2014 Tulsequah Project: Feasibility Study Optimization

APPENDIX I

FEASIBILITY STUDY TECHNICAL REPORT TULSEQUAH CHIEF PROJECT, NORTHERN BRITISH COLUMBIA, CANADA

Tulsequah River Area
Northwestern BC
NTS 104K/12

Atlin Mining Division

Latitude 58°44'N, Longitude 133°35'W

BC Geological Survey
Assessment Report
35093b

By
Operator:

JDS Energy and Mining
#200 – 532 Leon Avenue,
Kelowna, British Columbia

For
Owner:

Chieftain Metals Inc.
2 Bloor Street West, Suite 2510
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Feasibility Study Technical Report
Tulsequah Chief Project,
Northern British Columbia, Canada

Effective Date: October 20, 2014
Report Date: November 27, 2014

Prepared for:

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NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Chieftain Metals Corp. The quality of information, conclusions and estimates contained herein are 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 Corp. 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 Corp. (Chieftain) to carry out an optimized and revised feasibility study (FS) of the Tulsequah Chief project (Tulsequah). The project encompasses two advanced stage polymetallic massive sulphide deposits known as the "Tulsequah Chief" and "Big Bull" deposits. The feasibility study solely focuses on the development of the Tulsequah Chief deposit. This technical report summarizes the results of the optimized FS and is prepared according to the guidelines of the Canadian Securities Administrators' National Instrument (NI) 43-101 and Form 43 101F1.

This FS technical report varies from the previous JDS FS published in 2012 (JDS 2012) in that it proposes a lower processing rate and the use conventional barging services to transport concentrate to market and supplies to site. The average process plant throughput in this report is 1,100 tpd whereas the processing rate in the 2012 FS was 2,000 tpd.

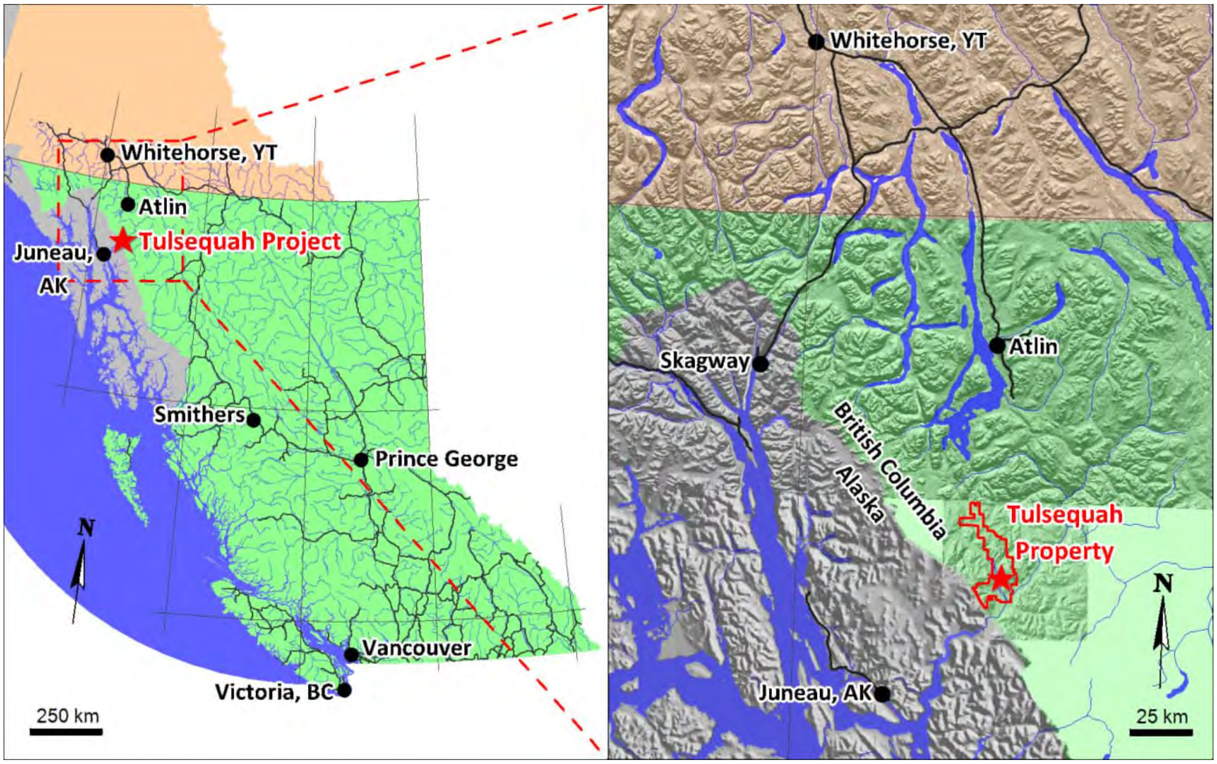
JDS managed the FS and completed the mining, infrastructure, metallurgy, processing and economics sections of the report. JDS was assisted by several principal-designated subcontractors providing report information as noted below:

- SRK Consulting (Canada) Inc. (SRK) Vancouver, Dr. Gilles Arseneau: property description, geology and mineral resource estimate;
- Marsland Environmental Associates (MEA): environmental and permitting;
- Klohn Crippen Berger Ltd. (KCB): geotechnical design for tailings management facility and potentially acid generating waste rock and pyrite containment facilities;
- David West Consulting (Dave West): underground geotechnical requirements analysis;
- Kovit Engineering Limited (Kovit): paste backfill testing assessment; and
- Ausenco Engineering Canada (Ausenco): barge and transportation logistics.

1.2 Property Description and 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 May to October from Juneau. The base camp is situated on the east bank of the Tulsequah River at an elevation of 55 meters above sea level (masl).

Figure 1.1: Project Location Map



Source: Chieftain 2014

Chieftain's property comprises 35 cell mineral claims and 25 overlapping crown granted mineral claims totalling 324.5km2. 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 Tulsequah Chief Mineral Claims

Property Area	Type	Tenure No.	Record No.	Area (ha.)	Good to Date
Tulsequah Chief	Mineral Claim	590422		420	December 31, 2023
	Crown Grant		5669	7.99	July 3, 2015
	Crown Grant		5668	20.90	July 3, 2016
	Crown Grant		5676	14.16	July 3, 2015
	Crown Grant		5670	20.90	July 3, 2015
	Crown Grant		5679	9.70	July 3, 2015

Source: Chieftain 2014

All mineral claims are in good standing, with the core Tulsequah Chief and Big Bull claims in good standing until year 2022. Mineral claims acquired directly by Chieftain through BC Mineral Titles Online have good to dates between 15th January 2015 and April 15 2017; these good to dates will be extended by Chieftain with the registration of technical exploration activities and subsequent filing of assessment reports with the BC Mineral Titles Office. Crown grants are maintained through annual tax payments due on July 2 of each year, and are in good standing through July 3, 2015. 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. However, a gold and silver streaming agreement with Royal Gold Inc. was established in December 2011.

1.3 Geology and 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, that are, in turn, covered by a thick package of sedimentary rocks, overlays the felsic volcanic host, 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 at Tulsequah Chief. In total, 10 surface holes and 50 underground diamond drill holes totaling 22,630 m were completed at Tulsequah Chief and included in the resource calculation. At Big Bull 22 surface holes totaling 8,827m were completed in 2011, upgrading inferred mineral resources and testing exploration targets.

Further exploration surface drilling was conducted at Tulsequah in 2013 with 3,450m in 9 surface holes. These holes tested new targets generated from re-interpreted legacy geophysical Induced polarization data, resulting in the intersected footwall stringer chalcopyrite VMS mineralisation in the newly named southwest zone, located 350m southwest of the known Tulsequah Chief sulphide lenses.

1.5 Mineral Processing & Metallurgical Testing

The ore is massive sulphide, dominated by pyrite, with barite, muscovite and minor quartz. Copper is contained in chalcopyrite, and tennantite/tetrahedrite, while lead is present in galena, and zinc is present mainly in sphalerite but also substituting for copper in tennantite/tetrahedrite. Gold is present as electrum containing on average 30% silver and 70% gold. Approximately 85% of the gold not recovered by gravity reports to the copper and lead concentrates. The silver in the gravity concentrate accounts for less than 1% of the silver in the ore, most of which occurs in tennantite/tetrahedrite, with a smaller amount in galena.

The upper zone composites prepared from the samples collected for the 2012 and 2014 test programs appear to be moderately fine grained and satisfactory liberation of the values is achieved by grinding to 80% finer than 53 microns (P_{80} 53µ). The upper zone massive sulphide ore has a relatively low work index so the grinding energy requirement is moderate and the ore is not very abrasive. To achieve 100% liberation of the lower zone ore a particle size of 80% passing 24 to 39 microns is required.

Historical test work programs were completed on the upper zone mineralization and included comminution, process mineralogy and gold recovery by gravity concentration and flotation processes.

In 2012, a metallurgical test program was conducted to assess the metallurgical performance of the mineralization to support the 2012 Feasibility Study. The test work indicated that sequential flotation would produce saleable copper, lead and zinc concentrates. The two composites provided by Chieftain, one from the upper zone and the one from the lower zone, responded well to gravity concentration and flotation.

In May 2014, additional test work was conducted to develop a flowsheet to produce two copper concentrates: a low arsenic, predominantly Chalcopyrite (Cp) product, and a high arsenic, predominantly Tennantite (Tn) product. The results proved that the copper could be separated but the recoveries were not as high as expected, possibly due to the oxidation of the aged samples. The decision was made to continue with one copper, lead and zinc concentrates yet designing flexibility in the plant layout and equipment to permit the split of copper concentrates option once the plant is in operation and additional test work is completed.

The mineralogical examinations and the bench scale test work done by ALS, Metallurgy 2012 Project T0662, were used to determine the best reagent regime, flotation conditions and flowsheet to develop the design criteria for the operating plant.

The results of the bench scale test work were used as a guide to plot best fit grade recovery curves for each metal. The resulting shape of the curve was projected to fit with the locked cycle tests, to provide the grade and recovery in the operating range, which is closer to the expected plant performance, taking into account the recirculation of cleaner tailings to the previous stage of cleaning.

Table 1.2: Projected Metallurgical Balance

Product	Wt (t)	Concentrate Assay Estimates					Recovery Estimates (%)				
		Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Cu	Pb	Zn	Ag	Au
Copper Conc	6.2	21	2.8	5.1	1300	22	89	13	4.5	78	47
Lead Conc	1.4	0.3	60	7.1	467	5.6	0.3	65	1.4	6.3	2.8
Zinc Conc	10.4	0.7	0.4	60	80	0.8	5	3.4	90	8	2.9
Pyrite Conc	33	0.2	0.3	0.6	22	0.3	3.6	8.5	2.9	6.9	3.6
Tailings	48.8	0.1	0.2	0.1	1.6	0.2	2	9	1	0.8	2.7
Feed	100	1.46	1.29	6.95	103.72	2.85	100	100	100	100	100
Gravity Concentrate	0.2	0.5	3	3	224	522	0.1	0.5	0.1	0.5	41

Source: JDS 2014

1.6 Mineral Resource Estimate

The Mineral Resource Statement presented in Table 1.3 represents the third mineral resource evaluation for the Tulsequah Chief project and second for the Big Bull deposit, prepared in accordance with the Canadian Securities Administrators' NI 43-101.

The mineral resource model prepared by SRK considers 818 core boreholes drilled by Cominco, Redfern and Chieftain at Tulsequah Chief and 313 at Big Bull during the period of 1940 to 2011. The resource estimation work was supervised 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 October 20, 2014.

Table 1.3: Tulsequah Chief and Big Bull Mineral Resources (Inclusive of Mineral Reserves) as of October 20, 2014

Tulsequah Chief

Category	MTonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn Eq (%)
Measured	0.787	1.57	1.5	8.6	2.81	105.5	30.9
Indicated	5.136	1.43	1.28	6.76	2.8	102.1	28.1
Total M+I	5.923	1.45	1.31	7	2.8	102.5	28.5
Inferred	0.439	0.79	1.03	5.54	2.33	80.6	21.6

Big Bull

Category	MTonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn Eq (%)
Indicated	0.653	0.34	1.54	4.11	3.03	125	23.8
Inferred	1.453	0.37	1.37	4.15	2.67	103.9	21.4

Total Combined Tulsequah Chief and Big Bull

Category	MTonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn Eq (%)
Measured	0.787	1.57	1.5	8.6	2.81	105.5	30.9
Indicated	5.789	1.31	1.38	6.46	2.83	104.7	27.6
Total M+I	6.576	1.34	1.33	6.71	2.82	104.8	28
Inferred	1.892	0.47	1.29	4.47	2.59	98.5	21.5

1. $100/tonne Net Smelter Return (NSR) cut-off used
2. The cut-off value is based on a price of US$ 1,250.00 per ounce of gold, US$ 19.00 per ounce for silver, US$ 0.90 per pound for zinc and lead and US$ 2.75 for copper and recoveries of 90.0 % for gold, 84.5 % for silver, 89.0 % for copper, 66.2% for lead and 89.0 % for zinc.
3. Resource: Zn EQ% = ((Au g/t*36.69x)+ (Ag g/t*0.5013)+ (Cu %*36.24)+ (Pb %*9.39)+ (Zn %*10.2))/10.2

Mineralized lenses were modeled 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 6.6 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 pass required that at least three drill holes and five composites be available within the search ellipse to estimate a grade within a block in the upper mine adjacent to the old workings or the lower H2 area of Tulsequah Chief where there is high density drilling information. 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 drill holes and three composites be available within the search ellipse to estimate a grade within a block, and again a maximum of eight composites were used to interpolate a grade value. The Third 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 grades for the Tulsequah Chief project were estimated by Dr. Gilles Arseneau, P. Geo and were classified according to the Canadian Institute of Mining's (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

SRK is satisfied that the geological modeling 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 m to 30 m.

