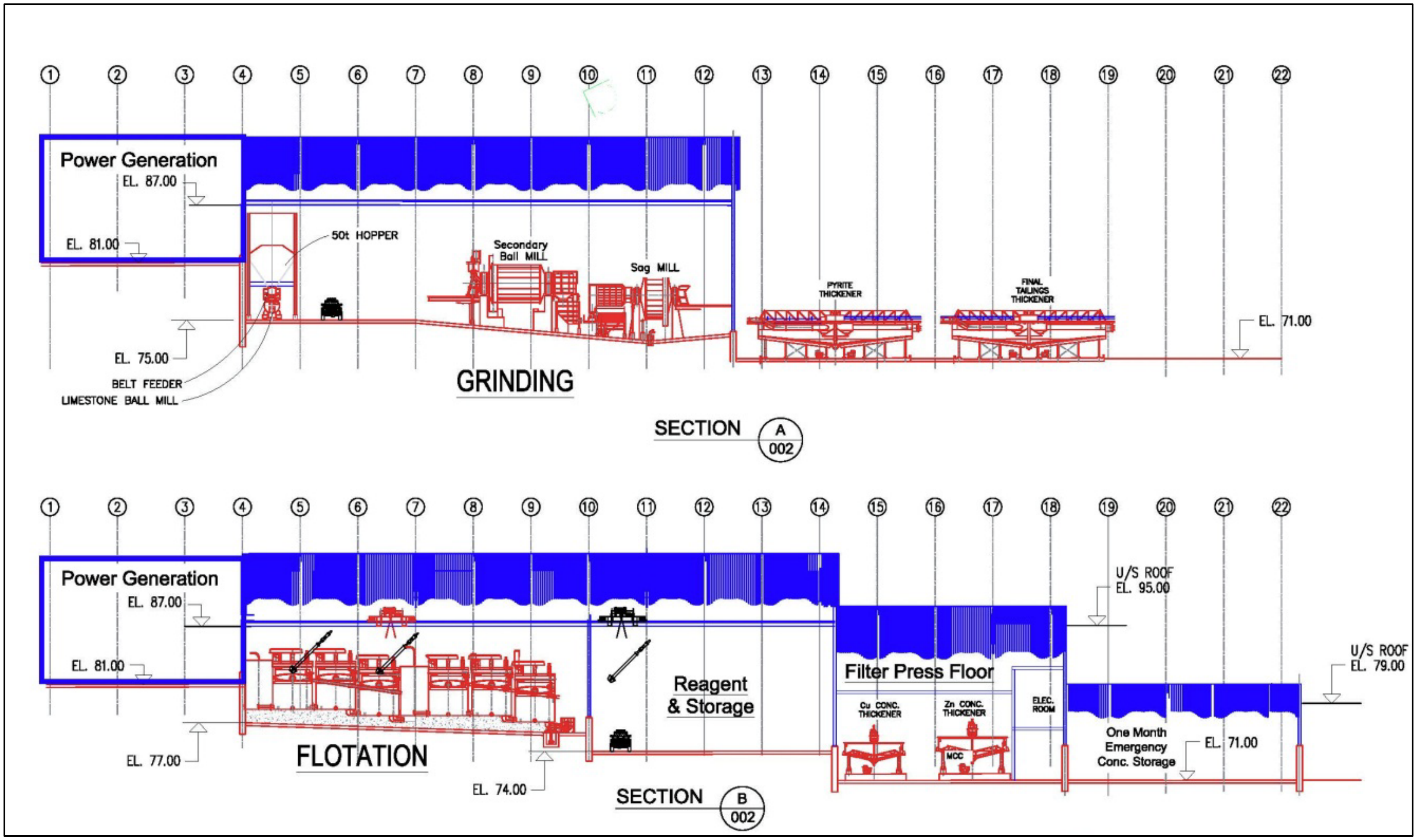


Figure 17-3: Mill Facility General Arrangement Sections



17.2 Mineralogy

The mineralogy of the ore can be summarized as follows:

- Copper minerals: Approximately 60% to 70% chalcopyrite and 30% to 40% tennantite/tetrahedrite. Note that the tennantite/tetrahedrite contains the bulk of the silver and all of the arsenic and antimony in the ore.
- Lead minerals: All as galena with some silver in the matrix.
- Zinc minerals: All as sphalerite with a low iron matrix, but with some minor mercury content.
- Iron minerals: Virtually all as pyrite; no arsenopyrite present.
- Gold: All as essentially free electrum; virtually no gold in the pyrite.
- Over 90% components liberated at 53 µm particle size.

Section 13 provides more details on mineralogy.

17.3 Grinding

Ore will be fed from a 2,000 tonne storage bin to a variable speed conveyor equipped with a weightometer and sampler. This will feed a 4.877 m diameter x 2.438 m long SAG mill powered by an 800 kW motor. A screen on the mill discharge will separate out pebbles to be recycled or discarded.

The design factor used was a work index of 10.0 kWh/t (Hazen, Figure 13-7 in Section 13), which indicated a power requirement of 650 kW, but this was enlarged to give commonality with the two ball mills.

The recycling of pebbles was included subject to further tests, but it is likely that the coarser, harder material will be barren dilutive rock. No pebble crushing was adopted, as the SAG work index and the rod mill work index were both lower than the ball mill index indicating that a buildup of intermediate size particles will not occur.

The SAG mill product will gravitate to a common pump sump with the primary ball mill.

The primary and secondary ball mills will be 3.35 m diameter x 5.76 m long powered by 800 kW motors a work index of 12.5 kWh/t was used for design.

A set of three 500 mm diameter cyclones will be fed from the SAG-primary ball mill sump. The cyclone overflow will flow to the discharge of the secondary ball mill.

The secondary ball mill will be in closed circuit with a set of seven 250 mm cyclones.

The overflow from these cyclones will flow to the copper flotation conditioner via a sampler. This stream is designed to have a P_{80} of 45 µm at 30% solids.

17.4 Gravity Gold Circuit

A portion of both cyclone underflows will be treated in Knelson concentrators. These units will be protected by trash screens.

Two 30 inch units will be used. The assessment done by Consep-Knelson indicated that 20 inch units should suffice, but that the larger units would give greater security in achieving the target of 42% gold recovery in this stream. The gravity recovery gold test gave a gold recovery of 53.7%. It is important to note that the gold is present as electrum with a composition of approximately 70% Au / 30% Ag. Electrum generally is less susceptible to plating than pure gold particles.

The gravity gold concentrate will be fed to a secure area containing a Consep-Knelson Acacia intensive leach reactor – the liquor will be electrolyzed to produce a doré.

The residue will be returned to the grinding circuit. Any excess liquor bleed will be fed to the copper flotation feed conditioner where the contained cyanide will be netted against the cyanide added as flotation reagents.

17.5 Copper Flotation

A copper conditioning tank will feed a set of three 50 m^3 tank cells as roughers. As with all the tank cells, these will be equipped with both external and internal launders to handle the very thin froths that will prevail. One stage of cleaning was sufficient to produce the desired concentrate grades in the laboratory but three stages have design into the plant to allow for greater flexibility. The copper circuit is largely open circuit with the first cleaner tail joining the rougher tail as a means of removing unwanted galena, sphalerite and pyrite.

The cleaner cells are a mixture of 5.1 and 2.8 m^3 units that will be fitted with cross launders to enhance the lip length for the thin selective froths.

Copper cleaner concentrate will be fed to a dedicated thickener.

The reagents used are lime, SMBS, Cytec 3510, zinc sulphate and MIBC.

At present, the philosophy is to recover both the chalcopyrite and the tennantite into one concentrate. This will mean that some pyrite will inevitably be recovered, thereby diluting the concentrate. The selectivity with respect to galena is good, but some unwanted sphalerite will report to the copper concentrate simply due to the overwhelming mass present (five times that of the copper minerals).

Nearly all of the silver in the ore is within the copper tennantite mineral, as is all of the arsenic.

It has been shown that the chalcopyrite and the tennantite can be effectively partitioned by flotation (e.g., over 60% of the total copper can be recovered into low arsenic (less than 0.5% As) concentrate). However, this greatly lowers the silver values of this stream, as all of the silver is in the tennantite. Trying to accomplish this delicate balance in a real time circuit was deemed impractical as it could easily upset the subsequent lead and zinc flotation steps. It was decided that any such chalcopyrite/tennantite separation was best accomplished in isolation using the mixed concentrate as a feed with no recycling into the current circuit.

This work is ongoing, but one clear advantage to this approach is that misreported sphalerite can be returned directly to zinc concentrate. It is important to note that none of the water from this possible future circuit can be returned as process water without treatment as the additional reagents used will be deleterious to the main process.

17.6 Lead Flotation

Copper flotation tails will be fed the lead flotation conditioner and thence to a bank of three 50 m^3 rougher cells. The rougher concentrate will be fed to three stages of cleaning incorporating five 2.8 m^3 cells. Testwork indicated that two stages should suffice. Lead cleaner concentrate will be fed to a dedicated thickener.

The reagents used are lime, NaCN, Zinc sulphate, Cytec 3418a and MIBC.

17.7 Zinc Flotation

Lead circuit tails will be fed to the zinc conditioner and thence to a bank of six 50 m^3 cells. The rougher concentrate will be fed to three cleaning stages incorporating ten 5.1 m^3 cells. Zinc cleaner concentrate will be fed to a dedicated thickener.

The reagents used are lime, Copper sulphate, Cytec 7021 and MIBC.

17.8 Pyrite Flotation

Zinc rougher tails will be fed to the pyrite flotation conditioner and thence to bank of four 50 m^3 rougher cells. The rougher concentrate will be fed to the pyrite thickener and the rougher tailings to the tails thickener.

The reagents used are PAX and MIBC.

Early testwork on pyrite flotation was conducted at acidic pH but later tests produced identical results at alkaline pH thereby relieving the burden of adding sulphuric acid as an activator and then using lime to readjust the pH of the rougher tailings.

17.9 Concentrate Handling

Copper, lead and zinc concentrates will be handled in a similar manner; each concentrate will have a dedicated thickener and filter and three discrete loadout areas. Copper and zinc will be treated as bulk and truck loaded, while the lead will be fed into ISO containers directly from the filter.

17.10 Tailings

Pyrite flotation tailings will report to the final tails thickener. Thickener underflow will be pumped either to the backfill plant or by a separate three stage pumping circuit to the tailings dam.

17.11 Process Control

An on-stream analyzer system will be incorporated within a distributed digital control system in order to give operators effective control of the process. The testwork did illustrate that effective macro control can be affected visually and it is an important aspect of the plant that clear line of sight of the flotation circuit is available to the operating staff.

18 Project Infrastructure

18.1 Access Road

The mine site access road consists of a 128 km all-season route from the Warm Bay Road to the mine site, typical of a remote mine site access road constructed through steep mountain terrain. This 128 km route between the town of Atlin, BC and the mine site's process plant will provide the necessary road access to and from the mine for the efficient transport of materials and equipment to support the planned construction activities. During operations, the access road will be used for exporting concentrate and for importing fuel and consumables to support mine, process plant, effluent treatment plant, and camp operations.

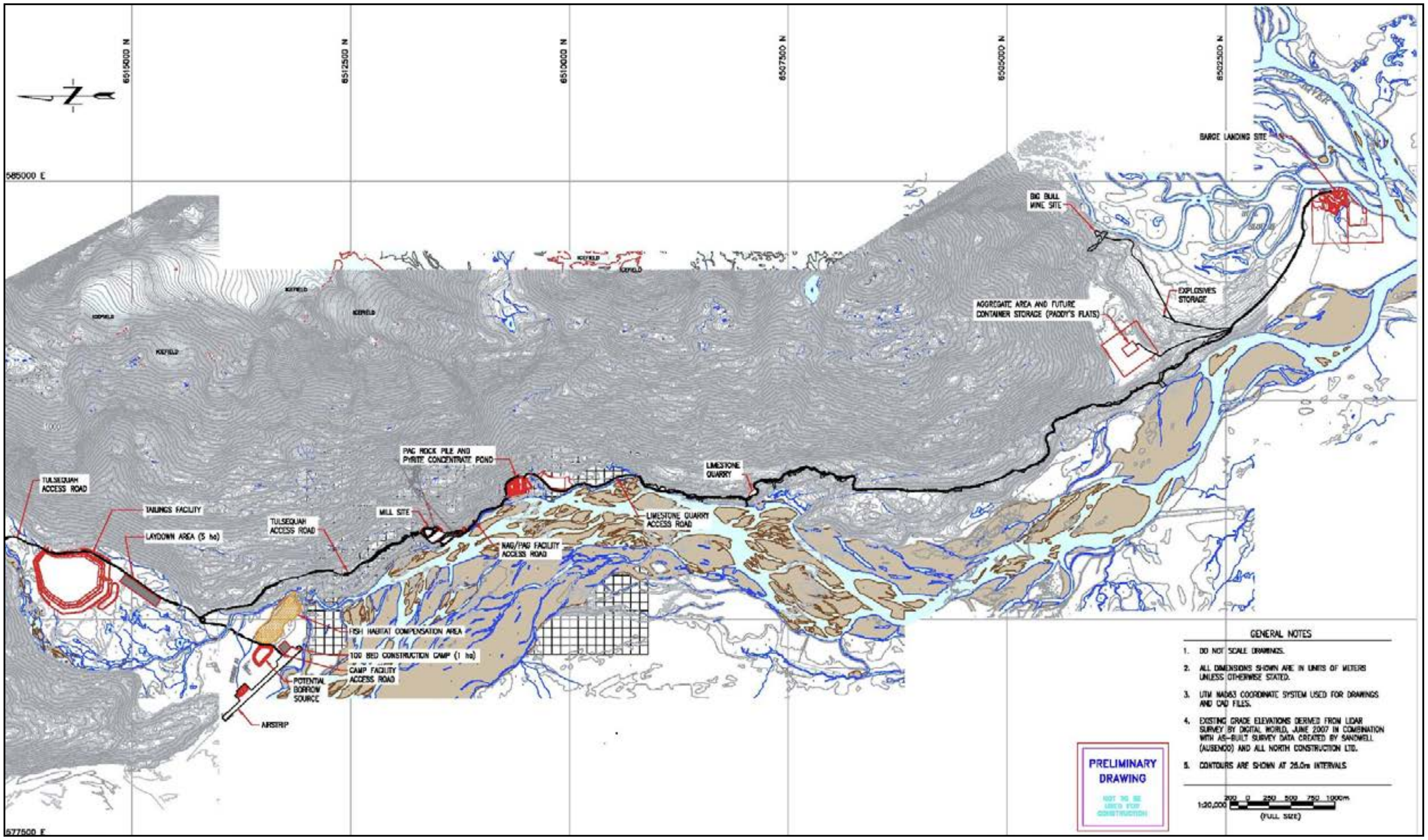
The access road design constraints were determined by McElhanney Consulting Services Ltd. (McElhanney) and Onsite Engineering Ltd. (Onsite) in consultation with SNT Engineering Ltd. (SNT) and JDS. The design speed limit for the access road is 30 km/h with the maximum adverse sustained grade of 8% with 10% allowed for distances up to 100 m in length. Favourable sustained grades were designed at a maximum of 12%, with 14% favourable grades allowed for distances of up to 150 m in length.

The elevation at the plant site is approximately 50 m above sea level and the summit on the access road will be at approximately 1,220 masl. Tridem drive trucks will be used for haulage of materials, supplies, and equipment along this access road. The planned access road route includes 24 bridges. The access road is designed with a 5 m width with four pullouts per kilometre to allow vehicles to pass each other.

The access road will be maintained year-round by a contract road maintenance service supplier. Access to the road will be controlled by Chieftain south of the town of Atlin for safety and environmental reasons.

Figure 18-1 shows the general arrangement of the project site.

Figure 18-1: Project Site General Arrangement



18.2 Power Supply

Power for the site will be supplied by diesel generators that are designed to run on a blended fuel mixture consisting of 70% liquefied natural gas (LNG) and 30% diesel fuel. The power plant will consist of six CAT 3516 generator sets, each with a peak rating of 2,250 kW. Each unit will include a fuel blending kit that will combine and blend the re-gasified LNG fuel with the diesel fuel to the recommended mix of 70% LNG and 30% diesel fuel.

The blended fuel power generation system was selected to take advantage of the lower cost LNG fuel while maintaining the flexibility to use the diesel fuel in the case of unexpected or prolonged access road closures.

The fully enclosed units will be located at the north end of the processing plant (the primary power draw). The generators will be supplied with exhaust heat recovery packages, which will be utilized to provide heat to the underground mine and processing plant, as needed.

LNG storage tanks and diesel storage tanks will be located in close proximity to minimize piping requirements and simplify fuel delivery to the generators. The LNG storage tanks will be located on the northwest end of the processing plant pad as close as allowable to the power plant. Two diesel storage tanks will be located in an area northeast of the camp. The power plant is the primary fuel consumer on the site. Anticipated site-wide power loading is estimated at 7.5 MW for the camp, processing plant, underground mine, and ancillary facilities.

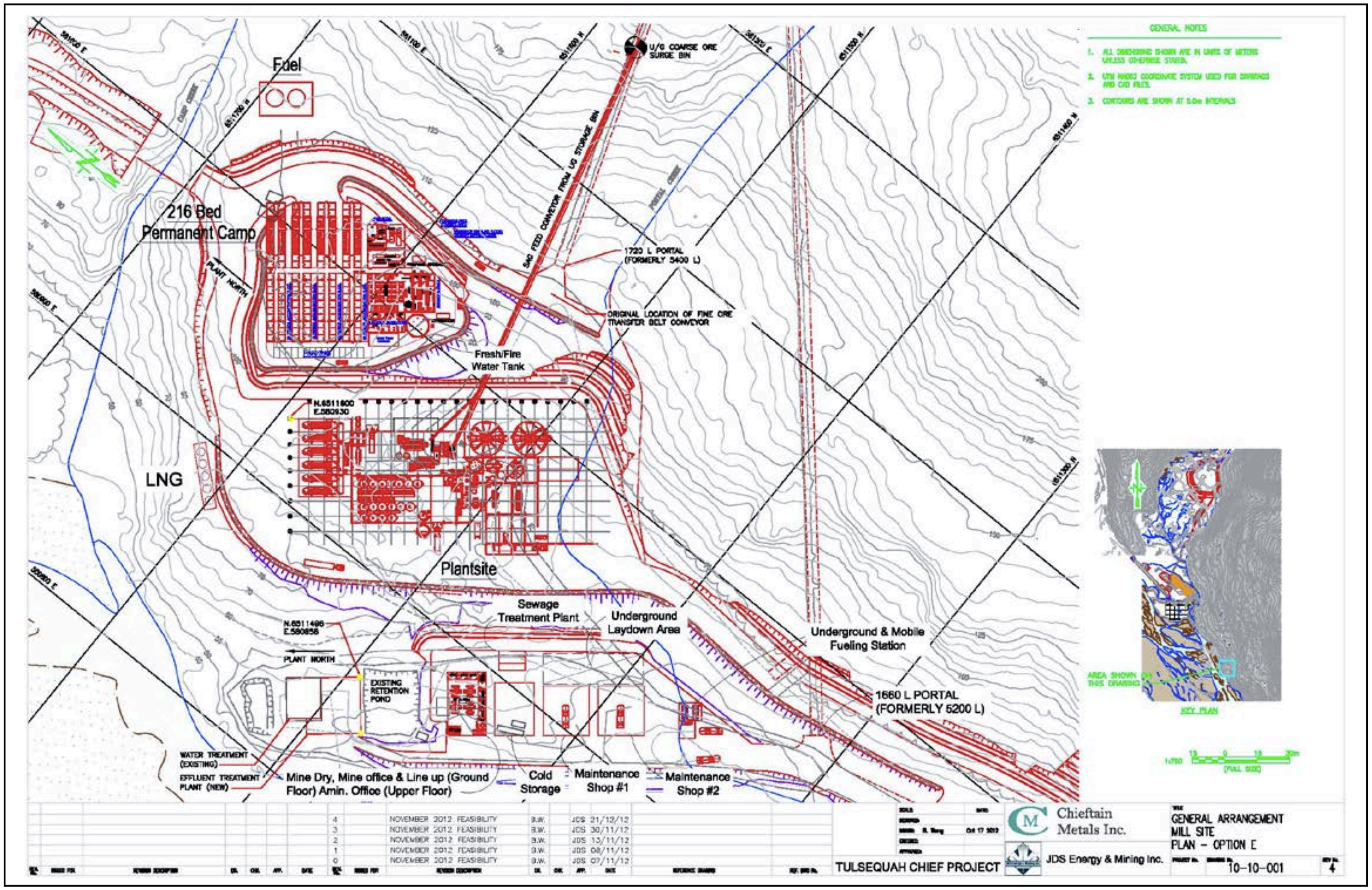
Table 18-1 summarizes the average power demand for the mine site by area.

Table 18-1: Estimated Power Demand by Area

Area	Power Demand
Mine	2.0 MW
Process Plant	4.2 MW
Camp, Admin, Shops	1.1 MW
Effluent Treatment	0.2 MW
Total Demand	**7.5 MW**

Figure 18-2 shows the mill site layout and the relative locations of the generators, LNG storage tanks and containment, diesel storage tanks and containments, camp, admin/mine dry, shops, cold storage warehouse, and the water treatment and sewage treatment facilities.

Figure 18-2: Mill Site Area Layout



18.3 Power Distribution

A 15 kV switchgear module is supplied with the vendor generator package. A 13.8 kV overhead power line will be constructed to supply power to the underground mine at the 5200 Portal, the mine maintenance shops, administration building/mine dry, and water treatment plant, and will continue south to the pyrite and PAG storage facilities and incinerator building. Underground power will be fed from redundant feeds to avoid a single point of failure, and an emergency generator will be provided to ensure ventilation is maintained in the event of a power outage. A second 13.8 kV overhead power line will be run north from the plant site to the airstrip and tailings storage facility.

A secondary power distribution substation and motor control center will be located in the process plant facility to energize and control the process equipment. A separate 5 kV switchgear center will be provided for power supply to the grinding mills. An emergency generator will also be provided for the critical loads in the process plant.

18.4 Heat Recovery

Based upon vendor performance data for the generators and the estimated load levels, an estimated 6.2 MW of heat energy will be rejected through the engine exhaust. The power generation system planned for the Tulsequah site will be equipped with heat recovery systems on each of the six generators. Heat will be recovered from the generator engine exhausts via exhaust gas/glycol water heat exchangers and will be distributed by insulated piping loops to heat the process plant and the underground fresh air as required. Additionally, an emergency backup boiler will be installed at the camp in case of a complete power station failure.

Additional opportunities exist to further exploit the recovered heat from the generator exhaust. For example, electrical loading requirements could be reduced by replacing some of the electrical heat loading with the recovered heat.

The estimated heat recovery for the mine ventilation air and for the processing plant is shown in Table 18-2. Also shown in the table are the potential opportunities for further savings from the use of more of the recovered heat.

Based on Tulsequah site climate data, conservative peak January heating requirements are estimated as shown in Table 18-2.

Table 18-2: Estimated Space Heating Requirements

Area	Estimated Heat Power
Mine Ventilation	1.2 MW
Process Plant	1.5 MW
Potential Areas of Further Opportunity	
Camp	2.0 MW
Offices and Dry	0.5 MW
Truck Shops	1.0 MW
Total Peak Demand	**6.2 MW**

18.5 Construction Camp

A 98-man construction camp will be mobilized to the site as part of the 2013 barge campaign to supplement the existing 45-man pioneer camp. The construction camp will be located on a new foundation pad near the existing camp and airstrip.

This camp will be a typical skid-frame style modular facility. It will consist of two, 49-man dormitory wings, each with its own communal washrooms, shower facilities, and laundry rooms. The kitchen and dining facility is a three-module complex capable of accommodating up to 48 people per seating.

The camp will be propane heated with power provided by suitably sized diesel generators. Potable water for the construction camp will be supplied from the existing water well on site. A water treatment unit will be installed to ensure potable water quality is achieved and maintained. A modular wastewater treatment plant will also be installed, as the existing plant for the pioneer camp cannot accommodate any additional load.

18.6 Main Camp

The main camp will provide approximately 55,000 ft^2 of modular living space. The camp will consist of modular 216 single bedrooms, two-storey dormitories with washrooms, showers, and laundry facilities located on each dormitory floor. The kitchen and dining facility will serve up to 140 people at one sitting. The recreation area will be a two-storey modular facility located adjacent to the dining area and will have a TV room area, washrooms, and exercise facilities.

A workshop and tool storage area and a first-aid/medical area with a few examination rooms and medical equipment are also included in the camp facility, adjacent to the dining area opposite the recreation area. A covered carport-type area will also be constructed alongside the first-aid/medical modules to house the ambulance and the fire truck.

A potable water treatment system will be located off the recreation area modules. The system will treat water gravity fed from Dawn Creek or pumped from the Tulsequah River or from a new water well (to be determined during detailed engineering). This potable water treatment plant will ensure that drinking water for the site meets potable drinking water quality requirements.

The camp has been designed for electric heat, but use of heat recovered from the power generation system's generator exhausts should be sufficient for most of the year to supply heat for the camp and this will be evaluated further during detailed engineering.

A 750 kW diesel generator will supply power for the camp until the main power generation system is commissioned.

18.7 Fuel Storage

Fuel storage for on-site power generation includes five 130 m^3 LNG storage pressure vessels and two 325,000 L capacity diesel storage tanks, each with offloading and distribution systems. A separate system consisting of two 75,000 L diesel storage tanks and a dispensing system for mobile equipment will be located near the 5200 portal.

The LNG storage is sufficient to operate the generator units (on blended fuel) for up to seven days without refueling. The dimensions of the LNG storage vessels are 12 ft diameter x 55 ft in length. These "bullets" will be installed on concrete footings and piers, and will have a suitably sized concrete slab and berm for spill containment. The bullets are piped to a common header, which is piped to a vaporizer unit to convert the LNG back to methane gas.

The diesel tanks provide sufficient storage to operate the mine (generators and mobile equipment) for approximately 40 days without refueling when LNG is available, or approximately 15 days in the event of an LNG supply disruption when the generators will have to be operated using 100% diesel fuel. The dimensions of the two diesel tanks are 26 ft diameter x 24 ft. The tanks will be field erected and contained in an HDPE lined area with pre-cast concrete retaining walls.

An offloading and fuel transfer facility consisting of two centrifugal pumps with 30 m^3/h capacity will be located outside of the fuel storage berm. A concrete slab and sump are provided at the offloading station for containment of any leaks during offloading and transfer. The fuel will be delivered to the mine site in conventional tanker trucks of 40,000 to 60,000 L capacity and offloaded into the storage tanks. The fuel is transferred to the diesel generator day-tanks by gravity flow through a steel pipeline, by opening a solenoid valve that is actuated by level switches in the day-tanks. The piping in the offloading module is configured so that the offload pumps can also be used to transfer fuel from the storage tanks if required.

The mobile equipment storage and dispensing system is currently on site and will be relocated to a suitable location near the 5200 portal. A concrete slab and sump will provide for spill containment during refuelling. A single offload/transfer pump will be provided at this location for unloading the tanker trucks into the storage tanks.

18.8 Water Supply

The water supply for site operations will be drawn from the Tulsequah River or from a new water well, to be determined in detailed engineering. Treatment and monitoring will be conducted to assure water quality standards are maintained that are appropriate for the end use of each water stream.

18.9 Sewage Water & Solids Disposal

A modular wastewater (sewage) treatment plant (WWTP) will be located on the lower pad level by the administration/mine dry, warehouse, and shops area and will treat sewage from all of the camp and other site facilities containing washrooms, showers, and/or laundry facilities. Treated effluent will be discharged.

Solids generated in the WWTP from the treatment of the sewage will be dewatered, bagged, and incinerated in the on-site incinerator facility that will be located near the PAG storage facility.

18.10 Mine & Effluent Water Treatment

18.10.1 Acid Treatment Plant

The acid treatment plant (ATP) was designed to treat 40 m^3/h of acidic mine discharge water generated in the old upper workings of the Tulsequah Chief mine. The water contains elevated concentrations of dissolved metals, total metals and non-metals, discharging from the historic 5200 and 5400 level portals. The water is directed to an on-site retention pond and then to the ATP. Portal discharge water quality parameters are shown in Table 18-3.

Table 18-3: Water Quality of Portal Discharge

Parameter	5400 Portal (mg/L) Maximum	5400 Portal (mg/L) Average	5200 Portal (mg/L) Maximum	5200 Portal (mg/L) Average	5900 Portal (mg/L) Maximum	5900 Portal (mg/L) Average
Aluminum	18.7	10.8	8.8	5.6	1.9	1.6
Arsenic	0.63	0.44	0.10	0.034	0.05	0.05
Cadmium	0.51	0.31	0.23	0.18	0.11	0.10
Copper	32.9	19.1	9.92	7.46	4.5	3.7
Iron	80	47.2	11	7.4	8.6	7.7
Lead	0.05	0.037	0.11	0.072	0.16	0.16
Zinc	127.0	74.6	78.7	45.6	27.0	23.7
SO_4	800	525	758	443	135	127
Ca	113	96.4	279	141	39	36
pH (no units)	2.9	3.2	4.0	4.5	7.1	6.7

1 Source of data: Redfern archived data in spreadsheet titled “Portal discharge quality 2005.xlsx” and spreadsheet titled “Water Treatment basis design V2.xls”. 2 Total concentrations are included in this table.

The treatment is a conventional process of neutralization of the acidic mine water with lime, followed by a solids separation in a clarifier. The water is directed to a neutralization tank where hydrated lime is added to precipitate dissolved metals in the form of hydroxides. Flocculent is then added to the slurry before it enters an Inclined-Plate Settler clarifier. The sludge is either recycled to the neutralization tank to act as a precipitation seed, or is removed from the system. Initially the sludge is transported to a pit adjacent to the airstrip, but will be incorporated into the paste backfill once operations commence. The clarified water is then passed through a polishing filter and a final pH adjustment stage. Zinc concentration, pH and turbidity are checked to determine if the water quality is sufficient for discharge to the environment. If water quality standards have not been met, the water is sent back to the retention pond to undergo retreatment.

Table 18-4: ATP Reagent Consumption Rates

Reagent	Units	Consumption
Hydrated Lime	g/m^3	160
Ferric Chloride	g/m^3	0
Flocculent	g/m^3	2.8
Hydrochloric Acid	g/m^3	0.5

18.10.2 Effluent Treatment Plant

The effluent treatment plant (ETP) was designed to treat the following water sources produced during mine operations: mill water, tailings water, neutral underground water, PAG containment water, site runoff and the ATP effluent. These water sources will be directed to the site retention pond and will then be pumped to the ETP. The ETP is designed to treat up to 260 m^3/h in two 130 m^3/h twin circuits. The process will produce between 13 and 23 kg/h of solids that will be incorporated into the paste backfill mix.

Treatment begins with water being pumped from the site retention pond into the mixing tank. Hydrated lime and coagulant are added to precipitate dissolved metals, total metals and non-metals. The water is then directed into an ACTIFLO® system comprised of four tanks. The first is the coagulation tank, the second is the injection tank where polymer and micro-sand is added, the third is the maturation tank, and the final is the settling tank with lamella plates across the top. Sludge is removed from the bottom of the settling tank and pumped to a cyclone. This removes micro sand from the sludge, recycling it back to the injection tank. The sludge exiting the cyclone will either be recycled back into the initial mixing tank to act as a seed, or pumped to the paste backfill plant for incorporation into the paste backfill material. The settling tank overflow is directed to the DUSENFLO® filtration unit using anthracite and sand as filter media. After filtration, the clean water undergoes a final pH adjustment in the post neutralization tank. From there, the water is stored for use as reagent preparation water, backwash water, and mill backup water supply, or is discharged to the environment through a diffuser.

Table 18-5: ETP Reagent Consumption Rates

Reagent	Units	Consumption
Hydrated Lime	g/m^3	80
Ferric Sulfate	g/m^3	150
Polymer	g/m^3	1
Sulfuric Acid (66 degree Be)	L/1000 m^3	0.64 (solution)

*Dosages have not been verified with laboratory testing.

18.11 Tailings Management Facility

18.11.1 Summary

Detailed design of the TMF is presented in the KCB Report entitled, “Tailings Management Facility – Detail Design,” (February, 2012). The TMF is located approximately 4 km upstream (north) of the main mine facilities on the east bank of the Shazah Creek. The TMF will store 3 Mt of non- acid-generating (NAG) tailings over the operating life of the mine. The 45 ha impoundment will be formed with a homogeneous compacted earthfill dam with a 1.0 mm (60 mil) LLDPE geomembrane liner. The majority of the impoundment dams will be constructed prior to operations using material excavated from within the impoundment area and raised later in the mine life, as required.

The perimeter embankment will have a 6 m wide crest at El. 80.0 m (up to 14.5 m high) and will be 2.2 km long. The upstream and downstream slope angles are both 2.5H:1V, and a stabilization berm will be constructed at the toe (the berm width varies based on stability requirements for the design earthquake). The dam is designed to the Canadian Dam Association Guidelines (2007) for a “High” consequence structure.

Water from the TMF will be recycled back to the process plant, and storage is provided for an environmental flood and an emergency spillway is provided for dam safety. An access road will be routed along the toe berm on the east side of the impoundment. Riprap armouring will be placed along the toe of the stabilization berm to protect against erosion from possible flooding of Shazah Creek or Chasm Creek. On closure, the TMF will be drained, capped with a soil cover, and revegetated. A general arrangement of the facility is shown on Figure 18-3.

18.11.2 Design Basis

The tailings dam is designed to National standards using the Canadian Dam Association – Dam Safety Guidelines (CDA, 2007). The dam classification and design criteria are discussed in the following subsections.

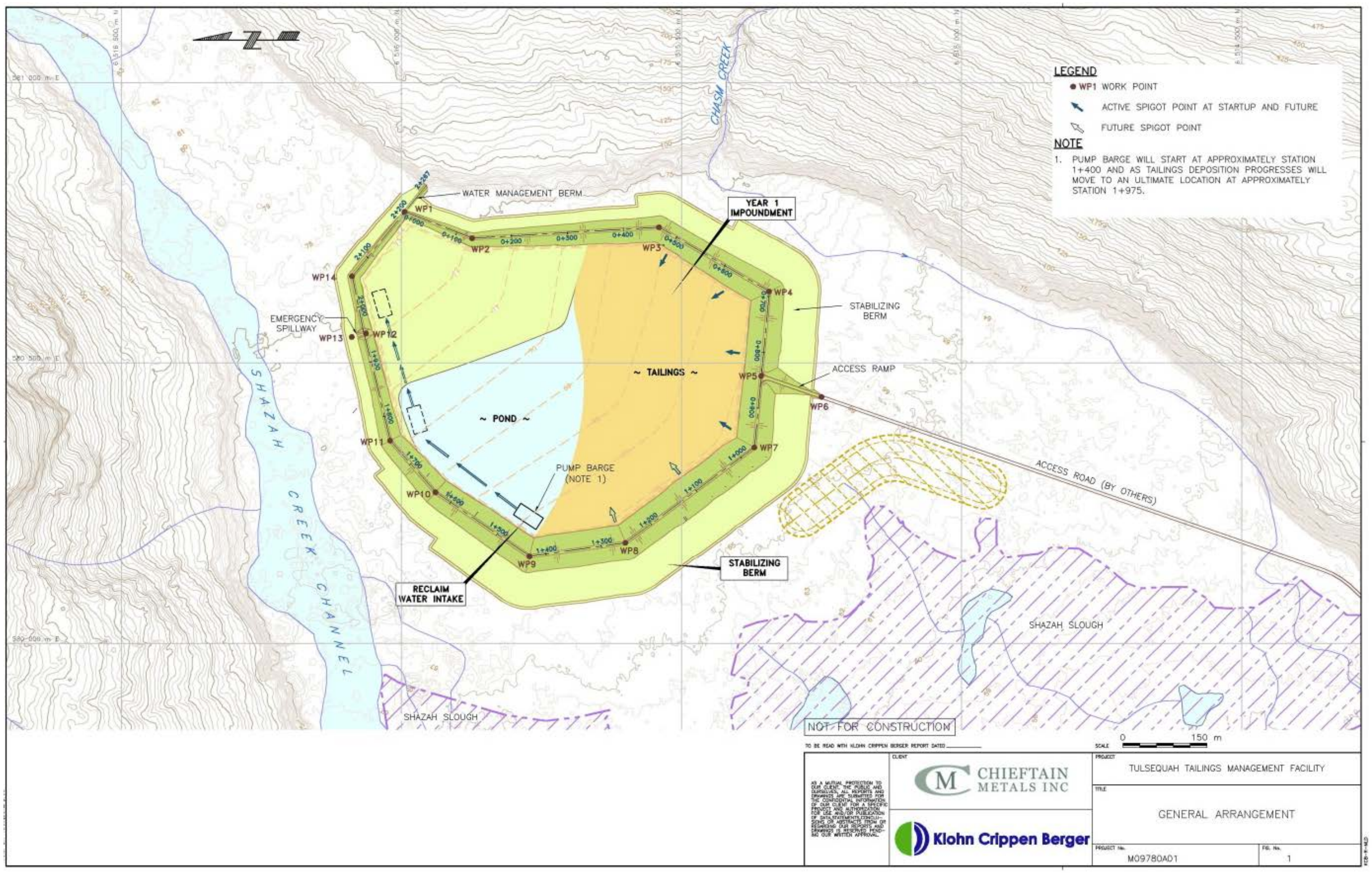
Tailings will be de-pyritized in the mill and have lime added in the milling process to provide an alkaline buffer. In addition, the tailings will be saturated within the impoundment, which will further reduce the risk of acid generation. The design of the impoundment is based upon minimizing the potential for seepage from the impoundment and keeping the tailings permanently saturated, and meeting the dam safety criteria. The Canadian Dam Association (CDA, 2007) Dam Safety Guidelines were used to determine the dam classification for seismic and flood protection criteria. Based on a dam break analysis, the selected dam classification is “High” during operations and “Significant” on closure.

For a “High” consequence dam, the maximum design earthquake (MDE) has annual exceedance probability of 1 in 2,500 years, which has an associated peak ground acceleration of 0.20 g and a magnitude M=6.7. The recommended flood design parameters are summarized as follows:

- The inflow design flood (IDF) is 1/3 between 1/1,000 year and the probable maximum flood (PMF).
- The TMF will store the 30-day 1/200 year precipitation event.
- The TMF will be designed with an emergency spillway to route the IDF during operations.
- The TMF will be designed with a permanent closure spillway to route the peak flow from the IDF on closure.
- A minimum of 2 m freeboard will be provided.

The dam toe will be designed to withstand the IDF in Shazah Creek and Chasm Creek, without catastrophic failure of the dam.

Figure 18-3: General Tailings Management Facility Arrangement



18.11.3 Site Conditions

The TMF site is located in mountainous terrain on the northern coast of BC, bordering Alaska. The mountaintops are steep, rugged ridges and peaks, while the valleys have forested, steeply sloping sides with networks of narrow streams and creeks flowing into larger rivers that meander across wide, flat floodplains. The TMF is located 3 km upstream of the Shazah Creek confluence with the Tulsequah River. The dam will be located in a steep-sided, glaciated valley with a gentle gradient of approximately 1% at the dam site. The surficial geology of the project area is dominated by glacio-fluvial processes accelerated by the high precipitation and steep mountainous topography. In general, the soils near the TMF consist of silty sands and gravels. Deposits of finer grained clay till are scarce.