1.7 Mineral Reserve Estimate

The mineral reserves identified in Table 1.5 comply with CIM definitions and standards. 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 proven and 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 1.4: Mineral Reserve Estimate

Category	Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn Eq (%)
Proven	684,000	1.48	1.36	7.84	2.71	100.59	29.4
Probable	3,752,000	1.45	1.28	6.78	2.88	104.39	28.9
Total P + P	**4,436,000**	**1.46**	**1.29**	**6.95**	**2.85**	**103.72**	**29.0**

1. Underground mineral reserves are reported at a NSR cut-off of US$200/tonne.
2. Cut-off grades are based on a price of US$1,250/oz of gold, US$19/oz for silver, US$0.90/lb for zinc and lead and US$2.75 for copper and recoveries of 90% for gold, 84.5% for silver, 87.8% for copper, 65.1% for lead and 89.3% for zinc.
3. Reserve: Zn EQ% = ((Au g/t*36.64x)+ (Ag g/t*0.4991)+ (Cu %*36.73)+ (Pb %*8.81)+ (Zn %*10.04))/10.04
4. Reserve tonnes and grades include dilution

Source: JDS 2014

1.8 Mining

The Tulsequah deposit will be accessed via the 120 m (former 5400 level) and 60 m (5200 level) portals. An additional portal will be driven at approximately 84 m level that will act as the SAG mill feed conveyor drift from the mine. The existing 5200 and 5400 levels will be slashed to 5.0 m x 5.0 m to accommodate the trackless equipment fleet. The main mine access will be via the 60 m level and connect to the main ramp that will access the mining levels. The main ramp is 5.0 m x 5.0 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° 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 1,100 tpd production rate by the second year of full production. The mine development and production plan is shown in Table 1.6 with ore, waste and backfill tonnages rounded to the nearest thousand tonnes.

Underground mine infrastructure will have a primary crusher and crushed ore bin. Crushed ore will be conveyed to the crushed ore bin, from where it will exit the mine via the 84 m 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; and
- 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 pre-production to be stored underground.

An insufficient volume of waste rock is available for the backfill requirement; hence, the use of paste fill has been incorporated into the mine plan. Paste fill consists of process tailings partially dewatered and mixed with cement. This material will be 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 will be added to produce cemented paste fill, which will be 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).

Table 1.5: Mine Development & Production Plan

Parameter	Unit	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	Totals
Total Mine Production	**kt**	**24**	**324**	**405**	**409**	**413**	**409**	**409**	**408**	**408**	**410**	**411**	**404**	**4,436**
Daily Production Rate	tpd	-	887	1,111	1,122	1,131	1,118	1,121	1,119	1,117	1,120	1,126	1,107	**1,098**
Gold Grade	g/t	2.91	2.26	2.49	2.82	3.06	2.86	3.11	3.09	3.18	2.34	3.27	2.74	**2.85**
Silver Grade	g/t	106.83	86.68	103.01	112.59	126.84	119.84	128.05	104.60	73.89	78.69	100.44	102.20	**103.72**
Copper Grade	%	1.50	1.28	1.43	1.51	1.83	1.79	1.85	1.36	1.11	1.55	1.22	1.07	**1.46**
Lead Grade	%	1.33	1.35	1.33	1.30	1.48	1.30	1.45	1.28	1.03	1.10	1.25	1.30	**1.29**
Zinc grade	%	9.87	7.18	7.31	7.13	8.12	7.72	7.85	7.27	6.37	5.30	5.54	6.47	**6.95**
Net Smelter Return	$/t	326	257	280	298	337	319	337	300	267	245	281	267	**291**
Total Lateral Development	**m**	3,233	3,772	3,064	3,327	2,242	1,944	1,602	1,204	1,201	1,069	891	877	**24,423**
	m/d	8.9	10.3	8.4	9.1	6.1	5.3	4.4	3.3	3.3	2.9	2.4	2.4	**5.1**
Raise Development	m	20	216	368	348	301	151	150	-	-	-	-	-	**1,554**
Mined Underground Waste	kt	177	171	157	172	115	74	84	50	23	19	8	19	**1,069**
Paste Backfill Placed	kt	-	102	116	209	59	171	132	151	192	189	219	203	**1,743**

Source: JDS 2014

1.9 Recovery Methods

The concentrator design includes standard crushing and grinding unit operations, gravity gold recovery and conventional froth flotation to recover mineral concentrates of chalcopyrite/tennantite (copper iron sulphide), sphalerite (zinc iron sulphide) and galena (lead sulphide) from the ground ore.

The concentrates will be transported to designated smelters worldwide for subsequent reduction into copper, zinc, and lead metal. Average mill throughput is planned to be approximately 1,100 dry tonnes per operating day (1,219 dry tonnes per day at 90% utilization). Total annual concentrate production is planned to be approximately 72,000 dry tonnes.

Listed below are the major process unit operations planned for the Chieftain mine site:

- Primary underground jaw crusher and 2,000 t capacity UG fine ore storage;
- Conveyance of material from the crusher to the main process facility;
- Semi-autogenous grinding (SAG) and two ball mills in series within closed circuit cyclone classification;
- Copper, lead and zinc sequential flotation to produce sulphide concentrates;
- Copper, lead, and zinc concentrate dewatering through thickening and filtration;
- Pyrite flotation, dewatering and tailings thickening for storage in the pyrite and tailings pond or deposition of paste to the underground workings;
- Process water, fire water, potable water distribution; utility air distribution:
- Storage areas for 2-tonne bags of copper, lead and zinc concentrate;
- Reagent storage and mixing; and
- Assay Laboratory.

The primary jaw crusher is planned to be located underground which also houses the jaw crusher discharge feeder and related ancillary systems, as well as the drive system for the conveyor belt that transfers crushed ore to the storage bin. The conveyors and mill building are planned to be completely enclosed.

The ore is proposed to be delivered by truck and dumped through a grizzly that feeds a dump pocket prior to the underground jaw crusher. The ore is planned to be crushed to a nominal 100 mm product size and conveyed to the underground 2,000 t storage prior to the SAG mill.

The comminution circuit is planned to consist of one SAG mill followed by two ball mills. The ball mill grinding circuits are designed to operate in closed circuit with the cyclones. Designated cyclone underflows feed the gravity concentrators in both ball mill circuits to recover approximately 41% of the free gold.

The cyclone overflow, at approximately 30% solids, P_{80} of 45 microns, is planned to flow by gravity to the copper circuit. Copper, lead and zinc concentrates would be produced from a conventional sequential flotation circuit in typical rougher and cleaner configuration. The tailings from each circuit are designed to feed the next.

The flotation concentrate products are planned to be dewatered in high rate thickeners with the under flow feeding filter feeding stock tanks. Dedicated pressure filters are designed to dewater the concentrates to a target moisture content of approximately 8%. The filter cake is planned to be placed in 2-tonne bags for storage in a designated area near the barge load-out facility.

The acid generating pyrite concentrate is proposed to be stored separately from the benign tailings. Thickened pyrite concentrate would report to the pyrite pond for temporary storage prior to being reclaimed as part of the paste fill underground. Thickened de-pyritized tailings are designed to report to the paste plant or TMF.

The process plant make-up water is planned to supply as fresh water or treated water from the effluent treatment plant. Fresh water is sourced from the Tulsequah River.

1.10 Project Infrastructure

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

- Diesel fueled power plant, heat recovery system and power distribution network;
- Bulk fuel storage tanks and containment;
- Construction and permanent camp (total 160 beds) including potable and wastewater treatment plants and incinerator to complement the existing 50 beds;
- Mill complex;
- Administration offices, mine dry and maintenance shop;
- Acid and effluent treatment plant;
- Tailings management facility; and
- Temporary PAG waste and pyrite concentrate storage facility.

In addition to the infrastructure listed above, improvements are planned for the existing barge facility, site road network and airstrip to better accommodate the operation.

These activities are scheduled to be completed during the two-year pre-production period.

1.11 Environmental Studies

The Tulsequah Chief Mine 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. The AMD 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 the winter of 2011 / 2012, 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 22, 2012 and the plant remains on care and maintenance, in contravention of the Fisheries Act and the EMA permit. The ATP will be restarted as part of Tulsequah project construction.

At the request of the British Columbia Ministry of the Environment, British Columbia based independent scientists from Palmer Environmental Consulting Group, Core6 Environmental Ltd. and Triton Environmental Consultants evaluated the water quality at four sites on the Tulsequah River near the confluence of the Taku River where the mine is located. The group studied the discharge impact on various types of fish including coho salmon, sockeye salmon, dolly varden and bull trout and chinook salmon. Fish tissue studies showed the fish were unaffected and that there was no discernable impact from the historic discharge resulting from previous mining operations. Overall, the potential risk to aquatic receptors as a result of mine discharge is considered low, the report stated. The report also stated the discharge does not affect the Taku River.

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 165 ha. The existing area of disturbance at the site is approximately 110 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 Tulsequah Chief project was issued a provincial Environmental Asses. Several permits related to the construction of the Tulsequah Chief project were issued to the previous owner, and have since been transferred to Chieftain.

1.11.1 Permitting

At the present time, Chieftain is waiting for the Minister of Environment's determination that the project has been substantially started. Once this decision has been received, the Environmental

Assessment Certificate (EAC) will remain valid throughout the project life. This determination is expected in December 2014. Subsequent to the determination, Chieftain will initiate a process for amending the Environmental Certificate to incorporate several changes to the project design including the reduced mill throughput, the TMF starter dam and using conventional river barges to haul concentrate down the river.

In parallel with the EAC amendment, Chieftain will resume permitting activities, primarily to amend the existing Mines Act permit and Environmental Management Act permit as needed, to encompass mill and TMF construction and operation.

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.

Chieftain continues to hold regular community meetings to update the community on the progress of the project.

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 project execution plan utilizes seasonal barging as the primary delivery method for equipment and materials that are required for the construction of the project. Previous studies called for the construction and utilization of an all-weather access road as the primary method for hauling of concentrate to port and fuel and supplies to the mine.

As part of the revised FS, Chieftain retained the services of Ausenco Engineering Canada Inc. to assess, analyze and report on the navigability of the Taku River. A logistics study and execution plan for access to the mine site for construction and operations via river transportation was developed. Based on estimates developed by JDS and the analysis conducted by Ausenco, it was determined that construction freight can be feasibly transported to site by means of river barging.

The following key objectives were respected during the development of the project execution plan:

- Utilize the revised 2014 FS to determine labour, equipment and material requirements;
- Assume project funding is received on January 1st, 2015 to maintain schedule;
- Minimize all cost impacts required to achieve the schedule; and
- Identify key risks and mitigation/contingency plans associated with the project execution plan.

1.13 Capital Cost

The initial capital cost estimate is $198.0M, as summarized in Table 1.6. Costs are expressed in Canadian dollars with no escalation (Q4 2014 dollars). The target estimate accuracy is (-10%/+15%).

Table 1.6: Capital Cost Summary

Pre-Production CAPEX	Pre-Production $M	Production $M	LOM $M
Underground Mining	18.4	61.0	79.4
Underground Infrastructure	10.5	0	10.5
Site Development	3.9	0	3.9
Processing Plant	44.6	0	44.6
Tailings & Waste Rock Management	6.6	12.7	19.2
On-Site Infrastructure	33.9	4.3	38.1
Off-Site Infrastructure	0	0	0
Project Indirects	15.2	0	15.2
Engineering & EPCM	13.5	0	13.5
Owner's Costs	21.4	0	21.4
Closure & Salvage	0	3.8	3.8
Pre-Production OPEX	11.7	0	11.7
Subtotal	**179.6**	**81.7**	**261.3**
Contingency (11.4%)	**18.4**	**2.4**	**20.7**
Total Capital Costs	**198.0**	**84.1**	**282.1**

Source: JDS 2014

1.13.1 Reclamation/Closure & Salvage Costs

Reclamation, closure and salvage costs are listed in Table 1.7 and show a new closure cost, after salvage credits, of $3.8M.

Table 1.7: Reclamation/Closure & Salvage Costs

Cost	$M
Reclamation/Closure	8.2
Salvage	-4.4

Source: JDS 2014

1.13.2 Basis of Capital Estimate

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 -2 and Y 1) and is based on self-performed mining (owner forces). No leasing contracts 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 rebuilding, and mining infrastructure installations as defined by the mine plan.

The closure/reclamation estimate is based on preliminary work scope determined through a design report issued by Gartner Lee (2008) and updated for this study.

1.14 Operating Cost

The average, Life of Mine (LOM) unit operating cost is estimated at $185.78 per tonne processed and is summarized in Table 1.8.

Table 1.8: Operating Cost Summary

Area	Unit Operating Cost $/tonne processed
Mining	29.36
Processing	32.24
Power	36.16
Transportation	33.23
G&A	28.50
Concentrate Transportation*	26.28
Total Unit Operating Cost Incl. Transportation	**185.78**

(*) Concentrate Transportation costs were estimated as part of the economic model. They are shown here to demonstrate all-in operating costs.