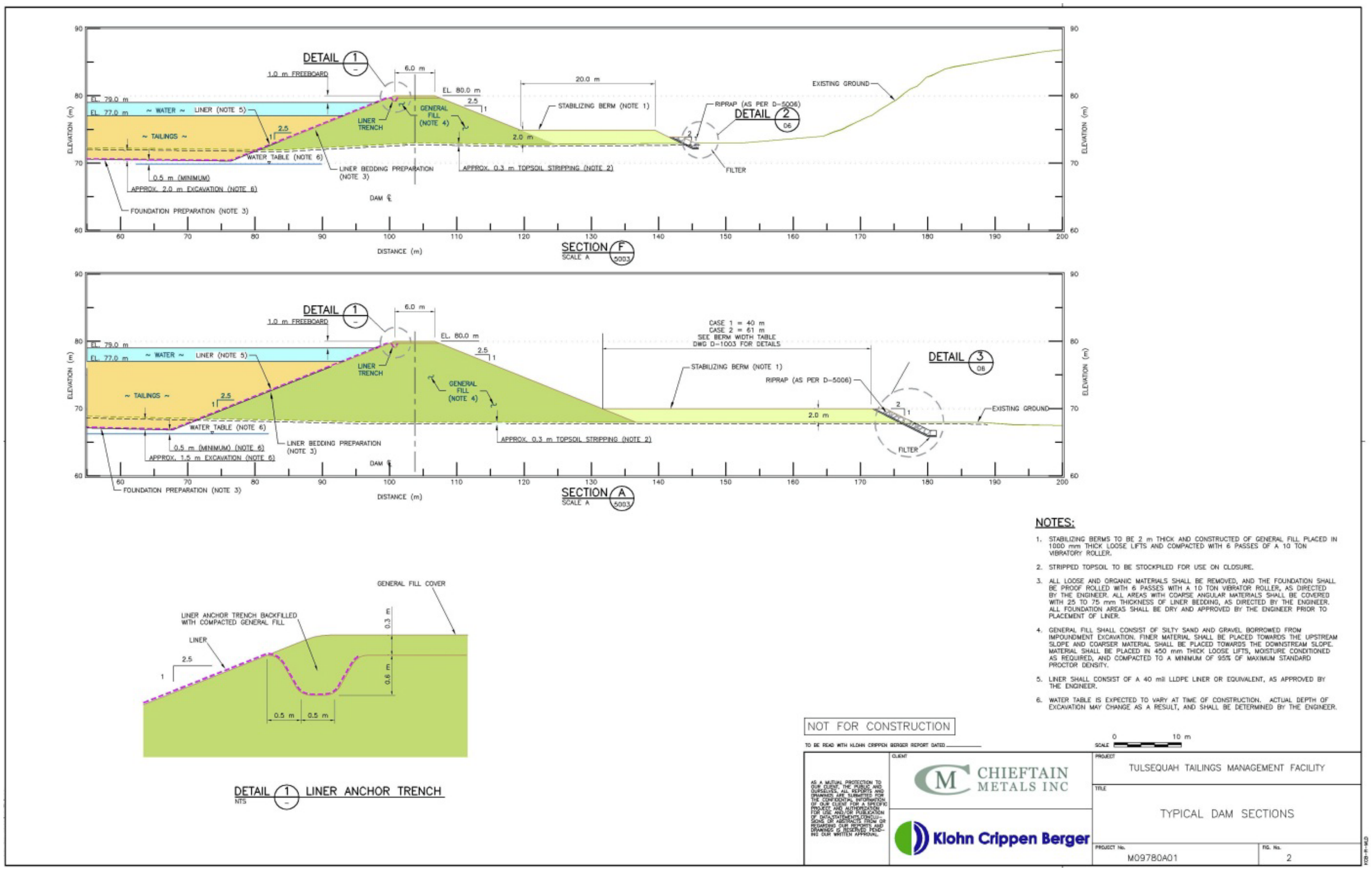
Site investigations include various programs carried out by KCB (2008), TBT (2007) and BGC (1995) and included drilling, static cone penetration tests (CPTs), dynamic CPTs and geophysical surveys. The soils in the upper 10 m of the foundation of the dam are inter-bedded and behave as sands or sand-silt mixtures. The CPT and standard penetration tests (SPT) data from these programs indicated that the average penetration resistance at the site was approximately (N1)60 = 37, with a reasonable lower bound of (N1)60 = 8. The average permeability in the deposits tested was 6 x 10-3 cm/s. Higher densities were associated with layers identified as sands, and lower densities were associated with layers identified as sand-silt mixtures. The CPT data suggested that the lower density sand-silt deposits might be potentially liquefiable.

A seismic hazard assessment was carried out to determine the appropriate seismic parameters for the selected seismic design criteria. Probabilistic and deterministic seismic hazard analyses were conducted to derive the MDE parameters for the design return period of 2,475 years. The peak ground accelerations of between 0.06 and 0.20 g were predicted.

18.11.4 Dam Design

The dam will be constructed of homogenous fill excavated from the footprint of the facility. The dam has a crest width of 6 m and is up to 13 m high (to 80.0 m elevation), with upstream and downstream slopes of 2.5H:1V. The toe berm will be 2 m thick and from 18 m to 61 m wide. A mine access road will be routed along the toe berm on the east side of the impoundment. Riprap armour will be placed along the toe of the toe berm adjacent to Shazah Creek and Chasm Creek. An LLDPE geomembrane liner (60 mil) will be placed on the upstream dam slopes and the base of the tailings impoundment. An emergency spillway will be built to pass storm events with greater than 200-year return periods, with a permanent spillway when the facility is closed. A typical TMF cross-section is shown on Figure 18-4.

Figure 18-4: Typical Tailings Management Facility Cross-section



The dam stability assessment is based on the conservative assumption that the foundation soils are potentially liquefiable, as suggested by the results of the site investigations. The design case is based on a composite strength in the foundation of 85% liquefiable, 15% non-liquefiable (but with the non-liquefiable strength reduced to account for pore pressure generation). The design was also checked for stability under static conditions. The size of the toe berm was selected to provide the design levels of stability (FS=1.5 under static conditions, and FS=1.2 under post-earthquake liquefied strengths).

The 60-mil LLDPE liner will be placed over the base of the tailings impoundment, extended up the upstream face of the dam, and keyed into the dam crest. The impoundment footprint and the dam will be graded and proof-rolled with a smooth drum roller to prepare a smooth surface, free of angular particles. It is likely that some bedding material (screened sand and fine gravel) will still be required.

18.11.5 Tailings Deposition & Water Management

Based on a total mine production of 5.4 Mt of ore over the life of mine, approximately 3.0 Mt of tailings will be deposited in the TMF. The average flow rate of tailings will be 850 t/d and the settled dry density of the tailings in the impoundment is expected to be 1.30 t/m^3. An ultimate dam crest elevation of 80.0 m would provide sufficient volume to store 3.0 Mt and provide freeboard for the closure spillway. A water pond will form at the southwest end of the TMF as soon as the liner is installed and this water will be used to float the pump barge and to provide water for start-up of the mill. Over time, spigotting will extend clockwise to drive the tailings pond to the north end of the TMF.

A monthly water balance was carried out for a typical year. The volume of the operational pond (required to provide for settling of tailings) varies between a minimum of 60,000 m^3 (in August) to a maximum of 260,000 m^3 (in March). The TMF will be operated as a zero-discharge system with all excess water recycled to the process plant, or sent to the water treatment plant for eventual discharge. The TMF is part of the site wide water management system and acts as an attenuation pond to manage seasonal variations in mine water. The TMF water balance will be reconciled annually with the site-wide water balance to confirm the design parameters.

The TMF is designed to store the 30-day precipitation (200-year return period) with 1.0 m freeboard to the spillway invert. The estimated 30-day, 200-year return period precipitation is approximately 770 mm, leading to an estimated volume of approximately 265,000 m^3. The spillway will pass the IDF, which is one-third between 1/1000 year and the PMF. The emergency spillway will be constructed at approximately 78.7 m elevation. The spillway is 4 m wide with 3H:1V downstream channel slope.

Since Shazah Creek and/or Chasm Creek channels may shift towards the TMF in the future, the toe of the TMF along the creeks will be armoured with riprap. Sufficient riprap is provided along the toe of the stabilizing berm to allow for potential scour below the existing ground level.

18.11.6 Closure Plan

The main areas of focus of the closure program will be erosion control, embankment stability, storm-water management, and revegetation. Establishing a surface cover of vegetation will reduce the potential for adverse environmental impacts such as erosion, as well as improving wildlife habitat and visual aesthetics. The TMF will occupy an area of approximately 45 ha, all of which will be reclaimed. Upon closure, the roads will be decommissioned and the dam crests will be recontoured to conform to the surrounding terrain and reduce the visual impact. Tailings deposits are not anticipated to require recontouring at closure. Selective spigotting will be used to infill any low areas and to maximize storage. The closure spillway will be constructed by lowering the elevation of the ultimate spillway from El. 78.7 m to approximately El. 76.5 m. A settling pond will be formed in the TMF during the reclamation stage to control potential sediment runoff, until vegetation is established on the tailings. The temporary settling area will be required on top of saturated tailings, at the inlet to the closure spillway. Additional rock armouring will be placed, if required, at the toe of the dam for closure to protect it from flood erosion due to extreme events in Shazah Creek and Chasm Creek.

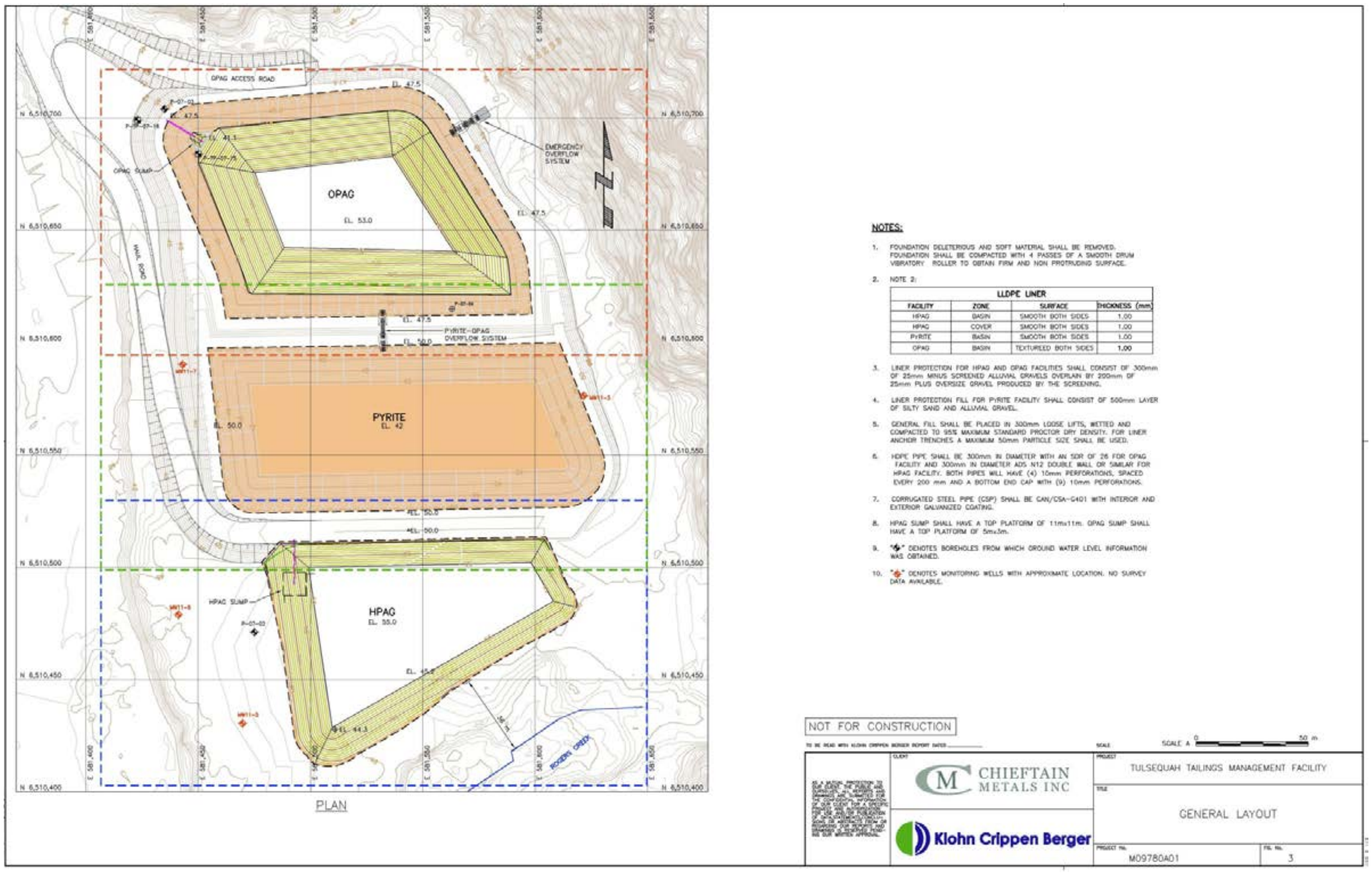
The TMF will be closed as a dry facility with no permanent pond. The top of the tailings will be graded towards the closure spillway constructed near the northwest corner of the TMF. The permanent closure spillway will direct runoff to Shazah Creek and will be constructed by lowering the operational spillway to approximately El. 76.5 m. A flow-through riprap berm will be constructed across the spillway outlet to minimize the potential for beavers to dam the spillway outlet. Several swales may be constructed on the surface of the tailings material to help to control rainfall erosion. The swales would direct water to the temporary settling area, where it would then discharge through the closure spillway. Until vegetation is established, a temporary settling area at the inlet of the closure spillway will help control potential sediment loads from surface water runoff on the TMF surface. The temporary settling area would be approximately 6 ha in size, located on top of the tailings, and would not be riprapped. The settling area will be revegetated when the closure spillway is lowered to El. 76.5 m.

18.12 PAG/Pyrite Facilities

18.12.1 Introduction

The historical potentially acid-generating (HPAG), operating potentially acid-generating (OPAG) and pyrite tailings storage facilities are located approximately 1 km south of the existing portals as shown on Figure 18-5. The area is a gently sloping floodplain, with a foundation of sand and gravel with occasional silty layers. These storage areas have been designed to store 140 kt of HPAG, 120 kt of OPAG and 75,000 tonnes of pyrite tailings. The design of these facilities targets a balance between cut-and-fill volumes. The design of these facilities was presented in more detail in the KCB report entitled, "Tulsequah Chief Mine Project PAG & Pyrite Facilities Design," (May, 2012).

Figure 18-5: PAG Facilities Arrangement



18.12.2 HPAG

The HPAG at the site will be stored in a lined storage facility until underground storage becomes available. Soil excavated from the footprint of the HPAG area will be used to construct embankments and ramps around the OPAG and pyrite tailings facilities.

HPAG will be placed into the excavation, which is to be lined with a 1.0 mm geomembrane (LLDPE) liner with a 0.5 m protective soil cover. The base of the excavation slopes at 2% to a sump located in the northwest corner. From the sump, leachate (which is mostly expected during fill placement and removal) will be pumped to the OPAG facility. When the HPAG has been placed to the design elevation of 55 m, the stockpile will be progressively covered with another LLDPE liner to minimize infiltration.

18.12.3 Pyrite Pond

Pyrite tailings slurry will be stored in a lined facility between the HPAG and OPAG areas until underground storage becomes available. As at the HPAG area, soil will be excavated from the base of the impoundment and used to construct perimeter embankments. The impoundment will be excavated to elevation 42 m, and the embankments will be built to a crest elevation of 50 m. The impoundment will be lined with a 1.0 mm LLDPE liner and 0.5 m protective soil cover will be placed over the liner in areas to be used as an access ramp. The pyrite tailings will be removed during operations and placed in the underground workings. Removal may be with slurry pumps or with a truck and shovel operation using the access ramp.

The pyrite tailings are to be covered by a 1.0 m water cap at all times, and the facility has been designed to store this cap with 0.8 m freeboard. During the environmental design flood (EDF), flood flows will discharge into the OPAG area through the spillway in the splitter dyke between the pyrite pond and the OPAG area.

18.12.4 OPAG

The OPAG facility will store 120,000 tonnes of operational PAG and up to 10,000 m^3 of contact water (seasonally). The impoundment will be excavated to elevation 42 m, and the embankments will be built to a crest elevation of 50 m. As at the HPAG and pyrite areas, the base impoundment and inside embankment slopes will be lined with a 1.0 mm LLDPE liner and 0.5 m cover.

The base of the OPAG area slopes at 1% to a sump in the northwest corner, where leachate will be pumped to the effluent treatment plant. The OPAG facility has been sized to store the EDF from the OPAG and pyrite areas with 0.9 m freeboard. The inflow design flood (IDF) will be discharged through an emergency spillway at the northeast corner of the facility. The IDF will reduce freeboard to 0.5 m in both the OPAG and pyrite areas.

18.12.5 Closure

At the end of mine operations all remaining OPAG, HPAG, pyrite tailings and any other contaminated fill will be removed from the storage facilities and stored underground. The embankments will be decommissioned and the basins filled, and the geomembrane liners will be disposed of appropriately. The disturbed areas will be recontoured, covered with topsoil and revegetated/re-seeded.

18.13 Waste Production Schedule

Table 18-6 (overleaf) shows the timing of the various types of waste that will be produced according to the current mine plan.

18.14 Concentrator Building

The concentrator building will be a purpose-designed multi-level building with a concrete footing and grade beam foundation. Overhead maintenance cranes will be incorporated into the design, along with required utility piping and HVAC systems loading. Process equipment and internal steel structures will be founded independently on concrete footings or thickened slabs placed on bedrock or engineered fill, as determined by the geotechnical engineers. Concrete floor slabs graded to catchment sumps will be provided throughout the processing areas.

18.15 Assay Laboratory

The assay laboratory will be a modular facility with a fire suppression system and a laboratory equipment package that will enable the laboratory staff to run the necessary tests and sample analyses to monitor the processing of the ore at various stages along the process plant’s sequence of processing steps.

18.16 Concentrate Storage

In normal production conditions, sufficient surge concentrate storage capacity is provided under the concentrate filters for trucking turnaround. To maintain production during unscheduled temporary access road closures, an additional one month of concentrate storage will be provided for the Zn, Cu and Pb concentrates. A roughly 24 m wide x 30 m long, engineered, fabric skinned, sprung building will be constructed adjacent to the southeast end of the processing plant facility, on a concrete slab foundation with 1 m high curb walls. Segregation of the Cu and Zn concentrates will be achieved using moveable concrete lock-blocks. The Pb concentrate will be loaded into sealed intermodal containers and stored in a safe location until they can be hauled to the port for shipping. The sprung fabric concentrate storage structure will be capable of storing up to 6,000 tonnes of Zn concentrate and 4,000 tonnes of Cu concentrate. It is estimated that up to 1,000 tonnes of Pb concentrate may need temporary storage and/or placement in the sealed intermodal containers if a 30-day access road closure were to occur during production.

Table 18-6: Waste Production Schedule

		2014	2015				2016	2017	2018	2019	2020	2021	2022	2023	2024	
		Q4	Q1	Q2	Q3	Q4	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
	Unit	92	90	91	92	92	366	365	365	365	366	365	365	365	366	3745
Ore Production	tonnes	-	-	-	-	50,401	556,697	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	**6,447,098**
Ore Production Rate	t/d	-	-	-	-	-	1,521	2,000	2,000	2,000	1,995	2,000	2,000	2,000	1,995	**1,946**
HPAG Placed Underground	t	-	-	-	-	-	-	-	-	-	24,000	24,000	24,000	24,000	24,000	**120,000**
OPAG Waste																
OPAG Produced	t	19,798	30,633	28,162	36,141	30,794	43,540	53,287	63,548	22,651	1,836	-	11,519	7,436	-	**349,345**
OPAG to Surface Waste Dump	t	19,798	30,633	23,162	25,613	20,794	-25,063	-30,369	-28,427	-36,141	-	-	-	-	-	**-**
OPAG to Backfill and Old Workings	t	-	-	5,000	10,528	10,000	68,603	83,656	91,975	58,792	1,836	-	11,519	7,436	-	**349,345**
NAG Waste																
NAG Produced	t	43,465	42,247	41,808	37,605	38,564	224,949	207,864	170,929	45,989	3,728	-	16,730	6,281	-	**880,159**
NAG to Surface Waste Dump	t	43,465	42,247	41,808	37,605	38,564	106,849	95,746	38,427	36,141	-	-	-	-	-	**480,852**
NAG to Backfill	t	-	-	-	-	-	118,100	112,118	132,502	9,847	3,728	-	16,730	6,281	-	**399,307**
Pyrite Concentrate																
Pyrite Concentrate Produced	t	-	-	-	-	-	156,098	204,692	204,692	204,692	204,692	204,692	204,692	204,692	204,692	**1,793,634**
Pyrite Concentrate to Surface Pyrite Surge Pond	t	-	-	-	-	-	19,630	-	-	-	-	-	-	-	-	**19,630**
Cumulative Pyrite Concentrate to Surface Surge Pond	t	-	-	-	-	-	19,630	-	-	-	-	-	-	-	-	**19,630**
Pyrite Concentrate Straight to Paste Backfill	t	-	-	-	-	-	136,468	204,692	204,692	204,692	204,692	204,692	204,692	204,692	204,692	**1,774,004**
Pyrite Concentrate from Surface Pyrite Surge Pond	t	-	-	-	-	-	-	19,630	-	-	-	-	-	-	-	**19,630**
Tailings																
Tailings Produced	t	-	-	-	-	-	310,303	406,902	406,902	406,902	406,902	406,902	406,902	406,902	406,902	**3,565,519**
Tailings to Tailings Pond	t	-	-	-	-	-	310,303	385,156	398,875	207,793	148,788	117,472	139,183	136,352	119,130	**1,963,052**
Tailings to Paste Backfill	t	-	-	-	-	-	-	21,746	8,027	199,109	258,114	289,430	267,719	270,550	287,772	**1,602,467**
Limestone to Tailings Pond	t	-	-	-	-	-	10,723	13,310	13,784	7,181	5,142	4,060	4,810	4,712	4,117	**67,838**
Total Tailings to Tailings Pond	t	-	-	-	-	-	321,026	398,466	412,659	214,973	153,930	121,532	143,993	141,064	123,246	**2,030,889**
Backfill																
OPAG	t	-	-	-	-	-	73,603	94,184	101,975	58,793	1,836	-	11,519	7,436	-	**349,345**
HPAG	t	-	-	-	-	-	-	-	-	-	24,000	24,000	24,000	24,000	24,000	**120,000**
NAG	t	-	-	-	-	-	118,100	112,118	132,502	9,847	3,728	-	16,730	6,281	-	**399,307**
Total Pyrite Concentrate to Paste Backfill	t	-	-	-	-	-	136,468	224,322	204,692	204,692	204,692	204,692	204,692	204,692	204,692	**1,793,634**
Tailings to Paste Backfill	t	-	-	-	-	-	-	21,746	8,027	199,109	258,114	289,430	267,719	270,550	287,772	**1,602,467**
Cement	t	-	-	-	-	-	4,094	7,382	7,790	12,114	13,884	14,824	14,172	14,257	14,774	**103,291**
Total Paste Backfill	**t**	**-**	**-**	**-**	**-**	**-**	**136,468**	**246,068**	**212,719**	**403,801**	**462,806**	**494,122**	**472,411**	**475,242**	**492,464**	**3,396,101**
Total Backfill	**t**	**-**	**-**	**-**	**-**	**-**	**328,171**	**452,371**	**447,196**	**472,441**	**468,370**	**494,122**	**500,659**	**488,959**	**492,464**	**4,144,753**

18.17 Maintenance Shops & Cold Storage Warehouse

The maintenance shops for the mine equipment fleet will be located on the lower bench of the mine site facilities near the 5200 portal. There will be two insulated shops for equipment repairs and one unheated storage warehouse. The buildings will be 60' x 80' pre-engineered structural steel frames with 26 gauge galvalume cladding exteriors, founded on a concrete floor slab. The shops will be insulated to R30 in the roof and R20 in the walls, the interior wall will be lined for the bottom 8 ft to protect the insulation from damage. Each building has one 14 ft x 14 ft overhead door, two mandoors and six windows.

Heating of the shops will be by diesel fired unit heaters, waste oil blending will be given consideration during detailed engineering.

No overhead cranes are planned for the shops. Mobile gantry cranes will be utilized for equipment hoisting requirements. One of the shops will be provided with an exhaust hood for a welding bay.

18.18 Mine Office & Dry

The mine office and dry will be housed in a two-storey modular building with the mine offices distributed around the top floor and the mine dry situated on the ground floor. The mine dry area will have a shift line-out or bull-pen area for the shift supervisors to line-up their crews prior to the start of shift. The mine dry will also have lockers, washroom and shower facilities so the employees can shower and change into clean cloths after their shift and change into their work clothes and gear prior to the start of their shift. The second floor office area will also have a small training area for times when mine or process plant crews need to go through specialized training.

18.19 Tulsequah Airstrip

The airstrip will be used for shuttling workers back and forth from Whitehorse, Yukon. Extension of this airstrip by approximately 50 m will provide sufficient length to enable the DHC-5 Buffalo aircraft to safely utilize the airstrip. Adding this aircraft to the handling capabilities of the airstrip will enable more efficient transport of personnel and supplies to and from site by air.

No major works have been undertaken to upgrade this airstrip to meet NavCan standards. Present operations are restricted to visual take-off and landing. The Buffalo aircraft will be the largest aircraft able to use the existing facility once the extension in length is completed.

Upgrades will include the addition of high-intensity strip lighting. Because of the location, dispensation may also be sought for relief from standard approach/take-off angles and turning radii for this airstrip.

18.20 Communication

Site communications will be achieved through a satellite-based site LAN and telephone system. This will provide internet, electronic/data communications and telephone connectivity for the site.

A leaky feeder communication system will be used as the communication system for mine and surface operations. Telephones will be located at key infrastructure locations in the mine. Key personnel such as mobile mechanics, crew leaders, shift supervisors and mobile equipment operators will be supplied with an underground radio for contact with the leaky feeder network.

18.21 Limestone Quarry

An on-site limestone quarry will provide limestone for the processing plant to raise the pH of the tailings material and control acid production potential of the tailings. The mining and stockpiling of limestone from this quarry will be conducted by a contractor on an as-needed basis. The contractor will mine the limestone, building a stockpile that can be used as needed for mixing with tailings materials before placement in the tailings facility. Blasted and stockpiled limestone will be transported via highway haul truck to the processing plant at a rate of up to 40 t/d by mine personnel.

18.22 Fire Protection

Fire protection will be provided through a site-wide firewater distribution system supplied from a 10 m diameter x 8 m tall combined fresh/fire water tank with a capacity of approximately 620,000 L. The firewater reserve level in the tank will contain sufficient volume to supply the largest calculated fire flow for the site for a minimum period of two hours. The distribution system will be provided with a dedicated firewater pump package that will be comprised of an electric jockey pump, an electric primary pump and a standby diesel driven pump. A mobile fire truck will also be provided for this site.

The accommodations complex will have an addressable fire alarm system that will communicate to a central control panel that will shut down the heating and ventilation fans and alert the entire complex in the case of a fire. Fire suppression in the complex will consist of hose reels and heat activated sprinkler heads throughout the facility and a stand-alone suppression system for the kitchen grease hoods.

The plant site will be protected by a closed loop distribution system that will feed yard hydrants and supply buildings with firewater for sprinkler systems, wall hydrants, and standpipes. The process plant and ancillary buildings will primarily be protected by standpipes and hose reels. Areas where firefighting with handheld fire hoses is impractical will be set up with sprinklers, such as conveyors located inside buildings and tunnels, oil storage areas, and elsewhere as required by code.

18.23 Incinerator

A 1 t/d batch incinerator for food waste and sewage sludge will be housed in its own pre-engineered structural steel building located south of the mine site near the PAG rock storage facility. The building dimensions will be approximately 15 m x 18 m. The sizing of the incinerator will be reviewed during detailed engineering to ensure it is correctly sized for this site. The current sizing is based on correspondence between Tetra-Tech and Eco-Waste Solutions who provided preliminary sizing and a budget. The batch incineration operation is estimated to take approximately 6 h/d and consume approximately 380 L of diesel fuel.

19 Market Studies & Contracts

19.1 Market Studies

19.1.1 Concentrate Sales

Preliminary market studies on the potential concentrate sales from the Tulsequah Chief project were completed by independent leading industry participants who have provided Chieftain with indicative terms and an analysis of the market conditions with respect to the concentrates to be produced. The participant names have been withheld for confidentiality, but the studies and indicative terms were reviewed and found to be acceptable by Qualified Person Doerksen. The terms have been confirmed with Chieftain.

A study on the Tulsequah Chief copper concentrate revealed that due to the elevated arsenic level, the concentrate could not be sold in China and suggests alternatives in India, Europe, or the Far East. The copper concentrate is also low in terms of copper content; however, the high levels of gold and silver compensate for this. In addition, the study suggests that the marketing of the copper concentrate should focus on spreading the material concentrates among a number of smelters with each smelter taking only a portion of the tonnage in any one year. It is recommended that rather than selling the concentrates in the spot market, Chieftain should market the concentrates on a long-term basis with fixed annual terms in order to avoid market demand fluctuations. A penalty for deleterious elements in the copper concentrate has been incorporated in the economic analysis based on estimated average values.

Smelter terms were identified for both zinc and lead concentrates and the terms are considered to be in line with the current market conditions and have been considered in the economic analysis.

Concentrate transportation will be conducted using trucks from the mine site to the Port of Skagway, Alaska where it will be loaded onto ships for transportation to overseas ports. Truck transportation costs were developed from contractor quotes while port and ocean freight charges were estimated from current experience. The Port of Skagway currently handles copper concentrates from the Minto mine and would be expanded to handle Tulsequah concentrate.

No contractual arrangements for concentrate trucking, port usage, shipping, smelting or refining exists at this time.

Tables 19-1 and 19-2 outline the concentrate transportation costs and smelter terms used in the economic analysis.

Table 19-1: Concentrate Treatment & Refining Cost Parameters Used in the Economic Analysis

Treatment Charges/Refining Charges (US$)		
Zinc Concentrate		
Zn Concentrate Treatment	$/dmt concentrate	145.00
Copper Concentrate		
Cu Concentrate Treatment	$/dmt concentrate	125.00
Cu Refining Charge	$/lb Cu payable	0.00
Ag Refining Charge	$/payable oz	1.50
Au Refining Charge	$/payable oz	25.00
Deleterious Elements Penalty	$/dmt	52.20
Lead Concentrate		
Pb Concentrate Treatment	$/dmt concentrate	100.00
Pb Refining Charge	$/payable oz	0.00
Ag Refining Charge	$/payable oz	1.50
Au Refining Charge	$/payable oz	25.00
Au Doré		
Au Refining Charge	$/payable oz	6.00
Ag Doré		
Ag Refining Charge	$/payable oz	1.50

Table 19-2: Concentrate Transport Cost Parameters Used in Economic Analysis (US$)

Zinc Concentrate		
Transport to Port	$/wmt	80.00
Container and Port charges	$/wmt	14.02
Ocean Freight	$/wmt	65.00
Building Capital – Port	$/wmt	23.65
Total	**$/wmt**	**182.68**
	$/dmt	**197.29**
Copper Concentrate		
Transport to Port	$/wmt	80.00
Container and Port charges	$/wmt	14.02
Ocean Freight	$/wmt	65.00
Building Capital - Port	$/wmt	23.65
Total	**$/wmt**	**182.68**
	$/dmt	**197.29**
Lead Concentrate		
Transport to Port	$/wmt	80.00
Container and Port charges	$/wmt	17.77
Ocean Freight	$/wmt	65.00
Building Capital - Port	$/wmt	23.65
Total	**$/wmt**	**186.43**
	$/dmt	**201.34**

19.1.2 Metal Prices

The base and precious metal markets are highly liquid and benefit from terminal markets around the world (London, Net York, Tokyo and Hong Kong) on an almost continuous basis. Historical metal prices are shown in Figures 19-1 to 19-5, which demonstrate the change in metal prices from 1998 to 2012.

Figure 19-1: Historical Gold Price (US$/oz)

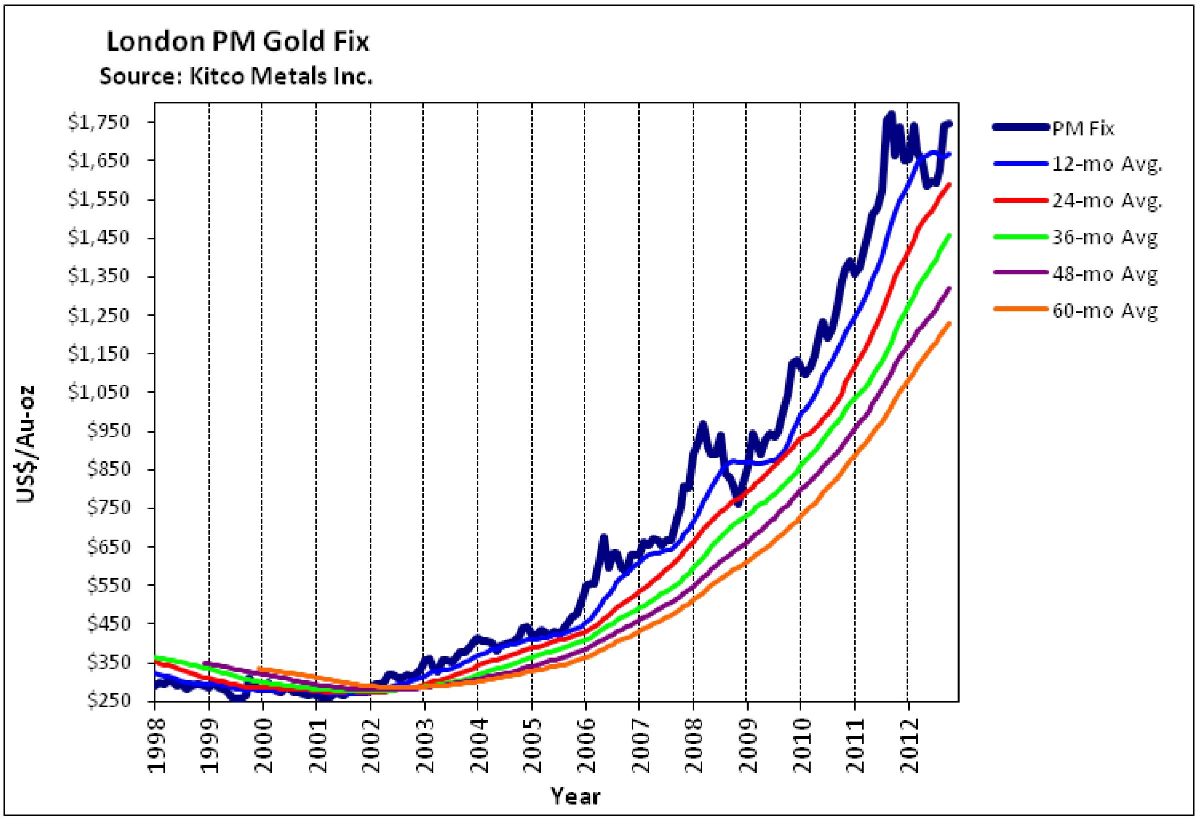


Figure 19-2: Historical Silver Price (US$/oz)

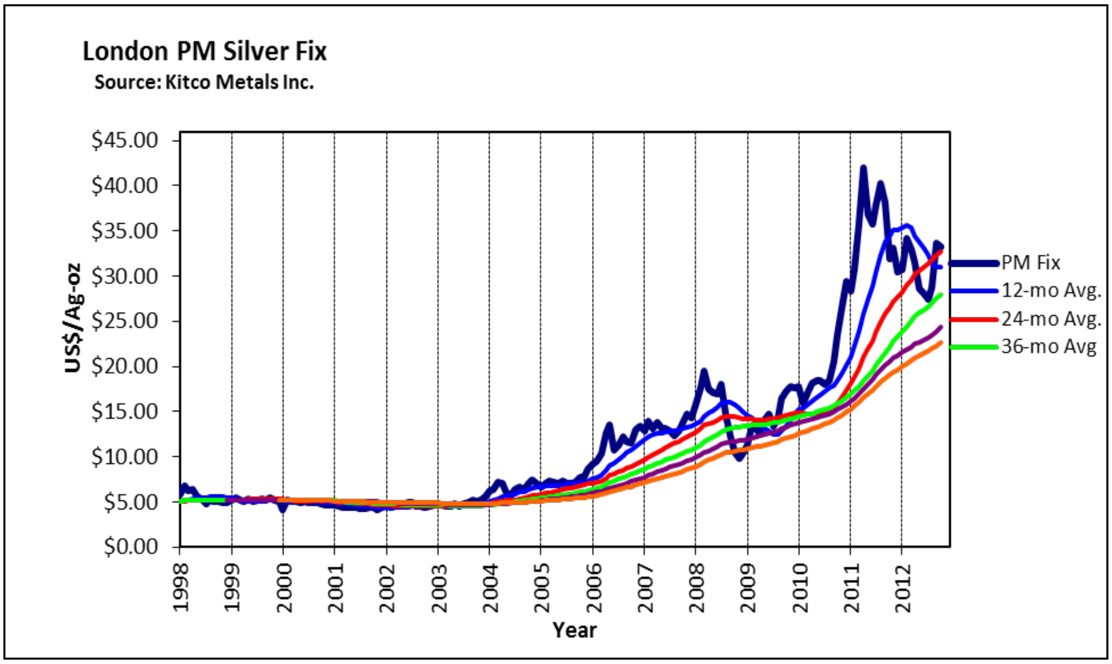


Figure 19-3: Historical Copper Prices

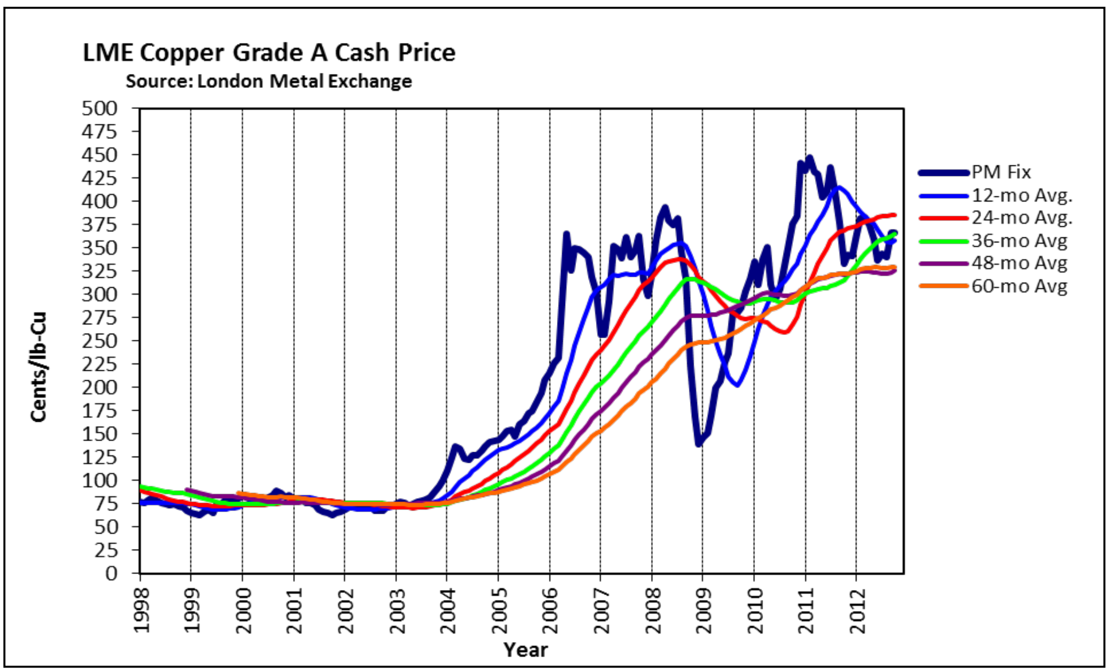


Figure 19-4: Historical Lead Prices

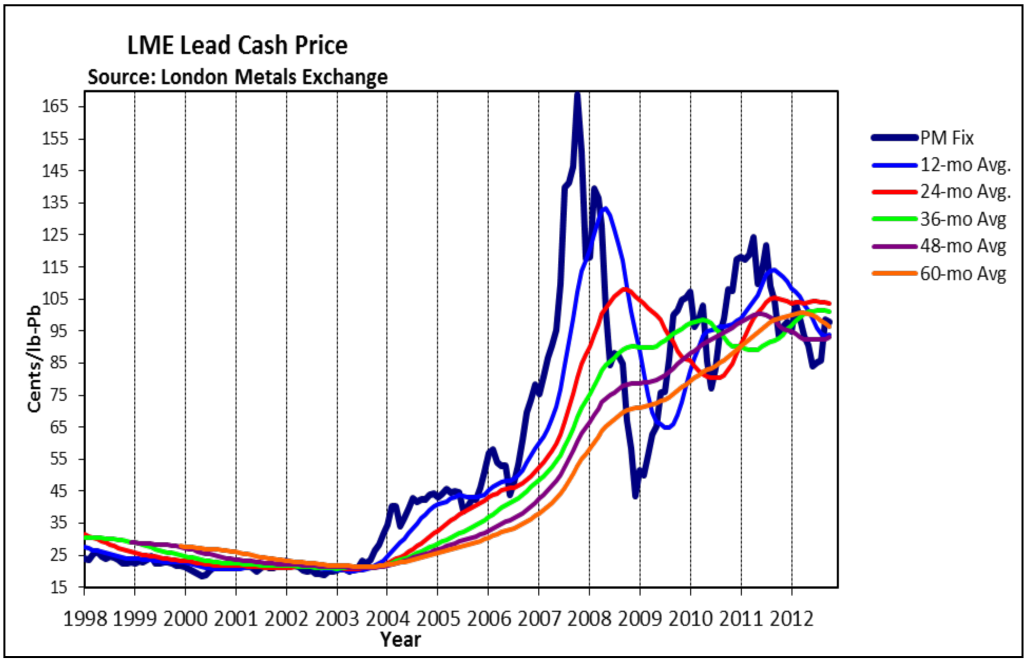
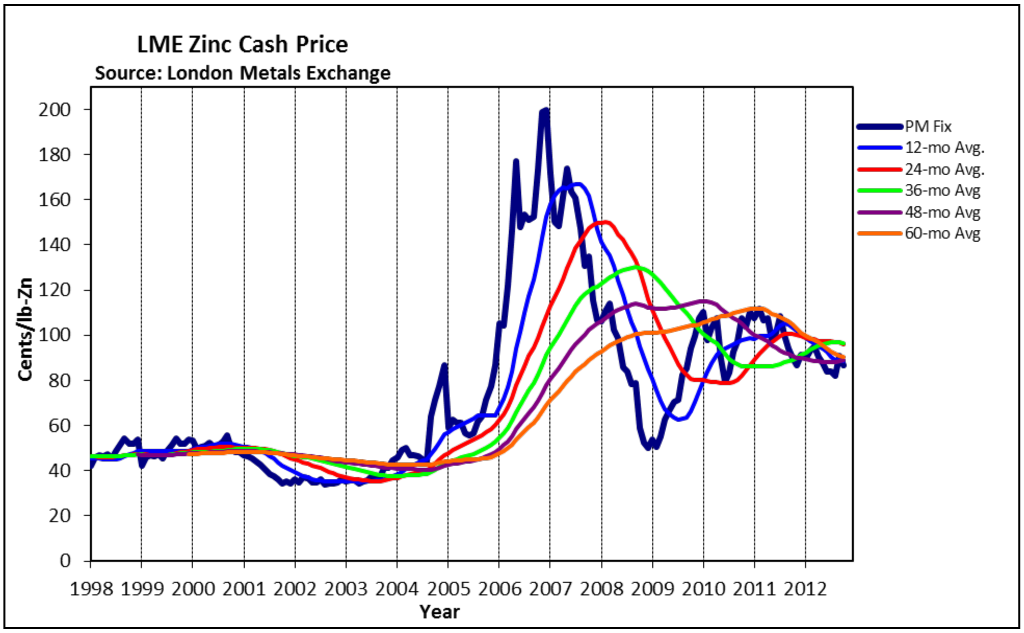


Figure 19-5: Historical Zinc Prices



The metal prices used in the Base Case economic analysis are the three-year rolling averages to October 31, 2012. Two additional cases were calculated, using the two-year rolling average (as of Oct. 31, 2012) and forecasted metal prices from Consensus Economics (Oct. 2012), an independent macroeconomic survey firm that prepares monthly compilations of metal prices using more than 30 analysts.

Table 19-3 summarizes the two- and three-year rolling averages of all three metals as at October 31, 2012 and lists the long-term forecast metal prices prepared by Consensus Economics during October 2012.

Table 19-3: Three-Year Rolling Average Prices for Gold, Silver, Copper, Zinc & Lead

Metal	Unit	Two-Year Rolling Average	Three-Year Rolling Average	Consensus Economics, Forecast Prices
Gold	US$/oz	1,592.00	1,455.00	1,338.00
Silver	US$/oz	33.00	28.00	22.00
Copper	US$/lb	3.85	3.66	2.95
Zinc	US$/lb	0.96	0.97	1.09
Lead	US$/lb	1.03	1.01	1.02

19.2 Contracts

19.2.1 Streaming Contract with Royal Gold

In December 2011, Chieftain entered into a gold and silver purchase transaction with Royal Gold to sell a portion of the precious metals expected to be produced at the Tulsequah Chief mine. Chieftain received a US$10 M up-front payment upon the signing of the contract, and will receive an additional US$50 M for the project build (upon certain conditions being met) that will be pro-rated during the development of the project.

The advance payments and future proceeds will allow Royal Gold to purchase, upon production of the Tulsequah Chief mine:

- 12.50% of payable gold at US$450/oz for the first 48 koz delivered, decreasing to 7.50% thereafter at US$500/oz
- 22.50% of payable silver at US$5.00/oz up to 2.8 Moz, decreasing to 9.75% thereafter at US$7.50/oz.

This contract has been included in the economic analysis of the project. Total silver and gold ounces expected to be sold to Royal Gold Inc. under this contract total 2.7 Moz and 49.5 koz, respectively.

19.2.2 MOU with China Engineering Co. Ltd. (CAMCE)

As at September 2012, Chieftain signed a non-binding memorandum of understanding with China CAMC Engineering Co. Ltd. (CAMCE) and its majority owned subsidiary Procon Holdings (Alberta) Inc. (Procon) for a comprehensive collaboration to build and operate Chieftain's Tulsequah Chief project. The principal features of the proposed collaboration are:

- CAMCE proposes to acquire a 30% interest in the project for a cash contribution equal to 30% of the project's net asset value. Chieftain will own the remaining 70%.
- Chieftain will enter into an engineering, procurement and construction contract with CAMCE to develop the project on market terms and upon completion of development, an underground mining contract with Procon renewable for successive three-year terms for the "life of mine."
- CAMCE will seek to arrange senior long-term debt for the development of the project from a Chinese financial institution for 70% of the agreed upon project development costs (Senior Debt Loan).
- The project entity will enter into long-term, arm's length off-take commitments with CAMCE for 30% of the zinc, copper and lead concentrates produced by the project.

20 Environmental Studies, Permitting & Social/Community

20.1 Environmental Issues

The Tulsequah Chief project is located at a historical brownfields site with visible acidic mine drainage (AMD). Potential historic environmental liabilities include the PAG waste rock piles located on surface outside the entrances to the 5200, 5400, 5900, 6400, and 6500 portals, as well as the AMD from the underground workings.

There is a plan in place, which has been permitted by the Ministry of Energy and Mines, to clean up the historical waste rock at the 5200 and 5400 portals. This plan, as presented, is dependent on project development, since the intention is to dispose of the historical waste rock in underground workings that will be flooded during mine closure. At present, there are no management plans for the PAG waste rock located at the 5900, 6400 and 6500 level portal openings. There is a possibility that this small volume of waste rock may eventually need to be addressed.

The acidic mine drainage at the Tulsequah Chief Site had been subject to an Environment Canada Directive. In response, Chieftain installed and commissioned an acidic water treatment plant (ATP) in late 2011. Through last winter, most of the acidic underground drainage was directed to the ATP and successfully treated. Treated effluent is discharged under a Waste Discharge Authorization issued by the BC Ministry of Environment under the *Environmental Management Act* (EMA). The operation of the treatment plant was suspended on June 23, 2012 and the plant remains on care and maintenance, in contravention of the *Fisheries Act* and the EMA permit.

The long-term solution for managing AMD is to backfill the historic stopes early during mine operations to stop the acidic underground flow by mine closure. If this mitigation strategy is unsuccessful, there could be the need for the long-term treatment of AMD at this site.

20.2 Waste Management Plan

20.2.1 PAG Waste

The mine plan for the Tulsequah Chief site has all PAG waste material being returned underground throughout the mine life for disposal in the underground working, which will eventually be flooded at mine closure. Section 18.12 provides a detailed explanation of the PAG waste rock management system for operations.

20.2.2 Tailings

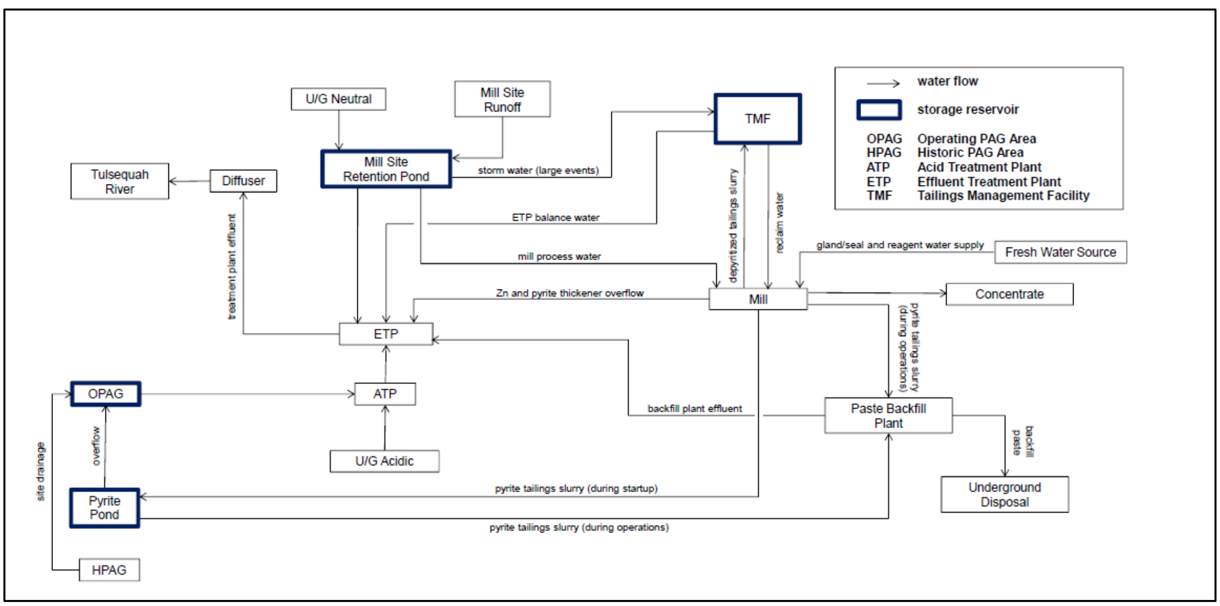
All tailings from the Tulsequah Chief mine that are not required for backfill will be desulphidized and then co-disposed with limestone in the TMF to provide an adequate NPR that will mitigate the possible future generation of AMD. For a detailed discussion of tailings disposal, refer to Section 18.11. The pyrite concentrate produced in the desulphidization process will be deposited in mine voids as backfill, below the ultimate groundwater table, preventing future AMD.

20.3 Water Management Plan / Water Balance

A site-wide water balance model (WBM) was developed to simulate the operation of the proposed water management system under the processing demands, and environmental conditions that can be expected to occur at the mine. The estimated rates of pumping, milling, and waste generation were combined with site precipitation, runoff, evaporation, and system storage capacity to optimize the operating logic of the proposed water management system. The model was designed to simulate water flows at the site for eight years of production, as well as a one-year start-up and a one-year closure period.

The WBM is illustrated schematically in Figure 20-1. This represents site-wide flow paths within the proposed project. The WBM was developed based on the inflows and outflows between the various mine system components. The WBM preserves the intention of the process flow diagram (PFD) developed by Tetra Tech as much as possible.

Figure 20-1: Water Balance Schematic



The following sections provide an overview of the site wide water balance. Details of the water balance model are included in the Tulsequah Chief Operations Water Balance report, prepared by Marsland Environmental Associates (MEA, 2012a).

20.3.1 Water Balance Modeling Platform

A robust Monte-Carlo Simulation Program, Goldsim, was used to set-up the WBM. It divided the balance up into five main containers and each container was broken down into sub-containers, defining key inputs to the water balance system. Below is a summary of the key components in the balance.

20.3.1.1 Weather Dataset

The dataset comprised of 59 years' worth of precipitation, temperature and snowmelt data collected at the Juneau Airport weather station from 1950 to 2008. The Juneau data was then adjusted to reflect the inland conditions experienced at the Tulsequah mine site. Fifty realizations were then created to test the WBM using real, historic data, providing a realistic approximation of environmental conditions that can be expected at the site.

20.3.1.2 Underground Workings

The Tulsequah Chief mine was operated by Cominco from 1951 to 1957, during which time five portals were developed 5200, 5400, 5900, 6400 and 6500. Water enters the old underground workings and discharges out 5200 and 5400 portals as an acidic, metal laden drainage.

20.3.1.3 Tailings Management Facility

The TMF is a proposed structure to permanently store the de-pyritized tailings from the mill. It will be operated as a zero discharge system with all excess water recycled to the process plant prior to discharge in the receiving environment. The TMF will also act as an attenuation pond to balance seasonal and operational variations in the site water balance.

20.3.2 PAG Waste Storage Facility (HPAG/OPAG/Pyrite Pond)

Three surface facilities have been designed for temporary storage of PAG waste: existing PAG rock from historic mining activities will be stored in the HPAG facility; PAG rock created during operations will be stored in the OPAG facility; and pyrite tailings during initial operations will be stored in the pyrite pond. All three will ultimately see final disposal underground. During operations, the drainage from all three facilities will be collected and directed through the water treatment system.

Non-acid generating rock will be stored in a waste rock dump. Dump drainage does not require storage and subsequent treatment. As such, it has not been incorporated into the proposed water management system.

20.3.3 Process Facilities

The process facilities refer to the following infrastructure: the mill, the paste backfill plant and the site retention pond. The ore will be milled to produce concentrate. The resultant two main tailings streams will be the pyrite tailings (PAG) and de-pyritized tailings (NAG). The retention pond collects mill site runoff and underground neutral water so that it can be used in the mill, or be treated by the ETP and discharged in the receiving environment. The paste backfill plant will fill the historic upper workings with de-pyritized fill in year one and the lower workings with pyritic fill thereafter.

20.3.4 Acid Treatment Plant & Effluent Treatment Plant

The water treatment system is comprised of two treatment facilities: the acid treatment plant (ATP) and the effluent treatment plant (ETP). The ATP will treat the acidic discharge from the underground and the drainage from the PAG Waste Storage Facility. The ETP will treat all other mine site effluents including the ATP effluent, prior to discharging in the receiving environment through a diffuser buried below the scour depth in the Tulsequah River flood plain.

20.3.5 Contingency Measures

The mine water management system will undergo intermittent events when there is a deficit or excess in water supply created by mine system malfunctions or wet/dry environmental conditions. Various contingency measures have been developed by Chieftain with the goal of minimizing the likelihood of disruptions to production, or uncontrolled discharges to the environment of water impacted by the mine. Contingency measures are summarized in Table 20-1.

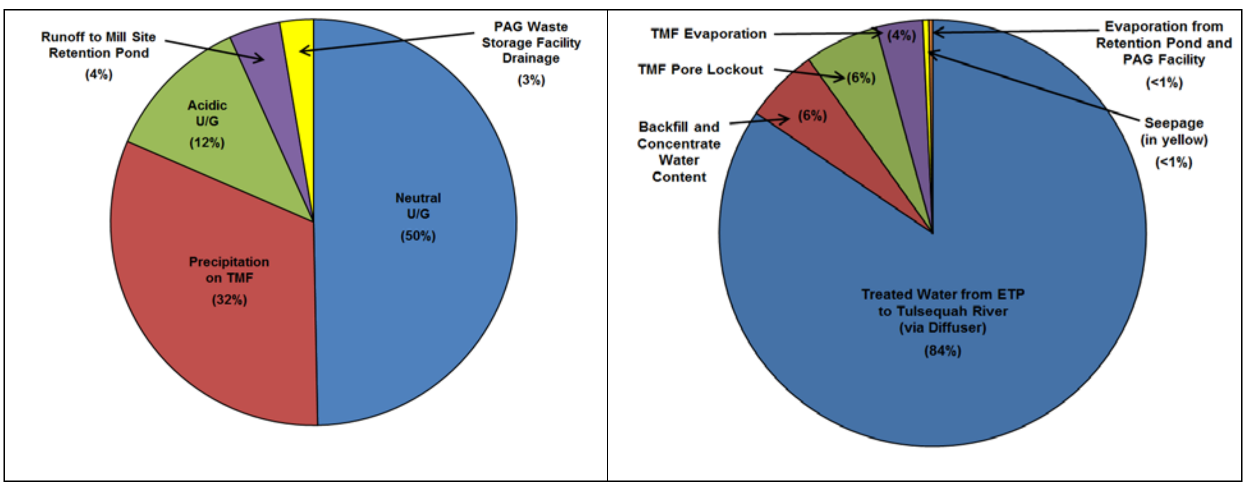
20.3.6 Site-wide Inflows & Outflows

The water balance model was run and a summary of the site-wide inflows and outflows to the water management system over the operating period are summarized in Figure 20-2.

Table 20-1: Contingency Measures for Water Management during Intermittent Events

Mine Facility Component	Base Case Plan	Possible Intermittent Event	Contingency Measure
TMF (Mill makeup water supply)	Water reclaimed from TMF to Mill.	Shortage of supply of water from the TMF to the Mill	Use neutral drainage from the underground workings as Mill makeup water.
TMF (storage of surplus mine site water)	TMF stores excess mine site effluents during wet conditions.	Excess volume in the TMF due to wet environmental conditions.	Treat surplus TMF water in the ETP and discharge to Tulsequah River (via diffuser)
Mill Site Runoff	Conveyed to Mill as makeup water.	Excess storm water inflow exceeds the storage capacity of the retention pond.	Pump excess volume from the retention pond to the TMF for storage.
ETP Plant	Treat site effluents and discharge to Tulsequah River (via diffuser)	ETP Shutdown or failure.	ETP inflows are routed to the TMF for storage until the ETP is back online.

Figure 20-2: Water Balance Inflows & Outflows



20.3.7 Water Balance Model Sensitivity Analysis

The WBM includes inputs derived from assumed or average operating conditions, and those values can be expected to vary over the duration of operations. The sensitivity of these WBM inputs was assessed by running the model using a range of input values as summarized below.

20.3.7.1 Environmental Conditions

The sensitivity of the model was tested using 50 unique weather scenarios. The development of the WBM was iterative; with each change, the WBM was run with all 50 realizations. Parameters were constantly adjusted to ensure that no uncontrolled releases occurred during the design process.

20.3.7.2 Evaporation Rates

Historical evaporation data was not available so average monthly estimates were applied in the 10-year operational period for the WBM. The model tested the uncertainty associated with the evaporation input by adjusting the values by ±20%. All 50 realizations were run and the WBM results showed that the proposed system components were able to meet mine water demands and uncontrolled releases were prevented.

20.3.7.3 Underground Mine Inflow

The underground mine flow at 110 m^3/h was adjusted by +100/-20% to test the WBM against variation in the underground flow. Since underground water normally reports to the ETP, it and the discharge piping were sized to handle the potential additional increase in flow.

20.3.7.4 Processing Variation

The ore processing base case, 2000 t/d, and parameters directly influenced by the processing rate were adjusted by ±20%. Sensitivity analysis showed that the variation in the processing rates applied over the entire period of operations would not affect the overall functionality of the proposed system. That is, storage, pumping and treatment rates were able to meet system demands, given the variation in model inputs associated with mill processing rates.

20.3.8 Results Summary

The WBM was developed to represent the proposed site-wide water management system and was run for a realistic range of operational and environmental conditions to assess the performance of the proposed system and to develop a set of procedures to be followed during operations.

- The neutral drainage from underground is the largest source of water to the site-wide water management system (50% of total), followed by precipitation on the TMF (32%), acidic underground drainage (12%) and drainage from the PAG waste storage facility and mill site (7%).
- More water is expected to enter the system than is expected to be lost to processing or the surrounding environment ensuring a sufficient supply of water throughout operations. During

dry periods when the TMF supply to the mill is insufficient, additional water can be diverted from the neutral underground discharge.

- The mill make-up return will be comprised from the TMF and neutral underground water. If water quality is acceptable, the treated effluent from the ETP will supply the mill with water for the gland, seal and reagent water. If the water quality is insufficient, it will be necessary to draw on freshwater sources.
- Uncontrolled releases did not occur from the retention pond, the PAG waste storage facility or the TMF during any of the 50 model realizations.
- The sensitivity analysis showed that the WBM was able to handle variations of at least ±20% in the key parameters: precipitation data, evaporation rates, neutral underground flow and mill processing rates. In all of these cases, the proposed system showed no uncontrolled releases and was able to meet all storage, pumping and treatment demands.

Overall, the WBM is a robust system able to meet the dynamic conditions that may be experienced during operations.

20.4 Mine Closure Plan

The project is expected to result in a total disturbance at end of mine life of approximately 162 ha. The existing area of disturbance at the site is approximately 106 ha. Remaining on surface at mine closure will be a TMF containing non-acid generating tailings, a non-acid generating waste rock storage facility and a demolition debris landfill associated with the waste rock dump.

Chieftain's closure goal is to return the site to as near to original pre-mining conditions as practical so that the site does not require ongoing control and maintenance, and the environment is not impacted following mine closure. This will be achieved through decommissioning mining, milling and related facilities, and reclaiming lands and watercourses disturbed by the project.

Chieftain will undertake, where possible, progressive reclamation activities during mine operations. For example, by the end of mine life, all PAG material brought to surface during operations will have been progressively backfilled into underground workings that will subsequently become flooded upon mine closure. Additionally, the historic workings will have been backfilled during mine operations with paste backfill. Therefore, at mine closure, the remaining decommissioning and reclamation closure work to be completed will consist of securing the mine portals, dismantling and removing mine infrastructure and fixtures, disposing of hazardous and non-hazardous wastes and obsolete equipment, restoring drainage patterns and shaping the land to approximate original contours, where possible. Recontouring will shape the ground to facilitate natural drainage patterns before being covered with a suitable growth medium that will be seeded to prevent erosion and encourage reestablishment of native species.

Figures 20-3 and 20-4 summarize the closure concepts. Details are provided in the Tulsequah Chief Mine Closure Plan (MEA, 2012b).

Figure 20-3: Reclamation Plan – South

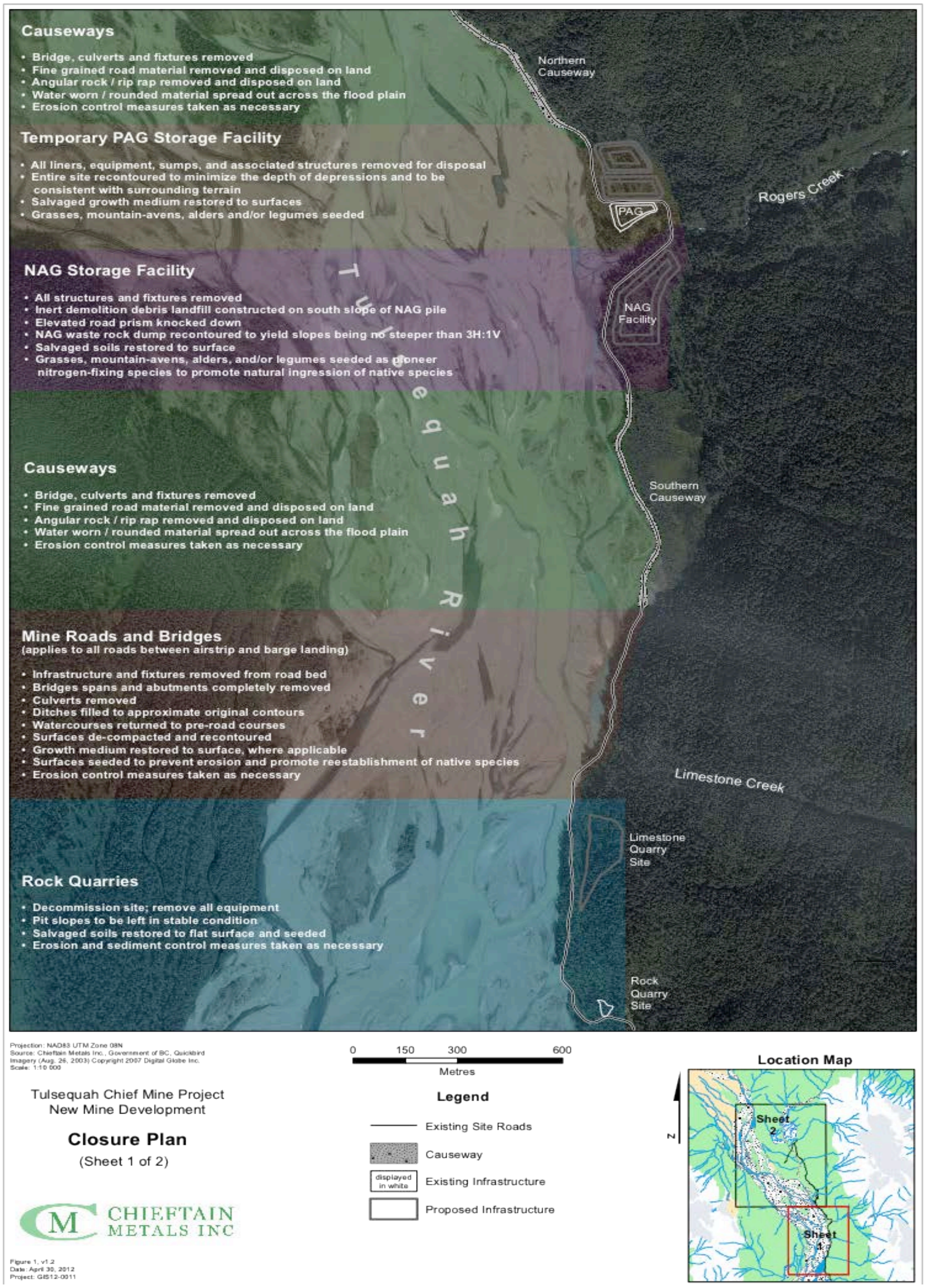
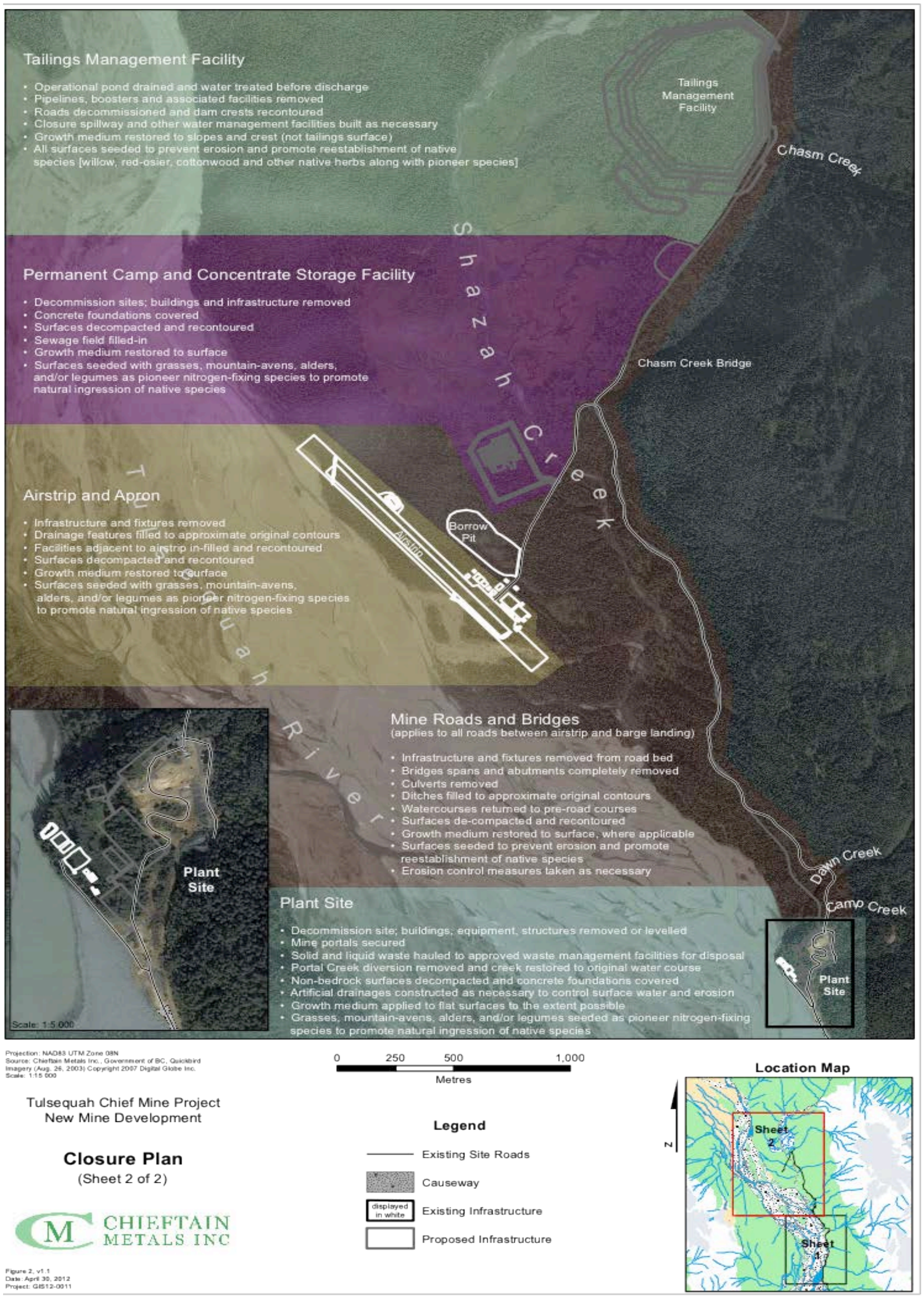


Figure 20-4: Reclamation Plan – North



The costs associated with closure implementation, which form the basis for the reclamation bonding discussed in Section 20.8, are provided in the capital cost estimate chapter of the feasibility study.

20.5 Access Road Deactivation

The proposed access road to the Tulsequah Chief Mine site from Atlin will be 119 km long and include 20 bridge crossings of either timber or concrete deck construction, three major culverts and eight fish passage culverts. A deactivation plan, and associated costing, was prepared by SNT Engineering Ltd. and submitted with the Special Used Permit (SUP) amendment application. Bonding for the decommissioning of the SUP will be levied under the Ministry of Forests, Lands, and Natural Resource Operations, independent of the *Mines Act* permit bonding.

The objective of the deactivation plan for the access road will be to leave the road in a self-maintaining state where natural surface drainage patterns are established and the impact of sediment transport is minimized, ensuring adjacent resources are protected indefinitely. This will be achieved through the removal of bridges and stream culverts, stabilizing the road prism and barricading the road in identified locations in a clearly visible manner to restrict motorized vehicle access.

Specific proposed deactivation activities include removing bridges, removing all stream culverts, re establishing stream banks (with armour) where disturbed, removing cross drain culverts where required (in moderate or high consequence areas), removing/breaching windrows, pulling back unstable fills, installing cross ditches, intercepting road surface and ditch line water and conveying it across the road to stable, non-erodible slopes. The road surface will be decompacted for the establishment of vegetation and scarified for the establishment of woody species in areas required either for visuals or to restrict access. Stockpiled salvaged soils will also be replaced. Gully restoration will occur as required to prevent landslides. General deactivation requirements are laid out in section 82 of the Forest Planning and Practices Regulation.

Motor vehicle access will be restricted by establishing full fill pullback and recontouring at five identified locations. These sites were selected where fill pullback would make it very difficult for off road vehicles to bypass thereby controlling access to parks and other sensitive areas along the road right of way.

20.6 Permitting

The Tulsequah Chief project was issued a provincial Environmental Assessment Certificate M02-01 and a *Canadian Environmental Assessment Act* screening approval. A condition of the certificate is that the proponent must have substantially started the project by December 12, 2012; otherwise, the certificate expires and is no longer valid. On May 30, 2012, the Associate Deputy Minister of the BC Environmental Assessment office determined that the project has been substantially started The Environmental Assessment certificate thus remains effective for the life of the project.

Chieftain has secured all necessary permits to commence construction at the mine site, and on October 19, 2012, received Amendment #5 to Environmental Assessment Certificate 02-01, approving the re-routing of the mine access road. All road construction permit application documentation has been submitted to the BC government for review, and the company anticipates a decision in early course.

Several permits related to the construction of the Tulsequah Chief project were issued to the previous owner and have since been transferred to Chieftain. Tables 20-2 and 20-3 provide a detailed listing of these permits, licenses and authorizations, along with those permit, license and authorization applications that are currently under review or remain to be submitted. The tables use the following colour legend:

Permit Issued	**Permit Pending**	**Permit to be applied for at a later date**

20.7 Monitoring & Surveillance Plan

A detailed Monitoring and Surveillance Plan was developed by Redfern Resources to support the *Mines Act* permit amendment application for full operations (Environmental Monitoring and Surveillance Plan, December 18, 2008). Chieftain has a scaled back version of this plan in place to address the current water treatment operations, supporting the existing EMA Discharge Permit. This plan will be updated to address the planned and permitted preconstruction relocation of the historic PAG rock, and initial site development. The updated plan will draw heavily on the original version prepared by Redfern. This prior plan is available for review. The current plan was most recently issued in April 2012 (Chieftain, 2012).

Table 20-2: Status of Project Permits, Licenses & Authorizations Required During Construction

	Permit	Permit Number	Issuing Authority	Description	Status/Issue date	Comments
1	BCEAA Environmental Assessment Approval	M02-01	BC Environmental Assessment Office	Overall project Environmental Approval to proceed	Valid for life of project	Transferred to Chieftain on November 1, 2010 Project deemed substantially started on 30 May 2012; certificate valid for life of mine.
2	BCEAA Environmental Assessment Approval Amendment #1	Amendment #1 to EAC M02-01	BC Environmental Assessment Office	Amendment to extend period of EA Certificate for a further 5 years	20-Sep-07	Transferred to Chieftain on November 1, 2010
3	BCEAA Environmental Assessment Approval Amendment #2	Amendment #2 to EAC M02-01	BC Environmental Assessment Office	Amendment to allow changes to project design	20-Sep-07	Transferred to Chieftain on November 1, 2010
4	BCEAA Environmental Assessment Approval Amendment #3	Amendment #3 to EAC M02-01	BC Environmental Assessment Office	Amendment to allow Barge access for project	26-Feb-09	Transferred to Chieftain on November 1, 2010. No longer required
5	BCEAA Environmental Assessment Approval Amendment #4	Amendment #4 to EAC M02-01	BC Environmental Assessment Office	Amendment to transfer EA Certificate to Chieftain Metals	01-Nov-10	
6	BCEAA Environmental Assessment Approval Amendment #5	Amendment #5 to EAC M02-01	BC Environmental Assessment Office	Amendment to realign project access road	19-Oct-12	
7	CEAA Screening Environmental Assessment Approval	36077	Canadian Environmental Assessment Agency	Overall project Environmental Approval to proceed	05-Jul-05	Already in Place
8	Special Use Permit	S23154	BC Ministry of Forests, Lands and Natural Resource Operations	Permits all-weather access road from Atlin Public Highway to Tulsequah Mine site	21-May-99	Assignment to Chieftain completed 17 February 2012; Revision nearing completion; expected February 2013
9	Occupant License to Cut and amendments	L47498, Amendments #1-5	BC Ministry of Forests, Lands and Natural Resource Operations	Required for removal of timber from construction areas	December 6, 2007.	Amendment #5 for new road alignment nearing completion; expected February 2013
10	Parks Use Permit	N/A	BC Parks	Permits all-weather access road through Nakina-Inklin Reserve	Pending	Nearing completion; expected February 2013
11	Roadworks Permit	N/A	BC Ministry of Transportation and Infrastructure	Permit to undertake roadworks on unmaintained BC MoTI right of way for mine access	Pending	Nearing completion; expected February 2013
12	Intersection Permit	N/A	BC Ministry of Transportation and Infrastructure	Permit for mine access road to intersect with existing BC road network	Pending	Nearing completion; expected February 2013
13	*Navigable Waters Protection Act* Approval	8200-99-8393	Transport Canada	Approval of final permanent bridge design for Shazah Creek Crossing	Exp. Dec 31, 2012	Extension of previous permit, needs to be extended again
14	*Navigable Waters Protection Act* Approval	8200-04-8669	Transport Canada	Approval of final permanent bridge design for Rogers Creek Crossing	Exp. Sept 30, 2012	Extension of previous permit, needs to be extended again
15	North Causeways Fisheries Authorization	5300-10-005-#2	Fisheries and Oceans Canada	Authorize construction of north causeway on Tulsequah River floodplain	04-Jul-08	Received Feb 2011
16	South Causeways Fisheries Authorization	5300-10-005-#4	Fisheries and Oceans Canada	Authorize construction of south causeway on Tulsequah River floodplain	24-Oct-08	Received February 2011
17	CEAA Screening Environmental Assessment Approval		Canadian Environmental Assessment Agency	Amendment to allow Barge access for project	Pending	Finalized, ready to be issued on request and determination that barging will be required. Still awaiting clarification from DFO re: necessity for construction barging campaigns.
18	*Mines Act* Permit Initial (MA1)	M232-1	BC Ministry of Mining, Energy and Natural Gas	All surface roads and infrastructure development	08-Feb-08	Received February 2011
19	*Mines Act* Permit Amendment (MA2)	M232	BC Ministry of Mining, Energy and Natural Gas	Inclusion of Waste storage, plant site surface development, underground preparatory work (slash adits)	14-Nov-08	Received February 2011
20	*Mines Act* Permit Amendment	M232	BC Ministry of Mining, Energy and Natural Gas	Approving Acid Water Treatment Plant	07-Jul-11	
21	*Mines Act* Permit Amendment	M232	BC Ministry of Mining, Energy and Natural Gas	Approving road bridge and camp construction activities	07-Jun-12	
22	*Mines Act* Permit Amendment (MA3)	M232	BC Ministry of Mining, Energy and Natural Gas	Inclusion of Tailings Impoundment and all underground development	Planned	Re-submission planned to ensure issuance prior to commencement of operations
23	MX-2 Permit - Full release	MX-2	BC Ministry of Mining, Energy and Natural Gas	Release of Exploration road permit to coverage under *Mines Act* Permit	Completed	Received February 2011
24	Discharge Diffuser Authorization		Fisheries and Oceans Canada	Installation of buried diffuser pipe in Tulsequah River floodplain		Required. Submission planned to ensure issuance prior to operations
25	Airstrip Extension Authorization		Fisheries and Oceans Canada	Extension of airstrip 150 m		Re-submission planned upon completion of detailed engineering to ensure issuance prior to construction
26	Stream 2 diversion (PAG waste site)		Fisheries and Oceans Canada	Divert intermittent stream from construction site		Documentation prepared; re-submission planned upon detailed engineering to ensure issuance prior to construction
27	Barge Landing Authorization		Fisheries and Oceans Canada	Barge landing fisheries habitat alteration		Documentation prepared; submission planned upon detailed engineering to ensure issuance prior to construction barging campaigns.
28	Interim Waste Discharge Approval	#105719	BC Ministry of Environment	Authorization for discharges during construction period	01-Apr-12	Original Expired. New permit application submitted in May 2011, approval received April 2012

Table 20-3: Status of Project Permits, Licenses & Authorizations Required During Operation

	Permit	Permit Number	Issuing Authority	Description	Status/Issue Date	Comments
29	Operations Discharge Approval		BC Ministry of Environment	Operations Discharge approvals		To be applied for pending finalization of site water balance and operating parameters
30	Water License Approval, Portal Creek Diversion	A600966	BC Ministry of Environment	Diversion of Portal Creek	13-Jun-08	
31	Water License, Portal Creek diversion	C126606	BC Ministry of Environment	Diversion of intermittent creek	12-Jul-11	
32	Conditional Water License	C120434	BC Ministry of Environment	Diversion of Camp Creek and potable water extraction	16-Feb-05	Transferred to Chieftain April 28, 2011
33	Other Water Licenses (Tulsequah/Dawn creek)		BC Ministry of Environment	Water Usage licenses		Pending finalization of use volumes. Submission planned to ensure issuance prior to operations
34	Water license conversion of Section 9 approval North causeway	A600968	BC Ministry of Environment	Removal of large woody debris, alteration of floodplain, construction of two bridges	Work Complete	Documentation process only.
35	Water license conversion of Section 9 approval South causeway	A600977	BC Ministry of Environment	Removal of large woody debris, riparian vegetation, construction of two bridges and six culverts	Work Complete	Documentation process only.
36	Section 9 approval diffuser		BC Ministry of Environment			Submission planned to ensure issuance prior to operations
37	Section 9 approval airstrip extension		BC Ministry of Environment			Submission planned to ensure issuance prior to operations
38	Section 9 approval barge landing		BC Ministry of Environment			Submission planned to ensure issuance prior to operations

Legend:

Permit Issued	Permit Pending	Permit to be applied for at a later date

20.8 Financial Securities

Chieftain has posted required securities totalling $2,022,000 as follows:

- Under *Mines Act* permit number MX-1-355, a reclamation security in the amount of $50,000, for reclamation costs associated with mineral exploration activities conducted outside the area covered by *Mines Act* permit M-232
- Under *Mines Act* permit M-232, a reclamation security in the amount of $1,200,000, for reclamation costs associated with the works permitted under M-232 as of July 2011; and
- Under *Fisheries Act* Authorization # 5300-10-005, a letter of credit in the sum of $772,000, for costs to decommission the causeways and complete construction of the compensatory fish habitat compensation works which are tied to the authorization.