Source: JDS 2014

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

- Mining operations will be performed by Owner forces utilizing Owner purchased equipment;
- All electrical power will be generated at site using diesel generators with a long-term delivered price of diesel of $1.112/L yielding an estimated LOM power cost of $0.326/kWhr;
- The process plant will process 1,100 tpd (~408,000 tpa) of ore and produce approximately 72,000 dry tonnes of concentrate per year;
- Seasonal barge supply and concentrate transportation augmented by multiple weekly freight and personnel flights;
- Tailings will be disposed of in a lined, conventional tailings dam;
- The mine will utilize a peak workforce of approximately 228 people (including all contract labour); and
- Concentrate production in containerized bags will be barged to a transfer point at the mouth of the Taku River then barged to the port of Seattle where it will be loaded on ocean going ships to Asia.

1.15 Economic Analysis

An engineering economic model was developed to estimate the project value and investment return. 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, operating costs, and capital costs to determine their relative importance as project value drivers.

The economic analysis presented does not include financial securities required to be posted by Chieftain for the Tulsequah project for the purposes of permitting.

This technical report contains forward-looking information resulting from projected mine production rates and resulting forecasted cash flows as part of this study. The grades are based on sufficient sampling that is reasonable 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.

Other economic factors include:

- Discount Rate of 8% (sensitivities using other discount rates have been calculated for each scenario);
- Nominal 2014 dollars;
- No Inflation;
- Numbers are presented on 100% ownership and do not include management fees or financing costs; and
- Exclusion of all pre-development and sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.). However, pre-development and sunk costs are utilized in tax calculations.

Metal Prices

The metal prices used in the Base Case economic analysis are spot metal prices as at October 15, 2014. An additional scenario was evaluated, utilizing forward-looking metal prices published by Consensus Economics (October 2014), an independent macroeconomic survey firm that prepares monthly compilations of metal prices using more than 30 analysis covering over 25 commodities.

Table 1.9 summarizes the spot metal prices and exchange rate as of October 15, 2014, and Consensus Economics forward-looking prices and exchange rates.

Table 1.9: Metal Prices Used in the Economic Analysis

Commodity	Unit	Base Case Spot as at 15-Oct-14	Forward Pricing Consensus Economics Publication Oct-14
Copper Price	US$/lb	3.08	3.38
Lead Price	US$/lb	0.93	1.10
Zinc Price	US$/lb	1.06	1.18
Gold Price	US$/oz	1,238	1,373
Silver Price	US$/oz	17.00	23.07
Exchange Rate	US$:C$	0.89	0.90

Source: JDS 2014

Life of Mine Production

Recovered metals for both economic scenarios are shown in Table 1.10.

Table 1.10: Life of Mine Plan Summary

Parameter	Unit	Value
Mine Life	Years	11.1
Total Ore	M tonnes	4.4
Mill Throughput Rate	tpd	1,100
Average Head Grade		
Cu	%	1.46
Pb	%	1.29
Zn	%	6.95
Au	g/t	2.85
Ag	g/t	103.72
Metal Production		
Cu Concentrate Produced	dmt	274,256
	Average dmt/yr	24,760
Pb Concentrate Produced	dmt	61,868
	Average dmt/yr	5,586
Zn Concentrate Produced	dmt	462,089
	dmt/yr	41,718
Au Payable	k oz committed	62
	k oz uncommitted	294
	Total k oz	**356**
	Average Total k oz/yr	**32**
Ag Payable	k oz committed	2,739
	k oz uncommitted	8,217
	Total k oz	**10,956**
	Average Total k oz/yr	**989**

Source: JDS 2014

1.15.1 Streaming Contract

In December 2011, Chieftain entered into a gold and silver purchase transaction with Royal Gold Inc. (Royal Gold). The agreement assigns a portion of the precious metals expected to be produced at the Tulsequah Chief mine to Royal Gold. Chieftain has received $10M in upfront payments upon the signing of the contract. In July 2014, Chieftain amended the agreement and will receive an additional US$45M for the project build (down from the US$50M in December 2011 and to be received upon progression of construction completion for the project).

The advance and future proceeds will allow Royal Gold to purchase, upon production of the Tulsequah Chief mine:

- 17.50% of payable gold up to 65,000 oz, payable at 30% of the daily London price quotation, and 8.75% of the gold production thereafter; and
- 25.00% of payable silver up to 3,000,000 oz, payable at 25% of the daily London price quotation, and 12.50% of the silver production thereafter.

The contract has been included in the economic analysis of the project. Total gold and silver ounces expected to be sold to Royal Gold Inc. under this contract total 62.3koz and 2.7Moz, respectively.

1.15.2 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; and
- 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; and
- Streaming Revenues – Streaming revenues were adjusted according to income tax regulations to appropriately determine the taxable income for the project.

The tax analysis completed amount to a LOM taxes payable of $136.6 M. The after-tax values are determined solely for project valuation purposes.

1.15.3 Financial Performance

Pre-tax and after-tax financial performance is summarized in Table 1.11. Pre-tax results provide a point of comparison with similar project and are not intended to represent a measure of absolute economic value.

Table 1.11: Summary of Economic Results

Category	Unit	Base Case Metal Prices	Forward Metal Prices
Net Revenues	$M	1,421	1,629
Operating Costs	$M	708	708
Cash Flows from Operations	$M	713	921
Capital Costs	$M	282	282
Up-Front Streaming Revenues	$M	51	50
Net Pre-Tax Cash Flow	$M	482	689
Pre-Tax $NPV_{8\%}$	$M	212	334
Pre-Tax IRR	%	25	33
Pre-Tax Payback	Years	3.8	3.2
Total Taxes	$M	137	211
Net After-Tax Cash Flow	$M	345	478
After-Tax $NPV_{8\%}$	$M	146	228
After-Tax IRR	%	22	29
After-Tax Payback	Years	3.9	3.2

Source: JDS 2014

1.15.4 Sensitivity Analysis

A sensitivity analysis was conducted on after-tax net present values (NPV8%) and average annual operating cash flows for individual parameters including metal prices, grades, operating costs, and capital costs. The results are shown in Tables 1.12 through Table 1.15. The project proved to be most sensitive to changes in metal prices and head grades, followed by operating costs. The project showed least sensitive to capital costs.

Table 1.12: After-Tax $NPV_{8\%}$ Sensitivity Test Results – Base Case Pricing

Factor	-10%	100%	10%
Metal Price	76.6	145.6	213.2
Head Grade	87.7	145.6	203.1
OPEX	175.0	145.6	115.9
CAPEX	170.5	145.6	120.7

Source: JDS 2014

Table 1.13: Average Annual Operating Cash Flow (M$) Sensitivity – Base Case Pricing

Factor	-10%	100%	10%
Metal Price	49.7	64.8	79.9
Head Grade	52.1	64.8	77.6
OPEX	71.0	64.8	58.6

Source: JDS 2014

Table 1.14: After-Tax $NPV_{8\%}$ Sensitivity Test Results – Forward Pricing

Factor	-10%	100%	10%
Metal Price	152.8	228.4	303.1
Head Grade	163.3	228.4	293.2
OPEX	257.3	228.4	199.4
CAPEX	253.2	228.4	203.5

Source: JDS 2014

Table 1.15: Average Annual Operating Cash Flow Sensitivity (M$) – Forward Pricing

Factor	-10%	100%	10%
Metal Price	66.5	83.4	100.3
Head Grade	68.8	83.4	98.1
OPEX	89.6	83.4	77.2

Source: JDS 2014

The project was also evaluated using various after-tax discount rates to determine the effect on project NPV. As expected, project NPV declined as the discount rate increased.

Table 1.16 and Table 1.17 demonstrate the summary of the discount rate sensitivity results on both metal price scenarios evaluated.

Table 1.16: Discount Rate Sensitivity Test Results – Base Case Pricing

Discount Rate (%)	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0	481.7	345.2
5	290.5	204.6
8	**211.7**	**145.6**
10	169.5	113.8
12	134.0	86.7

Source: JDS 2014

Table 1.17: Discount Rate Sensitivity Test Results – Forward Pricing

Discount Rate (%)	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0	689.0	478.4
5	438.3	302.4
8	**334.4**	**228.4**
10	278.7	188.3
12	231.6	154.2

Source: JDS 2014

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 $713.7M and an after-tax cash flow of $345.2 M over an eleven-year mine life. This results in an after-tax IRR of 21.8% and a $145.6 M NPV at 8%.

Several opportunities exist that could improve the economics of the project. They include:

- Expansion of resources and reserves;
- Potential to increase production;
- Reduced mining dilution;
- Improved metallurgical recoveries; and
- Hydroelectric power.

The main internal risks (excluding external risks such as metal price, exchange rates, regulatory changes, financing, etc.) associated with the project include:

- CAPEX and OPEX cost;
- Mining dilution and extraction factors;
- Metallurgical recoveries;
- Deleterious elements;
- Barge system capacity;
- Construction schedule; and

Ability to hire and retain experienced professionals.

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

2. INTRODUCTION

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 summarized scope of work for each company is listed below.

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;
- Establish recovery values based on metallurgical testing results;
- Design process plant to realize the predicted recoveries;
- Identify proper site plant facilities and other ancillary facilities;
- Estimate all initial and sustaining capital expenditures requirements and operating costs for processing; and
- Estimate all initial and sustaining capital expenditures requirements and operating costs for waste storage, tailings disposal and water storage.

SRK Consulting (Canada) Inc.

- Project setting, history and geology description; and
- Mineral resource estimate.

Marsland Environmental Associates Ltd.

- Prepare site wide water balance with supporting report; and
- Summarize status of existing and anticipated permits.

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.

David West Consulting

- Assess the mining rock geomechanics of the project.

Kovit Engineering Limited

- Complete the paste backfill testing; and
- Preliminary paste backfill plant and distribution system.

Ausenco Engineering Canada

- Barging logistics and transportation.

2.3 Qualifications & Responsibilities

Qualified persons are listed in Table 2.1. Qualified Person certificates are provided in Appendix A at the end of this technical report.

Table 2.1: Qualified Person Responsibilities

Author	Company	Report Section(s) of Responsibility
Mr. Gordon 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 16.9
Mr. Scot Klingmann, P. Eng.	JDS	18, excluding 18.14.1 but including paragraph “Potential Shortfall Effects, and excluding 18.25 and 18.26
Ms. Kelly McLeod P. Eng.	JDS	1.5, 1.9, 13, 17, 26.1.1
Mr. Gilles Arseneau, Ph.D., P.Geo	SRK	4,5, 6, 7, 8, 9, 10,11,12,14, 23
Mr. Robert Marsland, P. Eng.	MEA	20
Mr. Harvey N. McLeod, P. Eng.	KCB	18.25, 18.26
Mr. Dave West, P. Eng.	David West	16.3
Mr. Frank Palkovits, P. Eng.	Kovit	16.9
Ms. Nadia Krys, P. Eng.	Ausenco	18.2, 18.14.1 (except paragraph “Potential Shortfall Effects” under 18.14.1)

Source: JDS 2014

2.4 Site Visits

- Gordon Doerksen visited the project on March 20-21, 2011;
- Michael Makarenko visited the project site November 5-6, 2012;
- Scot Klingmann visited the site on June 10, 2014;
- Gilles Arseneau visited the project on May 18 - 19, 2006; September 13 - 14, 2006 and on October 25 - 26, 2011;
- Rob Marsland was last on site July 24-29, 2014 and May 12-15, 2014. He also visited the site November 6-9, 2012, September 11-14, 2012, June 6, 2012 and May 14-17, 2012;
- Dave West visited the site on March 14-15, 2012;
- Harvey McLeod has not visited the site;
- Nadia Krys visited the site on July 1 to 2, 2014; and
- Frank Palkovits has not visited the project site.

2.5 Currency

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

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

µ	Micron	km^2	square kilometer
°C	degree Celsius	kPa	kilopascal
°F	degree Fahrenheit	kVA	kilovolt-amperes
µg	Microgram	kW	kilowatt
A	Ampere	kWh	kilowatt-hour
a	Annum	l	liter
bbl	Barrels	l/s	liters per second
Btu	British thermal units	m	meter
C$	Canadian dollars	M	mega (million)
cal	Calorie	m^2	square meter
cfm	cubic feet per minute	m^3	cubic meter
cm	Centimeter	min	minute
cm^2	square centimeter	MASL	meters above sea level
d	Day	mm	millimeter
dia.	Diameter	mph	miles per hour
dmt	dry metric tonne	MVA	megavolt-amperes
dwt	dead-weight ton	MW	megawatt
ft	Foot	MWh	megawatt-hour
ft/s	foot per second	m^3/h	cubic meters per hour
ft^2	square foot	opt, oz/st	ounce per short ton
ft^3	cubic foot	oz	Troy ounce (31.1035g)
g	Gram	ppm	part per million
G	giga (billion)	psia	pound per square inch absolute
Gal	Imperial gallon	psig	pound per square inch gauge
g/L	gram per liter	RL	relative elevation
g/t	gram per tonne	s	second
gpm	Imperial gallons per minute	st	short ton
gr/ft^3	grain per cubic foot	stpa	short ton per year
gr/m^3	grain per cubic meter	stpd	short ton per day
hr	Hour	t	metric tonne
ha	Hectare	tpa	metric tonne per year
hp	Horsepower	tpd	metric tonne per day
in	Inch	US$	United States dollar
in^2	square inch	USg	United States gallon
J	Joule	USgpm	US gallon per minute
k	kilo (thousand)	V	volt
kcal	kilocalorie	W	watt
kg	kilogram	wmt	wet metric tonne
km	kilometer	yd^3	cubic yard
km/h	kilometer per hour	yr	year

Source: JDS 2014

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.