Additional financial security under the *Mines Act* will be payable as more activates related to mine development and construction are advanced and permitted. Additional payments of $200,000 for local roads and construction camp and $1,200,000 for underground development are already permitted but not paid. It is further anticipated, based on correspondence between the Ministry of Energy and Mines and Redfern Resources, that there will be incremental payments beginning on or before the commencement of underground development and reaching completion on or before four years of mill operations. The timing and quantification of these payments has not yet been established for Chieftain. The detailed estimate of the reclamation bond requirements is provided in the Capital Cost estimate section of the Feasibility. It is estimated that an additional $11,500,000 in security will likely need to be posted over the approximately five-year period. In addition, a security before commencement of construction of the access road will also be payable. This amount is estimated at approximately $5,000,000.

20.9 Social & Community

20.9.1 Requirements & Plans

The project site is in a remote area with limited land uses consisting of past mining activities, hunting and trapping. A small amount of logging activity occurred in conjunction with past mining. Mining has occurred on two deposits located on the property and on a former producing gold deposit on the west side of the Tulsequah River. Downstream of the project, commercial and subsistence fisheries are active in the May to October period each year in the Taku River.

The Company has undertaken an extensive community consultation program and provided numerous opportunities for stakeholders to gather information and comment on the project. A Consultation Report was prepared as part of the Environmental Assessment Amendment process and the consultation program has been deemed acceptable and approved by the Provincial Government.

20.9.2 Status of Negotiations & Agreements

The Tulsequah Chief Mine lies within the traditional lands of the Taku River Tlingit First Nation (TRTFN) and falls under the jurisdiction of the Atlin Taku Land Use Plan (LUP). The Atlin Taku LUP has been ratified by the BC government and the TRTFN has partnered with the province in a shared decision-making process.

An LUP for the area has been ratified by the TRTFN and was ratified by the provincial government in 2012. Upon legislation, the LUP established a number of protected areas (PA) and Resource Management Zones (RMZ) for specific land uses. The Tulsequah project resides in the Tulsequah Valley RMZ, in which mining is a permitted activity. No restrictions exist which are incompatible with the project design described in this report. Furthermore, the LUP provides for an access corridor for overland access to the Tulsequah project that encompasses the existing SUP for the access road. As described in the LUP and permitted by the BC government, the mine access road joins the provincial road network at the end of the Warm Bay Road.

Chieftain has signed a letter of Understanding with the TRTFN governing the establishment of a future Impact Mitigation and Mutual Benefit Agreement (IMMBA) focused on the project impacts and opportunities. Chieftain has progressed IMMBA discussions with the TRTFN and will continue to engage meaningfully with the TRTFN with a view to finalizing the IMMBA.

21 Capital & Operating Costs

21.1 Capital Costs

21.1.1 Introduction & Summary Data

Preparation of the capital cost estimates are based on the JDS philosophy that emphasizes *accuracy over contingency* and utilizes *defined and proven project execution strategies*. The estimates were developed by using first principles and applying direct applicable project experience and avoiding the use of general industry factors. Virtually all of the estimate inputs were derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The target accuracy of the capital cost estimate is in the range of -10%/+15%, which represents a JDS Feasibility Study Budget / Class 3 Estimate.

The following cost estimates are described within this section:

- Initial Capital Cost – This includes all costs incurred to develop the property to a state of nameplate production (2,000 t/d).
- Sustaining Capital Cost – These are costs incurred during operations for ongoing waste development, underground equipment acquisitions and underground infrastructure installations.
- Closure and Reclamation Cost – This refers to the cost incurred to permanently close and reclaim the site.
- Salvage Value – This refers to the monies recuperated through the sale of project assets at the end of the mine life.

The following project costs are not discussed in this section:

- Sunk costs are not considered in this study.
- Owners reserve is not considered in this study.
- Working capital is detailed in Section 22, Economic Analysis.

All cost estimates are based on the following key parameters:

- Owner-performed preproduction mining (contractors will be used only for Alimak raises).
- The specific scope and execution plans described in this study. Deviations from these plans will affect the capital costs.

21.1.2 Initial Capital Cost Estimate

21.1.2.1 Summary of Costs & Distribution

Initial capital costs include all costs to develop the property to a nameplate production of 2,000 t/d. Initial capital costs total $439.5 MM over three years (2013, 2014 and 2015).

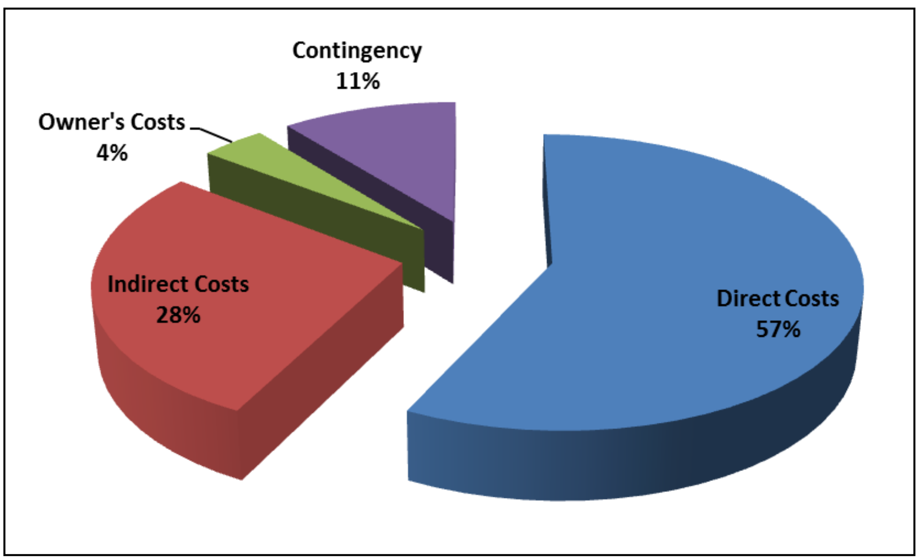
Table 21-1 summarizes the initial capital cost estimate by cost category; Figure 21-1 presents the initial capital cost distribution.

Table 21-1: Initial Capital Cost Estimate Summary by Category

Cost Category	Site Manhours	Total Cost (C$)	%
Direct Costs	979,700	252,046,000	57.4
Indirect Costs	416,500	122,994,000	28.0
Owners Cost	95,800	17,344,000	3.9
Contingency (12%)	-	47,086,000	10.7
Total	**1,491,900**	**439,470,000**	**100.0**

* All cost data are presented in Q4 2012 dollars.

Figure 21-1: Initial Capital Cost Distribution



A Level 4 work breakdown structure (WBS) was established for the initial capital cost estimate. Costs have been classified into the various WBS areas to ensure that 100% of the project scope has been captured. Table 21-2 summarizes the initial capital estimate by Level 3 WBS area.

Table 21-2: Initial Capital Cost Estimate by WBS (Level 3)

WBS	WBS Area Description	Site Manhours	Total C$	%
	Direct Costs			
10	**Site Development**	**34,900**	**7,418,000**	**1.7**
1010	Preproduction and Earthworks	21,300	4,708,000	1.1
1020	Mobile Mining Equipment	6,500	1,226,000	0.3
1030	Mobile Mining Equipment	7,100	1,484,000	0.3
15	**Underground Mining**	**264,900**	**50,236,000**	**11.4**
1505	Preproduction Mining Labour	222,000	13,853,000	3.2
1510	Underground Mining Supplies and Consumables	-	7,543,000	1.7
1520	Underground Equipment	-	11,229,000	2.6
1530	Underground Mining Infrastructure	3,900	4,891,000	1.1
1540	Ventilation and Services	-	979,000	0.2
1560	Underground Processing Facilities	39,000	11,741,000	2.7
20	**Limestone Quarry & Crushing**	**700**	**$159,000**	**0.0**
2010	Limestone Quarry and Crushing	700	$159,000	0.0
25	**Processing Plant**	**202,700**	**63,054,000**	**14.3**
2505	Concentrator Building	113,300	23,070,000	5.2
2510	Grinding And Classification	17,800	10,667,000	2.4
2520	Separation / Concentrating	26,400	14,392,000	3.3
2530	Concentrate Dewatering/Drying/Loadout	14,200	5,586,000	1.3
2540	Reagents and Lime Slaking	10,900	4,001,000	0.9
2550	Tailings and Paste	6,100	1,870,000	0.4
2560	Process Plant Utilities	14,000	3,467,000	0.8
30	**Tailings & Waste Rock Management**	**58,700**	**15,503,000**	**3.5**
3010	Tailings Area	52,000	13,964,000	3.2
3020	HPAG, PAG, and Pyrite Storage Area	6,700	1,539,000	0.4
35	**On-Site Infrastructure**	**96,900**	**61,479,000**	**14.0**
3510	Accommodation and Administration Facilities	22,200	15,248,000	3.5
3520	Ancillary Facilities	6,700	3,327,000	0.8
3530	Power Plant	26,800	22,502,000	5.1
3540	Fuel Storage And Distribution	9,200	5,602,000	1.3
3550	Fresh, Fire, and Potable Water Systems	10,500	2,717,000	0.6
3560	Water Treatment	15,500	4,307,000	1.0
3570	Garbage and Waste Management	5,000	1,610,000	0.4
3580	Plant Mobile Fleet	-	5,440,000	1.2
3590	Miscellaneous Infrastructure	1,000	725,000	0.2
40	**Off-Site Infrastructure**	**321,000**	**54,197,000**	**12.3**
4010	Mine Access Road	321,000	54,197,000	12.3

WBS	WBS Area Description	Site Manhours	Total C$	%
	Direct Costs Subtotal	**979,700**	**252,046,000**	**57.4**
	Indirect Costs			
90	**Project Indirects**	**287,300**	91,642,000	20.9
9010	Construction Indirects - Access Road	4,000	36,224,000	8.2
9015	Main Access Road Maintenance	12,100	3,027,000	0.7
9020	Camp Catering and Janitorial Services	138,800	7,380,000	1.7
9025	Construction Generator Fuel	-	6,260,000	1.4
9030	Construction Indirects - Others	118,800	9,173,000	2.1
9040	Freight	-	19,621,000	4.5
9050	Passenger Transportation	-	5,350,000	1.2
9060	Capital Spares and Initial Fills	-	2,610,000	0.6
9070	Commissioning and Start-up	13,400	1,996,000	0.5
95	**Engineering & EPCM**	**129,200**	**31,351,000**	**7.1**
9510	Detailed Engineering and Procurement Management	-	17,105,000	3.9
9520	Project and Construction Management	129,200	14,246,000	3.2
	Indirect Costs Subtotal	**416,500**	**122,994,000**	**28.0**
	Owners Costs			
90	**Owners Costs**	**95,800**	**17,344,000**	**3.9**
9810	Owners Costs - Access Road	25,300	5,029,000	*1.1*
9820	Owners Costs - Preproduction Labour	70,500	5,182,000	*1.2*
9830	Owners Costs - On-Site Items and Misc Costs	-	6,793,000	*1.5*
9840	Owners Costs - Sat Offices and Off-Site Warehousing	-	340,000	*0.1*
	Owner Costs Subtotal	**95,800**	17,344,000	3.9
	Totals			
	Direct Costs	979,700	252,046,000	57.4
	Indirect Costs	416,500	122,994,000	28.0
	Owners Costs	95,800	17,344,000	3.9
99	**Contingency (12%)**	-	47,086,000	10.7
	Grand Total	**1,491,900**	**439,470,000**	**100.0**

* All cost data are presented in Q4 2012 dollars

Costs were also categorized by commodity group in accordance with standard resource types and selected deliverables. Table 21-3 summarizes the initial capital cost estimate by commodity grouping and cost type.

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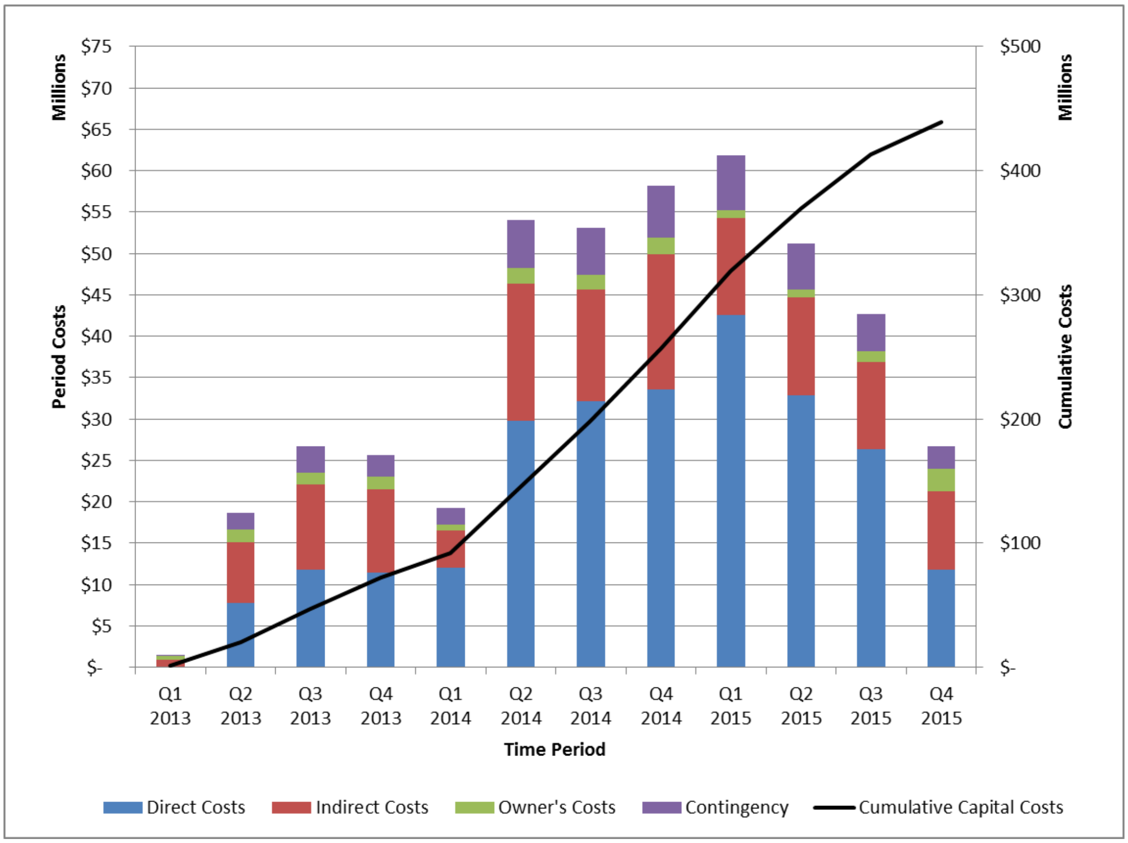
Table 21-3: Initial Capital Cost Estimate Summary, by Commodity Group

Commodity Group	Site Manhours	Labour $	Material $	Equipment $	Equip Usage $	Other $	Total $
Mining	225,900	14,231,000	8,462,000	15,737,000	-	65,000	38,495,000
Mine Access Road	321,000	24,389,000	8,130,000	-	21,679,000	-	54,197,000
Earthworks	80,000	7,762,000	1,122,000	-	11,486,000	-	20,370,000
Concrete	77,600	7,531,000	4,608,000	-	123,000	-	12,262,000
Architectural and Buildings	50,100	4,860,000	1,394,000	18,440,000	782,000	285,000	25,761,000
Structural Steel	23,000	2,234,000	3,361,000	-	196,000	-	5,792,000
Mechanical and Equipment	77,700	7,531,000	895,000	37,950,000	462,000	-	46,838,000
Plate-work	8,300	809,000	653,000	25,000	66,000	-	1,554,000
Piping	61,400	5,960,000	4,235,000	52,000	474,000	-	10,722,000
Electrical and Instrumentation	54,600	4,909,000	3,357,000	21,107,000	255,000	990,000	30,617,000
Surface Mobile Equipment	-	-	-	4,590,000	-	851,000	5,440,000
Indirects	416,500	29,762,000	17,292,000	2,540,000	8,107,000	65,293,000	122,994,000
Owner Costs	95,800	7,637,000	-	-	-	9,707,000	17,344,000
Contingency	-	-	-	-	-	47,086,000	47,086,000
Grand Total	**1,491,900**	**117,615,000**	**53,511,000**	**100,440,000**	**43,629,000**	**124,275,000**	**439,470,000**

* All cost data are presented in Q4 2012 dollars

Figure 21-2 presents the initial capital cost profile. More than 60% of the project costs are incurred after completion of the mine access road, which allows the start of equipment delivery, mechanical/electrical installations and underground mining. Refer to Section 24 for more information on the project development schedule.

Figure 21-2: Quarterly Initial Capital Costs, by Category



21.1.2.2 Basis of Initial Capital Estimate

The initial capital cost estimate was compiled utilizing input from engineers, contractors, and suppliers with experience delivering projects in northern Canada. Wherever possible, the bottom-up first principle estimates were top-down benchmarked against other projects of similar size with similar climate and logistical conditions.

Table 21-4 summarizes the basis of estimate for each key WBS area of the initial capital estimate.

Table 21-4: Basis of Initial Capital Estimate Summary

Commodity	Estimate Basis
General	
Contract Labour Rate	Estimated from first principles utilizing benchmarked base labour rates and applying standard burdens (EI, CPP, etc) as required by law. Additional indirect cost burdens (supervision, small tools, etc.) were added for items not estimated separately. Labour rates assume a 3&1 shift rotation.
Concrete Supply and Installation	Preliminary concrete quantities were provided by the engineer and verified by JDS. Costs were estimated from first principles utilizing quoted cement/rebar pricing and other estimates for projects with similar conditions. Concrete unit rates include batching costs.
Site Development	
Site Earthworks	Material take-off quantities were provided by the engineer. Costs were estimated from first principles, leveraging quoted unit and hourly rates from the March 2012 access road tender.
Underground Mining	
U/G Development Labour and Consumables	Estimated from first principles utilizing the same methodologies described in the Operating Cost section.
U/G Equipment Supply	Multiple budget quotations and firm prices were received based on project specific specifications and data sheets.
U/G Infrastructure Supply	Budget quotations or firm prices were received based on brief specifications or standard off-the-shelf equipment requests.
Limestone Quarry	
Limestone Crushing and Stockpiling	Limestone quantity for the first year of operations was determined from the mining waste schedule. Costs for the quarry development and contract crushing were estimated from first principles, leveraging quoted unit rates from local contractors.
Processing Facilities	
Concentrator Building Supply/Install	Budget quotations were provided by a building contractor based on preliminary project specific layouts and specifications.
Structural Steel Supply/Install	Material take-offs were provided by the engineer. Costs were estimated from first principles, leveraging other project estimates for similar projects.
Major Process Equipment	Multiple budget quotations and firm prices were received based on project specific specifications and data sheets prepared by the engineer.
Minor Process Equipment	Budget quotations or firm prices were received based on brief specifications or standard off-the-shelf equipment requests.
Mech/Plate/Pipe/Elect Installation	Material take-offs were provided by the engineer. Costs were estimated from first principles, leveraging other project estimates for similar projects.
Tailings & PAG Facilities	
Tailings and PAG Earthworks	Material take-off quantities were provided by the engineer. Costs were estimated from first principles, leveraging quoted unit rates from local contractors.
Mech/ Pipe/Elect Installation	Material take-offs were provided by the Engineer. Costs were estimated from first principles, leveraging other project estimates for similar projects.

Commodity	Estimate Basis
Surface Infrastructure	
Construction Camp and Administration Facilities	Budgetary quotations were received for supply based on project specific requirements. Installation estimated from first principles, leveraging vendor input and other estimates for projects of similar size.
Mine Shops and Warehouse	Multiple budgetary quotations were received for supply and installation based on project specific requirements.
Power Plant	A budgetary quotation was received for the generator engines. Installation costs estimated from first principles, leveraging other estimates for similar sized projects.
Fuel Storage System	Fuel storage requirements were determined by the engineer. Firm quotations were received for the diesel storage & off-loading systems and the LNG storage & vaporization systems.
Fire and Potable Water Systems	Fire water tank was sized to meet all code requirements. Firm quotations were received for the tank and pumping system. Budget quotations were received for the potable water system based on project specific requirements.
Water Treatment Facility	A budgetary cost estimate was received from vendor based on preliminary test-work.
Incinerator and STP Facility	Budgetary quotations were received from several vendors based on project specific requirements.
Plant Mobile Fleet	The mobile equipment fleet was sized based on required operations tasks and professional experience. Firm quotations were received for all pieces of equipment. Used equipment pricing was used for low utilization equipment.
Miscellaneous Infrastructure	Cost allowances were used based on experience from similar sized projects.
Mine Access Road	
Mine Access Road Construction	Multiple quotations were received based on initial design and specifications prepared by the engineer. New quantities were applied to quoted unit rates based on the revised design.
Project Indirect Costs	
Access Road Indirect Costs (Fuel)	Fuel quantities were provided by the contractors through the tendering process. Diesel quotations were received, including air transport and offloading at site.
Mine Access Road Maintenance	Costs were estimated from first principles, leveraging quoted unit rates from local contractors. An avalanche control contractor was engaged to provide a budgetary quotation.
Camp Catering and Janitorial Services	Multiple quotations were received based on project specific conditions. The site manhours within estimate were grouped into categories to determine total required catering man-days.
Construction Generator Fuel	Fuel quantities were determined based on OEM data sheets and required operating hours. Diesel quotations were received, including air or road transport components and offloading at site.
Miscellaneous Construction Indirects	Costs were estimated from first principles leveraging other estimates for similar sized projects. The largest cost areas are within Contractor Travel, Orientation, and Training and Site Services Labour.
Freight – Barge	A complete freight study was completed by JDS. A barging contractor was engaged to provide shipping constraints, exclusions, and costs.
Freight – Road and Air	A complete freight study was completed by JDS. Budget quotations were received for common road shipping routes and airfreight per load.

Commodity	Estimate Basis
Passenger Transportation	The site manhours within the estimate were grouped into categories to determine total required passenger movements. Quotations were received from air charter contractors.
Capital Spares and Initial Fills	Capital spares were factored based on the capital cost of appropriate process equipment. Initial grinding media charge was calculated by the engineer, initial reagent fills were factored.
Commissioning and Start-up	Cost allowances were used based on experience from similar sized projects.
Engineering & EPCM	
Detailed Engineering	Budgetary quotations were received for the major detailed engineering and procurement service packages. Allowances were added for specialty engineering tasks.
EPCM	Costs were estimated from first principles using a blended team of Owner and contract labour. Owner labour rates were determined utilizing the methods described in the Operating Cost section. Industry standard contract rates were utilized for contract project management labour.
Owners Costs	
Access Road Owners Costs	Costs were estimated from first principles based on support requirements for the access road construction, which were realized through the access road tendering process.
Other Owners Costs	Other Owners Costs include specific cost items described within the Operating Cost section that occur during the project development phase.

21.1.2.3 Contingency

A blended contingency was applied to the estimate through constructing and executing a probability analysis model. Costs were logically grouped by type and the P_5 and P_{95} cases were defined for both quantity and unit price growth risk. The model utilized PERT distribution curves and Monte-Carlo sampling (5,000 iterations) to determine the P_{85} contingency amounts for each cost grouping. Results concluded that the use of a 12.0% blended contingency was appropriate.

21.1.3 Sustaining Capital Cost Estimate

21.1.3.1 Summary of Costs& Distribution

The primary sustaining capital cost is capital mine development occurring during the operations phase. Capital mine development represents the mine’s permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, ore pass accesses and permanent explosive storage cut-outs, as well as main ventilation raises.

Other sustaining capital cost items include additional mine equipment, additional underground mine infrastructure (dewatering pumps, electrical equipment, paste backfill piping), and additional mine ventilation fans and equipment.

Table 21-5 summarizes the total sustaining capital costs by area; Figure 21-3 presents the distribution of these costs.

Table 21-5: Sustaining Capital Cost Estimate Summary, by Area

Cost Category	Total Cost (C$)	%
Development	49,435,000	77.3
Mine Equipment	7,805,000	12.2
Mine Infrastructure	5,343,000	8.4
Ventilation and Services	1,401,000	2.2
Total	**63,984,000**	**100.0**

* All cost data are presented in Q4 2012 dollars

Figure 21-3: Sustaining Capital Cost Estimate Distribution

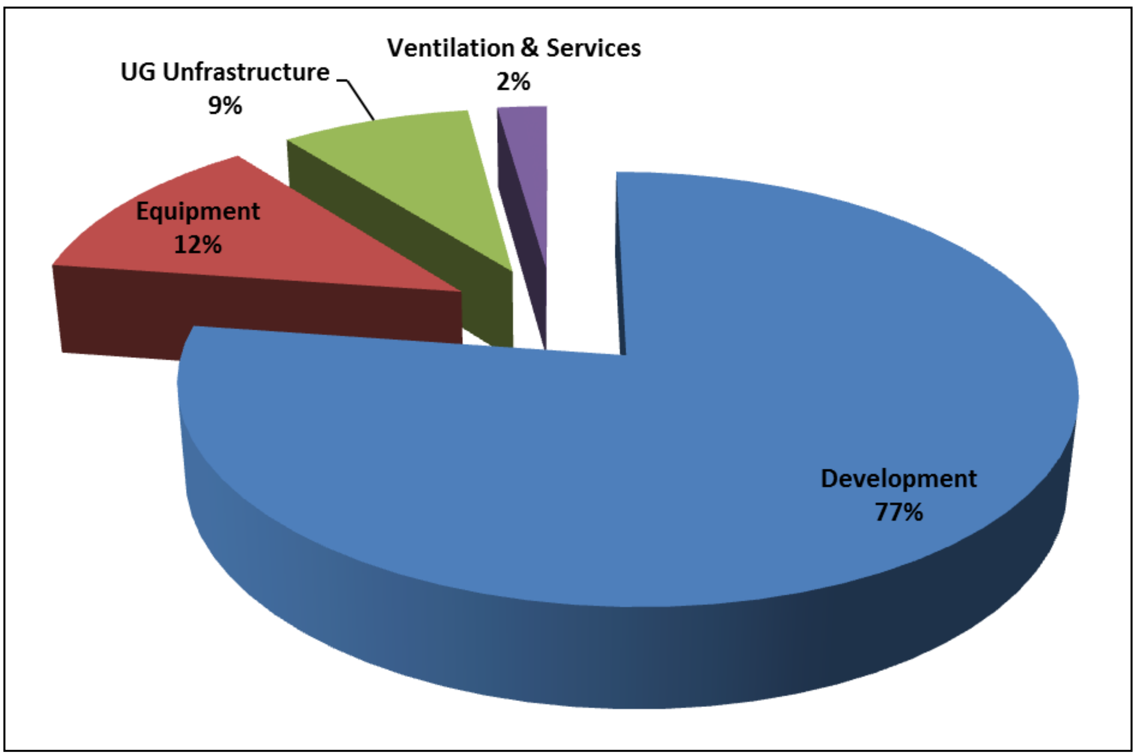


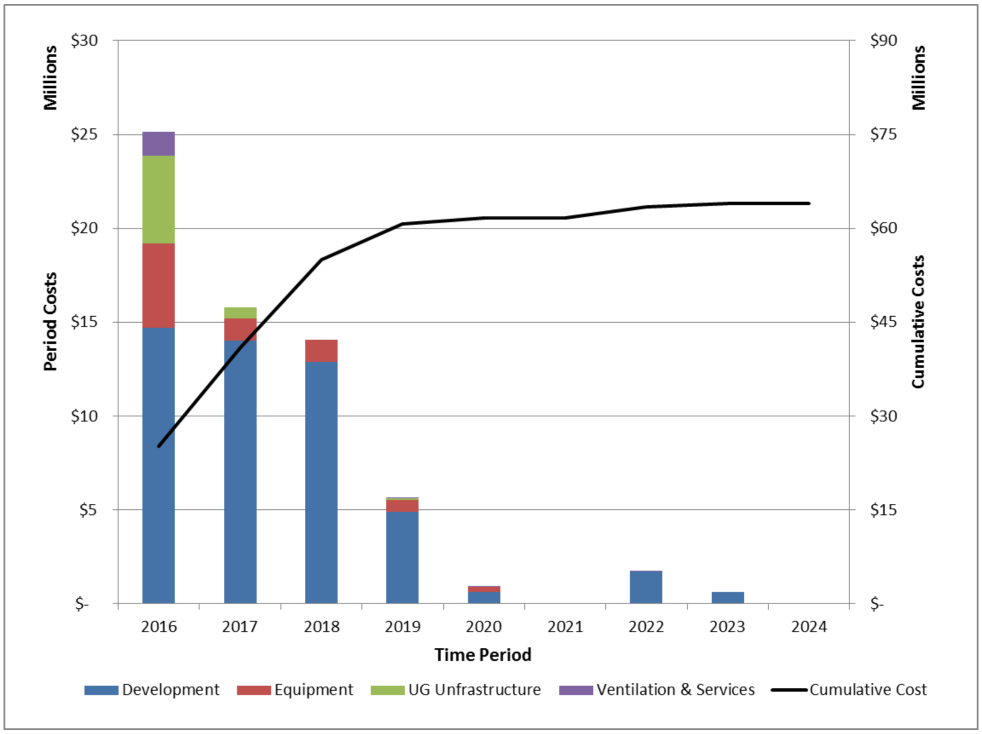
Table 21-6 and Figure 21-4 detail the annual sustaining capital costs by activity.

Table 21-6: Annual Sustaining Capital, by Activity

Area / Activity	2016	2017	2018	2019	2020	2021	2022	2023	2024	Total
Development										
Labour	8,254,000	7,735,000	7,099,000	2,886,000	473,000	-	1,260,000	443,000	-	28,150,000
Lateral Development	4,828,000	4,991,000	4,708,000	1,444,000	157,000	-	451,000	167,000	-	16,746,000
Vertical Development	1,596,000	1,281,000	1,092,000	570,000	-	-	-	-	-	4,539,000
Subtotal – Development	**14,678,000**	**14,007,000**	**12,899,000**	**4,900,000**	**630,000**	**-**	**1,711,000**	**610,000**	**-**	**49,435,000**
Mine Equipment										
Drilling Equipment	1,185,000	-	-	60,000	60,000	-	-	-	-	1,306,000
Loading Equipment	1,813,000	-	-	-	-	-	-	-	-	1,813,000
Hauling Equipment	1,080,000	1,080,000	1,080,000	-	-	-	-	-	-	3,240,000
Support/Utility Equipment	450,000	105,000	105,000	555,000	230,000	-	-	-	-	1,446,000
Subtotal – Equipment	**4,529,000**	**1,185,000**	**1,185,000**	**616,000**	**290,000**	**-**	**-**	**-**	**-**	**7,805,000**
Mine Infrastructure										
Dewatering	384,000	-	-	-	-	-	-	-	-	384,000
Underground Electrical	4,160,000	-	-	-	-	-	-	-	-	4,160,000
Paste Backfill Distribution	-	600,000	-	-	-	-	-	-	-	600,000
Safety and Mine Rescue	100,000	-	-	100,000	-	-	-	-	-	200,000
Subtotal – Infrastructure	**4,644,000**	**600,000**	**-**	**100,000**	**-**	**-**	**-**	**-**	**-**	**5,343,000**
Ventilation & Services										
Primary Ventilation Fans	1,176,000	-	-	-	-	-	-	-	-	1,176,000
Secondary Fans/Equipment	92,000	-	-	33,000	33,000	33,000	33,000	-	-	225,000
Subtotal – Ventilation	**1,268,000**	**-**	**-**	**33,000**	**33,000**	**33,000**	**33,000**	**-**	**-**	**1,401,000**
Totals										
Development	14,678,000	14,007,000	12,899,000	4,900,000	630,000	-	1,711,000	610,000	-	49,435,000
Mine Equipment	4,529,000	1,185,000	1,185,000	616,000	290,000	-	-	-	-	7,805,000
Mine Infrastructure	4,644,000	600,000	-	100,000	-	-	-	-	-	5,343,000
Ventilation and Services	1,268,000	-	-	33,000	33,000	33,000	33,000	-	-	1,401,000
Total	**25,119,000**	**15,791,000**	**14,084,000**	**5,649,000**	**953,000**	**33,000**	**1,744,000**	**610,000**	**-**	**63,984,000**

* All cost data are presented in Q4 2012 dollars

Figure 21-4: Annual Sustaining Capital Cost, by Area



21.1.3.2 Basis of Sustaining Capital Cost Estimate

Sustaining capital was estimated in the same manner as the initial capital costs.

21.1.3.3 Contingency

No contingency factor was applied to sustaining capital costs.

21.1.4 Closure & Reclamation Cost Estimate

21.1.4.1 Summary of Activities & Distribution

Progressive reclamation will begin during mine construction and continue throughout the operating life of the mine. When mine operations cease, all new PAG material will have been backfilled into underground workings and flooded, and the historic workings will be backfilled with neutral paste backfill. The remaining reclamation work will consist of decommissioning the facilities and re-contouring land surfaces. Re-seeding and monitoring programs will continue after the closure of the mine. The detailed scope of the closure, reclamation, and post-closure monitoring programs is provided below.

Mine closure and reclamation activities include:

- constructing an on-site demolition landfill
- demolishing and disposing of or removing all structures and equipment
  - demolition wastes consisting of clean inert material will be disposed of in an inert waste landfill on site
  - salvageable obsolete equipment and recyclables (e.g., steel structures and pipes) will be transported off site
  - hazardous and toxic wastes and liquid wastes will be hauled to approved waste management facilities for disposal
- disposing of or removing all liners, equipment, sumps, and associated structures at the PAG facilities
- disposing of or removing all bridges and culverts
- decommissioning of all site roads
- decommissioning of the airstrip
- re-contouring site areas consistent with surrounding landforms
- installing erosion control measures as necessary
- sealing all mine portals permanently
- draining and treating of all water from the TMF
- covering the tailings material in the drained TMF with salvaged soils
- constructing a closure spillway at the TMF
- decommissioning the mine access road
  - removing all bridges
  - removing all culverts.
  - removing all jersey barriers and other concrete structures
  - reestablishing natural creek channels
  - scarifying and seeding road surfaces.

Post-closure activities include:

- re-seeding the land annually for five successive years
- monitoring post-closure vegetation regrowth twice per year for two years and one final inspection five years following closure
- monitoring the geotechnical conditions of the TMF dam to ensure dam safety
- performing water quality monitoring regularly for ten years following closure
- performing remedial measures as required from monitoring program findings.

Table 21-7 summarizes the closure and reclamation costs by category and Figure 21-5 presents the cost distribution.

Table 21-7: Closure & Reclamation Cost Summary, by Category

Cost Category	Total Cost (C$)	%
Closure and Reclamation	10,272,000	74.3
Post-Closure Activities	1,761,000	12.7
Contingency	1,803,000	13.0
Total	**13,826,000**	**100.0**

* All cost data are presented in Q4 2012 dollars.

Figure 21-5: Closure & Reclamation Cost Distribution

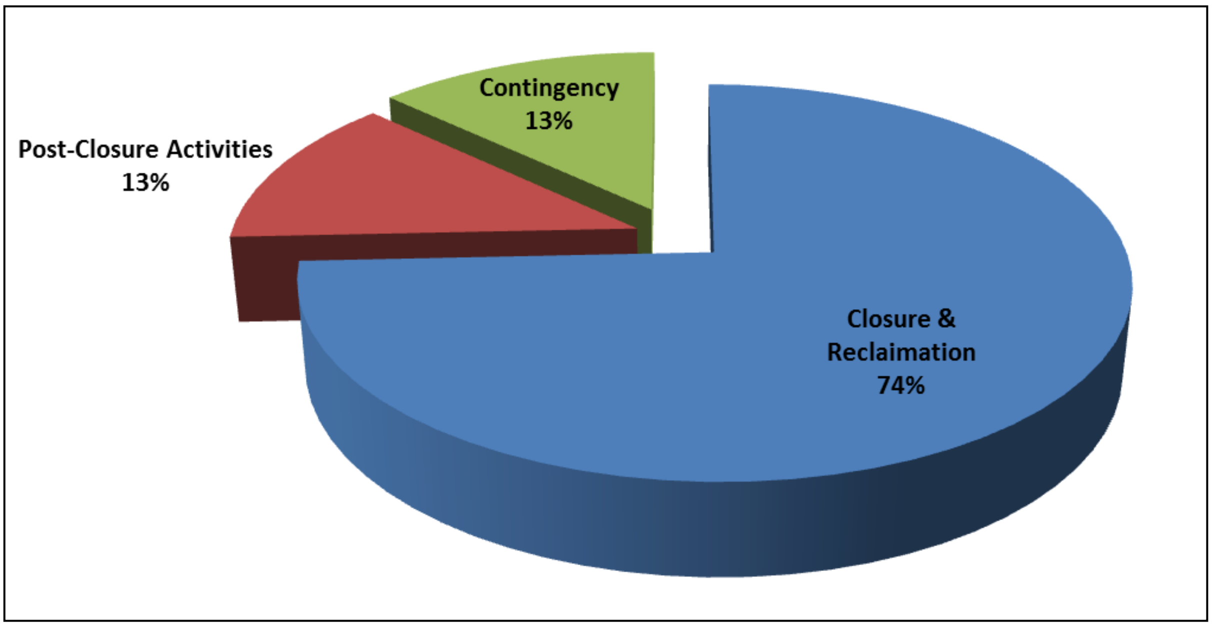


Table 21-8 details the annual closure and reclamation costs by activity.

Table 21-8: Closure & Reclamation Annual Cost Estimate, by WBS

Phase / Activity	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	Total
Site Closure											
Equipment	2,254,000	-	-	-	-	-	-	-	-	-	2,254,000
Labour	1,738,000	-	-	-	-	-	-	-	-	-	1,738,000
Waste Disposal/Removal	225,000	-	-	-	-	-	-	-	-	-	225,000
Tailings Facility Drainage	122,000	-	-	-	-	-	-	-	-	-	122,000
Portal Plugs (3)	900,000	-	-	-	-	-	-	-	-	-	900,000
Access Road Closure	3,911,000	-	-	-	-	-	-	-	-	-	3,911,000
Indirects	1,122,000	-	-	-	-	-	-	-	-	-	1,122,000
Subtotal – Site Closure	**10,272,000**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**10,272,000**
Post Closure Activities											
Monitoring and Maintenance	293,000	768,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	1,508,000
Revegetation	126,000	62,000	31,000	16,000	8,000	-	-	-	-	-	243,000
Subtotal – Post Closure	**420,000**	**831,000**	**87,000**	**71,000**	**64,000**	**56,000**	**56,000**	**56,000**	**56,000**	**56,000**	**1,751,000**
Contingency											
Contingency (15%)	1,604,000	125,000	13,000	11,000	10,000	8,000	8,000	8,000	8,000	8,000	1,803,000
Total	**12,295,000**	**955,000**	**100,000**	**82,000**	**73,000**	**64,000**	**64,000**	**64,000**	**64,000**	**64,000**	**13,826,000**

21.1.4.2 Basis of Closure Cost Estimate

Table 21-9 presents additional details of the closure and reclamation cost estimate, arranged by WBS.

Table 21-9: Basis of Closure Estimate Summary, by WBS

Phase / Item	
Closure	**Estimate Basis**
Schedule	The closure schedule was conservatively estimated based on the required trucking hours to remove/dispose of demolished items, with an allowance for re-contouring and mobilization/demobilization.
Equipment	Owner equipment operating costs were estimated as per the operating cost basis of estimate. Contractor equipment costs were estimated using rates from local contractors. Fuel requirements were estimated based on operating hours and delivered fuel commodity rates from the capital estimate.
Labour	Owner labour costs were estimated as per the operating cost basis of estimate. Contract labour costs were estimated at the blended rate as calculated in the initial capital estimate.
Waste Disposal/Removal	On-site landfill disposal costs are included in the equipment and labour costs. It is assumed that the value of salvageable materials will offset the cost of hauling. An allowance of 100 tonnes of toxic waste removal has been included.
Tailings Facility Drainage	An allowance was used based on current effluent treatment plant operations.
Portal Plugs	The estimate contained within the KCB design report from 1994 was escalated to 2012 dollars per the consumer price index.
Access Road Closure	Costs were estimated from first principles by SNT Engineering leveraging local contractor equipment rates.
Indirect Costs	Costs were calculated based on the level of effort required to perform the site closure activities; estimated per the same basis of estimate parameters as the initial capital estimate.
Post Closure Activities	
Monitoring/Maintenance	Conservative cost allowances were used based on similar projects.
Revegetation	Costs were estimated using historic pricing of seed and seedlings with an assumed 50% re-seeding rate

It was determined that site closure activities will occur during the initial five months following mine closure. Operations labour and equipment will be utilized the greatest extent possible and supplemented with contract labour and equipment as required.

The amount of solid waste generated in demolition activities was estimated based on the preliminary design information. It is estimated that 19,524 m^3 or 9,164 tonnes of waste will be disposed of at a landfill constructed on site and 8,119 m^3 or 3,466 tonnes of materials will be salvageable and shipped off site. An allowance of 100 tonnes of hazardous waste disposal was included in the estimate.

Indirect costs to support site closure were calculated based on the level of effort required to perform the site closure activities and were estimated per the same basis of estimate parameters as the initial capital estimate. Indirect cost items for the mine closure include mine access road maintenance, camp catering, personnel flights, project management, and environmental supervision.

A standalone mine access road closure cost estimate was completed by the design engineer (SNT engineering) from first principles, utilizing local contractor labour and equipment rates.

21.1.4.3 Contingency

A blended 15% contingency was applied to the closure and reclamation estimate utilizing professional judgement, based on the level of scope definition.

21.1.5 Salvage Value Estimate

Much of the capital equipment brought to site will have some resale value even at the end of mine life. Table 21-10 presents a summary of the purchase price of the equipment and the expected resale value after considering the costs of disassembly and hauling off site to Skagway, Alaska.

Table 21-10: Salvage Value Estimate

Item	Capital Cost	% Residual Value	Cash Value†
Underground Mining Equipment Fleet	19,034,000	5%	952,000
Jaw Crusher	375,000	10%	38,000
Grinding Mills	7,470,000	10%	747,000
Pressure Filters	1,227,000	10%	123,000
Paste Backfill Equipment	1,025,000	10%	103,000
Power Plant	15,370,000	25%	3,843,000
Other Generators	630,000	10%	63,000
Construction Camp Complex	1,949,000	5%	97,000
Main Camp Complex	9,180,000	10%	880,000
Administration/Dry Complex	1,485,000	15%	220,000
Ancillary Buildings	985,000	10%	99,000
Assay Lab	705,000	10%	71,000
Effluent Treatment Equipment	1,722,000	5%	86,000
Surface Equipment Fleet	3,960,000	5%	198,000
Bridges	600,000	10%	60,000
Total	**65,716,000**	**12%**	**7,577,000**

*All cost data presented in Q4 2012 dollars. † "Cash Value" denotes net cash value of salvageable equipment - after consideration of disassembly and shipment costs to Juneau, AK.

21.1.6 Capital Cost Exclusions

The following costs are not included in the capital cost estimate:

- GST/PST
- schedule acceleration costs
- schedule delays and associated costs, such as those caused by:
  - unexpected site conditions
  - latent ground conditions
  - labour disputes
  - force majeure
  - permit applications
- development fees and approval costs beyond those specifically identified
- cost of any disruption to normal operations
- foreign currency changes from project exchange rates
- commodity specific escalation rates
- economy factors/pressure on labour productivity
- event risk
- financing costs
- cost associated with third party delays
- working capital (discussed in Section 22)
- sunk costs
- Owner's reserve
- escalation (cost data are presented in Q4 2012 dollars).

21.2 Operating Costs

21.2.1 Introduction & Summary Data

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies. The estimate was developed using first principles and applying direct applicable project experience, and avoiding the use of general industry factors. The operating cost is based on Owner owned and operated mining/services fleets and minimal use of permanent contractors except where value is provided through expertise and/or packaged efficiencies/skills. Virtually all of the estimate inputs were derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The target accuracy of the operating cost estimate is -10%/+15%, which represents a JDS Feasibility Study Budget / Class 3 Estimate.

The operating cost estimate is broken into five major sections:

- Mining
- Processing
- Power Plant
- G&A
- Concentrate Transportation.

Certain items within the operating costs begin during the construction phase (2013 through 2015) and continue through the life of the mine. All costs incurred during the construction phase have been capitalized and are included as part of the capital cost estimate. Operating costs have been compiled in accordance with industry standards.

Underground lateral and vertical waste development after the preproduction period has been capitalized and will not appear as an operating cost (refer to Section 21.1.3, Sustaining Capital Cost). Capital waste development represents the mine's permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, ore pass accesses and permanent explosive storage cut-outs, as well as main ventilation raises.

The total operating unit cost is $125.70 per tonne processed; average annual operating costs and total unit costs are summarized in Table 21-11. Figure 21-6 presents the operating cost distribution.

Table 21-11: Total Average Operating Cost by Area

Cost Group / Item	Average Annual Operating Cost (C$)	Average Unit Operating Cost ($/t milled)	%
Underground Mining	21,534,000	30.06	23.9
Processing	16,491,000	23.02	18.3
Power Plant	16,173,000	22.58	17.9
General and Administration	16,094,000	22.47	17.8
Concentrate Transportation	19,936,000	27.83	22.0
Total	**90,228,000**	**125.96**	**100.0**

* All cost data are presented in Q4 2012 dollars.

Figure 21-6: Total Operating Cost Distribution

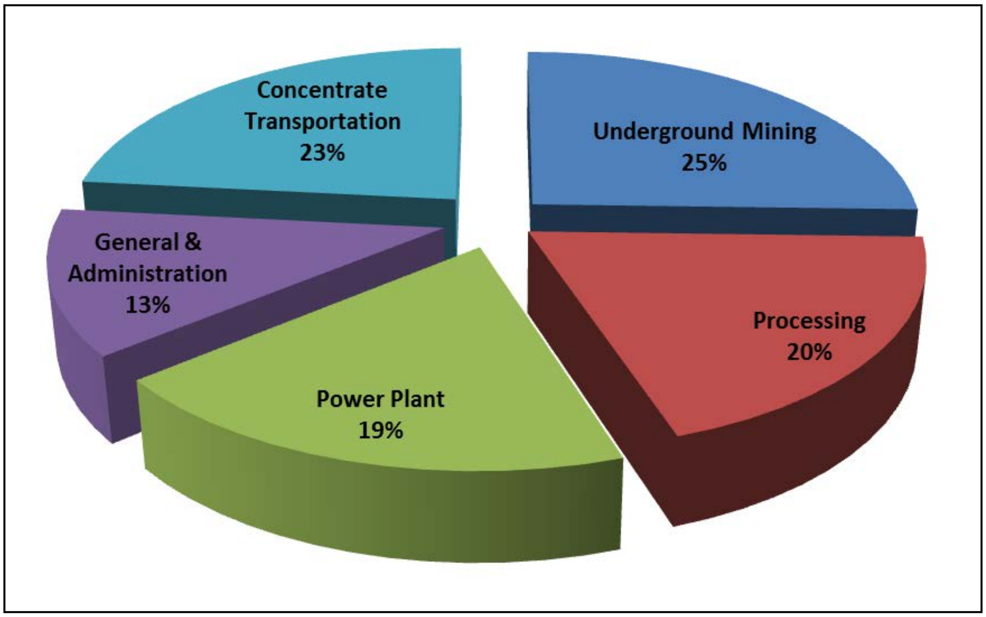


Table 21-12 (overleaf) and Figure 21-7 below details the annual operating costs per year by area.

Figure 21-7: Annual Operating Costs, Summary by Area

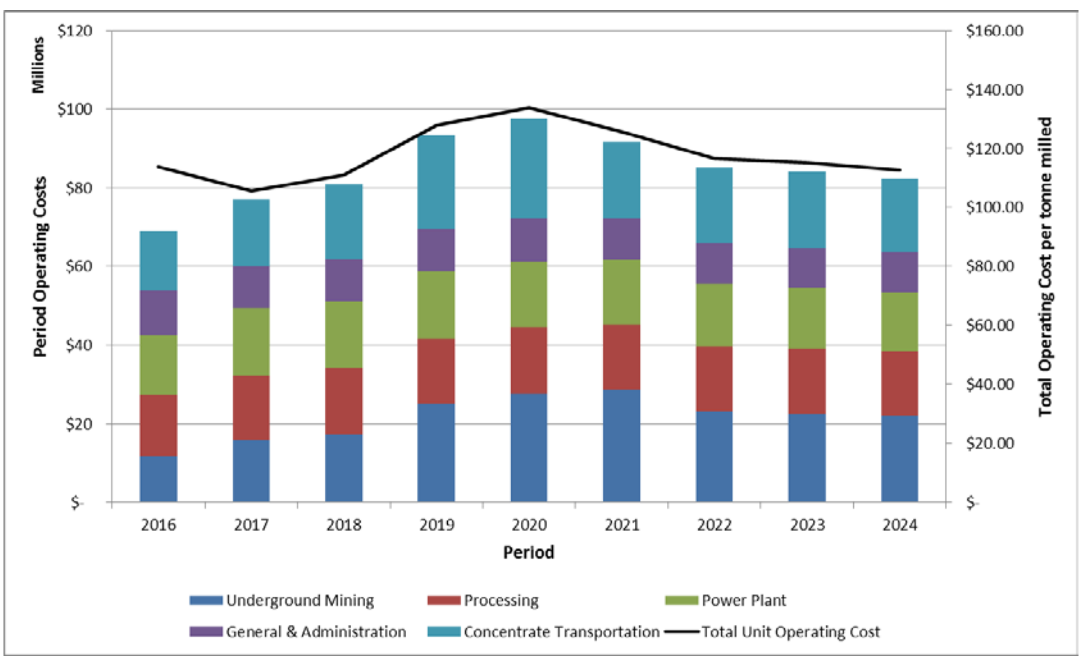


Table 21-12: Annual Operating Costs, Summary by Area

Activity	2016	2017	2018	2019	2020	2021	2022	2023	2024	Total
Annual Operating Costs										
Mining (C$)	11,778,000	15,777,000	17,191,000	25,079,000	27,509,000	28,755,000	23,242,000	22,546,000	21,930,000	193,808,000
Processing (C$)	15,702,000	16,488,000	16,874,000	16,485,000	16,935,000	16,484,000	16,484,000	16,484,000	16,484,000	148,421,000
Power Plant (C$)	14,985,000	17,096,000	17,078,000	17,049,000	16,898,000	16,345,000	15,702,000	15,422,000	14,977,000	145,553,000
G&A (C$)	16,817,000	16,190,000	16,184,000	16,419,000	16,443,000	16,289,000	15,747,000	15,395,000	15,368,000	144,850,000
Concentrate Transportation	15,382,000	17,178,000	19,312,000	24,205,000	25,654,000	19,626,000	19,521,000	19,671,000	18,872,000	179,422,000
Total Operating Cost (CAD)	**74,537,000**	**82,603,000**	**86,462,000**	**98,989,000**	**103,163,000**	**97,330,000**	**90,525,000**	**89,336,000**	**87,438,000**	**812,054,000**
Unit Operating Costs										
Mining ($/t milled)	19.40	21.61	23.55	34.36	37.68	39.39	31.84	30.88	30.04	30.06
Processing ($/t milled)	25.86	22.59	23.12	22.58	23.20	22.58	22.58	22.58	22.58	23.02
Power Plant ($/t milled)	24.68	23.42	23.39	23.35	23.15	22.39	21.51	21.13	20.52	22.58
G&A ($/t milled)	27.70	22.18	22.17	22.49	22.52	22.31	21.57	21.09	21.05	22.47
Conc. Transport ($/t milled)	25.34	23.53	26.45	33.16	35.14	26.89	26.74	26.95	25.85	27.83
Total Unit Cost (/t milled)	**122.78**	**113.15**	**118.44**	**135.60**	**141.32**	**133.33**	**124.01**	**122.38**	**119.78**	**125.96**

* All cost data are presented in Q4 2012 dollars.

21.2.2 Operations Labour

Operations labour cost is contained within each sub-section of the operating costs. This section serves to provide an overview of total workforce and the methods used to build the labour rates.

Table 21-13 summarizes the total planned workforce during project operations.

Table 21-13: Planned Operations Workforce

Department	Total Persons Employed (peak)
Mining	123
Processing	62
Site Services	13
G&A	23
Contractors	42
Total	**265**

Total contractor level is an average of all contractors utilized throughout the year (to account for intermittent contractors). Services to be contracted include camp catering and cleaning, access road maintenance incl. avalanche control, limestone quarrying, diesel generator overhauls.

Labour base rates were determined through experience and benchmarked against similar operations in BC. Labour burdens were assembled using first principles. The following items are included in the burdened labour rates:

- scheduled overtime costs based on individual employee rotation
- unscheduled overtime allowance of 10% for hourly employees
- travel pay of eight paid hours per rotation for hourly employees
- production bonus for underground production and development miners
- CPP, EI, and WCB as required by law
- statutory holiday allowance of 6% of scheduled hours
- vacation pay allowance of 6% of scheduled hours
- RSP allowance of 6% of scheduled hours
- health and welfare allowance of $2,500 per year for all employees.

21.2.3 Mining Operating Costs

The mine operating costs are broken down into the following functional areas:

- Production – Costs include equipment parts, fuel, oil and lube, explosives and ground support and other consumables for lateral ore development and LH and MCF stoping.
- Backfill – Costs include cement, piping and paste plant labour and consumables.
- Mine General – Costs include support equipment costs (parts, fuel, oil and lube), mining labour for stoping, technical services and miscellaneous supplies.
- Mine Maintenance – Costs include labour and shop consumables to maintain and repair the underground mining mobile equipment.

The estimated total mining operating unit cost is $30.06 per tonne processed; average annual operating costs and total unit costs are summarized in Table 21-14 (overleaf). Figure 21-8 below presents the mining operating cost distribution. The subsections below further describe the basis of estimate for major items within each grouping.

Figure 21-8: Mining Operating Cost Distribution

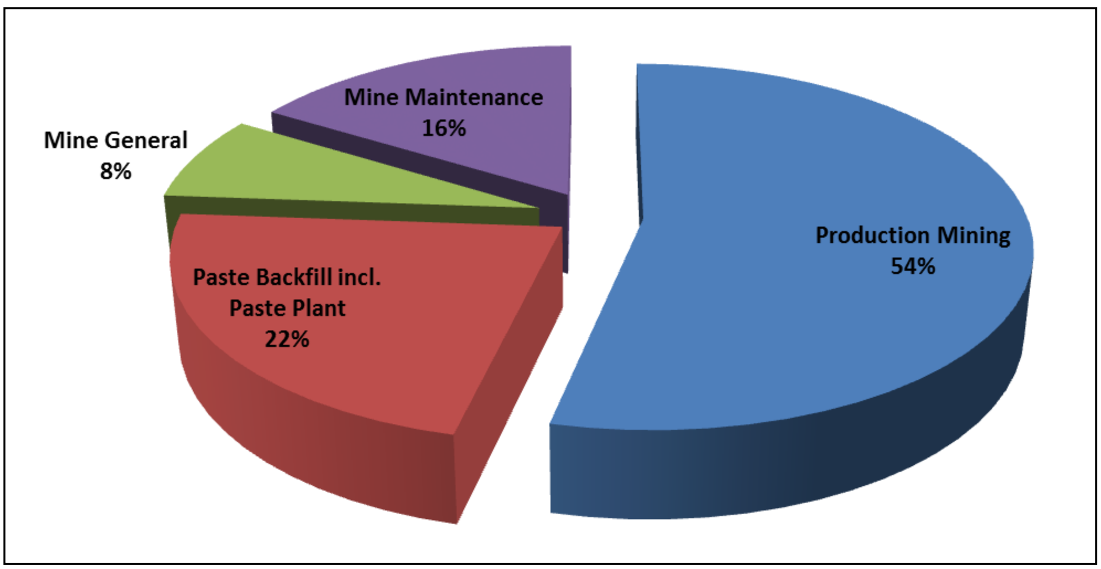


Table 21-14: Average Mining Operating Costs

Cost Group / Item	Average Annual Operating Cost (C$)	Average Unit Operating Cost ($/t milled)	(%)
Production			
Labour	6,815,000	9.51	31.6
Fuel	1,112,000	1.55	5.2
Equipment	2,031,000	2.84	9.4
Consumables	701,000	0.98	3.3
Explosives	891,000	1.24	4.1
Production - Total	**11,550,000**	**16.12**	**53.6**
Backfill			
Piping	154,000	0.21	0.7
Plant Operators	381,000	0.53	1.8
Consumables	150,000	0.21	0.7
Flocculant	34,000	0.05	0.2
Cement	4,132,000	5.77	19.2
Backfill - Total	**4,850,000**	**6.77**	**22.5**
Mine General			
Technical Services	1,524,000	2.13	7.1
Misc. Supplies	100,000	0.14	0.5
Mine General - Total	**1,624,000**	**2.27**	**7.5**
Mine Maintenance			
Labour	3,318,000	4.63	15.4
Shop Consumables	192,000	0.27	0.9
Mine Maintenance - Total	**3,510,000**	**4.90**	**16.3**
Total Operating Cost - By Area			
Production	11,550,000	16.12	53.6
Backfill	4,850,000	6.77	22.5
Mine General	1,624,000	2.27	7.5
Mine Maintenance	3,510,000	4.90	16.3
Operating Cost - Total	**21,534,000**	**30.06**	**100.0**

21.2.3.1 Mining Labour

Mining labour was calculated using the personnel numbers summarized in Section 16.10 of this report. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations. Table 21-15 summarizes the mining workforce labour rates.

Table 21-15: Mining Labour Rates

Group / Position		$/year	$/h
Mine Operations			
Mine Superintendent	Salary	176,800	-
Mine Captain	Salary	153,200	-
Mine Supervisor	Hourly	-	57.32
Production Drill Operator	Hourly	-	70.08†
Jumbo Operator	Hourly	-	70.08†
Ground Support	Hourly	-	70.08†
Development Services	Hourly	-	65.08†
Blaster	Hourly	-	70.08†
LHD Operator	Hourly	-	65.08†
Truck Driver	Hourly	-	60.08†
Backfill Labourers	Hourly	-	44.01†
Mine Labourers	Hourly	-	35.76
Paste Backfill Plant			
Paste Plant Operators	Hourly	-	52.15
Mining Maintenance			
Maintenance Superintendent	Salary	153,200	-
Maintenance Shift Supervisors	Salary	-	66.07
Maintenance Foreman	Salary	141,400	-
Maintenance Planner	Salary	129,600	-
Mechanics and Welders	Hourly	-	65.08
Electrician	Hourly	-	57.32
Bit and Lamp Man	Hourly	-	40.70
Mining Technical Services			
Chief Mining Engineer	Salary	141,400	-
Senior Mine Engineer	Salary	129,600	-
Mine Engineer	Salary	106,000	-
Ground Control Engineer	Salary	117,800	-
Senior Mine Technician	Salary	93,900	-
Surveyor/Mine Technician	Salary	93,900	-
Chief Geologist	Salary	129,600	-
Mine Geologist	Salary	106,000	-
Geotechnical Technician/Sampler	Salary	81,300	-

† Includes production bonus.

21.2.3.2 Equipment & Consumables

Drilling, mucking and hauling operating costs were developed from first principles from the mine plan and required equipment operating hours. Haulage profiles were developed for ore and waste rock to determine required haulage hours.

Equipment fuel and factored oil and lube consumption cost are based on Original Equipment Manufacturers (OEM) recommendations for the expected operating conditions. Diesel fuel pricing was provided by ARD Petroleum and includes delivery and off-loading at site.

Parts costs were provided by OEMs based on the life expectancy of the equipment. These include the following:

- major components (engine, torque converter, transmission, final drives, etc.)
- major hydraulic/suspension cylinders (suspension, hoist/steering cylinders, etc.)
- minor components (hydraulic pumps, motors, turbo chargers)
- all parts to remove and install components
- preventative maintenance (including filters, seals, screens, midlives)
- system parts (hydraulic, steering, transmission, cooling, cab, rear axle, suspension, brake, front axle, enclosures)
- scheduled and unscheduled repair parts
- hoses and fittings
- electrical wiring, sensors.

Life expectancy for major underground mining equipment is summarized in Table 21-16.

Table 21-16: Major Underground Equipment Life Expectancy

Equipment Type	Expected Life (hours)
Two Boom Jumbo	45,000
LH Drill	45,000
7 m^3 LHD with Remote	50,000
50 Tonne Truck	45,000
Mechanized Bolter	45,000
ANFO Loader	60,000

Tire replacement costs are included within the equipment unit rates and are based on expected tire life hours. Management of tires is considered to be of critical importance for the operation of the mine. Allowances for clean-up of drift floors and roadways, plus a grader, are included in mining costs. Table 21-17 summarizes the major underground equipment tire life expectancy, while major underground equipment operating costs per hour, excluding labour and drill tooling, are shown in Table 21-18.

Table 21-17: Major Underground Equipment Tire Life Expectancy

Equipment Type	Expected Life (hours)
7 m^3 LHD with Remote	1,750
50 tonne Truck	3,500

Table 21-18: Major Underground Equipment Hourly Operating Cost

Equipment Type	Fuel $/h	Oil/Lube $/h	Parts $/h	Tires $/h	Total $/h
Two Boom Jumbo	12.40†	4.34	5.00	1.25	22.99
LH Drill	13.64†	4.77	5.00	1.25	24.66
7 m^3 LHD with Remote	26.04	9.11	74.99	10.29	120.43
40 Tonne Truck	30.38	10.63	44.80	9.14	94.95
Mechanized Bolter	11.16†	3.91	6.00	1.25	22.32
ANFO Loader	6.82	2.39	10.00	0.50	19.71

* All cost data are presented in Q4 2012 dollars. † When tramming.

Consumables usage was based on required drift and stope services, explosives quantities, ground support patterns and drilling equipment tooling. Consumables usage by major drift and stope type are summarized in Table 21-19.

Table 21-19: Underground Mining Consumables Unit Cost

Drift/Stope Type	Ground Control	Services	Jumbo/ Bolter Drilling	LH Drilling	Explosives	Total
Ramp (5 m x 5.3 m)	$116.91/m	$266.32/m	$150.12/m	-	$396.63/m	$929.98/m
Ore Drift (5 m x 4 m)	$105.09/m	$143.81/m	$97.45/m	-	$266.93/m	$613.29/m
Level Access (4.6 m x 4.6 m)	$111.65/m	$249.17/m	$123.91/m	-	$337.97/m	$822.70/m
MCF Access (4 m x 4 m)	$90.40/m	$195.77/m	$87.41/m	-	$253.41/m	$626.99/m
LH Stoping	-	-	-	$0.36/t	$0.85/t	$1.21/t

* All cost data are presented in Q4 2012 dollars.

21.2.3.3 Paste Backfill

Paste backfill costs were based on an average cement content of 3% by weight. Other consumables use including flocculent, pipe, barricades and an allowance for emergency. Underground drilling was estimated from experience at similar mining operations utilizing paste backfill.

21.2.4 Processing Operating Costs

Operating costs for the 2,000 t/d concentrator plant were assembled using first principles and include the costs for process operations, maintenance, and technical service labour, as well all operating and maintenance supplies for both the process plant and the effluent treatment facility. The energy costs for the process plant are included within the power plant operating costs (estimated separately).

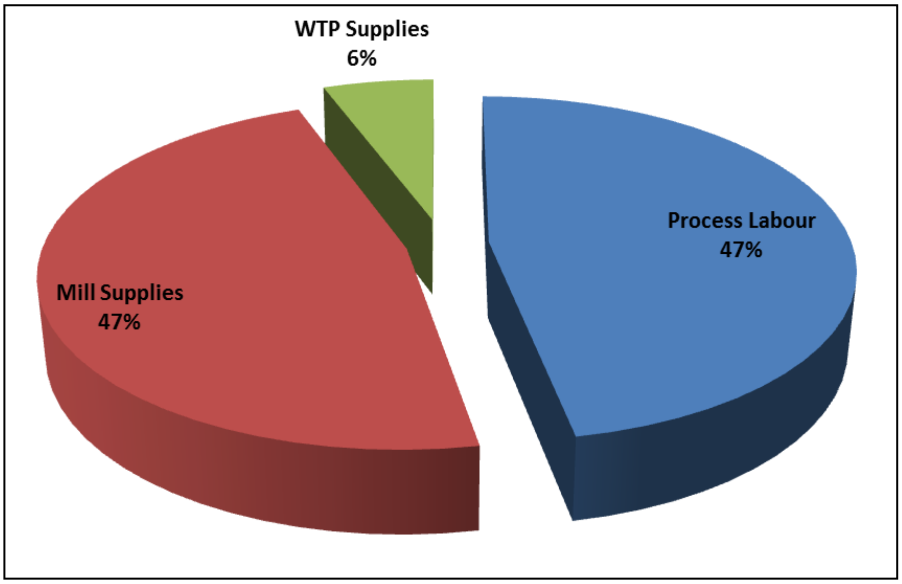
The estimated total processing operating unit cost is $23.02 per tonne processed; average annual operating costs and total unit costs are summarized in Table 21-20. Figure 21-9 presents the processing operating cost distribution.

Table 21-20: Average Processing Operating Costs

Cost Group / Item	Average Annual Operating Cost (C$)	Avg. Unit Operating Cost ($/t milled)	%
Process Labour			
Mill Operations	4,051,000	5.66	24.6
Water Treatment Plant Operations	476,000	0.66	2.9
Mill and WTP Maintenance	1,961,000	2.74	11.9
Process Technical Services	1,270,000	1.77	7.7
Process Labour - Total	**7,759,000**	**10.83**	**47.0**
Mill Supplies			
Contract Limestone Crushing/Stockpiling	135,000	0.19	0.8
Mill Operating Supplies	5,903,000	8.24	35.8
Mill Maintenance Supplies	1,747,000	2.44	10.6
Mill Supplies - Total	**7,785,000**	**10.87**	**47.2**
WTP Supplies			
Operating Supplies	889,000	1.24	5.4
Maintenance Supplies	58,000	0.08	0.4
WTP Supplies - Total	**948,000**	**1.32**	**5.7**
Total Operating Cost - By Area			
Process Labour	7,759,000	10.83	47.0
Limestone Quarry	135,000	0.19	0.8
Mill Supplies	7,649,000	10.68	46.4
WTP Supplies	948,000	1.32	5.7
Operating Cost - Total	**16,491,000**	**23.02**	**100.0**

* All cost data are presented in Q4 2012 dollars.

Figure 21-9: Processing Operating Costs Distribution



The subsections below further describe the basis of estimate for major items within each grouping.

21.2.4.1 Processing Labour

Processing labour includes a peak of 31 operations employees: four effluent treatment plant employees, 15 process maintenance employees, and 12 process technical services employees. The majority of the processing workforce will operate on a 2&2 rotation, with supervisory personnel (foreman and above) working 4&3 rotations. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations. Table 21-21 (overleaf) summarizes the processing workforce labour rates.

21.2.4.2 Operating & Maintenance Supplies

Reagent consumption rates for the concentrator and effluent treatment plant were determined through metallurgical testwork and water test reports, respectively. Consumption of grinding media and mill liners was based on vendor input and historical information for ore with similar work and abrasion indices. Quotations were received for all operating supplies.

Limestone crushing and stockpiling will be performed on a contract campaign basis approximately three times through the life of the mine. Costs were estimated based on typical contract rates for contract quarrying at similar operations.

Table 21-21: Processing Labour Rates

Group / Position		$/a	$/h
Processing Operations			
Mill Process Superintendent	Salary	176,800	-
Mill General Foreman	Salary	141,400	-
Mill Shift Foreman	Salary	129,600	-
Crushing Operator	Hourly	-	67.93
Crushing Helpers	Hourly	-	47.34
Grinding Mill Operators	Hourly	-	61.35
Gold Room/Gold Recovery Operators	Hourly	-	61.35
Control Room Operators	Hourly	-	68.35
Flotation Operators	Hourly	-	61.35
Concentrate Dewatering Operators	Hourly	-	61.35
Reagent Prep	Hourly	-	54.35
Tailing Delivery/General Labours	Hourly	-	40.08
Limestone Prep Plant Operators	Hourly	-	54.35
Effluent Treatment Plant Operations			
Effluent Treatment Plant Operations	Hourly	-	54.35
Process Maintenance			
Mill Maintenance Shift Foreman	Salary	141,400	-
Mechanics/Welders	Hourly	-	68.35
Mechanic Apprentice	Hourly	-	47.34
Lead Electrician (Shared)	Hourly	-	68.35
Electricians	Hourly	-	61.35
Electrical Apprentices	Hourly	-	47.34
Welders	Hourly	-	61.35
Instrument Technicians	Hourly	-	61.35
Crane / Equipment Operators (Shared)	Hourly	-	61.35
General Labour for Maintenance	Hourly	-	40.08
Process Technical Services			
Senior Metallurgical Engineer	Salary	141,400	-
Plant Metallurgist	Salary	117,800	-
Metallurgical Technicians	Salary	106,000	-
Chief Chemist	Salary	117,800	-
Assay Technicians	Salary	106,000	-
Sample Preparation	Hourly	-	40.08

* All cost data are presented in Q4 2012 dollars.

An allowance was made in each processing area for maintenance supplies, based on the capital cost and complexity of the equipment within each area. The total annual allowance for maintenance supplies between the concentrator and effluent plant is $1.8 M.

21.2.5 Power Plant Operating Costs

Operating costs for the power plant include the fuel and maintenance costs to provide energy for the entire mine operation. Power will be generated by four operating dual-fuel (LNG/diesel) generator sets. The average unit energy cost was estimated at $0.261/kWh.

The primary energy consumer on site is the concentrator plant, followed by the mining operations and then the site infrastructure. Table 21-22 summarizes the site energy consumption by area.

Table 21-22: Average Annual Energy Consumption, by Area

Area	Energy Consumption (kWh)	%
Underground Mine	14,826,000	23.9
Processing Facilities	36,112,000	58.2
Effluent Treatment Plant	1,664,000	2.7
Ancillary Buildings	9,443,000	15.2
Total	**62,045,000**	**100.0**

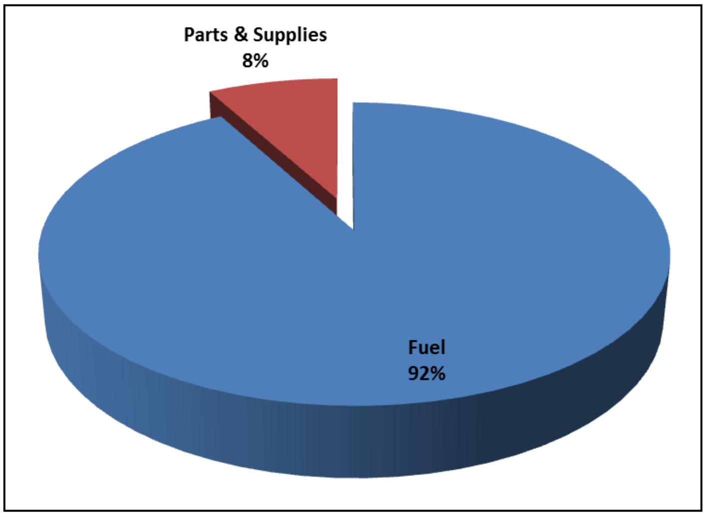
The total estimated power plant operating unit cost is $22.58 per tonne processed; average annual operating costs and total unit costs are summarized in Table 21-23. Figure 21-10 presents the power plant operating cost distribution.

Table 21-23: Average Power Plant Operating Costs

Cost Group / Item	Average Annual Operating Cost (C$)	Average Unit Operating Cost ($/t milled)	%
Fuel			
Diesel	6,387,000	8.93	39.5
LNG	8,495,000	11.84	52.5
Fuel - Total	**14,882,000**	**20.77**	**92.0**
Parts, Supplies & Contract Service			
Engine Oil	135,000	0.19	0.8
Engine Coolant	6,000	0.01	0.0
Grease	2,000	0.00	0.0
Regular Service Intervals	45,000	0.06	0.3
Top End Overhauls	510,000	0.71	3.2
In Frame Overhauls	593,000	0.83	3.7
Parts. Supplies & Service - Total	**1,291,000**	**1.80**	**8.0**
Total Operating Cost - By Area			
Fuel	14,882,000	20.77	92.0
Parts and Supplies	1,291,000	1.80	8.0
Operating Cost – Total	**16,173,000**	**22.58**	**100.0**

* All cost data are presented in Q4 2012 dollars.

Figure 21-10: Power Plant Operating Costs Distribution



The subsections below further describe the basis of estimate for major items within each grouping.

21.2.5.1 Fuel

Fuel consumption rates were determined using manufacturer consumption curves and account for the average load factor per operating year (average 79% loading). Seventy percent of the energy produced by the power plant will be via natural gas, the remaining 30% will be produced by diesel fuel. Average consumption rates are an estimated 435,000 GJ of LNG per year, and 4.8 ML of diesel per year.

Diesel and liquefied natural gas supply was quoted at $1.32/L and $19.50/GJ, respectively; the commodity rates include transport to site via the mine access road. Table 21-24 details the commodity rate buildup for LNG.

Table 21-24: LNG Commodity Rate Buildup

Component	**Unit Cost ($/GJ)**
Gas Cost	4.50
BC Carbon Tax	1.50
Liquefaction	5.50
Transportation (Delta, BC to site)	8.00
Total	**19.50**

* All cost data are presented in Q4 2012 dollars.

21.2.5.2 Parts, Supplies & Contract Service

Engine oil, coolant, and grease consumption rates were provided by the manufacturer. Quotations were received for all fluids.

Regular, top-end, and in-frame overhaul intervals were provided by the manufacturer accounting for the expected engine loadings. Regular service interval costs include parts only, as mill maintenance personnel (estimated within the processing operations area) will perform the regular maintenance. Top-end and in-frame overhaul costs include contract service labour and average costs for each service were provided by the manufacturer based on installations with similar infrastructure.

21.2.6 General & Administrative Operating Costs

G&A operating costs are grouped into the following categories:

- G&A Labour
- Camp Catering and Cleaning
- Mine Access Road Maintenance
- Miscellaneous On-Site Items
- Support Equipment Fleet
- Satellite Offices
- General Freight
- Employee Travel.

The total G&A operating unit cost is an estimated $22.47 per tonne processed; average annual operating costs and total unit costs are summarized in Table 21-25 (overleaf). Figure 21-11 below presents the G&A operating cost distribution.

Figure 21-11: G&A Operating Costs Distribution

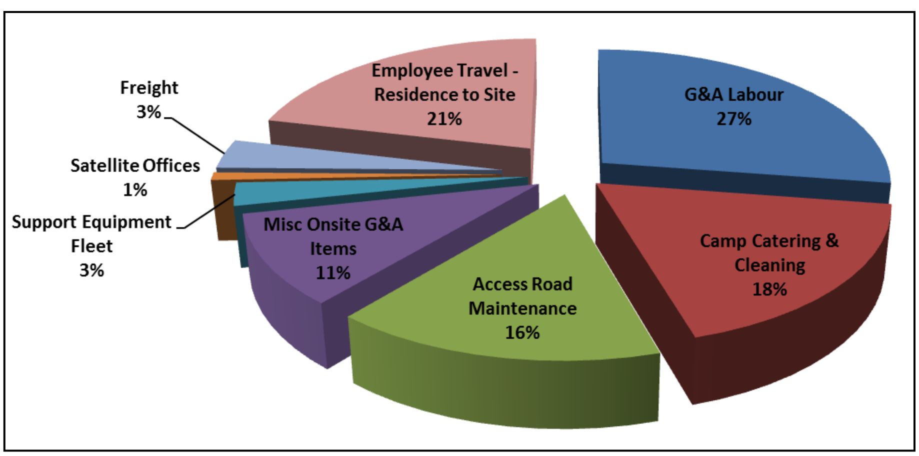


Table 21-25: Average G&A Annual Operating Costs

Cost Group / Item	Average Annual Operating Cost (C$)	Average Unit Operating Cost ($/t milled)	%
G&A Labour			
Surface Infrastructure and Maintenance	1,689,000	2.36	10.5
First Aid	283,000	0.39	1.8
Environmental	438,000	0.61	2.7
Administration	929,000	1.30	5.8
HSE - Health and Safety	236,000	0.33	1.5
Human Resources	377,000	0.53	2.3
IT and Communications	141,000	0.20	0.9
Security	317,000	0.44	2.0
G&A Labour - Total	**4,410,000**	**6.16**	**27.4**
Camp Catering & Cleaning			
Camp Catering and Cleaning	2,912,000	4.07	18.1
Camp Catering & Cleaning – Total	**2,912,000**	**4.07**	**18.1**
Access Road Maintenance			
Access Road Maintenance	2,511,000	3.50	15.6
Access Road Maintenance - Total	**2,511,000**	**3.50**	**15.6**
Miscellaneous On-Site G&A Items			
Health and Safety, Medical, and First Aid	256,000	0.36	1.6
Environmental	176,000	0.25	1.1
Human Resources	84,000	0.12	0.5
Land and Permitting	30,000	0.04	0.2
Insurance and Legal	732,000	1.02	4.5
External Consulting	78,000	0.11	0.5
IT and Communications	152,000	0.21	0.9
Office and Miscellaneous Costs	155,000	0.22	1.0
Housing and Car Allowance	24,000	0.03	0.1
R&D Allowance	10,000	0.01	0.1
Miscellaneous On-Site G&A - Total	**1,696,000**	**2.37**	**10.5**
Support Equipment Fleet			
Construction Equipment	237,000	0.33	1.5
Lifting and Support Equipment	67,000	0.09	0.4
Buses	25,000	0.04	0.2
Vans	27,000	0.04	0.2
Pickups	72,000	0.10	0.4
Emergency Vehicles	8,000	0.01	0.1
Support Equipment Fleet - Total	**437,000**	**0.61**	**2.7**
Satellite Offices			
Atlin Office	45,000	0.06	0.3
Whitehorse Office	103,000	0.14	0.6
Support Equipment Fleet - Total	**148,000**	**0.21**	**0.9**
General Freight			
Ground Freight	557,000	0.78	3.5
General Freight - Total	**557,000**	**0.78**	**3.5**

Cost Group / Item	Average Annual Operating Cost (C$)	Average Unit Operating Cost ($/t milled)	%
Employee Travel			
Residence to YXY (Commercial)	2,292,000	3.20	14.2
YXY to Site (Charter)	1,132,000	1.58	7.0
General Freight - Total	**3,424,000**	**4.78**	**21.3**
Total G&A Operating Costs			
G&A Labour	4,410,000	6.16	27.4
Camp Catering and Cleaning	2,912,000	4.07	18.1
Access Road Maintenance	2,511,000	3.50	15.6
Miscellaneous On-site G&A Items	1,696,000	2.37	10.5
Support Equipment Fleet	437,000	0.61	2.7
Satellite Offices	148,000	0.21	0.9
Freight	557,000	0.78	3.5
Employee Travel - Residence to Site	3,424,000	4.78	21.3
Total G&A Operating Cost	**16,094,000**	**22.47**	**100.0**

* All cost data are presented in Q4 2012 dollars.