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.

Elements of Section 20 were provided by Keith Boyle of Chieftain. Rob Marsland of MEA reviewed this section and assumed responsibility for its content.

Sections 7-12 were provided by Brett Armstrong of Chieftain. Gilles Arseneau of SRK reviewed these sections and assumed responsibility for its content.

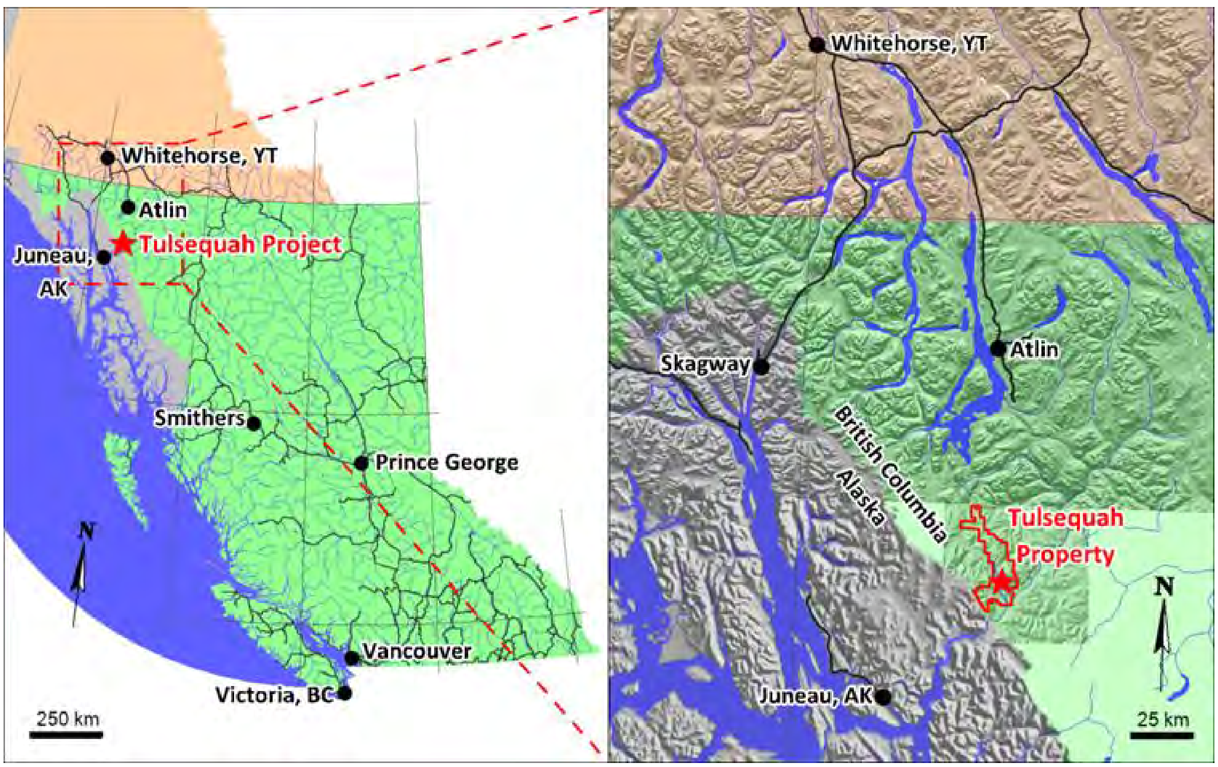
The results and opinions expressed in this report are conditional on the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein. The authors reserve the right, 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 AND LOCATION

Chieftain's Tulsequah Property is located at 58°43'N and 133°35'W on the Tulsequah River in northwestern British Columbia (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 on the Taku River May to October from Juneau. The base camp is situated on the east bank of the Tulsequah River at an elevation of 55 metres above sea level (masl).

Figure 4.1: Property Location Map



Source: Chieftain 2014

4.1 Mineral Tenure

The Tulsequah Chief property comprises 35 cell mineral claims (Table 4.1 and Figure 4.2) and 25 crown granted mineral claims (Table 4.2) totaling 324.53 km^2. All mineral claims are in good standing, with the core Tulsequah Chief and Big Bull claims in good standing until 2022. Mineral claims acquired directly by Chieftain through BC Mineral Titles Online have good to dates between 15th January 2015 and April 15 2017; these good to dates will be extended by Chieftain with the registration of technical exploration activities and subsequent filing of assessment reports with the BC Mineral Titles Office. The Crown grants are maintained through annual tax payments due on July 2 of each year, and are in good standing through July 3, 2015. 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. Chieftain's Mineral Claim 513828 was expropriated by the BC Government under the Park Act on July 6th 2012 with the establishment of the Taku River/T'akú Téix' Conservancy, and as such Chieftain does not hold the mineral title rights to this claim, the Gold Commissioner has commenced compensation discussions.

Table 4.1: Chieftain Tulsequah Chief Cell Mineral Claims

Tenure No.	Area (ha)	Good to Date
513806	1,241	31-Dec-22
513807	1,242	31-Dec-22
513809	1,393	31-Dec-22
513812	622	31-Dec-22
513813	807	31-Dec-22
513814	1,160	31-Dec-22
513815	1,311	31-Dec-22
513818	1,616	31-Dec-22
513819	841	31-Dec-22
513820	1,094	31-Dec-22
513821	842	31-Dec-22
590422	420	31-Dec-23
1011222	151	15-Apr-17
1017199	17	15-Apr-17
1017642	1,508	15-Apr-17
1017643	1,594	15-Apr-17
1017644	1,659	15-Apr-17
1017645	1,506	15-Apr-17
1017646	838	15-Apr-17
1017647	1,673	15-Apr-17
1017696	1,671	15-Apr-17
1017697	1,619	15-Apr-17
1017699	168	15-Apr-17
1017700	420	15-Apr-17
1017701	84	15-Apr-17
1017702	202	15-Apr-17
1017907	1,533	15-Apr-17
1017909	1,633	15-Apr-17
1017910	1,682	15-Apr-17
1022381	33	15-Apr-17
1025125	268	15-Jan-15
1025514	1,487	28-Jan-15
1026167	50	21-Feb-15
1026326	17	27-Feb-15
1031109	50	23-Sep-15
Total	**32,453**	**35 Claims**

Source: Chieftain 2014

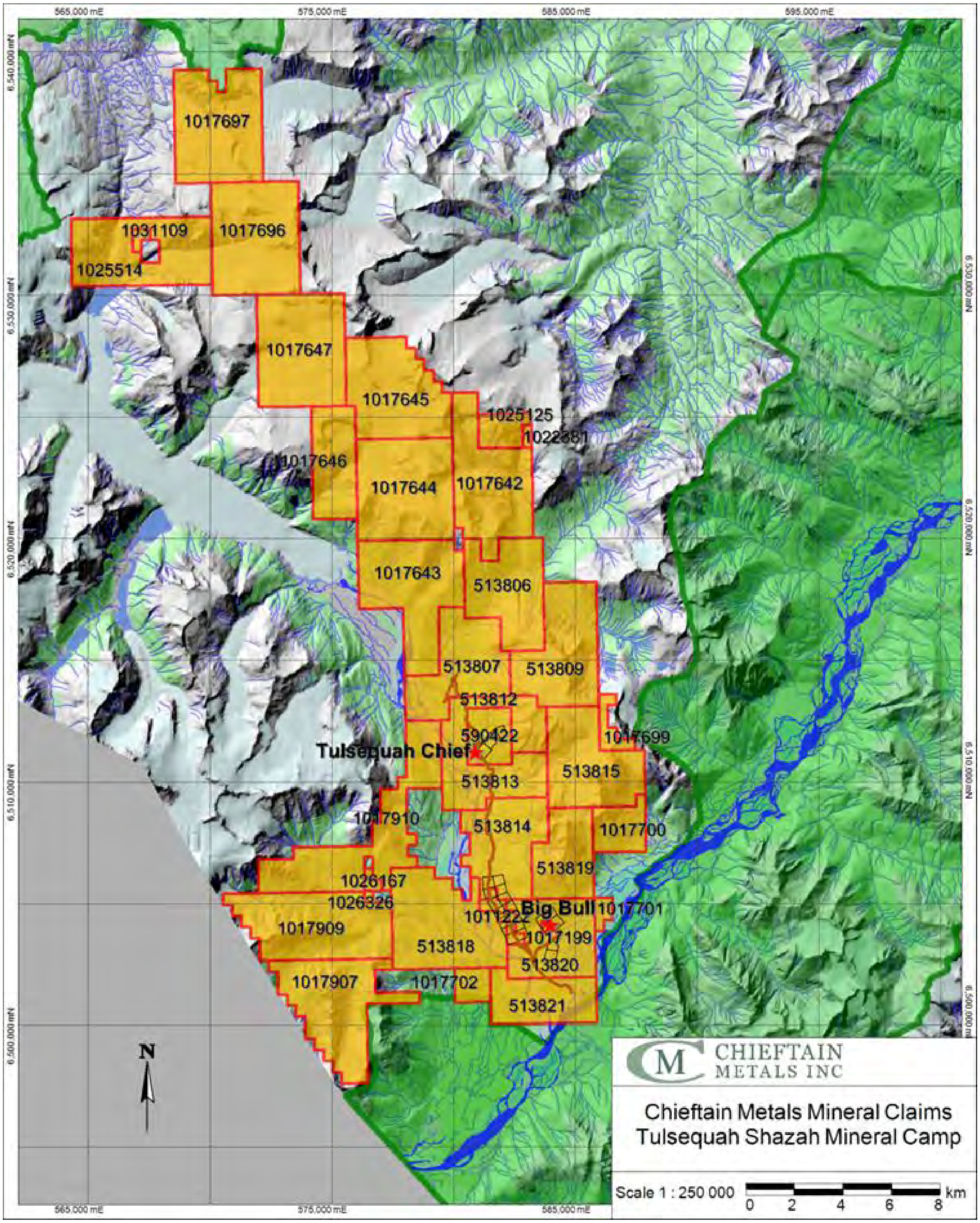
Table 4.2: Tulsequah Chief Crown Grants

Property Area	Record No.	Units	Area (ha)	Expiry Date
Crown Grants 1				
River Fr.	5669	1	7.99	3-Jul-15
Tulsequah Bonanza	5668	1	20.9	3-Jul-15
Tulsequah Bald Eagle	5676	1	14.16	3-Jul-15
Tulsequah Chief	5670	1	20.9	3-Jul-15
Tulsequah Elva Fr.	5679	1	9.7	3-Jul-15
Big Bull Crown Grants 1				
Big Bull	6303	1	20.65	3-Jul-15
Bull No. 1	6304	1	16.95	3-Jul-15
Bull No. 5	6306	1	14.57	3-Jul-15
Bull No. 6	6305	1	17.22	3-Jul-15
Hugh	6308	1	20.71	3-Jul-15
Jean	6307	1	17.02	3-Jul-15
Banker Crown Grants 1				
Vega No. 1	6155	1	20.9	3-Jul-15
Vega No. 2	6156	1	17.62	3-Jul-15
Vega No. 3	6157	1	18.97	3-Jul-15
Vega No. 4	6158	1	19.85	3-Jul-15
Vega No. 5	6159	1	14.94	3-Jul-15
Janet W. No. 1	6160	1	18.95	3-Jul-15
Janet W. No. 2	6161	1	18.75	3-Jul-15
Janet W. No. 3	6162	1	16.6	3-Jul-15
Janet W. No. 4	6163	1	20.76	3-Jul-15
Janet W. No. 5	6164	1	18.2	3-Jul-15
Janet W. No. 6	6165	1	19.02	3-Jul-15
Janet W. No. 7	6166	1	18.78	3-Jul-15
Janet W. No. 8	6167	1	17.98	3-Jul-15
Joker	6169	1	16.6	3-Jul-15

1. Maintained through annual tax payments due July 2 of each year .All Crown grants overlie mineral clams held by Chieftain.

Source: Chieftain 2014

Figure 4.2: Tulsequah Chief Mineral Claims



4.2 Underlying Agreements

Chieftain holds a 100% interest in both the mineral claims and the crown grants. There are no back-in rights or 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

Mining operations in the 1950s by a prior owner of the Tulsequah property have left a residual acid mine drainage (ARD) problem resulting from oxidation of in-mine sulphides and acidic waters carrying dissolved metals draining into the Tulsequah River. Previous remediation efforts by Redfern Resources Ltd. (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 them to install an ATP 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..

Chieftain 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 constructed the ATP at the Tulsequah Chief Mine site in the second half of 2011.

During ATP operations, treatment of mine-impacted water showed significant improvement in discharge water quality, with test results showing a reduction of greater than 98% of the total metals load into the receiving environment when compared with untreated mine water. Operations were curtailed at the plant on June 22, 2012, as the ATP had been operating outside its design parameters. Although the effluent had been meeting guidelines, the ATP had been operating below designed levels of efficiency, with higher than budgeted operating costs. The ATP is planned to be re-started upon project financing.

Accredited third party consultants completed an Aquatic Ecological Risk Assessment report in December 2013. It concludes that the fish resource downstream from the Tulsequah Chief mine site is at a healthy level and the 60 years of historic discharge posed low risk to fish.