The subsections below further describe the basis of estimate for major items within each grouping.

21.2.6.1 G&A Labour

G&A labour includes 36 employees, which are summarized by functional grouping in Table 21-26.

Table 21-26: Planned G&A Operations Workforce

Department	Total Persons Employed (peak)
Surface Infrastructure and Maintenance	13
First Aid/Site Security	2
Environmental	3
Administration	8
HSE – Health and Safety	2
Human Resources	3
IT and Communications	1
Access Road Security	4
Total	**36**

G&A labour includes a blend of on and off-site positions. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations. Table 21-27 summarizes the G&A workforce labour rates.

Table 21-27: G&A Labour Rates

Group / Position	Location	Hourly/Salary	$/a	$/h
Surface Infrastructure & Maintenance				-
Maintenance Superintendent	Site	Salary	165,000	-
Chief Electrician	Site	Salary	153,200	-
Surface Foreman	Site	Hourly	-	68.35
Mechanic - Surface Shops	Site	Hourly	-	61.35
Labourer - Surface Shops	Site	Hourly	-	47.34
Mobile Equipment Operator	Site	Hourly	-	61.35
First Aid/Site Security				
Nurse/First Aid/Security	Site	Salary	141,400	-
Environment				
Sustainability Manager	Site	Salary	153,200	-
Environmental Technician	Site	Hourly	-	68.25
Administration				
Mine/General Manager	Site	Salary	188,600	-
Controller/Accountant	Off site	Salary	129,600	-
Payroll Supervisor	Site	Salary	117,800	-
Transport/Dispatch Supervisor	Site	Salary	117,800	-
Warehouse Clerk/Tech	Site	Salary	93,900	-
Health & Safety				
Safety Training Coordinator	Site	Salary	117,800	-
Human Resources				
Human Resources Manager	Off site	Salary	141,400	-
Human Resources Clerk	Off site	Salary	106,000	-
Community Relations Coordinator	Off site	Salary	129,600	-
IT & Communications				
IT/Telecom. Technician	Site	Salary	141,400	-
Security				
Protective Services Officials	Atlin	Hourly	-	36.13

* All cost data are presented in Q4 2012 dollars.

21.2.6.2 Camp Catering

Camp catering costs were calculated based on the estimated average daily camp loading (with an allowance for specialty contractors and visitors) and proposals received from camp catering contractors.

21.2.6.3 Access Road Maintenance

The Mine Access Road will be maintained on a contract basis from both the Site and Atlin sides. Costs were estimated using first principles and current contractor equipment and labour rates. Approximately 30% of the annual Access Road maintenance costs are associated with avalanche control.

21.2.6.4 Miscellaneous On-Site Items

The miscellaneous G&A grouping includes 47 small items within the following areas:

- Health and Safety, Medical, and First Aid
- Environmental
- Human Resources
- Land and Permitting
- Insurance and Legal
- External Consulting
- IT and Communications
- Office and Miscellaneous Costs
- Housing and Car Allowance
- R&D Allowance.

The single largest expense within this grouping is operations insurance, estimated at $646,560 per year.

21.2.6.5 Support Equipment Fleet

Costs for fuel and maintenance for each piece of support equipment were estimated based on an allowance of operating hours per year.

21.2.6.6 Satellite Offices

Small satellite offices will be established in Atlin and Whitehorse, primarily to support Human Resources, payroll, and accounting. Costs were estimated for the building lease, building maintenance, and office supplies for each location.

21.2.6.7 General Freight

General freight includes trucking costs for the following items:

- bulk cement
- bulk mining supplies
- mining equipment purchased during operations
- process plant supplies
- miscellaneous (allowance).

An average of 14,500 t/a of trucking was estimated for these items. When possible, these supplies will be backhauled from the port of Skagway on the empty concentrate haul trucks. An allowance was added for freight of items that will likely be sourced outside of Skagway; primarily mining equipment and process plant supplies (operating spare parts).

Trucking costs associated with the following items are included elsewhere in the operating cost estimate:

- Diesel fuel is included in commodity rates.
- Liquefied natural gas is included in commodity rates.
- Reagents are included in commodity rates.
- Concentrate is estimated separately (refer to Section 21.2.7).

An allowance for airfreight was not included, it has been assumed that any supplies that will be transported by air (primarily fresh foods) will be loaded in passenger planes, and thus the costs are considered incidental to the passenger transportation costs.

21.2.6.8 Passenger Transportation

All workers will be transported to site via charter aircraft departing from Erik Nielson International Airport (YXY) in Whitehorse. Employees who do not reside in the Whitehorse area will be transported to YXY via commercial airline in economy class. An estimated annual average of 2,292 commercial flights and 306 charter flights will be required during operations.

The required commercial flight schedule was calculated based on shift durations of the various employees and assumes the following:

- The camp catering contract labour will be sourced in the Whitehorse area and will not require commercial flights.
- All employees on 4&3 rotation will reside in the Whitehorse area and will not require commercial flights.
- 15% of the remaining employees will reside in the Whitehorse area and will not require commercial flights.

Charter flight costs were based on contractor quotations for the utilization of a 19 passenger plane, and the average round-trip rate includes the initial costs for any required repositioning. The required charter flight schedule was calculated based on shift durations of the various employees (with an allowance for contractors). An 80% capacity factor was included to allow for cancellations and visitors.

21.2.7 Concentrate Transportation Costs

Concentrate will be hauled to Atlin via off-highway tractor and then transferred to highway B-trains to continue to the port of Skagway, Alaska. Concentrate will then be loaded onto ocean barge and transported overseas. Costs were estimated through obtaining vendor quotations for the road transport portion of the concentrate haul and through negotiations with the Port of Skagway for container and port charges. Concentrate transportation costs also include a component for the improvement of the Skagway facility, which is required to accept concentrates for the project.

The total concentrate haulage unit cost is an estimated $27.57 per tonne processed. Average annual operating costs and total unit costs are summarized in Table 21-28 on the following page. Figure 21-12 presents the concentrate haulage cost distribution by area.

21.2.8 Contingency

No separate operating contingency was applied to the operating cost estimates; however, several allowances are built into the operating cost estimate through conservative estimating:

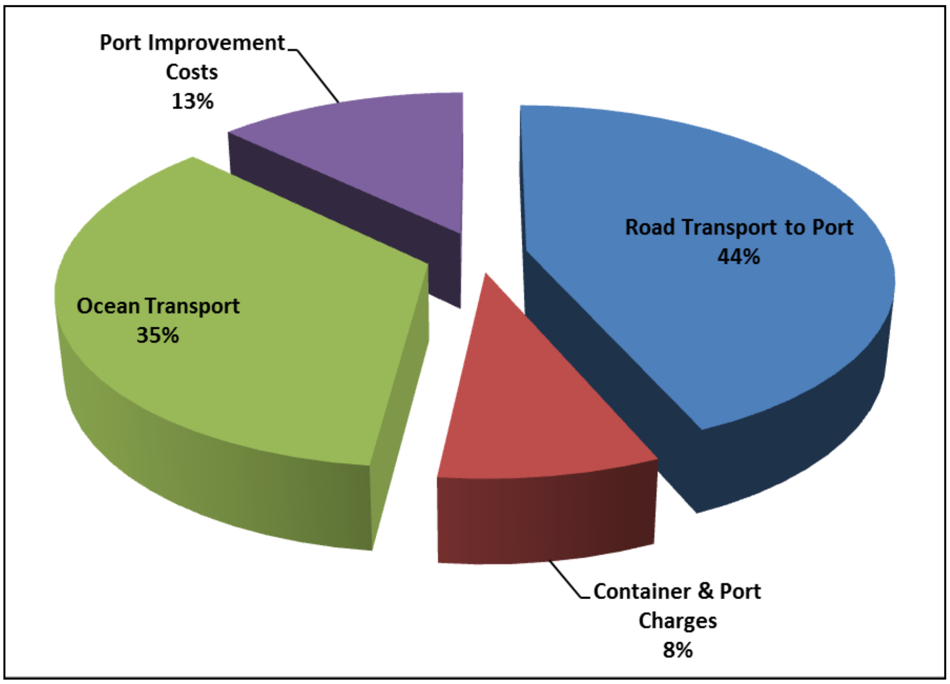
- A conservative estimate was used for grinding media consumption and there is opportunity to reduce this once operations begin.
- Reagent consumptions were based on lab testwork and scaled up for the full sized process plant. There is an opportunity to reduce these consumption rates once operations begin.
- Once the power plant is operational, it is possible that the fuel mixture of 30% diesel could be somewhat reduced, thereby reducing the operating cost of the power plant.
- Camp power consumption includes electric heat. It may be possible to heat the camp via the heat recovery system, thereby reducing power plant operating costs.

Table 21-28: Average Annual Concentrate Haulage Costs

Cost Group / Item	Average Annual Operating Cost (C$)	Average Unit Operating Cost ($/t milled)	%
Road Transport to Port	8,671,000	12.10	43.7
Container and Port Charges	1,656,000	2.31	7.7
Ocean Transport	7,045,000	9.83	35.5
Port Improvement Costs	2,564,000	3.58	13.1
Total	**19,936,000**	**27.83**	**100.0**

* All cost data are presented in Q4 2012 dollars.

Figure 21-12: Concentrate Haulage Costs Distribution



22 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variation in metal prices, operating costs, capital costs and discount rates to determine their relative importance as project value drivers. The economic analysis presented does not include financial securities that have been posted by Chieftain for the Tulsequah project with respect to permitting.

This technical report contains forward-looking information regarding mine production rates, construction schedule and forecast of resulting cash flows. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Sections 20 and 21 of this report in 2012 dollars. The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

Three metal price and exchange rate scenarios were evaluated to estimate the economic value potential of each and to use the results as a comparative tool to better understand the value drivers in each scenario. All costs and economic results are reported in Canadian dollars (C$), unless otherwise noted, while metal pricing is reported in US dollars (US$). All three cases use identical life of mine (LOM) plan tonnage and grade estimates (Table 22-1). On-site and off-site costs and production parameters were also held constant for each case.

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

- discount rate of 8% (sensitivities using other discount rates have been calculated for each scenario)
- reclamation costs of $13.8 M
- salvage value of $7.6 M
- nominal 2012 dollars
- no inflation

- revenues, costs, taxes are calculated for each period in which they occur rather than the actual outgoing/incoming payment
- working capital calculated as three months of operating costs in the first year of production; this amount is recuperated at the end of the mine life
- numbers are presented on 100% ownership and do not include management fees or financing costs
- exclusion of all pre-development and sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.); however, pre-development and sunk costs are used in tax calculations to determine the project's taxable income.

Table 22-1: LOM Plan Summary for all Cases

Component	Units	Value
LOM Tonnes	Mt	6.45
LOM Grade		
Gold	g/t	2.30
Silver	g/t	81.38
Zinc	%	5.59
Copper	%	1.12
Lead	%	1.04
LOM Payable Metals		
Gold (pre-streaming)	koz	403.9
Silver (pre-streaming)	koz	11,971.8
Zinc	Mlbs	601.5
Copper	Mlbs	135.5
Lead	Mlbs	93.0

Table 22-2 outlines the metal prices and exchange rate assumptions that were used in the economic analysis. Metal streaming contract prices are not included in the table.

Table 22-2: Non-Streamed Metal Prices & Exchange Rates (as of October 31, 2012) by Case

		Base Case	Case 2	Case 3
Parameter	**Units**	**Three-Year Trailing Average as of Oct 31, 2012**	**Two-Year Trailing Average as of Oct 31, 2012**	**Consensus Economics, Long-term Forecast Prices based on 5 Major Canadian Banks (National Bank, October 2012)**
Gold price	US$/oz	1,455.00	1,592.00	1,338.00
Silver price	US$/oz	28.00	33.00	22.00
Copper price	US$/lb	3.66	3.85	2.95
Zinc price	US$/lb	0.97	0.96	1.09
Lead price	US$/lb	1.01	1.03	1.02
Exchange Rate	C$:US$	1.01	1.00	1.06

22.2 Mine Production Statistics

Mine production is reported as the ore and waste material resulting from the mining operation. Annual production figures were obtained from the mine plan developed for this study. The LOM ore and waste quantities and head grades are presented in Table 22-3.

Table 22-3: LOM Mine Production & Head Grades

Parameter	Unit	Value
Waste	Mt	1.23
Ore	Mt	6.45
Head Grade		
Au	g/t	2.30
Ag	g/t	81.38
Zn	%	5.59
Cu	%	1.12
Pb	%	1.04

22.3 Revenues & NSR Parameters

Mine revenue is derived from the sale of concentrates and doré into the international marketplace. No contractual arrangements for concentrate smelting or refining exist at this time, however, preliminary market studies on the potential concentrate sales were completed by independent participants who have provided Chieftain with indicative terms and an analysis of the market conditions specific to the anticipated Tulsequah concentrate characteristics. These details can be found in Section 19 of this report. Concentrate production and sale of concentrate is assumed to begin in 2016 for a period of nine years, ending in 2024. Tables 22-4 to 22-7 indicate the NSR parameters that were used in the economic analysis. Table 22-8 and Figures 22-1 to 22-3 show the amount of concentrate produced during the mine life.

In addition to the sale of concentrate and doré, the economic analysis incorporates Chieftain's streaming contract with Royal Gold Inc. This contract was signed with Royal Gold in December 2011, details can be found in Section 19.0 of this report.

Figure 22-4 demonstrates the breakdown of revenues LOM by concentrate and incorporates revenues received from Chieftain's streaming contract. Total revenues for the base case amounted to C$1,713 M. Total NSR for the base case amounted to C$1,626 M.

Table 22-4: NSR Parameters used in Economic Analysis – Zinc Concentrate

Parameter	Unit	Value
Flotation Recovery		
Zinc Recovery	%	89.0
Silver Recovery	%	7.8
Gold Recovery	%	3.0
Concentrate Grade		
Zinc	%	62.0
Moisture Content	%	8.0
Smelter Payables		
Zinc Payable	%	85.0
Min. Zinc Deduction	%	8.0
Silver Payable	%	0.00
Gold Payable	%	0.00
Treatment/Refining Charges	US$/dmt concentrate	145.00
Transport Costs		
Transport to Port	US$/wmt	80.00
Port Charges	US$/wmt	14.02
Ocean Freight	US$/wmt	65.00
Building Capital (at Port)	US$/wmt	23.65
Subtotal	**US$/wmt**	**182.68**
	US$/dmt	**197.29**
Escalator (Base Case)	US$/dmt	5.54
Escalator (Case 2)	US$/dmt	4.66
Escalator (Case 3)	US$/dmt	16.24

Table 22-5: NSR Parameters used in Economic Analysis – Copper Concentrate

Parameter	Unit	Value
Flotation Recovery		
Copper Recovery	%	89.0
Silver Recovery	%	75.0
Gold Recovery	%	44.0
Concentrate Grade		
Copper	%	21.0
Moisture Content	%	8.0
Smelter Payables		
Copper Payable	%	96.5
Minimum Copper Deduction	% copper/tonne	1.0
Silver Payable	%	90.0
Minimum Silver Deduction	g/t in concentrate	30.0
Gold Payable	%	92.0
Minimum Gold Deduction	g/t in concentrate	0.0
Treatment Charge	US$/dmt concentrate	125.00
Silver Refining Charge	US$/payable oz	1.50
Gold Refining Charge	US$/payable oz	25.00
Penalty for deleterious elements	US$/dmt concentrate	52.20
Transport Costs		
Transport to Port	US$/wmt	80.00
Port Charges	US$/wmt	14.02
Ocean Freight	US$/wmt	65.00
Building Capital (at Port)	US$/wmt	23.65
Subtotal	**US$/wmt**	**182.68**
	US$/dmt	**197.29**
Escalator (Base Case)	US$/dmt	0.00
Escalator (Case 2)	US$/dmt	0.00
Escalator (Case 3)	US$/dmt	0.00

Table 22-6: NSR Parameters used in Economic Analysis – Lead Concentrate

Parameter	Unit	Value
Flotation Recovery		
Lead Recovery	%	66.2
Silver Recovery	%	9.0
Gold Recovery	%	4.0
Concentrate Grade		
Lead	%	60.0
Moisture Content	%	8.0
Smelter Payables		
Lead Payable	%	95.0
Minimum Lead Deduction	% lead/tonne	3.0
Silver Payable	%	95.0
Minimum Silver Deduction	g/t in concentrate	50.0
Gold Payable	%	95.0
Minimum Gold Deduction	g/t in concentrate	1.5
Treatment Charge	US$/dmt concentrate	100.00
Silver Refining Charge	US$/payable oz	1.50
Gold Refining Charge	US$/payable oz	25.00
Penalty for deleterious elements	US$/dmt concentrate	0.00
Transport Costs		
Transport to Port	US$/wmt	80.00
Port Charges	US$/wmt	17.77
Ocean Freight	US$/wmt	65.00
Building Capital (at Port)	US$/wmt	23.65
Subtotal	**US$/wmt**	**186.43**
	US$/dmt	**201.34**
Escalator (Base Case)	US$/dmt	18.13
Escalator (Case 2)	US$/dmt	21.66
Escalator (Case 3)	US$/dmt	19.90

Table 22-7: NSR Parameters used in Economic Analysis – Gold & Silver Doré

Parameter	Unit	Value
Recovery		
Gold	%	42.0
Silver	%	0.50
Smelter Payables		
Gold	%	98.0
Minimum Gold Deduction	Units	0.0
Silver	%	98.0
Minimum Silver Deduction	Units	0.0
Refining Charge		
Gold	US$/payable oz	6.00
Silver	US$/payable oz	1.50

Table 22-8: LOM Concentrate Production

Concentrate	Unit	Value (000s)
Zinc	dmt	517.7
Copper	dmt	307.3
Lead	dmt	74.0
Total	**dmt**	**899.0**

Figure 22-1: Concentrate Production by Year

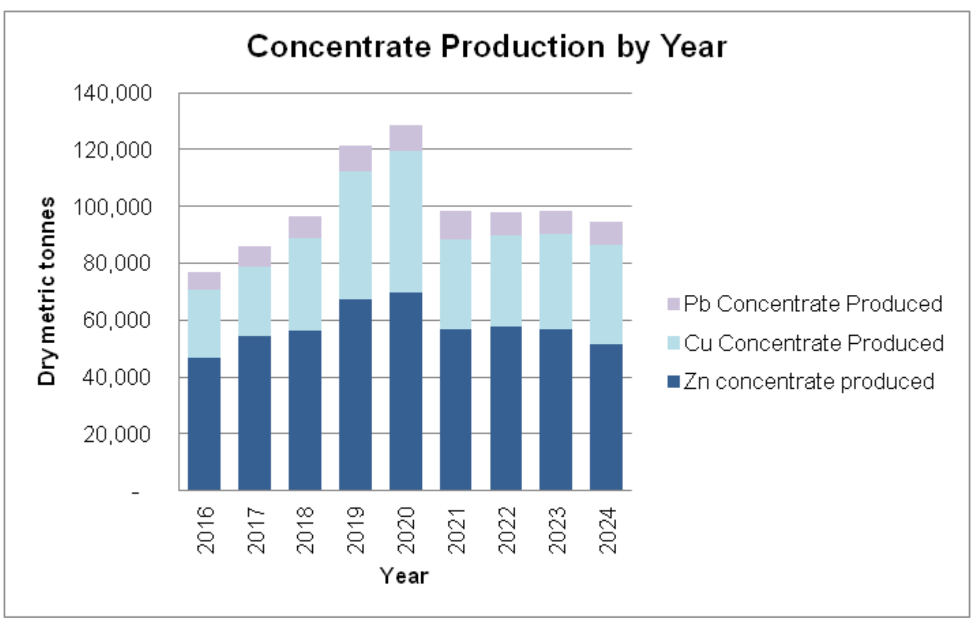


Figure 22-2: Base Case LOM NSR by Concentrate & Doré Production

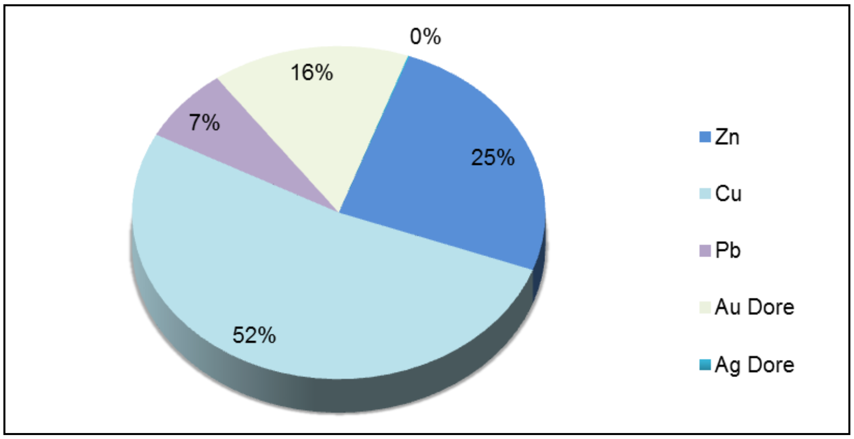


Figure 22-3: LOM NSR Value by Metal Produced

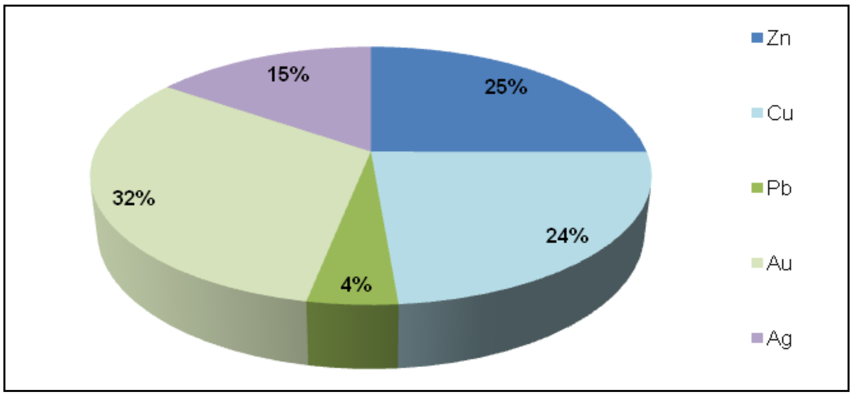
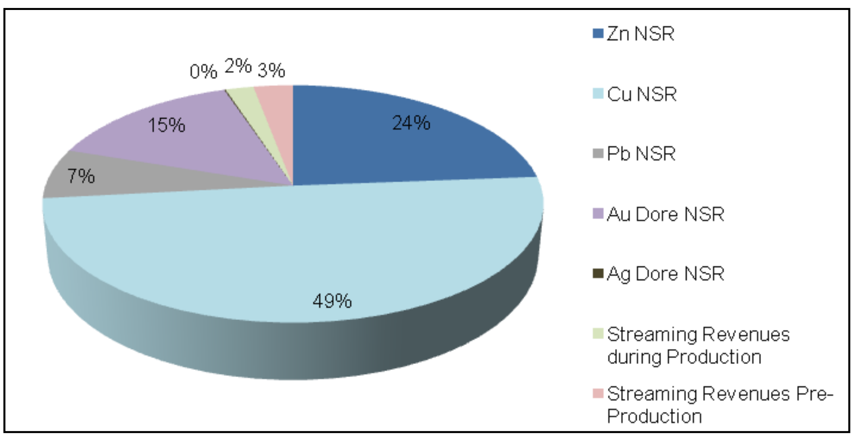


Figure 22-4: LOM Revenues by Concentrate – Base Case (incl. from Streaming Contract)



22.4 Summary of Capital Costs

The capital costs listed below were used for the economic analysis. Detailed information can be found in Section 21.

- From 2013 to 2015, the preproduction capital costs amount to C$439.5 M. This includes costs for site development, processing plant, on-site infrastructure, mine access road, etc. A blended 12% contingency is included in this total amount.
- Sustaining capital costs amount to C$64.0 M and occur from 2016-2024. These costs account primarily for development and underground mining equipment.
- Reclamation costs amount to C$13.8 M with most of the cost occurring in 2025.
- The salvage value of the project totals C$7.6 M.

22.5 Summary of Operating Costs

Total operating costs amount to C$812.1 M including transport costs. This translates to an average cost of $125.96/t processed over the life of mine. These costs are shown in Table 22-9.

Table 22-9: Summary of Operating Costs

Area	$/t processed	LOM Cost $M
Mining	30.06	193.8
Processing	23.02	148.4
Power	22.58	145.6
G&A	22.47	144.9
Transport	27.83	179.4
Total	**125.96**	**812.1**

Transport costs were accounted for in the NSR calculation as the costs are driven by the amounts of concentrate produced. Detailed operating cost information can be found in Section 21 of this report.

22.6 Taxes

The project has been evaluated on an after-tax basis to reflect a more indicative, but still approximate, value of the project. Both BC Mineral Tax and Federal and Provincial Income Tax rates were applied to the project. A detailed tax analysis was completed by independent tax consultants for the purpose of the after-tax valuation of the project.

A detailed tax analysis was completed specifically to evaluate the Tulsequah Chief project. Specific assumptions and methodology in the analysis includes the following:

- BC Mineral Tax:
  - The BC Mineral tax is comprised of two tiers. Tier 1 Tax is 2% of net current proceeds defined as (the current year's gross revenue less operating costs). Operating costs are all current operating costs, but do not include expenses due to capital investment such as preproduction exploration and development expenses. If the mine has an operating loss, no net current proceeds tax (Tier 1 Tax) is payable.
  - After the company's investment plus as extra 33.3% for the new mine allowance, the company must pay the Tier 2 Tax of 13% of adjusted net revenue, essentially the net current proceeds from Tier 1 Tax computations from the mine. The Tier 1 Tax is deducted from the Tier 2 Tax owed, so the maximum tax does not exceed 13%. Any previous Tier 1 Tax paid is creditable against the Tier 2 Tax owed. It can be carried forward indefinitely.
- Federal and Provincial Corporate Income Tax – Federal tax rate of 15.0% and a BC tax rate of 10.0% were used to determine a blended 25.0% rate that was used to calculate income taxes.
- Mineral Property Tax Pools – Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with appropriate opening balances to calculate income taxes.
- Federal Investment Tax Credits – Appropriate opening balances were used to calculate the Federal Investment Tax Credits for the project with respect to the preproduction capital costs of the project.
- Capital Cost Allowance (CCA) – Capital cost specific CCA rates were applied to and used to calculate the appropriate amount of CCA the Company can claim during the life of the project.
- Streaming Revenues – Streaming revenues were adjusted according to income tax regulations to determine the taxable income for the project.
- Corporate Expenses – To provide a more accurate indicative value of the project on an after-tax basis, taxable income was adjusted to include corporate overhead expenses that will be incurred in the project over the life of mine. These costs were calculated by Chieftain and amounted to $168.1 M LOM. Since Chieftain is primarily a one-asset company, this assumption was deemed reasonable for the purpose of calculating the taxable income. These costs were not treated as project costs when determining the net present value of the project cash flows.

The tax analysis completed amount to a LOM taxes payable of $119.5 M. The after-tax values are determined solely for project valuation purposes.

22.7 Streaming Contract with Royal Gold

In December 2011, Chieftain entered into a gold and silver purchase transaction with Royal Gold to sell a portion of the precious metals produced at the Tulsequah Chief mine (see Section 19 for details). Under the agreement, Chieftain received an up-front payment of US$10 M (at closing – December 2011) and will receive an additional US$50 M for the project build (upon certain conditions being met) that will be pro-rated during the development of the project. A summary of the streaming contract with Royal Gold is shown in Table 22-10. Figures 22-5 through 22-11 further depict the breakdown of gold and silver production and revenue.

Table 22-10: Summary of Streaming Contract with Royal Gold

Area	Unit	Silver	Gold
Precious metal sold via streaming contract	oz (000)	2,693.7	49.5
Total revenue from streaming during production	US$M	13.5	22.3
Total precious metal available (pre-streaming)	oz (000)	11,971.8	403.9
Total precious metal uncommitted (post-streaming)	oz (000)	9,278.1	354.4

Figure 22-5: Sale of Total Silver Produced LOM

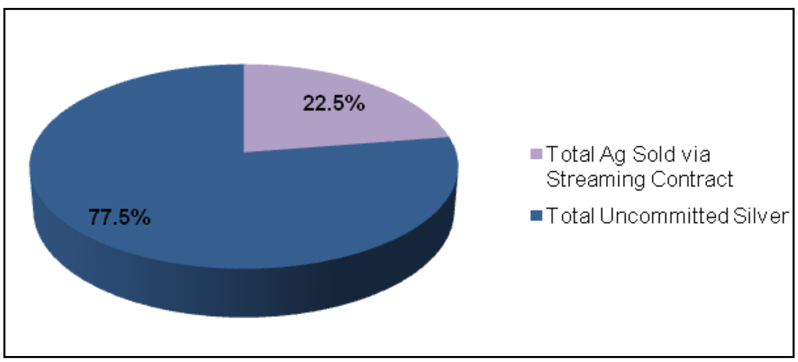


Figure 22-6: Sale of Total Gold Produced LOM

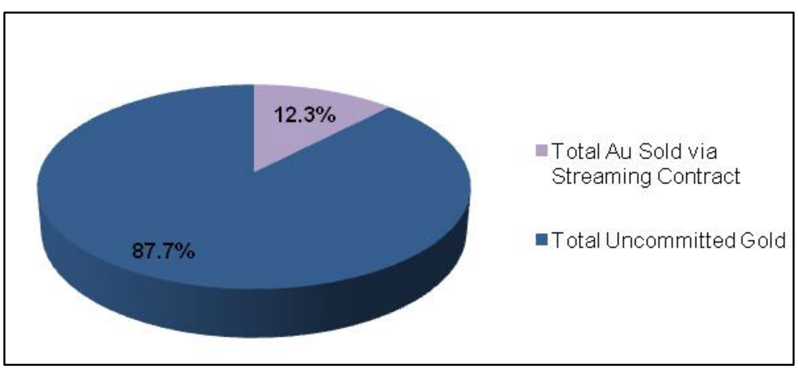


Figure 22-7: Cash Revenues from Streaming Contract by Precious Metal

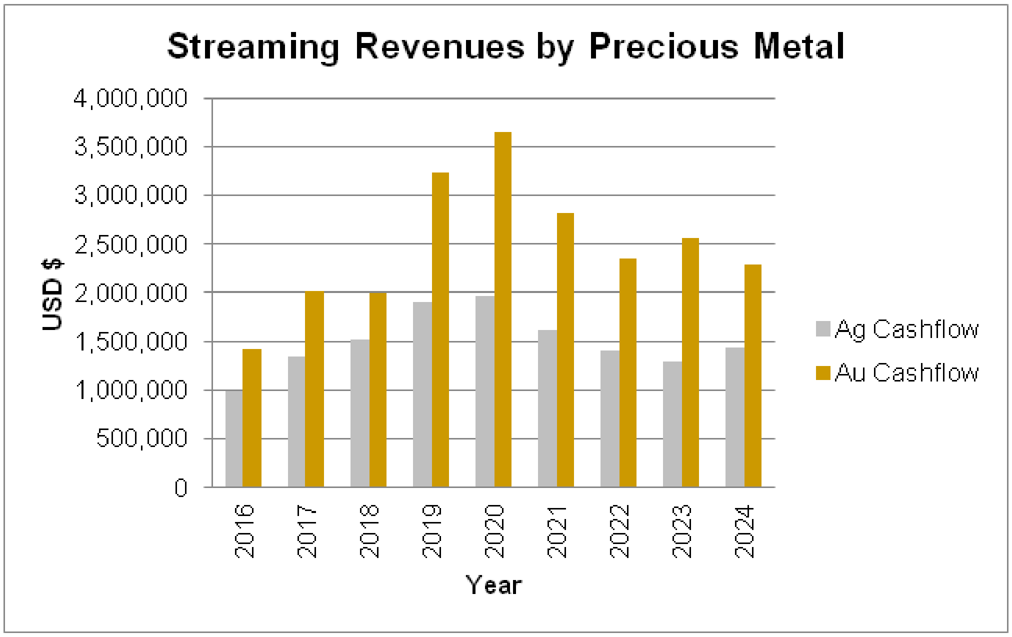


Figure 22-8: Total Silver Sold Through Streaming Contract

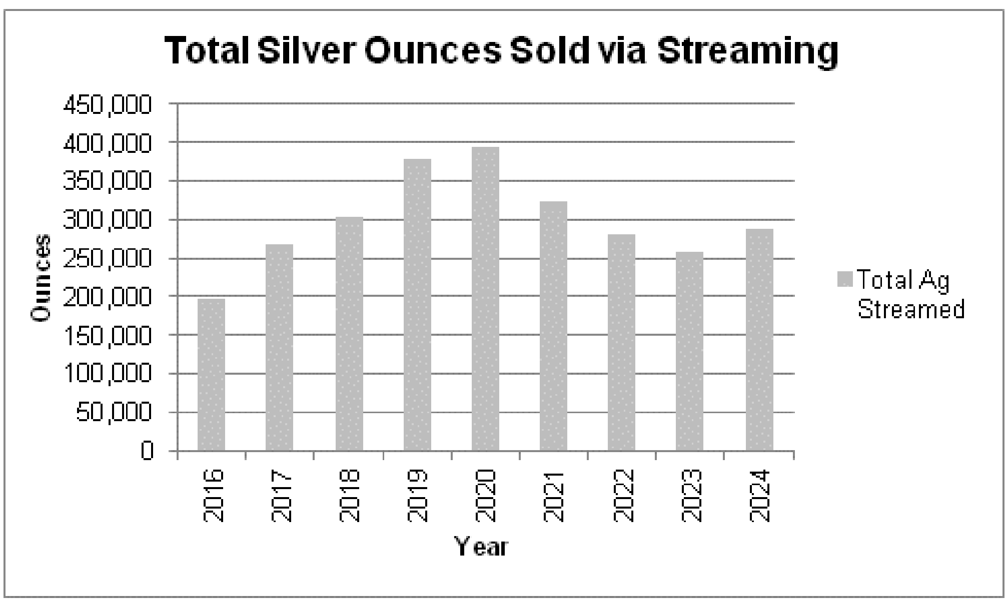


Figure 22-9: Source of Silver Sold Through Streaming Contract

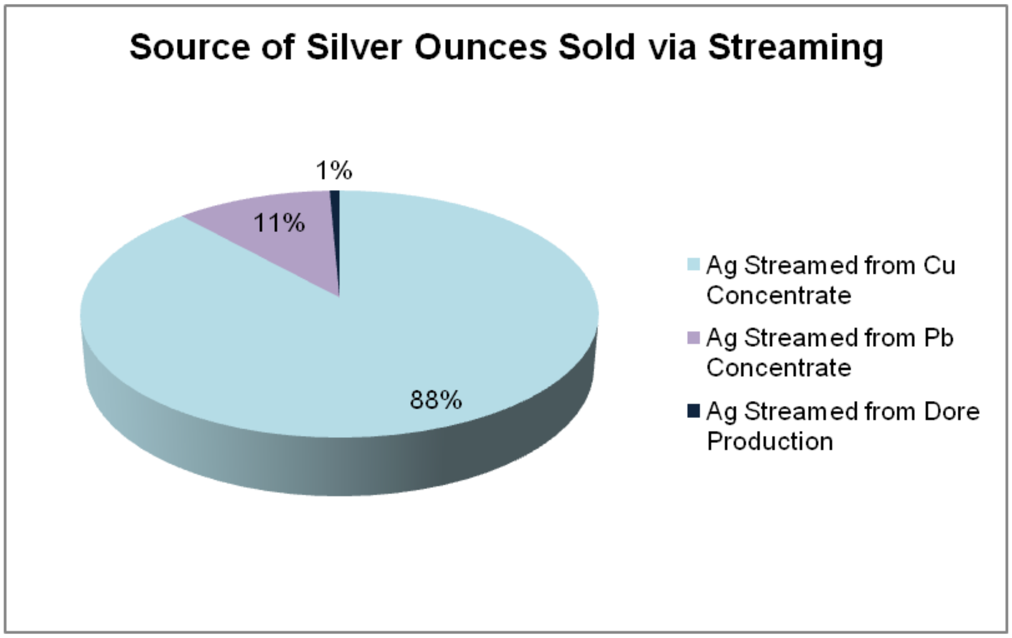


Figure 22-10: Total Gold Sold Through Streaming Contract

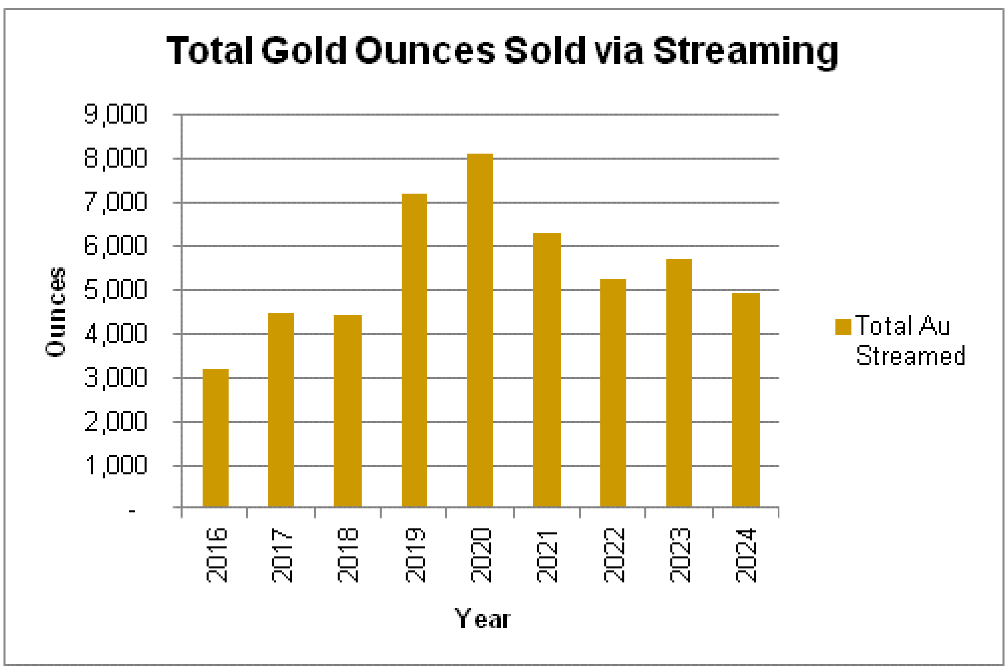
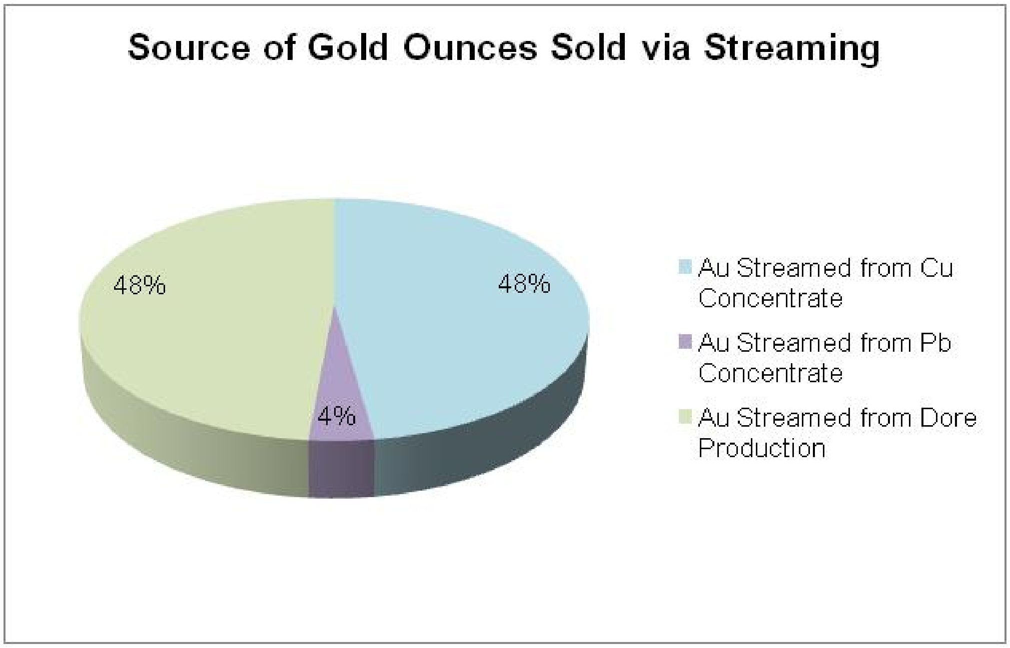


Figure 22-11: Source of Gold Sold Through Streaming Contract



22.8 Economic Results

The project is economically viable with an after-tax internal rate of return (IRR) of 14.7% and a net present value at 8% ($NPV_{8\%}$) of C$138.7 M for the Base Case, which was calculated based on three-year trailing average metal prices and three-year average CAD:USD exchange rate as of October 31, 2012. Two additional cases were measured based on the two-year trailing average of metal prices and two-year exchange rate (Case 2), and projected long-term metal prices published by Consensus Economics' October 2012 report with exchange rates provided to Chieftain by National Bank identifying the long-term forecast of the five major banks in Canada (Case 3).