4.5 Mining Rights in British Columbia

Under the BC Mineral Tenure Act, Chieftain can maintain the cell mineral claims in good standing by filing assessment work between $5-20 per Ha 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 AND 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 main access to the mine site is by air. Access is easiest by fixed-wing aircraft or by helicopter from Atlin or Juneau. 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 Chief Mine site and south to the Taku River barge landing area. Helicopters are intermittently based in the Tulsequah Valley, but otherwise must be chartered from Atlin or Juneau.

Barging on the Tulsequah River is the only transportation route to and from the Mine for inbound construction equipment, annual supplies and outbound ore concentrate. The River originates in northwest British Columbia's Boundary Range and flows about 265 km before emptying into the Taku Inlet southeast of Juneau, Alaska. This is a shallow river, which is primarily fed by snowmelt. The majority of the annual water flow occurs during May through October. A significant amount of sand and silt is deposited each year on the riverbed due to glacial melt that continuously changes the depth and course of the River. Customized fleet of shallow draft river barges and tugs are specified for this project. The use of forward sound sonar, differential GPS units and weekly reconnaissance trips to map out the best barging route is recommended.

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

5.2 Local Resources & Infrastructure

The property is remote and currently only accessible by air or shallow-draft boat. 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 bed via sandwells.

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

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

Waste rock disposal areas have been identified 1 km south of the mine portals near Rogers Creek on the east side of the Tulsequah River. Site construction for the temporary storage of the historic potentially acid generation rock near completion, only requiring liner installation.

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 communities 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 ice fields: 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

Source: Chieftain 2014

Figure 5.1 is looking south down the Tulsequah River towards the confluence with the Taku River. The Shazah camp and air strip are in the foreground with the Tulsequah Chief mine site 4 km distant, and the ATP visible adjacent to the shore.

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 meters above mean sea level (AMSL).

In 1923, the Tulsequah Chief was bonded to the Alaska Juneau Gold Mining Company which did 60 feet of unsuccessful tunneling and relinquished its option. In 1928, a syndicate represented by W.A. Eaton and Dan J. Williams of Juneau optioned the Tulsequah Chief Property and turned the previous tunnel to the left and penetrated “good grade ore over an exceptionally promising width” (J.T. Mandy, BC resident mining engineer, report to the minister of mines 1929). In spring of 1929, they optioned the property to the United Eastern Mining Company which conducted diamond drilling and efficient and aggressive development with the two uppermost adits, the historic ‘A’ (6500) and ‘B’ (6400) levels complete.

The Big Bull deposit was staked by V. Manville in 1929 with massive sulphide outcropping in a small creek over a width of 1.8-7.6 m and a strike of 140 m. The later discovery in 1929 of the Potlatch (Sparling), Banker, Ericksen-Ashby, and the Whitewater (Polaris Taku) deposits contributed further to the favorable publicity given to the area.

The Big Bull portion was optioned by the Alaska Juneau Gold Mining Company (Juneau Gold) in 1929, which completed a 610 m adit on the Big Bull occurrence and ten cross cuts, relinquishing their option in 1930. Leta Exploration optioned the Big Bull property in 1944, completing six underground drill holes (L-1 to L-6), and declined to make their option payment and abandoned the property.

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 tpd. Total production was 932,926 t (572,463 tonnes from the Tulsequah Chief mine and 360,473 tonnes from the Big Bull deposit). Average grade of ore was 1.57% Cu, 1.53% Pb, 6.93% Zn, 4.09 g/t Au, and 126.1 g/t Ag. The mines produced 29.76 Mlbs Cu, 28.15 Mlbs Pb, 126.81Mlbs Zn, 96,675 oz Au, and 3,364,528 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. There were no listed reserves at Big Bull at Shutdown in 1956. 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 1971, the deposit was reinterpreted as volcanogenic massive sulphides, rather than fault controlled hydrothermal replacement. Very little work is reported in the Tulsequah and Big Bull areas between the 1957 Tulsequah Chief shut-down and 1980. Cominco re-commenced exploration in 1980 at Big Bull, with surface mapping and soil geochemical sampling at an 8.9 km cut grid.

Redfern commenced a reconnaissance exploration joint venture with Comaplex Resources International Ltd. (Comaplex) in 1980, which ultimately resulted in the staking of a block of claims surrounding Cominco claims 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 volcanogenic massive sulphide (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 between the Tulsequah Chief and Big Bull deposits. Redfern acquired its partner's interest in their Joint Venture Tulsequah Chief claims and initiated discussions with Cominco concerning joint exploration. In 1987, an agreement was signed whereby Redfern could acquire a 40% interest in Cominco's amalgamated claims by funding the first $3M 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, and in 1994 initiated full feasibility studies, completed in 1995 by Rescan Environmental Services Ltd. (Rescan) and updated in 1997. 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 exploration 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.

In 2006, Redcorp commissioned Wardrop Engineering Inc. (Wardrop) 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 93.7 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 tpd. 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 $160M and an IRR of 30% based on an 8% discount rate. Wardrop also estimated a mineral resource for the Big Bull Deposit in 2007 in the indicated category of 211,000 t grading 3.33% Zn, 0.40 % Cu, 1.25 % Pb, 3.04 g/t Au and 162 g/t Ag, and additional 699,000 t were estimated in the inferred category.

The Wardrop mineral resource and reserves estimates were prepared in accordance with NI 43 101 and used categories for mineral resources and reserves as stipulated in NI 43-101. The estimates are believed to be reliable but are no longer relevant as it is replaced by the estimates presented in Sections 14 and 15 of this report.

Subsequent to the completion of the feasibility study in 2007, Redfern, the 100% owned subsidiary of Redcorp Ventures Ltd. (Redcorp), undertook a comprehensive mine permitting and development program at the Tulsequah property. Construction activities included: two construction camps, 25 km site roads, 1,050 m air strip, mill site surface striping, drilling and blasting, waste rock storages areas and construction laydown areas. 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 (Companies' Creditors Arrangement Act) 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 crossings and bridge authorizations from Fisheries and Oceans Canada, Transport Canada and BC Ministry of Environment.

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 British Columbia Supreme Court approved the purchase and a Vesting Order issued to Chieftain granting full, unencumbered ownership of the Tulsequah claims, crown grants and 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.

In 2011, Chieftain conducted an exploration drilling program at Tulsequah Chief of 22,630 m, directed at increasing indicated resources, and 8,527 m drilling at Big Bull directed at small exploration step outs and increasing indicated resources. In 2012, JDS Energy & Mining Inc. (JDS) was commissioned by Chieftain to carry out a feasibility study of the Tulsequah Chief deposit; this included an update of the mineral resource incorporating the 2011 Drilling. SRK Consulting Inc. (SRK) estimated the Indicated Mineral Resource at 6.75Mt grading 1.19 Cu(%), 1.10 Pb(%), 5.89 Zn(%), 2.40 Au(g/t) and 85 Ag (g/t), and 0.20Mt Inferred resources grading 0.20Mt grading 0.67 Cu(%), 0.76 Pb(%), 4.02 Zn(%), 1.81 Au(g/t) and 62 Ag (g/t). JDS estimated a Probable Mineral Reserve of 6.45Mt grading 1.12 Cu(%), 1.04 Pb(%), 5.59 Zn(%), 2.30 Au(g/t) and 81.38 Ag (g/t).

This study was based on a 2,000 tpd underground mine with a 9-year mine life, and a pre-production capital expenditures of $439.5 million, including approximately $125 million for 128 km road to connect the mine site to the Provincial road network at Atlin BC. Operating costs were estimated at $NSR 126/tonne. The feasibility study yielded a pre-tax NPV8% of $192.7 million and an IRR of 16.5% and post-tax NPV8% of $146 million and an IRR of 14.9%, using three year trailing average price deck of: Cu $3.66/lb, Pb US$ 1.01/lb, Zn $0.97/lb, Au US$ 1,455/oz, and Ag $20/oz and an exchange rate of CAD/USD = 1.01. The JDS mineral resource and reserves estimates were prepared in accordance with NI 43-101 and used categories for mineral resources and reserves as stipulated in NI 43-101. The estimates are believed to be reliable but replaced by the estimates presented in Section 14 of this report.

7. GEOLOGICAL SETTING AND MINERALIZATION

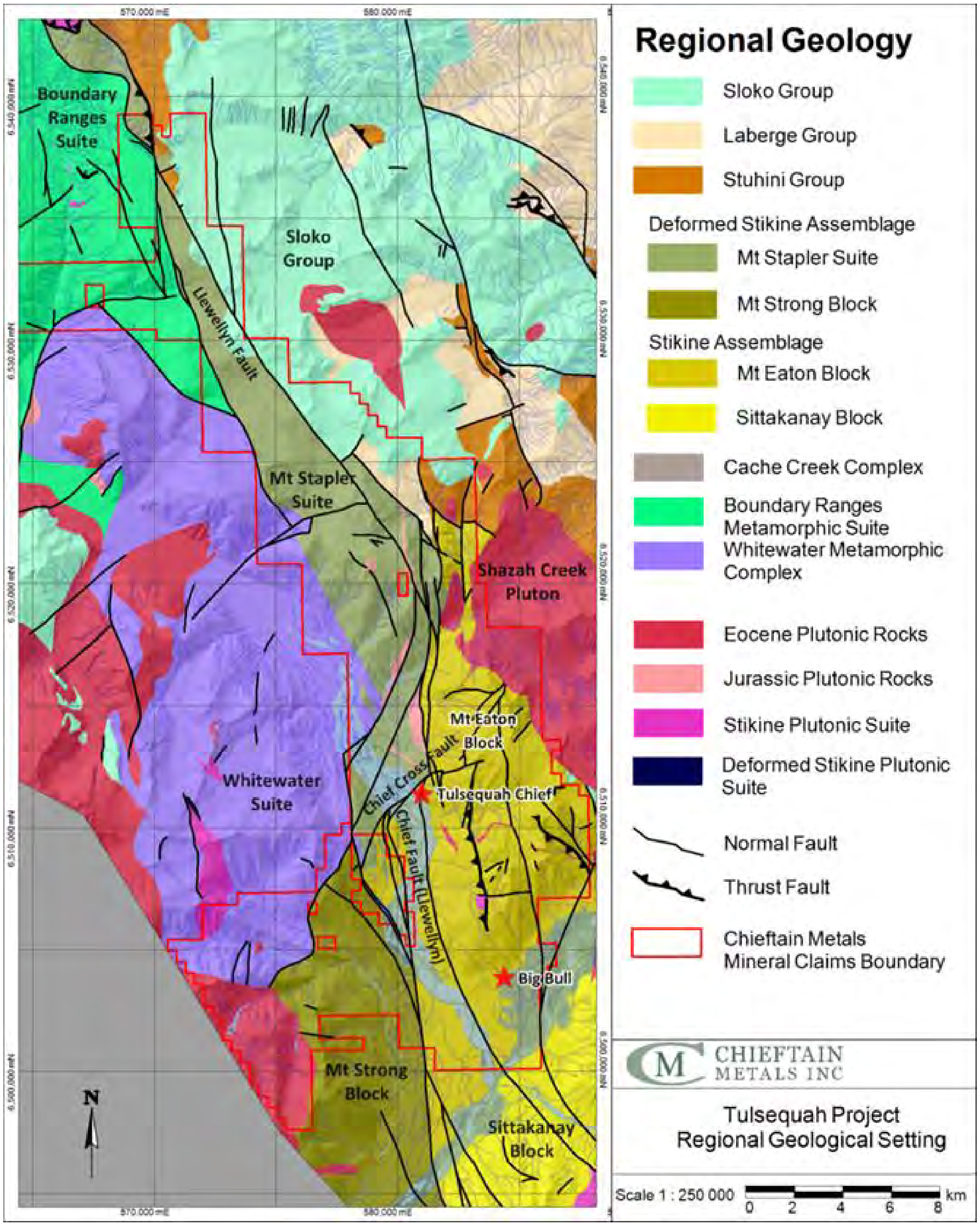
7.1 Regional Geology

The regional geology of the Tulsequah area (Figure 7.1,) is characterized by fault juxtaposition of several diverse Paleozoic to Mesozoic tectono-stratigraphic terranes, which have been variably deformed. Subsequent intrusions by Jurassic to Cretaceous age Coast plutons, and unconformable burial by Tertiary Sloko volcanics contributes further to the deformation and complexity in this region (Mihalynuk et al, 1994).

The dominant structural feature of the region is the Llewellyn Fault (known locally as the Chief Fault) which separates higher grade metamorphic rocks of Paleozoic and older ages on the west from weakly metamorphosed Paleozoic and Mesozoic rocks on the east. West of the fault three suites of rocks are recognized: the Whitewater Suite which consists of an amphibolite grade metamorphic sequence of sedimentary origin, the Boundary Ranges Suite, consisting of schists of volcanic and sedimentary origin, and the Mount Stapler Suite, a low-grade metamorphic package which shares characteristics of both the Whitewater and Boundary Range suites and may be gradational to both. East of the fault Paleozoic rocks of the Stikine Assemblage include the Mount Eaton Block comprising low metamorphic grade volcanic rocks of island arc affinity which host the Tulsequah Chief and Big Bull sulphide deposits.

Deformation and metamorphic grade in the Tulsequah region decrease from west to east. Lithologies range from polyphase deformed high grade gneisses in the Boundary Ranges suite to lower greenschist grade volcanic rocks of the Mount Eaton block. The latter have been affected by an upright to steeply overturned, north trending, open to isoclinal fold event. A second, less well developed fold event overprints the first. North trending, steeply dipping faults show evidence of numerous reactivations and intrusion by late Tertiary Sloko dykes.

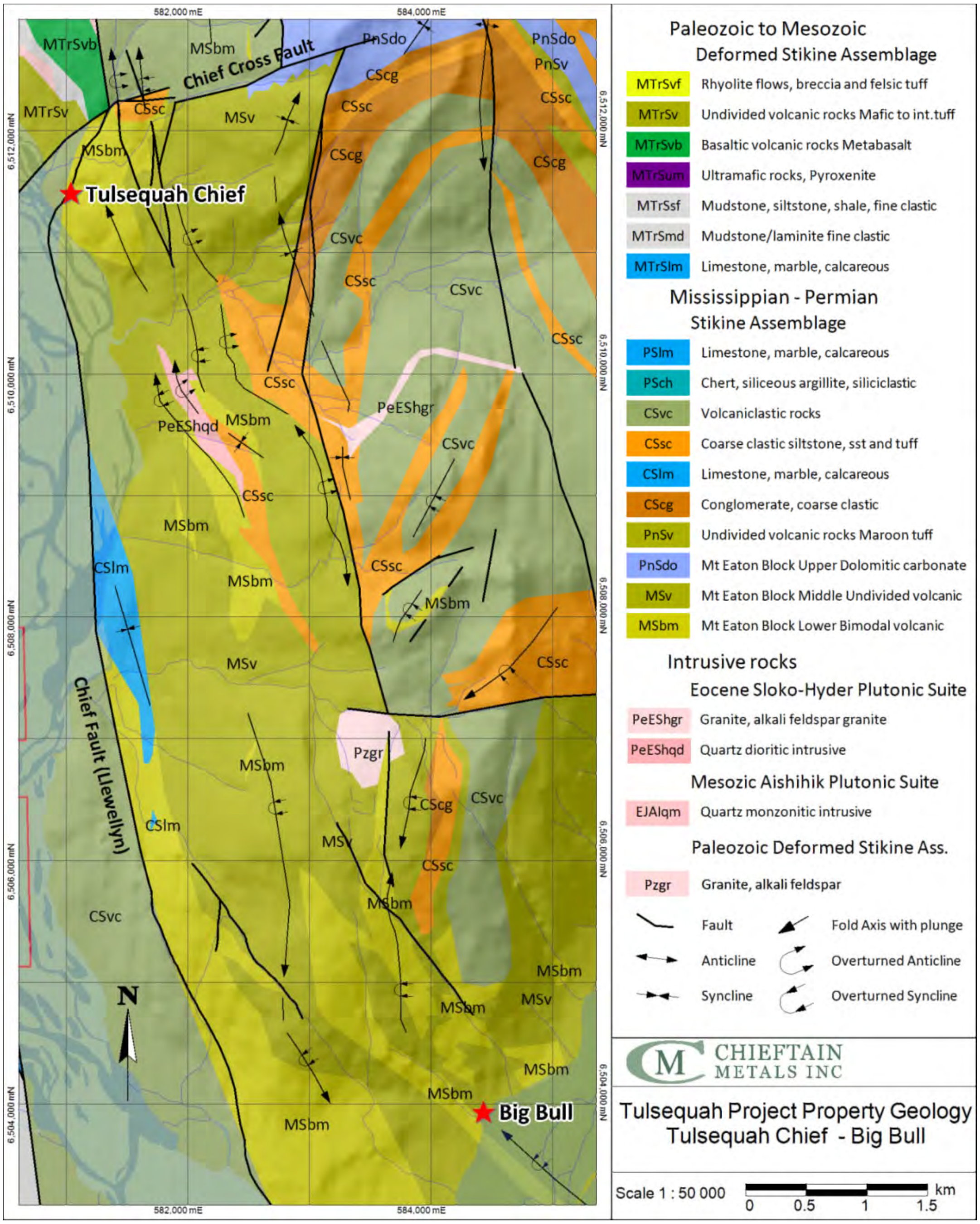
Figure 7.1: Tulsequah Property Regional Geologic Setting



7.2 Property Geology

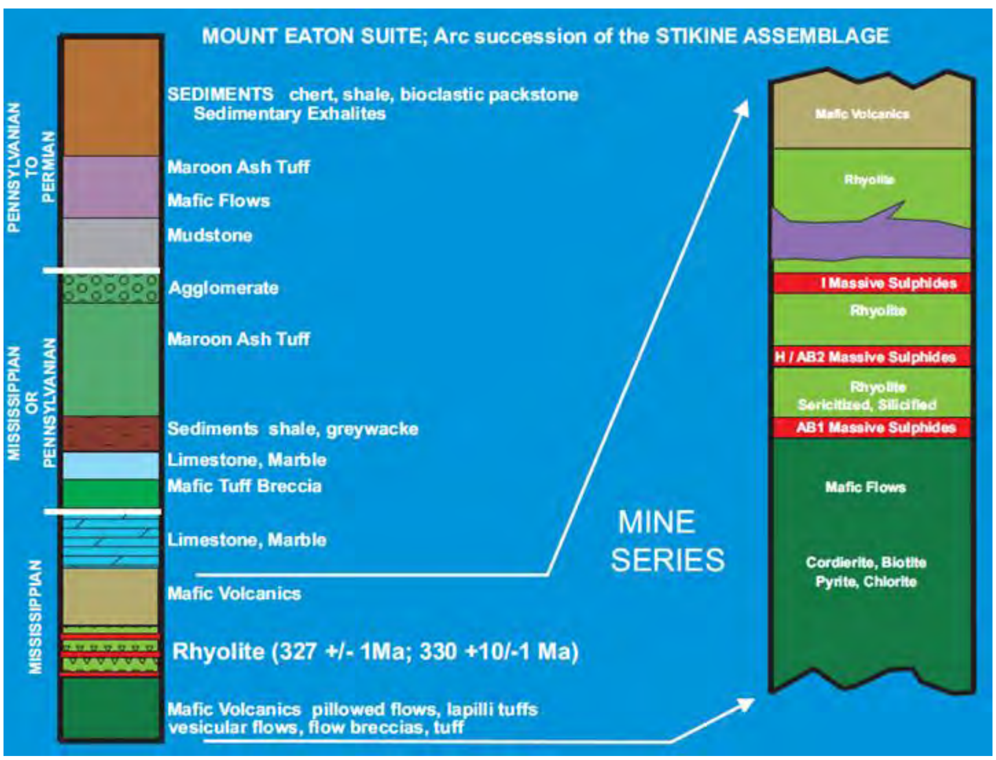
The Tulsequah property is dominantly underlain by rocks of the Mount Eaton Block, an low metamorphic grade island arc volcanic sequence of Devono-Mississippian to Permian age contained within the Stikine Terrane of northwest British Columbia (Mihalynuk et al, 1994). These rocks lie east of the Chief (Llewellyn) fault and are predominantly located north of the Taku River and east of the Tulsequah River, (Figure 7.2).

Figure 7.2: Tulsequah Chief Property Geology



Work by the BCGS Mihalynuk et al 1994, Mineral Deposits Research Unit (MDRU) Sherlock et al 1993 and Redfern Resources outlined a stratigraphy of the Mount Eaton block based on mapping, biochronology, lithogeochemistry and isotopic age determinations. The stratigraphy has been subdivided into three divisions, (Figure 7.3). 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 dominantly of pyroxene bearing mafic breccias and agglomerates with locally extensive accumulations of mafic ash tuffs and volcanic sediments. The transition from the Middle to Upper Divisions is marked by polymictic debris flows and/or conglomerate. The Upper Division, Pennsylvanian to Permian in age, consists primarily of volcanic derived and clastic sediments with lesser mafic flows. Distinctive bioclastic rudite and intercalated chert, shales and occasional sulphidic exhalite occur near the top of the Upper Division. The Mount Eaton suite is overprinted by sub-greenschist to middle greenschist facies metamorphism (Mihalynuk et al., 1994) characterized by the breakdown of pyroxene and amphibole to chlorite and epidote, and potassium feldspar to sericite. Late Tertiary Sloko rhyolite and mafic dykes cut the Paleozoic units and commonly intrude along re-activated north-trending faults.

Figure 7.3: Mount Eaton Suite Stratigraphic Column (modified after Mihalynuk et al 1994)



Source: Redfern 2005

Structure in the Mount Eaton block is dominated by the north trending, eastward verging Mount Eaton anticline which plunges moderately north and dips steeply west. A number of parasitic upright to overturned folds (F1) which range from open to near isoclinal occur on the western limb of this anticline. This first phase of folding (F1) is refolded by a second, east-west fold phase (F2) that is irregularly expressed across the property and locally produces a cross-cutting cleavage (S2). The F2 folds are generally upright and open. F1 folds are not significantly re-oriented by the F2 second phase of folding although they do exhibit variable plunge attitudes. F1 fold axes generally plunge to the north in the northern half of the property with southern plunges more common in the southern areas. In the Tulsequah Chief area, folds are open and plunge at 55° to 60° to the north with steep westerly dipping axial planes.

At Big Bull upright to overturned moderate to tight folds plunge at less than 40° to the northwest, with steep southwesterly dipping axial planes.

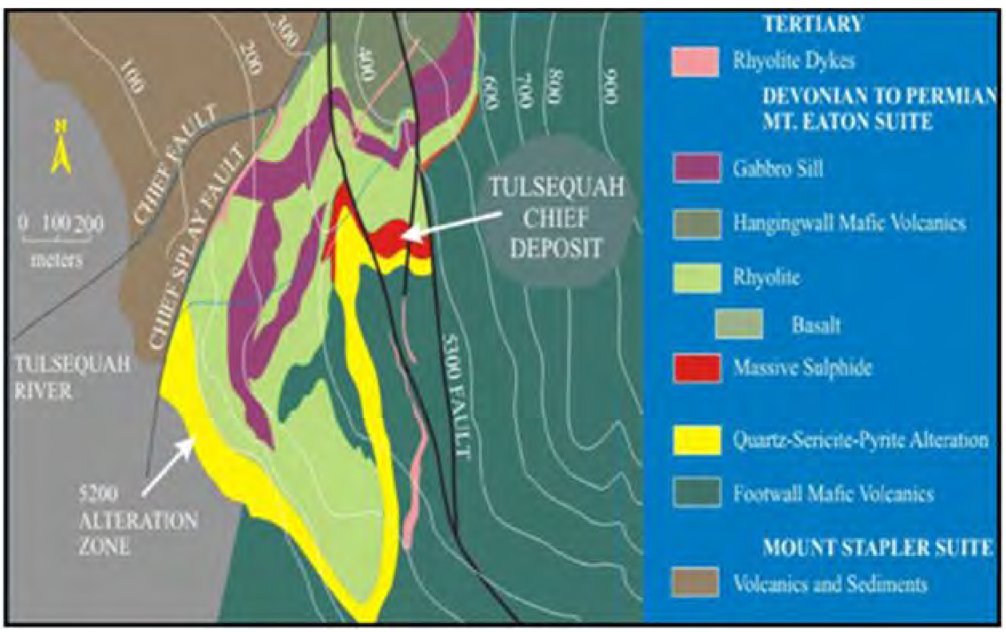
North to northwest-trending high-angle faults with complex displacement histories are common within the Tulsequah Chief region, with the largest and most significant being the Llewelyn fault. Displacement appears to be small on these faults except for the major Chief Fault. Most faults are marked by topographic depressions in the form of steep-sided gullies and ravines. The north trending faults are commonly intruded by Sloko rhyolite dykes.
Younger east-west faults are less common on the property. However, based on regional mapping (Mihalynuk et al, 1994), these faults may have significant displacements. In particular, the Chief Cross Fault was identified as potentially offsetting the regional Llewellyn (Chief) fault in a dextral sense by as much as 2 km.

The Mount Eaton suite is a weakly penetratively deformed sequence that is overprinted by subgreenschist to middle greenschist facies metamorphism (Mihalynuk et al. 1994). It is characterized by the breakdown of pyroxene and amphibole to chlorite and epidote, and potassium feldspar to sericite. Locally, the Mount Eaton suite in the Tulsequah Chief Mine area has undergone contact metamorphism. It is characterized by quartz + epidote, chlorite, actinolite, magnetite and garnet veinlets which crosscut pervasive biotite and cordierite. Biotite is fine grained to aphanitic and phlogopitic in composition (Raudsepp, 1992). Cordierite forms subhedral to euhedral porphyroblasts (4 cm) and often appears to be replacing quartz amygdules within altered basalt flows of unit 1.