Case 2 resulted in the highest performance and project value due to the use of two-year trailing average metal prices. These prices are higher for all of the other cases used in this study. The calculated Base Case resulted in the second highest project value of all three cases primarily due to the higher precious metal prices used compared to Case 3. Case 3, using the long-term metal prices, has the highest zinc price of all three cases, however, the higher price of zinc was not sufficient to compensate for the lower prices of all other metals.

Figure 22-12 shows the projected cash flows for the project used in the economic analysis. Tables 22-11 to 22-13 show the economic results of each of the three cases calculated. In addition, Tables 22-14 and 22-16 show the sensitivity of pre-tax and after-tax NPV to discount rates for each of the scenarios.

Figure 22-12: Annual & Cumulative Cash Flows for the Project Life during Preproduction & Production (C$)

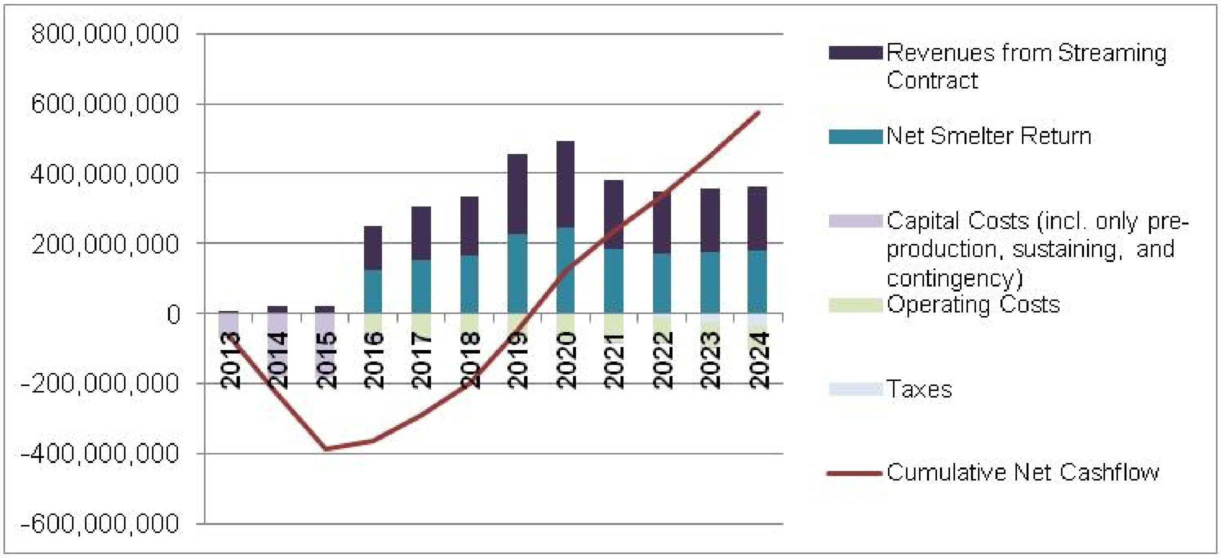


Table 22-11: Summary of Base Case Economic Results

Category	LOM Base Case C$M
Net revenue (Net Smelter Return + Streaming Revenues)	1,713.0
Operating costs (excluding transport costs)	632.6
Cash flow from operations	1,080.4
Capital (incl. sustaining capital, contingency, reclamation, salvage)	509.8
Net profit	570.7
Pre-Tax IRR (%)	16.5
Pre-Tax $NPV_{8\%}$	192.7
Pre-Tax Payback Period	4.3
After-Tax IRR (%)	14.7
After-Tax $NPV_{8\%}$	138.7
After-Tax Payback Period	4.3

Note: Three-year trailing average metal prices and exchange rate.

Table 22-12: Summary of Case 2 Economic Results

Category	LOM Case 2 C$M
Net revenue (Net Smelter Return + Streaming Revenues)	1,805.4
Operating costs (excluding transport costs)	632.6
Cash flow from operations	1,172.7
Capital (incl. sustaining capital, contingency, reclamation, salvage)	509.8
Net profit	663.0
Pre-Tax IRR (%)	18.5
Pre-Tax $NPV_{8\%}$	246.4
Pre-Tax Payback Period	4.1
After-Tax IRR (%)	16.5
After-Tax $NPV_{8\%}$	177.1
After-Tax Payback Period	4.1

Note: Two-year trailing average metal prices and exchange rate.

Table 22-13: Summary of Case 3 Economic Results

Category	LOM Case 3 C$M
Net revenue (Net Smelter Return + Streaming Revenues)	1,659.8
Operating costs (excluding transport costs)	632.6
Cash flow from operations	1,027.2
Capital (incl. sustaining capital, contingency, reclamation, salvage)	509.8
Net profit	517.5
Pre-Tax IRR (%)	15.4
Pre-Tax $NPV_{8\%}$	162.9
Pre-Tax Payback Period	4.4
After-Tax IRR (%)	13.8
After-Tax $NPV_{8\%}$	117.6
After-Tax Payback Period	4.5

Note: Consensus economics long-term projected metal prices and long-term exchange rate.

Table 22-14: Net Present Value for Various Discount Rates (Base Case)

Discount Rate	Pre-Tax NPV C$M	After-Tax NPV C$M
0%	570.7	451.1
6.5%	244.2	181.9
7.0%	226.3	166.8
7.5%	209.1	152.4
8.0%	192.7	138.7

Table 22-15: Net Present Value for Various Discount Rates (Case 2)

Discount Rate	Pre-Tax NPV C$M	After-Tax NPV C$M
0%	663.0	511.1
6.5%	303.3	223.5
7.0%	283.5	207.4
7.5%	264.5	191.9
8.0%	246.4	177.1

Table 22-16: Net Present Value for Various Discount Rates (Case 3)

Discount Rate	Pre-Tax NPV C$M	After-Tax NPV C$M
0%	517.5	416.6
6.5%	211.1	158.8
7.0%	194.3	144.5
7.5%	178.3	130.8
8.0%	162.9	117.6

22.9 Sensitivity

The sensitivity charts, Tables 22-17 to 22-20 and Figures 22-13 to 22-16, below, show IRR and NPV variations from the Base Case with respect to changes in metal prices, operating costs, and capital costs, holding all other inputs constant. The results below show that the project is most sensitive to metal prices and least sensitive to changes in operating costs for all three scenarios.

Table 22-17: Pre-Tax $NPV_{8\%}$ Sensitivity Test Results

Case	Variable	Pre-Tax $NPV_{8\%}$ C$M		
		-15% Variance	0% Variance	+15% Variance
Base Case	Capital Costs	260.0	192.7	125.4
	Operating Costs	249.5	192.7	135.9
	Metal Prices	22.1	192.7	359.8
Case 2	Capital Costs	313.7	246.4	179.1
	Operating Costs	303.2	246.4	189.6
	Metal Prices	67.8	246.4	421.5
Case 3	Capital Costs	230.2	162.9	95.6
	Operating Costs	219.7	162.9	106.1
	Metal Prices	-1.6	162.9	322.0

Table 22-18: After-Tax $NPV_{8\%}$ Sensitivity Test Results

Case	Variable	After-Tax $NPV_{8\%}$ C$M		
		-15% Variance	0% Variance	+15% Variance
Base Case	Capital Costs	206.0	138.7	71.4
	Operating Costs	179.8	138.7	97.5
	Metal Prices	11.0	138.7	257.2
Case 2	Capital Costs	244.4	177.1	109.8
	Operating Costs	217.6	177.1	136.0
	Metal Prices	47.5	177.1	300.5
Case 3	Capital Costs	184.9	117.6	50.3
	Operating Costs	158.8	117.6	75.8
	Metal Prices	-8.1	117.6	231.1

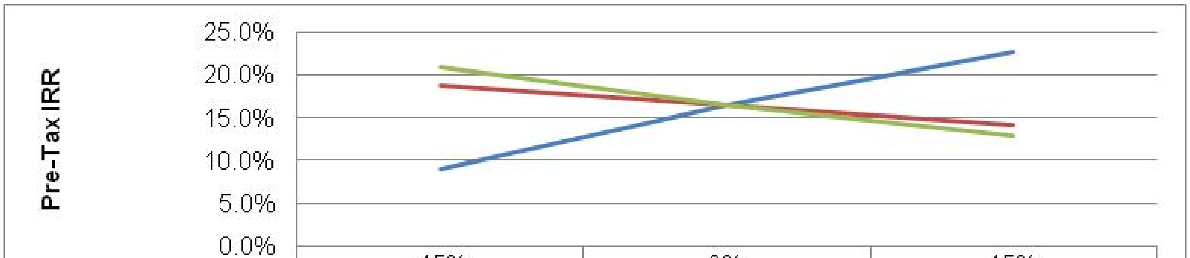
Table 22-19: Pre-Tax IRR Sensitivity Test Results

Case	Variable	Pre-Tax IRR%		
		-15% Variance	0% Variance	+15% Variance
Base Case	Capital Costs	20.9	16.5	13.0
	Operating Costs	18.7	16.5	14.1
	Metal Prices	9.1	16.5	22.6
Case 2	Capital Costs	23.0	18.5	14.9
	Operating Costs	20.7	18.5	16.3
	Metal Prices	11.2	18.5	24.6
Case 3	Capital Costs	19.7	15.4	11.9
	Operating Costs	17.7	15.4	12.9
	Metal Prices	7.9	15.4	21.4

Table 22-20: After-Tax IRR Sensitivity Test Results

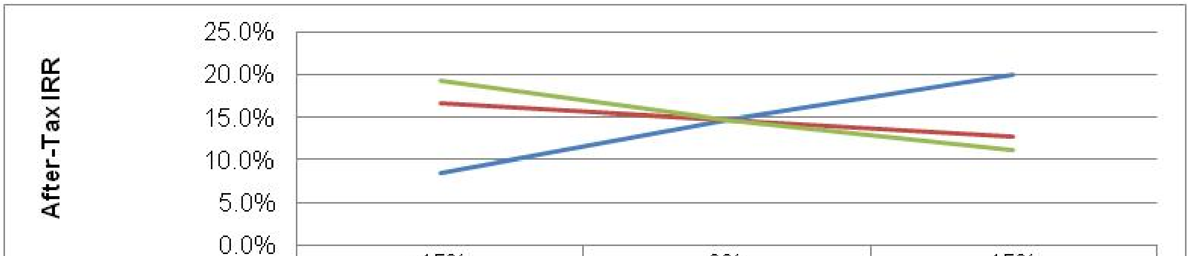
Case	Variable	After-Tax IRR%		
		-15% Variance	0% Variance	+15% Variance
Base Case	Capital Costs	19.2	14.7	11.1
	Operating Costs	16.7	14.7	12.8
	Metal Prices	8.5	14.7	20.0
Case 2	Capital Costs	21.1	16.5	12.8
	Operating Costs	18.3	16.5	14.5
	Metal Prices	10.3	16.5	21.8
Case 3	Capital Costs	18.2	13.8	10.2
	Operating Costs	15.8	13.8	11.7
	Metal Prices	7.6	13.8	19.0

Figure 22-13: Pre-Tax IRR Variation for Base Case



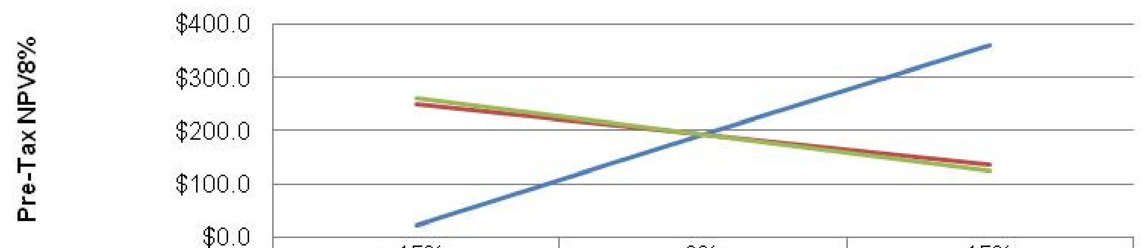
	-15%	0%	15%
Metal Price	9.1%	16.5%	22.6%
Operating Costs	18.7%	16.5%	14.1%
Capital Costs	20.9%	16.5%	13.0%

Figure 22-14: After-Tax IRR Variation for Base Case



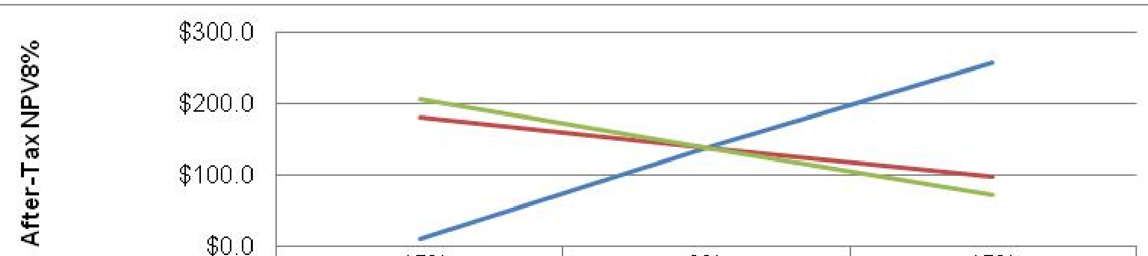
	-15%	0%	15%
Metal Price	8.5%	14.7%	20.0%
Operating Costs	16.7%	14.7%	12.8%
Capital Costs	19.2%	14.7%	11.1%

Figure 22-15: Pre-Tax NPV 8% Variation for Base Case



	-15%	0%	15%
Metal Price	$22.1	$192.7	$359.8
Operating Costs	$249.5	$192.7	$135.9
Capital Costs	$260.0	$192.7	$125.4

Figure 22-16: After-Tax NPV 8% Variation for Base Case



	-15%	0%	15%
Metal Price	$11.0	$138.7	$257.2
Operating Costs	$179.8	$138.7	$97.5
Capital Costs	$206.0	$138.7	$71.4

22.10 Metal Price Sensitivity Analysis

A sensitivity analysis was performed to test the volatility of the project based on the changes of a specific commodity price in the base case calculation.

The prices of gold, copper and zinc were each tested to show the changes in NPV and IRR assuming the prices of lead and silver remain unchanged. These metals were chosen as they each contribute to over 20% of the NSR value for the base case.

Table 22-21 and 22-22 show the results of these sensitivity tests.

Table 22-23 shows the details of the economic model for the Base Case scenario.

Table 22-21: Metal Price Sensitivity Analysis (Pre-Tax)

Gold		
Metal Price (US$/oz)	**Pre-Tax IRR (%)**	**Pre-Tax NPV (C$M)**
1,550	17.2	212.4
1,455	**16.5**	**192.7**
1,300	15.2	160.6
1100	13.5	119.1
600	8.8	15.4
Zinc		
Metal Price (US$/lb)	**Pre-Tax IRR (%)**	**Pre-Tax NPV (C$M)**
1.10	18.2	235.6
0.97	**16.5**	**192.7**
0.85	14.8	151.2
0.75	13.3	115.3
0.45	8.4	7.3
Copper		
Metal Price (US$/lb)	**Pre-Tax IRR (%)**	**Pre-Tax NPV (C$M)**
3.80	16.9	203.9
3.66	**16.5**	**192.7**
3.50	16.0	179.9
2.75	13.5	119.8
1.40	8.6	11.7

Table 22-22: Metal Price Sensitivity Analysis (After-Tax)

Gold		
Metal Price (US$/oz)	**After-Tax IRR (%)**	**After-Tax NPV (C$M)**
1,550	15.4	152.8
1,455	**14.7**	**138.7**
1,300	13.7	115.7
1,100	12.2	85.6
600	8.3	6.0
Zinc		
Metal Price (US$/lb)	**After-Tax IRR (%)**	**After-Tax NPV (C$M)**
1.10	16.2	169.7
0.97	**14.7**	**138.7**
0.85	13.3	108.8
0.75	12.0	82.4
0.45	7.9	-1.3
Copper		
Metal Price (US$/lb)	**After-Tax IRR (%)**	**After-Tax NPV (C$M)**
3.80	15.1	146.7
3.66	**14.7**	**138.7**
3.50	14.3	129.5
2.75	12.2	86.0
1.40	8.1	2.6

Table 22-23: Base Case Economic Analysis

			2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
		Prod. Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
	Unit	LOM																						
METAL PRICES																								
Au	$/oz	1,455.00	-	-	-	1,455.00	1,455.00	1,455.00	1,455.00	1,455.00	1,455.00	1,455.00	1,455.00	1,455.00	-	-	-	-	-	-	-	-	-	-
Ag	$/oz	28.00	-	-	-	28.00	28.00	28.00	28.00	28.00	28.00	28.00	28.00	28.00	-	-	-	-	-	-	-	-	-	-
Pb	$/lb	1.01	-	-	-	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	-	-	-	-	-	-	-	-	-	-
Cu	$/lb	3.66	-	-	-	3.66	3.66	3.66	3.66	3.66	3.66	3.66	3.66	3.66	-	-	-	-	-	-	-	-	-	-
Zn	$/lb	0.97	-	-	-	0.97	0.97	0.97	0.97	0.97	0.97	0.97	0.97	0.97	-	-	-	-	-	-	-	-	-	-
F/X RATE	CAD:USD	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01	1.01
METAL GRADE																								
Au	g/t	2.30	-	-	-	1.54	1.80	1.79	2.89	3.26	2.52	2.10	2.29	2.38	-	-	-	-	-	-	-	-	-	-
Ag	g/t	81.38	-	-	-	63.87	70.46	80.71	101.27	105.90	85.73	74.91	69.58	77.09	-	-	-	-	-	-	-	-	-	-
Pb	%	1.04%	-	-	-	0.96%	0.92%	0.98%	1.13%	1.14%	1.22%	1.00%	1.01%	0.98%	-	-	-	-	-	-	-	-	-	-
Cu	%	1.12%	-	-	-	0.93%	0.78%	1.05%	1.45%	1.61%	1.02%	1.03%	1.08%	1.13%	-	-	-	-	-	-	-	-	-	-
Zn	%	5.59%	-	-	-	5.36%	5.19%	5.38%	6.42%	6.63%	5.43%	5.53%	5.44%	4.93%	-	-	-	-	-	-	-	-	-	-
MINE PRODUCTION																								
Waste	tonnes	1,229,504	-	63,263	285,955	268,489	261,152	234,477	68,640	5,564	-	28,248	13,716	-	-	-	-	-	-	-	-	-	-	-
Total Ore Processed	tonnes	6,447,098	-	-	-	607,098	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	-	-	-	-	-	-	-	-	-	-
Strip Ratio	w:o	0.19	-	-	-	0.44	0.36	0.32	0.09	0.01	0.00	0.04	0.02	0.00	-	-	-	-	-	-	-	-	-	-
NSR CALCULATION																								
Zn CONCENTRATE																								
Payable Zn	lbs	601,460,418	-	-	-	54,270,313	63,170,125	65,458,004	78,163,960	80,768,803	66,076,851	67,291,321	66,276,309	59,984,731	-	-	-	-	-	-	-	-	-	-
	$	583,416,605	-	-	-	52,642,204	61,275,021	63,494,264	75,819,041	78,345,739	64,094,545	65,272,581	64,288,020	58,185,189	-	-	-	-	-	-	-	-	-	-
Total Payable in Zn Concentrate	$	583,416,605	-	-	-	52,642,204	61,275,021	63,494,264	75,819,041	78,345,739	64,094,545	65,272,581	64,288,020	58,185,189	-	-	-	-	-	-	-	-	-	-
TC/RCs																								
Zn Treatment Charge + Escalator	$	77,931,702	-	-	-	7,031,847	8,185,003	8,481,445	10,127,766	10,465,278	8,561,630	8,718,990	8,587,474	7,772,269	-	-	-	-	-	-	-	-	-	-
Total TC/RC	$	77,931,702	-	-	-	7,031,847	8,185,003	8,481,445	10,127,766	10,465,278	8,561,630	8,718,990	8,587,474	7,772,269	-	-	-	-	-	-	-	-	-	-
At-mine Revenues	$	505,484,903	-	-	-	45,610,356	53,090,018	55,012,819	65,691,275	67,880,461	55,532,916	56,553,592	55,700,546	50,412,920	-	-	-	-	-	-	-	-	-	-
Transport Charge	$	102,134,229	-	-	-	9,215,663	10,726,944	11,115,449	13,273,053	13,715,382	11,220,536	11,426,766	11,254,406	10,186,031	-	-	-	-	-	-	-	-	-	-
Penalties in Zn Concentrate	$	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL Zn NSR	$	403,350,674	-	-	-	36,394,693	42,363,075	43,897,369	52,418,223	54,165,079	44,312,380	45,126,826	44,446,140	40,226,889	-	-	-	-	-	-	-	-	-	-
Cu CONCENTRATE																								
Payable Cu	lbs	135,505,308	-	-	-	10,550,481	10,693,352	14,351,754	19,814,726	21,976,251	13,930,211	14,028,935	14,720,373	15,439,225	-	-	-	-	-	-	-	-	-	-
	$	495,949,427	-	-	-	38,614,761	39,137,667	52,527,420	72,521,898	80,433,079	50,984,572	51,345,902	53,876,566	56,507,562	-	-	-	-	-	-	-	-	-	-
Uncommitted Payable Ag	oz	8,183,831	-	-	-	602,253	814,560	923,043	1,149,608	1,196,249	986,627	853,370	784,704	873,416	-	-	-	-	-	-	-	-	-	-
	$	229,147,259	-	-	-	16,863,090	22,807,677	25,845,211	32,189,033	33,494,958	27,625,559	23,894,358	21,971,725	24,455,647	-	-	-	-	-	-	-	-	-	-
Uncommitted Payable Au	oz	169,324	-	-	-	10,647	14,971	14,869	24,019	27,099	20,956	17,446	19,064	20,254	-	-	-	-	-	-	-	-	-	-
	$	246,366,098	-	-	-	15,491,084	21,782,635	21,633,877	34,947,497	39,428,830	30,491,269	25,383,205	27,738,746	29,468,954	-	-	-	-	-	-	-	-	-	-
Total Payable in Cu Concentrate	$	971,462,784	-	-	-	70,968,935	83,727,979	100,006,509	139,658,428	153,356,868	109,101,400	100,623,465	103,587,037	110,432,163	-	-	-	-	-	-	-	-	-	-
TC/RCs																								
Cu Treatment Charge + Escalator	$	38,415,503	-	-	-	2,991,042	3,031,545	4,068,696	5,617,438	6,230,226	3,949,189	3,977,177	4,173,198	4,376,991	-	-	-	-	-	-	-	-	-	-
Cu Refining Charge	$	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag Refining Charge	$	12,275,746	-	-	-	903,380	1,221,840	1,384,565	1,724,412	1,794,373	1,479,941	1,280,055	1,177,057	1,310,124	-	-	-	-	-	-	-	-	-	-
Au Refining Charge	$	4,233,094	-	-	-	266,170	374,272	371,716	600,472	677,471	523,905	436,138	476,611	506,339	-	-	-	-	-	-	-	-	-	-
Total TC/RC	$	54,924,343	-	-	-	4,160,591	4,627,657	5,824,977	7,942,323	8,702,070	5,953,035	5,693,369	5,826,866	6,193,455	-	-	-	-	-	-	-	-	-	-
At-mine Revenues	$	916,538,440	-	-	-	66,808,344	79,100,322	94,181,532	131,716,105	144,654,798	103,148,366	94,930,095	97,760,171	104,238,708	-	-	-	-	-	-	-	-	-	-
Transport Charge	$	60,631,902	-	-	-	4,720,817	4,784,744	6,421,698	8,866,107	9,833,282	6,233,078	6,277,252	6,586,637	6,908,287	-	-	-	-	-	-	-	-	-	-
Penalties in Cu Concentrate	$	16,042,314	-	-	-	1,249,059	1,265,973	1,699,087	2,345,842	2,601,743	1,649,181	1,660,869	1,742,728	1,827,832	-	-	-	-	-	-	-	-	-	-
Total Cu NSR	$	839,864,225	-	-	-	60,838,468	73,049,604	86,060,747	120,504,156	132,219,773	95,266,106	86,991,974	89,430,807	95,502,590	-	-	-	-	-	-	-	-	-	-
Pb CONCENTRATE																								
Payable Pb	lbs	92,976,079	-	-	-	8,080,560	9,350,969	9,893,526	11,411,918	11,572,299	12,359,303	10,145,418	10,219,290	9,942,797	-	-	-	-	-	-	-	-	-	-
	$	93,905,840	-	-	-	8,161,365	9,444,479	9,992,461	11,526,037	11,688,022	12,482,896	10,246,872	10,321,483	10,042,225	-	-	-	-	-	-	-	-	-	-
Uncommitted Payable Ag	oz	1,030,234	-	-	-	74,996	100,777	116,199	146,740	153,792	121,678	106,943	98,590	110,518	-	-	-	-	-	-	-	-	-	-
	$	28,846,560	-	-	-	2,099,884	2,821,758	3,253,576	4,108,729	4,306,178	3,406,989	2,994,410	2,760,528	3,094,509	-	-	-	-	-	-	-	-	-	-
Uncommitted Payable Au	oz	12,921	-	-	-	742	1,107	1,080	1,891	2,175	1,573	1,314	1,464	1,576	-	-	-	-	-	-	-	-	-	-
	$	18,800,544	-	-	-	1,079,132	1,610,774	1,571,626	2,750,944	3,164,178	2,288,648	1,911,896	2,129,590	2,293,757	-	-	-	-	-	-	-	-	-	-
Total Payable in Pb Concentrate	$	141,552,944	-	-	-	11,340,380	13,877,011	14,817,663	18,385,710	19,158,377	18,178,533	15,153,179	15,211,601	15,430,490	-	-	-	-	-	-	-	-	-	-
TC/RCs																								
Pb Treatment Charge + Escalator	$	8,740,433	-	-	-	759,632	879,060	930,064	1,072,804	1,087,881	1,161,865	953,744	960,688	934,696	-	-	-	-	-	-	-	-	-	-
Pb Refining Charge	$	0	-	-	-	0	0	0	0	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-
Ag Refining Charge	$	1,545,351	-	-	-	112,494	151,166	174,299	220,110	230,688	182,517	160,415	147,885	165,777	-	-	-	-	-	-	-	-	-	-
Au Refining Charge	$	323,033	-	-	-	18,542	27,677	27,004	47,267	54,367	39,324	32,850	36,591	39,412	-	-	-	-	-	-	-	-	-	-
Total TC/RC	$	10,608,818	-	-	-	890,667	1,057,902	1,131,367	1,340,181	1,372,936	1,383,706	1,147,009	1,145,164	1,139,885	-	-	-	-	-	-	-	-	-	-
At-Mine Revenues	$	130,944,126	-	-	-	10,449,713	12,819,109	13,686,296	17,045,528	17,785,441	16,794,826	14,006,170	14,066,437	14,290,606	-	-	-	-	-	-	-	-	-	-
Transport Charge	$	14,896,912	-	-	-	1,294,692	1,498,241	1,585,171	1,828,452	1,854,149	1,980,245	1,625,530	1,637,366	1,593,065	-	-	-	-	-	-	-	-	-	-
Penalties in Pb Concentrate	$	0	-	-	-	0	0	0	0	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-
Total Pb NSR	$	116,047,214	-	-	-	9,155,021	11,320,868	12,101,125	15,217,076	15,931,292	14,814,581	12,380,640	12,429,071	12,697,541	-	-	-	-	-	-	-	-	-	-

			2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
		Prod. Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Au Dore Production																								
Uncommitted Payable Au	oz	172,168	-	-	-	10,826	15,222	15,118	24,422	27,554	21,308	17,739	19,385	20,594	-	-	-	-	-	-	-	-	-	-
	$	250,504,659	-	-	-	15,751,310	22,148,549	21,997,292	35,534,559	40,091,172	31,003,475	25,809,602	28,204,713	29,963,986	-	-	-	-	-	-	-	-	-	-
TC/RC																								
Au Refining Charge	$	1,033,009	-	-	-	64,954	91,334	90,710	146,534	165,324	127,849	106,431	116,308	123,563	-	-	-	-	-	-	-	-	-	-
At-Mine Revenues	$	249,471,650	-	-	-	15,686,356	22,057,215	21,906,582	35,388,025	39,925,848	30,875,625	25,703,171	28,088,405	29,840,423	-	-	-	-	-	-	-	-	-	-
Total Au NSR	$	249,471,650	-	-	-	15,686,356	22,057,215	21,906,582	35,388,025	39,925,848	30,875,625	25,703,171	28,088,405	29,840,423	-	-	-	-	-	-	-	-	-	-
Ag Dore Production																								
Uncommitted Payable Ag	oz	64,061	-	-	-	4,734	6,280	7,193	9,026	9,438	7,640	6,677	6,202	6,870	-	-	-	-	-	-	-	-	-	-
	$	1,793,715	-	-	-	132,557	175,848	201,415	252,719	264,277	213,934	186,943	173,651	192,373	-	-	-	-	-	-	-	-	-	-
TC/RC																								
Ag Refining Charge	$	96,092	-	-	-	7,101	9,420	10,790	13,538	14,158	11,461	10,015	9,303	10,306	-	-	-	-	-	-	-	-	-	-
At-Mine Revenues	$	1,697,624	-	-	-	125,456	166,427	190,625	239,180	250,119	202,473	176,928	164,348	182,068	-	-	-	-	-	-	-	-	-	-
TOTAL Ag NSR	$	1,697,624	-	-	-	125,456	166,427	190,625	239,180	250,119	202,473	176,928	164,348	182,068	-	-	-	-	-	-	-	-	-	-
Total NSR (All Concentrates)	US$	1,610,431,387	-	-	-	122,199,994	148,957,189	164,156,448	223,766,659	242,492,111	185,471,165	170,379,539	174,558,770	178,449,510	-	-	-	-	-	-	-	-	-	-
	$CAD	1,626,374,658	-	-	-	123,409,774	150,431,865	165,781,597	225,981,949	244,892,783	187,307,330	172,066,296	176,286,902	180,216,160	-	-	-	-	-	-	-	-	-	-
Cashflow from Streaming	$CDN	36,169,297	-	-	-	2,441,964	3,383,159	3,551,008	5,181,316	5,682,345	4,481,975	3,787,125	3,895,209	3,765,196	-	-	-	-	-	-	-	-	-	-
Upfront Cash Payment Received	$CDN	50,495,000	8,313,240	21,204,929	20,976,831	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Cashflow From Streaming	$CDN	86,664,297	8,313,240	21,204,929	20,976,831	2,441,964	3,383,159	3,551,008	5,181,316	5,682,345	4,481,975	3,787,125	3,895,209	3,765,196	-	-	-	-	-	-	-	-	-	-
Net Revenues	$CDN	1,713,038,955	8,313,240	21,204,929	20,976,831	125,851,737	153,815,025	169,332,605	231,163,266	250,575,128	191,789,305	175,853,421	180,182,111	183,981,356	-	-	-	-	-	-	-	-	-	-
OPEX																								
Mining Cost	$/t milled	30.06	-	-	-	19.40	21.61	23.55	34.36	37.68	39.39	31.84	30.88	30.04	-	-	-	-	-	-	-	-	-	-
Processing	$/t milled	23.02	-	-	0	25.86	22.59	23.12	22.58	23.20	22.58	22.58	22.58	22.58	-	-	-	-	-	-	-	-	-	-
G&A	$/t milled	22.47	-	-	0	27.70	22.18	22.17	22.49	22.52	22.31	21.57	21.09	21.05	-	-	-	-	-	-	-	-	-	-
Power	$/t milled	22.58	-	-	0	24.68	23.42	23.39	23.35	23.15	22.39	21.51	21.13	20.52	-	-	-	-	-	-	-	-	-	-
Operating Supplies - Tail. & Effl.	$/t milled	0.00	-	-	0	0	0	0	0	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-
G&A	$/t milled	0.00	-	-	0										-	-	-	-	-	-	-	-	-	-
Total Unit Cost	$/t milled	98.13													-	-	-	-	-	-	-	-	-	-
Mining Cost	$	193,808,343	-	-	$0	$11,778,372	$15,776,660	$17,191,250	$25,079,360	$27,509,252	$28,755,374	$23,242,367	$22,545,920	$21,929,788	-	-	-	-	-	-	-	-	-	-
Processing	$	148,421,212	-	-	-	15,702,379	16,488,064	16,874,399	16,485,260	16,935,248	16,483,832	16,484,118	16,484,054	16,483,858	-	-	-	-	-	-	-	-	-	-
G&A	$	144,850,061	-	-	-	16,816,661	16,190,434	16,183,530	16,418,646	16,442,707	16,288,594	15,746,703	15,394,632	15,368,155	-	-	-	-	-	-	-	-	-	-
Power	$	145,552,596	-	-	-	14,985,275	17,095,881	17,077,595	17,048,907	16,898,257	16,345,303	15,702,454	15,421,793	14,977,132	-	-	-	-	-	-	-	-	-	-
Total OPEX	$	632,632,213		-	-	59,282,686	65,551,039	67,326,774	75,032,173	77,785,463	77,873,102	71,175,642	69,846,400	68,758,933	-	-	-	-	-	-	-	-	-	-
Operating Cashflow	$	1,080,406,742	8,313,240	21,204,929	20,976,831	66,569,051	88,263,986	102,005,831	156,131,093	172,789,665	113,916,203	104,677,779	110,335,712	115,222,423	-	-	-	-	-	-	-	-	-	-
CAPEX																								
Mining	$	114,220,605	-	22,766,902	27,469,547	25,118,924	15,791,317	14,084,366	5,648,569	953,366	33,200	1,744,390	610,025	-	-	-	-	-	-	-	-	-	-	-
Site Development	$	7,417,819	2,871,306	4,546,513	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Limestone Crushing	$	159,237	-	$0	$159,237	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing Plant	$	63,054,443	1,643,474	$23,651,647	$37,759,321	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings & Waste Rock Mgmt	$	15,502,816	-	-	$15,502,816	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On-site Infrastructure	$	61,478,528	3,900,163	$24,881,524	$32,696,841	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Off-site Infrastructure	$	54,197,178	22,582,157	31,615,020	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Project Indirects	$	91,642,283	22,108,087	$37,444,897	32,089,299	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Engineering & EPCM	$	31,351,419	6,413,213	$13,538,746	$11,399,459	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's Costs	$	17,343,672	5,081,679	$6,332,874	$5,929,119	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Salvage Value	$	-7,577,409	-	$0	$0	-	-	-	-	-	-	-	-	-	-7,277,409	-	-	-	-300,000	-	-	-	-	-
Reclamation Costs	$	13,751,173	-	-	-	-	-	-	-	-	-	-	-	-	12,651,670	523,957	99,888	82,003	73,061	64,119	64,119	64,119	64,119	64,119
Pre-Contingency CAPEX	$	462,541,764	64,600,079	164,778,124	163,005,639	25,118,924	15,791,317	14,084,366	5,648,569	953,366	33,200	1,744,390	610,025	-	5,374,262	523,957	99,888	82,003	-226,939	64,119	64,119	64,119	64,119	64,119
Contingency	%	12.0%	-	-	-	-	-	-	-	-	-	-	-	-	-	12%	12%	12%	0%	12%	12%	12%	12%	12%
	$	47,209,234	7,752,010	19,773,375	19,560,677	-	-	-	-	-	-	-	-	-	-	62,875	11,987	9,840	-	7,694	7,694	7,694	7,694	7,694
CAPEX incl. Contingency	$	509,750,998	72,352,089	184,551,499	182,566,316	25,118,924	15,791,317	14,084,366	5,648,569	953,366	33,200	1,744,390	610,025	-	5,374,262	586,832	111,874	91,844	-226,939	71,813	71,813	71,813	71,813	71,813
Working capital	$	-	-	-	-	14,820,672	-	-	-	-	-	-	-	-14,820,672	-	-	-	-	-	-	-	-	-	-
Cashflow																								
Net Cashflow	$	570,655,744	-$64,038,849	$163,346,570	161,589,484	26,629,455	72,472,669	87,921,465	150,482,523	171,836,299	113,883,003	102,933,389	109,725,687	130,043,095	5,374,262	586,832	111,874	91,844	226,939	71,813	71,813	71,813	71,813	71,813
Cumulative Cashflow	$		-$64,038,849	$227,385,419	388,974,903	362,345,448	289,872,779	201,951,314	51,468,790	120,367,509	234,250,512	337,183,901	446,909,587	576,952,682	571,578,421	570,991,588	570,879,714	570,787,871	571,014,810	570,942,997	570,871,183	570,799,370	570,727,557	570,655,744
Pre-Tax Payback	Years	4.3				0	0	0	0.0	4.3	0.0	0.0	0	0	0	0	0	0	0	0	0	0	0	0
Pre-Tax IRR	%	16.5%																						
Pre-Tax NPV	8.0%	192,701,281																						
Taxes (Incl. BC Mineral Tax)	$	119,511,187	-	-	-	-	1,320,067	1,805,646	2,095,363	3,268,407	3,638,062	11,057,970	24,044,760	36,069,158	40,073,700	-3,571,616	-215,561	-47,047	-11,940	74,829	-25,055	-15,615	-16,551	-16,686
Net After-Tax Cashflow	$	451,144,557	64,038,849	163,346,570	161,589,484	26,629,455	71,152,602	86,115,819	148,387,160	168,567,892	110,244,941	91,875,419	85,680,927	93,973,937	45,447,961	2,984,783	103,687	44,797	238,879	146,642	46,758	56,198	55,262	55,127
Cumulative Cashflow	$		64,038,849	227,385,419	388,974,903	362,345,448	291,192,846	205,077,027	56,689,866	111,878,026	222,122,967	313,998,386	399,679,313	493,653,250	448,205,289	451,190,072	451,293,759	451,248,962	451,487,841	451,341,199	451,294,441	451,238,244	451,182,982	451,127,855
After-Tax Payback	Years	4.3				0	0	0	0.0	4.3	0.0	0	0.0	0	0	0	0	0	0	0	0	0	0	0
After-Tax IRR	%	14.7%																						
After-Tax NPV	8.0%	138,661,259																						

23 Adjacent Properties

The following information is taken from the Canarc Resources Corp. (Canarc) website. Canarc owns the Polaris Taku property, situated 6 km northwest of the Tulsequah Chief property. Canarc holds a 100% interest in 61 crown granted mineral claims and one modified grid claim totalling 1196 ha. Canarc is considering a feasibility program with the following objectives:

- drive a decline from surface down to the 1050 mine level (305 m below surface)
- develop one or more drifts and raises within the C vein
- conduct trial mining to extract a bulk sample
- ship and process a representative portion of the bulk sample for final metallurgical testing
- finalize the process flowsheet
- complete a feasibility study.

The author has been unable to verify the above information and the information on the Polaris Taku property is not necessarily indicative of the mineralization at the Tulsequah Chief deposit. This study does not rely upon any information from adjacent properties.