7.3 Tulsequah Chief Deposit Geology

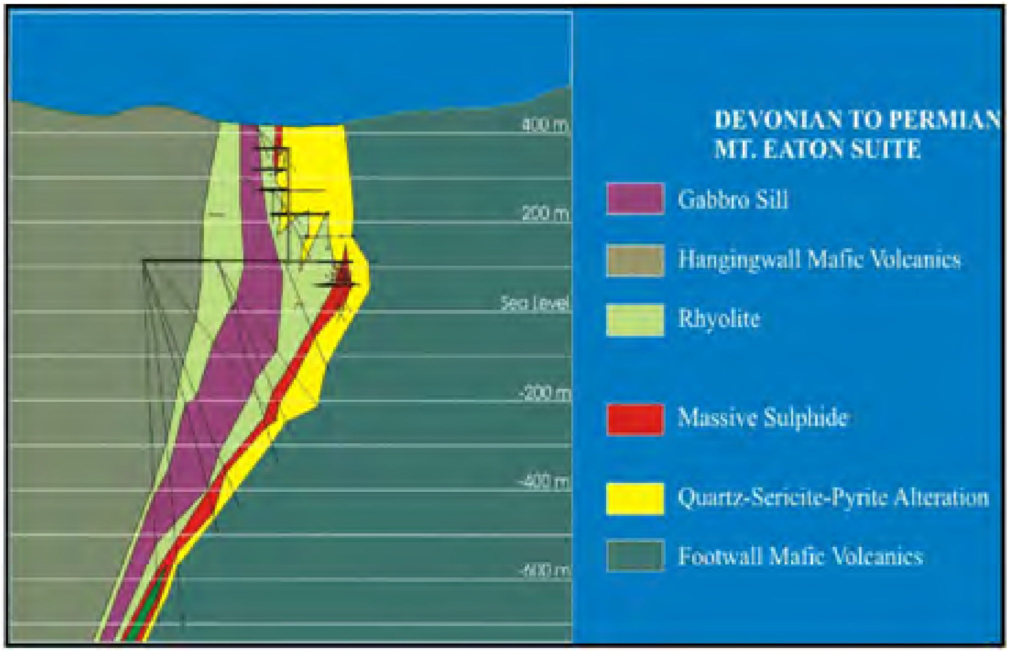
The Tulsequah Chief deposit is a precious metal-rich massive sulphide deposit hosted by the Devonian to Permian Mount Eaton suite, the deposit consists of numerous stacked sulphide lenses developed within the basal stratigraphy of a rhyolite-rich sequence of volcanic flows and fragmental units (Figure 7.4, Figure 7.5). These felsic volcanics rest above a thick assemblage of mafic volcanics (primarily basalt, and basaltic andesite). Above the assemblage of rhyolitic volcanic rocks, 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.

Figure 7.4: Tulsequah Chief Deposit Area Geology



Source: Redfern 2005

Figure 7.5: Schematic Vertical Section through Tulsequah Deposit



Source: Redfern 2005

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 ("forty four hundred east") and 5300E ("fifty three hundred east"), 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.

7.4 Tulsequah Chief Mineralization

Mineralization consists of massive lenses of pyrite and chalcopyrite, and semi-massive sphalerite, galena and pyrite in distinct sulphide lenses at different statigraphic levels. Accessory economic 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, 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 sulphide facies may occur within a single lens, typically with sharp boundaries between them. Despite the occurrences of several distinct sulphide types (based on the relative abundance of different sulphide minerals) no clear geographical zonation pattern has emerged.

The discrete mineralized lenses have been modeled in three zones separated by the 4400E and 5300E faults, A-extension (AEX) Zone, H zone and G zone. The AEX zone has been modeled as an upper, middle and lower zone of stacked lenses of semi-massive sulphides and is similar in character to the H8 lens. The H zone has been modeled as ten lenses H1-10, with the majority of the deposit tonnes in the H2, H3 and H4 lenses in the core of the H-syncline. The H10 lens is the lowest stratigraphic level and is dominantly massive pyrite/copper facies. The H2 and H3 lenses are at the mid stratigraphic level with the H4 lens located 10-20m stratigraphy above separated by felsic tuffs, these main H lenses are mostly zinc facies with primarily sphalerite and galena in a barytic gangue, with much less pyrite and chalcopyrite. The H8 lens is the uppermost stratigraphic level and is semi-massive sulphides with disseminated sphalerite, chalcopyrite and pyrite with fragmental lapilli and occasional massive sulphide clasts. The remaining H lenses are smaller tonnage with lateral offsets from the larger lenses. The G zone has 4 lenses G1-4, with G1 the largest with thinly banded sphalerite, galena, chalcopyrite and barite.

7.5 Tulsequah Chief Alteration

Footwall alteration associated with the massive sulphide horizons is mainly confined to the top of the mafic footwall series and within the felsic unit. The alteration is characterized by intense, texture-destructive quartz-sericite-chlorite-pyrite alteration, extending three to tens of meters. Fine-grained, exceptionally pale disseminated sphalerite is sometimes present in the intensely altered footwall rocks. Hanging wall alteration is poorly developed generally 1-3 m thick and is confined to flows and tuffs within and directly above the sulphide horizons.

Barrett (2006) identified that the addition of potassium (K), is one of the main characteristics of the alteration zone at Tulsequah Chief. Large amounts of K have been added to several lithologies near mineralized zones, including the normal mafic footwall (giving them a felsic appearance), The K is now largely in sericite and biotite/phlogopite. Some holes also show an identifiable pattern that could be used as an exploration vector with "overall barium (Ba) enrichment with a 'reversal' in silica (Si) mass changes close to mineralized horizons": strong Si addition and K addition (with sodium (Na) and calcium (Ca) depletion) over several tens of meters below a particular sulfide lens; followed by Si leaching and reduced K addition occurring up to about 20 m below and even above the lens. Notable Ba additions (more than 2000 ppm) occur throughout both zones, but especially in the upper one.

7.6 Tulsequah Chief Structure

Mount Eaton suite rocks are deformed into anticlinal-synclinal fold pairs, these folds are easterly verging, parasitic folds on the western limb of the regional Mount Eaton anticline. In the Tulsequah Chief Mine area the anticlinal-synclinal fold pairs are upright to steeply overturned parasitic folds north-westerly plunging. Small dextral off-sets in stratigraphy occur along faults, including the 4400 and 5300 faults, which run sub-parallel to the axial plane of these folds (Mihalynuk et al., 1994).

The 4400 fault has a prominent surface expression with a fault gully, and as a 1 m of clay gouge zone at in the underground crosscuts. Strikes range from 355° to 003° with easterly dips of 75° 80°. Stratigraphy is displaced less than 50 m right laterally (dextral) across this fault. Sloko rhyolite dykes are emplaced along part of this fault.

Surface expression of the 5300 fault is less pronounced with a faint surface expression that is traceable to the south where it intersects the 4400E fault 3.5 km south of the Tulsequah Chief Mine. Underground the fault is a 1 m clay gouge zone with a number of sub-parallel subsidiary splays that are identified in drilling and in underground workings. It strikes AZ 001° and dips 80°east, apparent displacement across this fault is less than 30 m in a right lateral (dextral) sense. Sloko Rhyolite dykes are also emplaced along this fault.

7.7 Big Bull Deposit Geology

The Big Bull property is dominantly underlain by moderately deformed rocks of the Mount Eaton Block, a low metamorphic grade island arc volcanic sequence of Devono-Mississippian to Permian age contained within the Stikine Terrane of northwest British Columbia (Mihalynuk et al, 1994).

The Volcanogenic massive sulphide mineralization at Big Bull occurs within a strongly foliated zone of intense sericite-pyrite alteration which is over-and underlain by laminated and chaotically banded dacite crystal tuffs. This sequence has been intruded by irregularly-shaped, aphanitic to fine-grained dark green diabase sills.

The Big Bull stratigraphy has been affected by two phases of folding and sits on the eastern limb of a northwest trending synclinal structure. Several brittle faults cut the deposit area.

The Big Bull stratigraphy has been subdivided into five main lithologic units by Dawson and Harrision (1993), and refined by Carmichael (1994): Footwall mafic volcanic rocks with mixed lapilli tuff and fine grained basalt flows, followed by felsic flows and tuffs that host the Big Bull deposit, that are locally chaotically banded dacites. The next unit is andesite tuff with the top of the unit marked by interbedded hematite and manganese chemical sediments. The upper unit is basalt tuff. Mafic intrusive diabase sills and dykes intrude the other lithologies at Big Bull. Late feldspar-phyric mafic dikes and a distinctive quartz feldspar porphyry dike postdate all other lithologies, and are thought to be related to the Eocene Sloko Group.

7.8 Big Bull Mineralization

The Big Bull deposit consists of two sub-parallel sulphide lens horizons and a lower alteration zone hosted within a dacite to rhyolite-rich sequence of volcanic flows and fragmental units. Mineralization consists of conformable lenses, with a moderate to strong planar to gneissic fabric in most blocks or fragments. The fabric is defined by different proportions of sulphides and barite as well as differing concentrations of talcose, phyllitic, lithic material forming streaks and bands, or layers.

The modeled Big Bull lenses consist of a Main zone with six lenses M1-6, an upper zone with two lenses U1-2 and a lower alteration zone with two lenses L1-2. The principal main zone lens M1 is about 1,000 m long, with an average width of about 2 m, a maximum width of about 8 m, and has been defined by drilling to 350 m below the surface. The M2 lens is slightly above the M1 lens separated by 10-20 m of quartz sericite altered felsic volcanic rocks. The mineralization is classic volcanogenic sulphide rock composed of a mixture of fine to medium-grained crystalline banded and disseminated sphalerite, pyrite, galena and chalcopyrite in a gangue of barite, quartz, some calcite. Sericitic fragments within the mineralized lenses may represent altered lithic fragments that were incorporated in the mineralized interval.

The Big Bull upper zone U1 lens is the high grade 60-62 zone with high grade sphalerite and galena with trace visible gold, the sulphides are recrystallized, with well-developed annealed textures that have obliterated any primary features. The U2 lens is a smaller upper alteration lens with disseminated sulphides and elevated precious metals. The Lower L1 and L2 lenses are also alteration zones with disseminated sulphides and elevated precious metals.

7.9 Big Bull Alteration

Adjacent to the mineralization is a strongly foliated sericite-quartz-pyrite assemblage, containing 5 to 20% disseminated and stringer pyrite, with local base metal sulphides and tetrahedrite. The quartz-sericite-pyrite alteration appears to form a stratiform layer near the top of the felsic tuffs, but may in places be discordant to stratigraphy. Chlorite is also present with a locally strong staining effect.

7.10 Big Bull Structure

Rocks in the Big Bull area have been affected by two phases of folding and several episodes of faulting, creating an area of structural complexity. Lithological contacts generally trend north-northwest, with steep dips to the southwest.

The first phase of folding (S1) is the dominant phase in the Big Bull area, and formed the Big Bull syncline. This phase is characterized by tight, approximately cylindrical moderately overturned folds. A second, weaker phase of folding is indicated by a spaced crenulation fabric which does not appear to have significantly reoriented either S0 or S1 fabrics.

Post-mineral brittle faulting has affected the mineralization at Big Bull. The Bull fault is a northwest striking, steeply west-dipping structure which is approximately axial planar to the Big Bull syncline. The Bull fault has disrupted the massive sulphide lenses in places, with brecciated and rotated mineralized blocks present in the fault gouge. The fault has had a long history involving several periods and directions of movement, the latest of which offsets a quartz feldspar porphyry dyke of probable Eocene age. Although the amount and direction of displacement across the fault is unknown, apparent offsets of lithologic units suggest sinistral strike-slip movement.

8. DEPOSIT TYPES

The Tulsequah Chief and Big Bull deposits are polymetallic volcanogenic massive sulphide (VMS) deposits of bimodal-felsic, island arc or arc-related affinity, similar to the Kuroko type deposits found in Japan. These deposits are generally associated with back-arc basins associated with subduction tectonic environments. The deposit consists of several distinct lenses of massive sulphide mineralization that were deposited at or slightly below the sea floor due to precipitation from the venting of metal-rich hydrothermal fluids. These fluids typically exploit fault planes as fluid pathways and create a large zone of hydrothermal alteration in the rocks below the deposits. VMS deposits are characterized by concordant massive to banded sulphide lenses, typically occurring within a distinct stratigraphic interval.

9. EXPLORATION

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

At Big Bull, in 2011, Chieftain drilled 8,827 m in 22 surface drill holes, to increase the confidence of several inferred areas and the understanding of the high grade 60-62 area. Small step outs to the west and the continuation of the Big Bull main trends to the northwest also identified extensions to the mineralization.

Further exploration surface drilling was conducted at Tulsequah in 2013 with 3,450 m in nine surface holes. A re-interpretation of historic induced polarization surveys generated new targets based on 3D geophysical inversion with iso-shell chargeability anomalies. This included drill hole TC13064 that intersected footwall stringer chalcopyrite VMS mineralization in the newly named southwest zone, located 350m southwest of the known Tulsequah Chief sulphide lenses.

10. DRILLING

10.1 Tulsequah Chief 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 totaling 3,526 m, was carried out. During the period from 1987 to 2013, a total of 120,453 m was drilled in 280 holes (Table 10.1). These holes generally range in length from 134 m to 1,000 m. There are 819 holes in the Tulsequah database, 361 were used to generate the resource estimate. The other 458 include: 20 1950's Cominco surface exploration holes; 333 1950's Cominco underground definition drill holes for historic production, 57 surface Redfern and Chieftain exploration drill holes and 46 Redfern and Chieftain underground exploration drill holes. Exploration drill holes were drilled along strike of the resource area.