24 Other Relevant Data & Information

24.1 Project Execution

24.1.1 Introduction & Philosophy

The project execution plan for the Tulsequah Chief project is based on principles tested and proven in the development of remote, logistically challenged projects in northern Canada. These principles include:

- safety in design, construction and operations is paramount to success
- simple, passive environmental solutions; minimizing disturbance footprint
- fit-for-purpose design, construction, and operation
- due to the high cost of transportation, consolidate construction and operational needs to the extent practical (i.e., “Bring it in – it stays”)
- common equipment fleet purchased by Owner at the outset and used for construction needs to the extent practical
- efficient operations; minimize site labour requirements
- negotiated contracts with suppliers, contractors, and engineers with proven track records in northern Canadian mine developments
- no nonsense project management; decisive decision-making
- early completion of project components turned over to operations
- elimination of superfluous management organizations
- same camp accommodation status applied to all site personnel (no management quarters).

24.1.2 Existing Site Development

There is currently a 950 m airstrip adjacent to the Tulsequah River near the mine site (Figure 24-1), which is in a reasonable condition to accommodate light aircraft such as Dornier and Caravans to provide passenger and freight service to the site. By lengthening the airstrip by approximately 50 m and carrying out minor re-surfacing, it would accommodate DHC-5 Buffalo (18,000 lb. payload) and DHC-8 Dash 8 – 100 (32 passenger) aircraft, which will provide better logistical flexibility during construction.

Figure 24-1: Tulsequah Chief Project – Existing Airstrip

A pioneer camp consisting of 45 beds complete with kitchen and dining facilities is located on site adjacent to the airstrip, which has been in continuous use for the past three years and is suitable for use to support the initial mobilization of construction crews and materials in 2013. Living quarters are a mix of ATCO style dormitories and containerized rooms housed in an all-weather structure, as well as some stick-built dorms. All units are currently single occupancy, which is a site standard anticipated to be maintained through construction and operations. Existing washroom facilities consist of one common washroom with four showers for the 18 unit all-weather dorm, and three single washrooms in the ATCO modular camp, one of which is designated for female use. Potable water is provided from a water well, and treated with a simple filtering and chlorination/UV system. It is anticipated that the existing well will accommodate additional demand providing that suitable surge capacity is provided. The wastewater treatment plant is sized for the current camp only, with no capacity for additional loading.

A laydown area adjacent to the Taku River approximately 14 km south of the mine site has been utilized for receiving and offloading river barges in the past, but the barge landing is unimproved (natural river bank) and is not deemed suitable for the intensified campaign required to support construction mobilization. Improvements to the barge landing are required to enable the barging campaign required for 2013 and 2014 construction activities.

There is an existing road from the barge landing area to the plant site, as well as from the plant site to the airstrip and camp area. These roads are currently passable for mobile equipment, but require upgrades for mobilization of the construction camp and materials.

All water discharged from the existing mine portals is currently captured in a lined containment pond and processed in an existing water treatment plant prior to discharge to the environment. The Catchment pond and water treatment facility are located on the proposed truck shop bench (Figure 24-2). Operations of the WTP must be maintained through the construction period.

Figure 24-2: Historic Mine Workings & Proposed Plant Site (Existing WTP bottom left)

24.1.3 Project Management Team

Project management will be an integrated team comprised of the Owners project management personnel and the project management consultant (PM consultant). The project management team (PM team) will oversee the detailed engineering, procurement, and construction management activities for the project. The PM team will also coordinate the work of the engineering subcontractor and other specialized consultants as required.

The PM team will be responsible for all project activities from detailed design through to commissioning and turnover to operations. The PM team will be available to backstop the operations teams with key supervision and management assistance when the operations personnel assume control of project components as they are completed.

24.1.4 Project Procedures

The PM team will prepare and publish a project procedures manual (PPM) early in the development of the project. This manual will describe standard project templates, procedures, and forms for use in the engineering, procurement, construction, and project disciplines.

Some of the major procedures are listed below for reference:

- engineering (supplemented by procedures utilized by selected engineering contractors)
- procurement
- designation of authority guideline
- purchase order and contract execution procedure
- purchase order and contract change procedure
- invoice approval and payment procedures.
- logistics
- procedures as required to support the freight and logistics plan.
- construction
- quality assurance procedures
- health and safety procedures
- environmental procedures.
- project controls
- project change procedure
- project cost procedures
- project schedule procedures
- project risk procedures.

24.1.5 Project Controls Systems

In keeping with the fit-for-purpose execution philosophy, a suitable Owner-approved cost and budget control system with minimum complexity will be utilized. As the Owner is embedded into the PM team, it is envisioned that project reporting will be concise and contain pertinent project progress information only. Project reporting will track budget, committed, actual and forecasted quantities and costs. Earned value will be implemented as required for specific critical sub-projects only (i.e., concrete installation or building erection).

The project management team will utilize Primavera as the primary scheduling software. All scheduling will be performed utilizing the critical path method (CPM).

24.1.6 Procurement Strategy

In general, the PM consultant will oversee the selection and tendering of all tagged equipment and bulk materials and commodities as a function of managing the engineering subcontractor. Tagged equipment is defined as uniquely designed and engineered equipment and assemblies required for the project as documented in the project equipment lists. Bulk materials are not generally specifically engineered items and are not identified on the project equipment list. All bulk materials for the project will be purchased, tracked and referenced to applicable specifications and standards.

Construction bulk materials (such as concrete and rebar for the maintenance shops and mill building foundations) and HDPE liner materials for the HPAG containment pond will need to be factored based on estimated quantities and procured by the Owner to meet the mobilization timeline of May 1, 2013, as detailed engineering will not be sufficiently advanced to generate MTOs.

Process equipment considered to be “long delivery,” will have to be selected and conditionally committed to earlier than required by site delivery schedules, in order to receive the vendor’s certified drawings and allow detailed design of the civil and structural components of the project to be completed in a timely manner to receive the equipment. Most long-lead delivery equipment and materials are targeted for delivery on the all-weather access road over the winter of 2014/15.

Structural steel for the process plant building will be delivered to site on the 2014 spring barge campaign; detailed design will need to be complete by the fall of 2013 to allow sufficient time for fabrication and shipping.

24.1.7 Freight & Logistics

A detailed Freight and Logistics Plan will be developed for the project. The plan will address the requirements for barge freight, airfreight and truck freight, as well as personnel transport to support the project schedule.

Freight movement for the project will require mobilization of equipment and bulk supplies by ocean and river barge for the first two years of construction, supplemented by airfreight prior to the completion of the all-weather access road. A barging contractor with experience moving freight up the Taku River from Prince Rupert has been consulted regarding the anticipated requirements and a cost estimate has been obtained. The barging component of the logistics plan is critical to the project success; the barge contractor must be engaged immediately upon project approval to work with the PM team on the coordination and management of this work.

The 2013 barge campaign will consist primarily of the access road construction contractor's camp and equipment as well as bulk rebar and formwork supplies to support the early 2014 concrete construction, and HDPE liner materials for the HPAG pond. Fuel supply for the first two years will be delivered by air, fuel storage and dispensing facilities will be located on the airstrip apron for this purpose.

The 2014 barge campaign will consist primarily of the process plant structural steel and maintenance shops so that they can be erected over the summer in advance of the access road completion.

For truck transport over the mine access road once opened in the fall of 2014, freight will be staged in a laydown area at the commencement of the road from Atlin. To minimize the number of operators and control the conduct of the drivers on the newly constructed road, a fleet of tridem tractors will be engaged to shuttle the loads into site from the laydown yard. Best efforts will be made to minimize double handling of freight by negotiating with freight contractors for trailer demurrage (highway transport trucks can then deck the trailers and have them transported to site by the tridem trucks). The PM team will position a freight manager at the laydown yard to manage and coordinate shuttling loads to site as well as the transfer of materials between freight contracts when required. An expediting contractor based in Whitehorse will be engaged to receive and consolidate less than trailer-load (LTL) freight and forward to site as warranted.

Airfreight and passenger transport will be tendered to qualified carriers on a unit cost basis; however, it is anticipated that multiple carriers will be required to support the project, particularly in advance of the access road completion. Passenger transport will be via commercial aircraft to Whitehorse and charter aircraft from Whitehorse directly to site. Airfreight will be received and consolidated by an expediting service provider operating either on a standalone contract or in combination with the air charter service.

24.1.8 Contracting Strategy

The contracting strategy will be established by the PM team at the onset of the project, which will address each contract battery limit, detailed scope of work and the cost structure of each. Contract work packages will be divided into manageable scopes, and awarded to contractors "best fit" for the work. Contractors will be pre-qualified by the PM team based on their ability to execute the work in a safe and efficient manner, as demonstrated by past performance.

Opportunities for qualified local and aboriginal contractors will be given consideration when determining the work packages, providing that they can meet bid requirements and are available to provide value to the project through competitive pricing.

It is currently envisioned that there will be a mixture of lump sum supply and install, time and material, cost plus fixed fee, and unit price contracts, as outlined in Table 24-1.

Table 24-1: Contracting Strategy

Contract	Contract Type	Compensation Strategy
Accommodations Complex	Supply and Install	Lump Sum
Catering	Labor and Material	Unit Price
Process Plant Building	Supply and Install	Lump Sum
General Contractor – Process Plant Mechanical / Electrical / Civil	Labor and N.O.S. Material *	Unit Price / Target Price
Underground Mining	Owner's Workforce	Time and Material
Underground Civil / Mechanical / Electrical	Labor and N.O.S. Material*	Unit Price / Target Price
Access Road Construction	Labor and Equipment / Materials	Cost + Fee
Site Earthworks	Labor and Equipment / Materials	Unit Price / Cost + Fee
Air Support - Fuel / Freight Haul	Labor and Equipment	Unit Price
Air Support – Passenger Transport	Labor and Equipment	Unit Price
Barge Freight Transport	Labor and Equipment	Time and Material
Road Freight Transport	Labor and Equipment	Unit Price

*Non-Owner supplied (N.O.S) material – to be determined by Procurement Strategy.

The compensation structures for the major works contracts will be developed with the intent of providing incentives for meeting schedule milestones and controlling cost. Such a compensation structure may include:

- base fee for overhead and profit
- incentive fee based on achievement of milestones
- penalty for non-achievement of milestones.

An open shop labour strategy will be adopted for the project, and the number of discipline contractors (such as concrete, structural steel and mechanical) will be minimized to mitigate the cost of separate administrations, duplication of temporary facilities and progress dictated by peer contractors. In the process plant and underground mechanical installations in particular, single general contractors will be selected and they will manage and coordinate the interfacing of the various trade disciplines, subcontractors and vendor representatives. The exception to this may be the early process building foundations and the supply and erection of the building, which may be awarded as stand-alone contracts given the need for early award.

Contracts that extend into operations, such as camp catering, will be structured in conjunction with the Owner's operations personnel to ensure that operational needs are properly addressed.

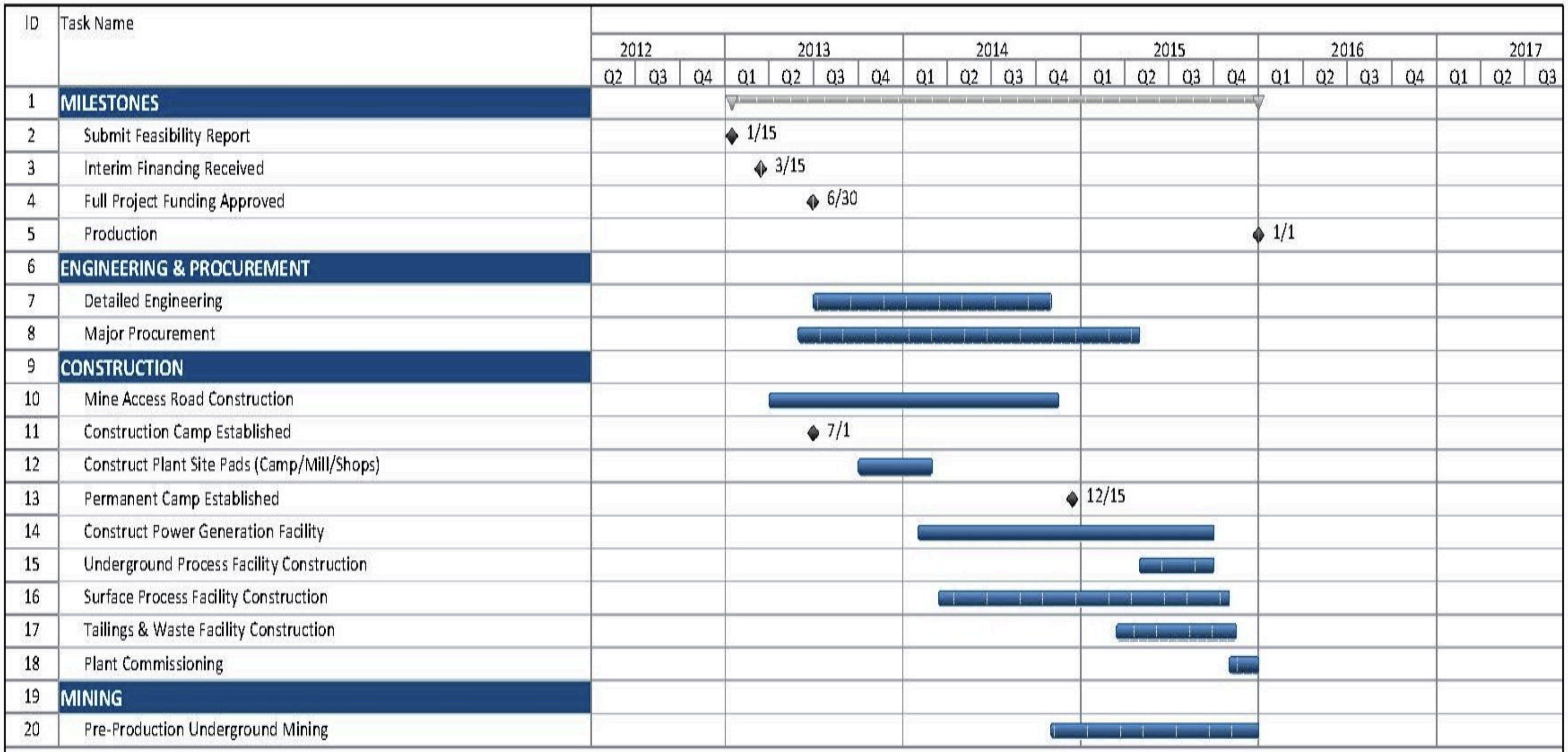
24.1.9 Development Schedule

The construction execution schedule is driven initially by the spring barging transportation windows, followed by the timely completion of the all-weather access road from Atlin to the mine site. A detailed, resource loaded schedule has been developed for the site construction activities, utilizing the feasibility cost estimate as the basis for the required manhours. This scheduling exercise indicates that mechanical completion and wet commissioning can be accomplished by the end of Q4 2015, providing that the construction of the access road is achieved by end of Q3 2014. A brief description of the construction sequence is presented in Table 24-2. The summary project execution Gantt chart schedule is shown on Figure 24-3.

Table 24-2: Construction Sequence

Period	Construction Sequence
April / May 2013	Establish Barge landing facility at the Taku River Laydown area
May 1 2013	Have all required equipment and materials to support Q3 2013 to Q2 2014 construction activities staged in Prince Rupert
May – June 2013	Barge Mobilization of Access Road Construction Fleet and Q2 2014 Foundation Concrete equipment and materials.
July – Oct 2013	Construct Access Road from Site to Km 30, and from Atlin to Km 90
October – March 2014	Construct Permanent Camp Pad and Plant Site
March – September 2014	Construct Process Plant Building and Mill Foundations
May – June 2014	Barge Mobilization of Process Plant Structural Steel and Mine Maintenance Shops
August – December 2014	Erect and enclose Process Plant Building
October 2014	Access Road Rough Grading Complete, priority Freight loads can access site
October – December 2014	Construct Permanent Accommodations Complex
October – May 2015	Development Mining for Underground Mechanical Systems
January – May 2015	Resume Concrete Foundations inside Process Plant
January – October 2015	Process Plant Mechanical and Electrical Systems
May – September 2015	Install Underground Crushing, Conveying Systems and Paste Backfill Plant
March – November 2015	Construct Tailings Facility and Pipelines
February – October 2015	Install Power Generation and Distribution Systems
November – December 2015	Commissioning

Figure 24-3: Summary Project Execution Schedule



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24.1.10 Schedule Risks & Mitigation Plans

- Regarding the 2013 (and to a lesser extent, 2014) barging campaign, there is a risk the river levels will not support a four to six week campaign. To mitigate this risk, an assessment of the snowpack should be undertaken in February 2013. If the snowpack is below normal levels, the potential of developing and utilizing the low draft boats or barges should be investigated as an opportunity to extend the barging season for critical equipment mobilization.
- If the road construction crew is not on schedule, then the start of critical load transport (particularly the permanent camp modules and associated equipment) over the rough graded road will be delayed. If the road construction schedule is behind in the summer of 2014, steeper grades should be accepted temporarily to ensure the schedule is maintained. Under this condition, the contractor would return to complete construction to design requirements in 2015.
- Camp loading in the fall of 2014 prior to the 216 man camp being ready for occupancy is very constrained and additional space may be required. To mitigate this risk, a portion or all of the Atlin road crew camp could be moved to the mine site if required.

24.1.11 Opportunities

- Although it is not on the critical path, if there is an opportunity to advance the development of the plant site earthworks during the summer of 2013, it should be pursued given that this work will be controlled blasting and very methodical. It is scheduled for completion during the winter months of 2013/2014, but could be done more efficiently in summer weather.
- The above may also present the opportunity to establish some of the infrastructure foundations (e.g., shops/warehouse, fuel dispensing pads, LNG storage tank foundations and firewater tank foundation, etc.) in the fall and early winter of 2013.
- Low draft barges, if available, should be utilized to mobilize a portion of the permanent camp and fuel and firewater tank steel to site. This will help prevent delays to the permanent camp setup schedule and mitigate constrained camp loading issues.

25 Interpretations & Conclusions

25.1 Conclusions

The financial analysis of the feasibility study demonstrates that the project has positive economics and warrants consideration for detailed engineering and construction by Chieftain.

Standard industry practices, equipment and processes were used in this study. The Qualified Persons for this report are not aware of any unusual significant risks or uncertainties that could affect the reliability or confidence in the project based on the data and information available to date.

25.2 Risks

As with most mining projects, many risks could affect the economic outcome of the project. Most of these risks are external and largely beyond the control of the project proponents. They can be difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. Table 25-1 identifies what are currently deemed to be the most important internal project risks, potential impacts, and possible mitigation approaches, excluding those external circumstances that are generally applicable to all mining projects (e.g., changes in metal prices, exchange rates, smelter terms, transport costs, investment capital availability, government regulations, First Nation support, etc.).

25.3 Opportunities

Significant opportunities exist that could improve the economics, timing and/or permitting potential of the project. Most of these opportunities are also potential risks, as explained in the previous section. For example, metallurgical recoveries present both a risk and opportunity.

Opportunities not previously mentioned are shown in Table 25-2, excluding those that are typical to all mining projects, such as increases in metal prices. Further information and evaluation is required before these opportunities can be included in the project economics.

Table 25-1: Preliminary Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Dilution and Extraction Factors	Ore not extracted would reduce the mine's reserves and require accelerated development to meet production demands. Excessive dilution is one of the most critical internal risks at most underground mines and can lead to excessive milling costs, lower head grades, lower metal recoveries, lower metal recovery, increased tailings requirements, etc. An 11% drop in head grade takes the project to an after-tax break-even level, using a discount rate of 8%.	Well-planned definition drilling coupled with a comprehensive dilution control plan should provide adequate dilution and extraction control of the ore. Development advance needs to be kept well ahead of production needs so stoping can be controlled properly.
Resource Modeling	Resource volumes that were estimated using industry standard methods, but are still subject to some variation. Variability of grade and discontinuity of orebodies can be the biggest issues of a resource model that is not representative of the orebody.	Further definition drilling, careful mapping and regular resource model upgrades can significantly reduce the risk of an un-representative model.
Metallurgical Recoveries	The metallurgical recoveries in this study are based on numerous tests but results may vary when the actual orebody is mined. A drop in recoveries would have a direct impact on the project economics.	Continued testwork and optimization during the plant operations would help improve recoveries if results are below expectations.
Deleterious Elements	The concentration of deleterious elements in the concentrate could present problems with concentrate marketing and/or smelter penalties that could reduce the value of the concentrate.	Modeling of the deleterious elements in the concentrates will help define expected concentrations.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. An increase in OPEX of 15% would reduce the after tax $NPV_{8\%}$ by about $41 M. If OPEX increases then the mining cut-off grade would increase and, all else being equal, the size of the reserves would decrease. An increase in CAPEX of 15% would reduce the after tax $NPV_{8\%}$ by about $67 M.	Well-developed and controlled construction and operating plans, along with experienced personnel will greatly mitigate potential cost overruns.
Development Schedule	The project development could be delayed for a number of reasons and a change in schedule would alter the project economics.	Well-developed and controlled construction and operating plans, along with experienced personnel will greatly mitigate potential schedule overruns. Contingency planning will be conducted for project execution to help mitigate variances.
Ability to Attract Experienced Professionals	The ability of Chieftain to attract and retain competent, experienced professionals is a key success factor for the project. High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	The early search for professionals as well as the potential to provide living arrangements other than in a camp may help identify and attract critical people. A well-planned, comprehensive training program for local people would help increase the local content and likely improve employee retention.

Table 25-2: Project Opportunities

Opportunity	Explanation	Potential Benefit
High Potential Benefit		
Exploration Potential	The expansion of known mineral resources and the addition of new deposits may be possible with further resource drilling and could potentially extend mine life.	The expansion of the deposit resources could potentially lead to a longer project life and/or greater operating flexibility and potentially higher throughput justification. This becomes particularly important if higher grade mineral resources are defined that defer lower grade mineral resources currently utilized in the economic analysis.
Project Strategy and Optimization	Typically, feasibility study mine planning and scheduling can be improved upon with detailed engineering. In addition, leasing financing, streaming and other financial factors can be improved with further investigation.	Detailed optimization of the mine plan could result in improved economics.
Hydro Power	The current plan is to use LNG/diesel power generation on site at a cost of about $ 0.261 /kWh.	If hydro power is used, especially from May to November, significant power cost savings could be realized and the project economics potentially improved.

26 Recommendations

Based on the results of this study, the project is economic, and a decision whether or not to proceed with construction should be taken. If Chieftain decides to proceed, various permitting, financing and detailed engineering tasks need to be conducted before most construction activities can begin. Some of this work is well underway and ongoing costs are captured in the project development capital contained in this report.

The work programs described in the following sections are suggested to advance the project through front-end engineering design (FEED).

26.1 Recommended Work Programs

26.1.1 Metallurgy Future Testwork

There are three main areas of planned metallurgical work, which are summarized in the following subsections.

26.1.1.1 Copper Minerals Separation

The aim is to use the knowledge gained in trying to separate chalcopyrite and tennantite to improve the quality of the copper concentrate. The removal of the misreporting sphalerite, which is known to be well liberated but is recovered by both minor copper ion activation and by mass split, is also an important issue. At present, maximizing the recovery of tennantite for both copper and silver values makes it difficult to suppress this effect, which similarly allows some pyrite to intrude into the same products. If a protocol can be devised to accomplish this, it will benefit copper concentrate grade and zinc recovery. Given the high degree of liberation of the sphalerite (MODA Pty Ltd, Burnie, Report #3, November 2011), any sphalerite removed will be of sufficiently high grade to be directed to the final zinc concentrate. Not only will this increase zinc recovery and metal revenue, it will also reduce the penalty metal levels in the copper concentrate.

26.1.1.2 Gold Leaching

The refinement of the leaching circuit for the gravity concentrates needs to be completed. This includes optimizing the desired weight and grade of the gravity concentrate and getting a fix on the recovery of the non-electrum silver. The gravity concentrate will contain some galena and tennantite, both of which are relatively insoluble in cyanide solution and will be returned to the grinding circuit. As such, the non-electrum silver contained in the tennantite and galena should be recovered to the respective concentrates, but this needs confirmation. The exact levels of cyanide consumption are also required —although this probably has little impact on the total cyanide usage.

26.1.1.3 Physical Properties

From the point of view of better operational understanding, it is necessary to look at treating a larger sample. The resultant larger subsamples will assist in the above exercises and generate sufficient tailings for cemented backfill tests. Such a test will be scaled to provide enough zinc concentrate (i.e., minimum 4.5 kg) to conduct statutory TML tests for shipping purposes. The same sample will also be used to establish process design criteria, such as bulk density and angle of repose. The issue of representivity is a major one; mining a bulk sample close to the adits is not ideal as clearly demonstrated in previous work. Such material will have suffered some alteration (as well as being a single point sample) as the use of older core. Obtaining fresh material requires planning and is probably best done with a large diameter drill to drill multiple holes over a larger area.

26.1.2 Permitting

Several permits related to project construction were issued to the previous owner and have since been transferred to Chieftain. Section 20 provides a detailed listing of these permits, licenses and authorizations. This includes permit, license and authorization applications that are currently under review or remain to be submitted, as well as the operations waste discharge permit amendment, which must be submitted at least six months before any of the permitted work is required.

26.1.3 Paste Backfill

This section describes the laboratory testing that is required to confirm the assumptions in this study and to verify the design and cost estimates. The testing program is also needed to identify and quantify any risks and/or opportunities that may exist. JDS recommends that the testing program described in the following subsections be conducted.

26.1.3.1 Material Characterization

Material characterization testing is conducted to determine the make-up of the tailings and sand products to ensure samples are representative of ore types or blends. Material characterization will cover the following:

- specific gravity
- particle size distribution (PSD)
- pH
- mineralogy and chemical composition (including ABA).

26.1.3.2 Rheological Index Testing

The purpose of conducting rheological index testing is to evaluate the flow and fluidic properties of the sample being tested. This testing can determine how the material is handled in the process. The index testing gives a good indication as to how the material will react to various processing and handling methods such as pumping and slump adjustment. The flowing rheological index tests are recommended:

- yield stress (Vane method) vs. wt% solids content
- solids content vs. slump
- paste stability/water bleed
- slurry segregating/non-segregating transition
- slump vs. moisture content determination and water retention
- viscosity vs. solids content.

26.1.3.3 Unconfined (Uni-axial) Compressive Strength Tests

The UCS testing program is based on an initial assessment or screening phase of UCS tests to identify trends and preferred tailings supply sources (if any) based on a number of criteria (e.g., operating costs, tailings management, environmental impact, etc.).

Strength and cost relationship with one or more binders, recognizing there is potential degradation of strength due to sulphide and cement reaction, should also be assessed.

26.1.3.4 Settling/Dewatering Tests

With regard to the applicability of paste dewatering technology, preliminary settling, thickening and filtration characteristics will be determined through bench-scale tests to establish equipment sizing and applicability for paste preparation and capital cost. This program is comprised of flocculent screening (pulp density, flocculent dosage, overflow clarity), along with an assessment of column settling rates and densification for ranges of underflow solids concentration (typical thickener to high density and paste production).

To limit variability on the feed end, two options using classification systems may be considered. Cyclones or screens may be used to reduce the variability of tailings and to produce a more consistent product.

26.1.4 Underground Geotechnical

Additional work will be required during the detailed design and implementation phase when more site-specific details are known. This includes the following:

- A groundwater hydrology study is required to determine the ground water inflow.
- The support requirement for multiple cut-and-fill panels should be investigated.

Trade-off studies on supplementary cable bolting versus either temporary or permanent or artificial pillars (i.e., shotcrete posts) should be completed.

26.1.5 Tailings

Excavation within the TMF will need to be closely monitored to ensure that sufficient allowance is provided for variable groundwater levels to reduce the risk of liner uplift from rising groundwater levels. It is important to maximize the excavation within the TMF to minimize use of borrow areas outside of the TMF. The removal of material within the TMF also provides additional storage capacity for tailings. If sufficient materials cannot be obtained for the full dam height, the dam height will be lowered and raised later in the mine life with borrow material from alternate sources.

An Operations, Maintenance and Surveillance Manual (OMS) will be prepared prior to operations. The OMS manual will document operating procedures, including QA/QC, monitoring and maintenance requirements. An Emergency Preparedness Plan (EPP) will also be prepared.

TMF construction should be monitored by qualified personnel and an as-constructed report should be prepared to document all aspects of construction, including QA/QC.

An annual reconciliation of the water and mass balance should be carried out to confirm the in-situ density, as well as water inputs and evaporation losses.

The groundwater quality monitoring plan (developed by others) should be implemented and the contingency plan implemented, if required.

26.1.6 PAG & Pyrite Facilities

An Operations, Maintenance and Surveillance Manual (OMS) should be prepared prior to operations. The OMS manual will document operating procedures, including QA/QC, monitoring and maintenance requirements. An Emergency Preparedness Plan (EPP) will also be prepared.

The facilities construction should be monitored by qualified personnel and an as-constructed report should be prepared to document all aspects of construction, including the QA/QC.

An annual inspection and review of the performance of the facility should be carried out by a qualified geotechnical engineer.

26.2 Work Program Cost Estimate

The overall cost estimate to complete the proceeding work programs is approximately $3.107 M, of which $2.617 M is included in the capital cost estimate contained in this report. Details of the individual work programs are outlined below.

26.2.1 Metallurgy Program Cost

An estimated $100,000 is required to further investigate the copper mineral separation and the gold leaching aspects of the recommended testwork programs. It is assumed that this program will use the remaining core sample from the 2011/12 drilling. The drill samples should also be enough to generate 20 to 30 kg of tails and pyrite concentrates for backfill filtering and strength tests. Copper minerals separation is viewed as an opportunity and is not essential for the feasibility study test program.

The physical properties recommendation is a view to the longer term need to create both un-oxidized and larger samples (and such subsamples) for further confirmation testwork, but mainly to involve the backfill issues. This is estimated to cost $300,000 and is not included in the feasibility study capital cost estimate.

26.2.2 Permitting Program Cost

The estimated cost for the remaining permits and authorizations and supporting technical work is approximately $1,637,000. This is included in this feasibility study capital cost estimate.

26.2.3 Paste Backfill Program Cost

The cost estimate for the paste testing program outlined in Section 26.1.3 is approximately $116,000. This cost is not included in the feasibility study capital cost estimate.

26.2.4 Geotechnical Program Cost

The estimated cost for the hydrology study is $75,000, while the cost for the other geotechnical studies is included in the mining operating costs. The hydrology study cost is not included in the feasibility study capital cost estimate.

26.2.5 Tailings Program Cost

The total amount in the feasibility study capital cost estimate for the tailings and PAG and pyrite programs is approximately $980,000.

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McArthur Ore Deposit Assessments Pty Ltd (MODA) 2011, Chieftain Metals Inc Tulsequah project, Composite 1 Mineralogy, Report #2 for ALS Ammtech Metallurgy, August 2011, Text and appendices.

McArthur Ore Deposit Assessments Pty Ltd (MODA) 2011, Chieftain Metals Inc Tulsequah project, Composite 4 Mineralogy, Report #3for ALS Ammtech Metallurgy, November 2011, Text and appendices.

McArthur Ore Deposit Assessments Pty Ltd (MODA) 2011, Chieftain Metals Inc Tulsequah project, Composite 4 Mineragraphy, Report#4 for ALS Ammtech Metallurgy,December 2011, Text and appendices.

McArthur Ore Deposit Assessments Pty Ltd 2011 (MODA), Chieftain Metals Inc Tulsequah project, Tennantite Mineralogy, Report #5 for ALS Ammtech Metallurgy, December 2011, Text and appendices.

McArthur Ore Deposit Assessments Pty Ltd. 2011 (MODA), Chieftain Metals Inc Tulsequah project, Tennantite Microprobing, Report #6 for ALS Ammtech Metallurgy, December 2011, Text and appendices.

TetraTech Wardrop 2012, Amended Reviewed report - Draft version 2,Text and appendices

Wardrop 2007, Technical Report on the Tulsequah Chief Property,Report to Redfern Resources Ltd., March 2007, Text and appendices.

Wardrop Consulting 2007, Technical report on the Tulsequah Chief Property, internal report for Redfern Resources Ltd., report number 0652460100-REP-R0007-02, 2007, Text and appendices

Watters Consulting Pty Ltd. 2011, Historical Testwork Data Summary, September 2011, Text.

Watters Consulting Pty Ltd. 2011, Tulsequah Metallurgical Test Composite, August 2011, Text.

Attachment 1

Qualified Person Certificates

CERTIFICATE OF AUTHOR

I, Gordon Edward Doerksen, P.Eng. do hereby certify that:

1. This certificate applies to the Technical Report entitled ”Technical Report for the Tulsequah Chief Project of Northern BC, Canada”, with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as V.P. Technical Services with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C2T6;

3. I am a graduate of Montana Tech in 1990, I obtained a B.Sc. (Mining). I have been involved in mining since 1985 and have practised my profession continuously since 1990. I have held senior mine production and mine technical positions in mining operations in Canada the US and in Africa. I have worked as a consultant for over six years and have performed mine planning, project management and economic analysis work, as a Qualified Person, for a Large number of engineering studies and Technical Reports covering a wide range of mineral commodities in Africa, South America, North America and Asia. ;

4. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I am also a current Registered Professional Mining Engineer is Yukon, Alaska and Wyoming. I am a Founding Registered Member of the Society of Mining Engineers of the AIME;

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have visited the Tulsequah Chief Project site in March 2011;

7. I am responsible for Sections 1,2,3,19,21,22,24,25,26 and 27 ;

8. I have had prior involvement with the property that is the subject of this Technical Report. I was a QP and Project Manager for the Preliminary Economic Assessment and Technical report done for the Tulsequah project by SRK Consulting (Canad) Inc. in 2011;

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

“Signed and Sealed”

Gordon Doerksen

Gordon E. Doerksen, P. Eng.

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as Project Manager, Mining with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C2T6;

3. I am a graduate of Unversity of Alberta with a BSc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I am a Registered Professional Mining Engineer in Alberta and the Northwest Territories;

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have visited the Tulsequah Chief Project site in November 2012;

7. I am responsible for and/or shared responsibility for Sections 15 and 16, excluding Sections 16.3 and 16.8;

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

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Michael Makarenko

Michael E. Makarenko, P. Eng.

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CERTIFICATE OF AUTHOR

I, Robert L. Matter do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;
2. I am currently employed as a Project Engineer with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C2T6;
3. I am a graduate of the Montana College of Mineral Science and Technology with a Bachelor of Science in Mining Engineering, 1993. I have practiced my profession continuously since 1993;
4. I am a Registered Professional Mining Engineer in the state of Arizona;
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
6. I have not visited the Tulsequah Chief Project site;
7. I am responsible for and/or shared responsibility for Section 18;
8. I have had no prior involvement with the property that is the subject of this Technical Report;
9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Robert L. Matter

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CERTIFICATE OF QUALIFIED PERSON

I, Dr. Gilles Arseneau do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;
2. I am an associate consulting with the firm of SRK Consulting (Canada) Inc. ("SRK") with an office at Suite 22200, 1066 West Hastings Street, Vancouver British Columbia, Canada;
3. I am a graduate of the University of New Brunswick in 1979, the University of Western Ontario in 1984, and the Colorado School of Mines in 1995; I obtained a B.Sc., an M. Sc. and a Ph.D. degree. I have practiced my profession continuously since 1995. I have over twenty years of experience in mineral exploration including work on volcanogenic sulphide deposits in British Columbia and throughout Canada. I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101. I have over ten years' experience estimating mineral resource estimates using three dimensional block modeling software;
4. I am a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia, License #25474;
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
6. I have visited the Tulsequah Chief Project site on May 18 to 19, 2006, September 13 to 14, 2006 and on October 25 to 26, 2011;
7. I am responsible for Sections 4 to 12, 14 and 23 of the Technical report;
8. I have had prior involvement with the subject property. I am the author of two technical reports on the property dated November 2010 and September 2007;
9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Gilles Arseneau

Dr. Gilles Arseneau, P. Geo.

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Haughley Green
Suffolk
United Kingdom
IP14 3RP

CERTIFICATE OF AUTHOR

I, Kenneth John Sangster do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as Managing Director, with Ken Sangster and Assoiates Ltd,3 Cobbs Place, Fir Tree Lane, Haughley green, Suffolk, United Kingdom

3. I am a graduate of The University of Strathclyde, Glasgow, Scotland with a Honours Degree in Metallurgy, 1965. I have practiced my profession in various research ,operational and managerial roles in Europe,Africa,Australia and Noth and South America for 47 years;

4. I am a Registered Professional Chartered Enginer and a member of the Institute of Mining Materials an Metallurgy;

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have visited the Tulsequah Chief Project site twice in 2011 3-6 May,13-16 September;

7. I am responsible for and for Sections 13 and 17;

8. I have been responsible for directing the metallurgical work for Chieftain Metals since 2010 as a contractor prior to this Technical Report;

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Ken Sangster

Kenneth John Sangster, C.Eng.
Ken Sangster and Associates Ltd.

Marsland
Environmental
Associates

CERTIFICATE OF AUTHOR

I, Robert Marsland, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as a Senior Environmental Engineer, with Marsland Environmental Associates, in Nelson, BC;

3. I am a graduate of the University of Alberta with a M.Sc. in Environmental Engineering, June 1991. I have practiced my profession in the mining industry since 1989;

4. I am a Registered Professional Engineer in BC with membership number 25110;

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have visited the Tulsequah Chief Project site in November 2012;

7. I am responsible for and/or shared responsibility for Section 20;

8. I have done work on the Tulsequah property that is the subject of this Technical Report for the previous owners and through prior employers dating back to April 1994;

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Robert Marsland

Robert Marsland, P.Eng.
Senior Environmental Engineer
Marsland Environmental Associates Ltd.

CERTIFICATE OF AUTHOR

I, Harvey N. McLeod, do hereby certify that:

1. This certificate applies to the Technical Report entitled ”Technical Report for the Tulsequah Chief Project of Northern BC, Canada”, with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as Principal, with Klohn Crippen Berger Ltd. of 2955 Virtual Way, Vancouver, BC. V5M 4X6, Canada;

3. I am a graduate of the University of British Columbia with a Bachelor of Applied Science Degree in Geological Engineering (1973); and I am a graduate of the University of London with a Masters Degree in Soil Mechanics (1980) and a Diploma of Imperial College (1980). I have practiced my profession of geotechnical engineering for dams and mining projects since 1973;

4. I am a Registered Professional Engineer and Geoscientist of the Association of Professional Engineers of BC (No. 10432);

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have not visited the Tulsequah Chief Project site;

7. I am responsible for and/or shared responsibility for Sections 18.11, and 18.12;

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

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

“Signed and Sealed”

Harvey N. McLeod

____

Harvey N. McLeod, P.Eng., P.Geo. ;
KLOHN CRIPPEN BERGER LTD.; Principal

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Dave West, P.Eng.
David West Consulting
105 David Street,
Sudbury,
Ontario P3E 1T2

CERTIFICATE OF AUTHOR

I, Dave West do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as a Consultant, with David West Consulting

3. I am a graduate of the University of Newcastle upon Tyne with a Masters Degree in Rock Mechanics and Excavation Engineering, 1977. I have practiced my profession for 35 years;

4. I am a Registered Professional Engineer in Ontario, membership number 90523853;

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have visited the Tulsequah Chief Project site in March, 2012;

7. I am responsible for and/or shared responsibility for Section 16.3;

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

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Dave West

____
Dave West, P. Eng.

Frank Palkovits
Kovit Engineering Limited
311 Harrison Drive
Sudbury, ON P3E 5E1

CERTIFICATE OF AUTHOR

I, Frank Palkovits do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report for the Tulsequah Chief Project of Northern BC, Canada", with an effective date of December 12, 2012 prepared for Chieftain Metals Inc.;

2. I am currently employed as President, with Kovit Engineering Limited 311 Harrison Drive, Sudbury, ON;

3. I am a graduate of Laurentian University with a [Degree] in [Mining Engineering, 1988. I have practiced my profession mining engineering since 1988 and exclusively paste backfill since 2000;

4. I am a Registered Professional Engineer in the Professional Engineers Ontario (PEO);

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have not visited the Tulsequah Chief Project site;

7. I am responsible for and/or shared responsibility for Sections 16.8;

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

9. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: December 12, 2012
Signing Date: January 22, 2013

"Signed and Sealed"

Frank Palkovits

Frank Palkovits, P.Eng.
President
Kovit Engineering Limited