Table 10.1: Summary of Drilling Campaigns at Tulsequah Chief

Year	Surface		Underground		
	No. holes	(m)	No. holes	(m)	Total (m)
1950-57 Cominco	20	3,138	518	27,077	30,215
1987 Cominco	6	3,526			3,526
1988 Cominco	2	486	11	3,046	3,531
1989 Redfern			10	4,890	4,890
1990 Redfern			9	6,991	6,991
1991 Redfern			6	3,088	3,088
1992 Redfern			11	4,252	4,252
1993 Redfern	6	1,812	14	6,238	8,051
1994 Redfern	4	1,700	11	4,241	5,942
2003 Redfern	2	1,069	21	9,040	10,109
2004 Redfern			54	30,463	30,463
2006 Redfern	20	6,225	10	2,802	9,027
2007 Redfern	15	4,782			4,782
2011 Chieftain	10	4,005	50	18,625	22,630
2013 Chieftain	8	3,170			3,170
Total	**93**	**29,913**	**725**	**120,754**	**150,668**

Source: Chieftain 2014

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 system or differential GPS. 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. 2011 and 2013 Core is palletized and racked at the Shazah Camp site, with historic 1950's-2007core palletized and stored at Paddy's Flats lay down area.

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 field visits in May 2006, and 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.

10.2 Big Bull Drilling

The first diamond drill campaign carried out at the Big Bull property was in the early 1940s. Drilling programs were ongoing from then until the mine closed in 1956. The property remained inactive until 1993 when a small drilling program, 12 holes totaling 3,556 m, was carried out. During the period from 1993 to 2005, a total of 40,295 m was drilled in 167 holes (Table 10.2). These holes generally range in length from 200 m to 600 m. There are 313 holes in the Big Bull database, 146 were used to generate the resource estimate. The other 106 include 16 1950's Cominco surface exploration holes; 117 1950's Cominco underground definition drill holes for historic production, 34 surface Redfern and Chieftain exploration drill holes. Exploration drill holes were drilled along strike of the resource area.

Table 10.2: Summary of Drilling Campaigns at Big Bull

Year	Surface		Underground		
	No. holes	(m)	No. holes	(m)	Total (m)
1940-57 Cominco	28	3,885	179	6,403	10,288
1993 Redfern	12	3,556			3,556
1994 Redfern	15	5,528			5,528
2006 Redfern	37	15,312			15,312
2007 Redfern	20	7,372			7,372
2011 Chieftain	22	8,527			8,527
Total	**134**	**44,181**	**179**	**6,403**	**50,583**

Source: Chieftain 2014

Collar locations for all the surface holes and all underground holes drilled since 1993 were surveyed in Universal Transverse Mercator (UTM) coordinates relative to established mine survey stations with a total station in 1993-1994 and differential GPS from 2006-2011. Down hole surveys were done using the Maxi Bore system for holes drilled since 2006. Light Log system was used for the 1993 to 1994 holes and Holes surveyed with the Maxi Bore and Light Log were also surveyed by Sperry Sun or EZ Shot as a backup. Surface Drill core was moved helicopter, to the Shazah Camp / Big Bull mine site 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. 2011 Core is palletized and racked at the Shazah Camp site, with 1993-2007 core cross-piled and racked at the Big Bull mine site.

The author has reviewed the sampling procedures and sample intervals for the Big Bull drilling and concluded that the sampling quality and methodologies utilized were appropriate for this type of deposit. The samples collected are representative of the mineralization and not apparent biases were observed in the sampling protocols or the samples collected.

Overall, the drilling over the main Big Bull deposit area is at a nominal 35 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 AND 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 tags 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 1m to 1.5m, 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 Tulsequah Chief 1987 and 1988 core was split with a manual core splitter. Half of the core was placed in labeled 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. Redfern and Chieftain both used the Ecotech Laboratory until late 2011, when ALS minerals acquired Ecotech and the analysis was transitioned to their facility.

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 cable ties, placed into rice bags which were sealed with tie straps, and transported by helicopter or fixed-wing aircraft to Atlin BC, and shipped by bonded carrier to Whitehorse preparation Lab facilities of Eco Tech Laboratory Ltd (Eco Tech) or ALS Minerals. Ecotech's analytical lab is situated in Kamloops, and ALS's lab is located in North Vancouver.

Eco Tech was 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 participated in the annual Canadian Certified Reference Materials Project (CCRMP) and Geostats Pty bi-annual round robin testing programs. The laboratory operated an extensive quality control/quality assurance program, which covered all stages of the analytical process from sample preparation through to sample digestion and instrumental finish and reporting.

ALS is an accredited laboratory with the Standards Council of Canada conforming to the requirements of CAN-P-1579 (Requirements for the Accreditation of Mineral Analysis Testing Laboratories) and CAN-P-4E (ISO/IEC 17025: General Requirements for the Competence of Testing and Calibration Laboratories - ISO/IEC 17025-2005).

At both EcoTech and ALS, 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 pulveriser 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; and
- 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:HN03:H20) 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).

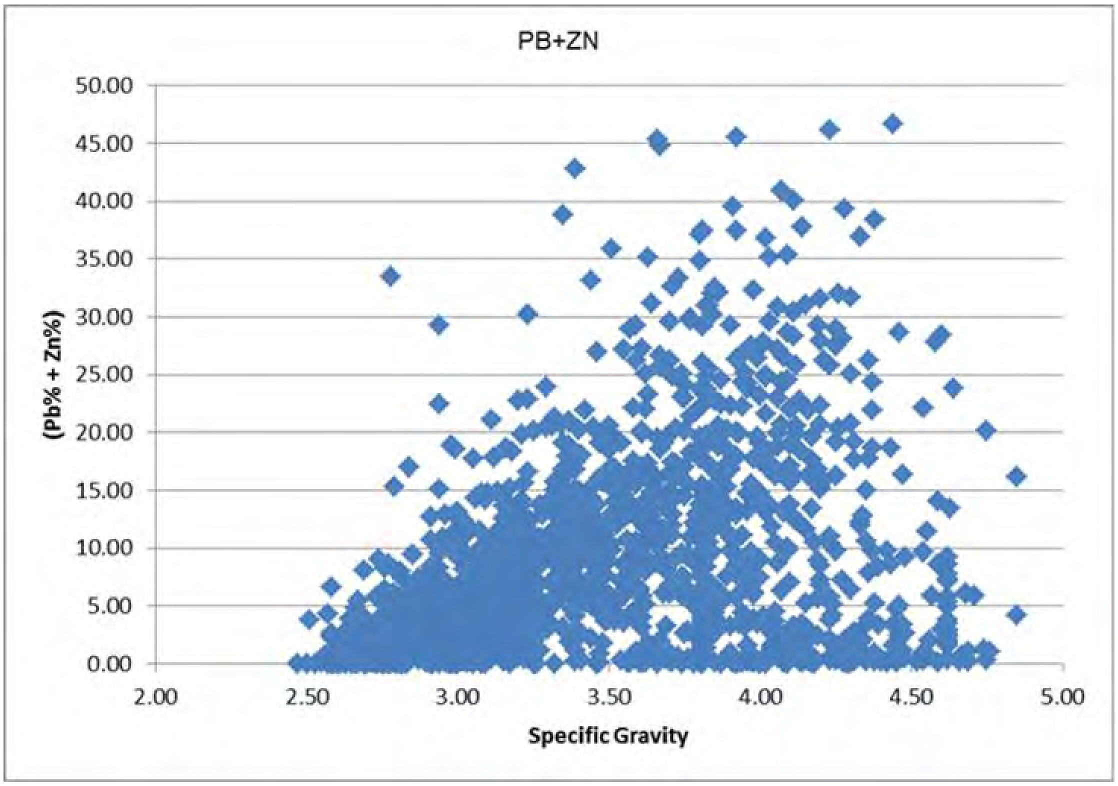
Any of the base metal elements (Ag, Cu, Pb, Zn) that are over limit (>1.0%) are run as an ore grade assay.

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 (SG) determinations have been carried out on Tulsequah Chief drill core, and 2600 Big Bull core samples. Of the total SG determinations, 1,776 samples were from the mineralized intervals at Tulsequah and 240 at Big Bull. The average of all mineralized SG measurement at Tulsequah is 3.44, which is slightly lower than the 3.5 that was used when the mine was in operation in 1957. At Big Bull the average of the SG measurements was 2.94.

While there is a broad correlation between grade (Pb + Zn) contents and SG, there are significant high SG values associated with little or no Pb-Zn values (Figure 11.1). For this reason, SRK decided not to weight the assays with the SG for grade interpolation.

Figure 11.1: Tulsequah Chief SG Values against (Pb + Zn) in Percent

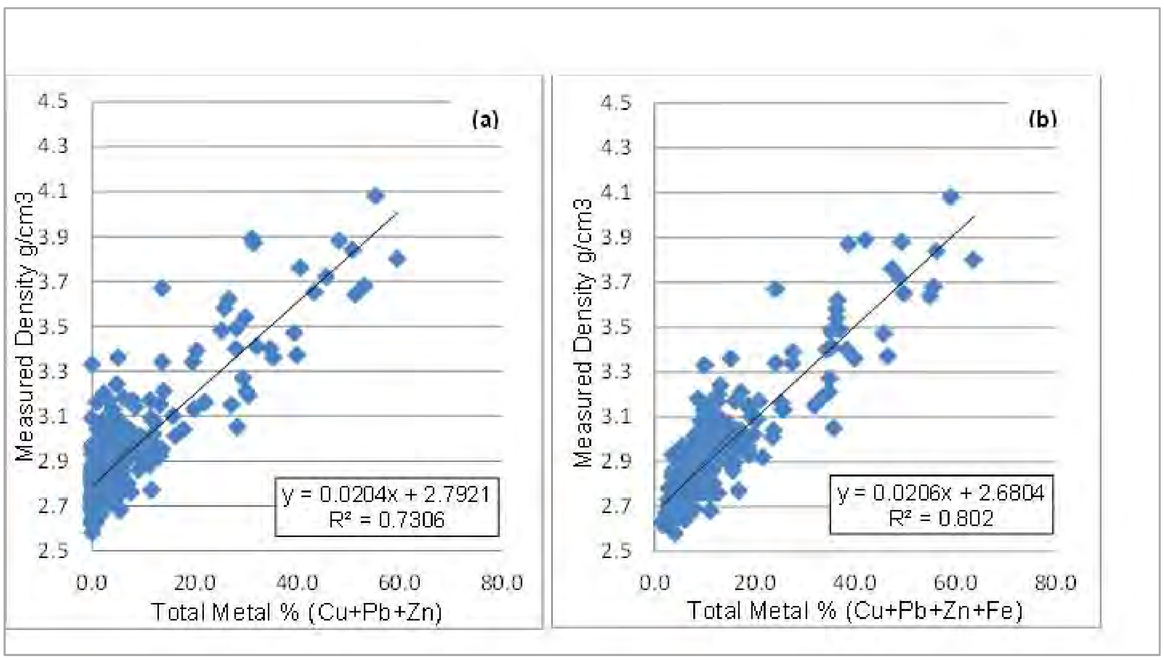


Source: SRK 2012

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

At Big Bull the specific gravity was modeled for 417 historical assays based on the relationship between the total metal content (Cu+Pb+Zn+Fe) and SG, based on the 240 samples within the modeled resource with SG data. A plot of total metal content (Zn + Cu + Pb grades, Figure 11.2a; and Zn + Cu + Pb + Fe grades, Figure 11.2b) against SG indicate that SG is directly proportional to total metal content. The modeled specific gravity for the 417 historical assays within the modeled zones used the formula for the slope of the trend line fitted to curve of SG against total metal content greater than 2% for Zn + Cu + Pb, or greater than 0% for Zn + Cu + Pb + Fe. The mean density of all 2,600 samples with 0% total metal content was 2.72 g/cm^{3}.

Figure 11.2: Big Bull Measured Specific Gravity vs. Total Metal Content



Source: Chieftain 2014

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 100. 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. Two packets for the base metal and gold standard continued to be used in 2011 with the reference material sourced from Canadian Resource Laboratories, Langley BC. Blanks consisted of sawn sections of drill core from a barren quartz-feldspar porphyry dyke that is commonly cut by drill holes, in 2011 landscaping granite grit was used for blank material. 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. The sampling quality and methodologies utilized were appropriate for this type of deposit. The samples collected are representative of the mineralization and no apparent biases were observed in the sampling protocols for the samples collected.

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 2006 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.

Thirty one sample results out of the 3,856 samples submitted were investigated. The majority of these were related to procedural errors caused by the change in principal laboratory halfway through the program.

12.2 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.

12.3 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 2-SD but within 3-SD of the expected SRM value. All the base metal low-grade standards pass the QC. The base metal high-grade standards pass the QC.

12.4 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.

12.5 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 an SG determination. For the entire population (n=80), Acme's gold assays were 21% higher than those of Eco Tech. 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.

12.6 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.

12.7 Verifications by SRK

The author has reviewed the sampling procedures and sample intervals for the Tulsequah Project 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.

12.8 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. METALLURGY AND TESTING

Chieftain is undertaking the completion of an optimized feasibility study for the Tulsequah Chief deposit. This feasibility study is based on underground mining to feed a processing facility at approximately 1,100 tpd to produce gravity gold and independent copper, lead and zinc concentrates.

An NI 43-101 report was published on Sedar in January 2013 and summarized the test work up to 2012. For detailed results see JDS 2012 FS Section 13 and referenced appendices of the report.

The test work is summarized as:

- 1953 - Previous Operation produced concentrates from the Chieftain Upper Levels and Big Bull Orebodies. Wardrop Technical Report reference to The Canadian Mining and Metallurgical Bulletin, Sept. 1953;
- 1992 - 1993 - Beattie Consulting Ltd. - three composites, high, low and average copper head grades, were tested incorporating gravity recovery followed by sequential flotation of lead, copper and zinc. Regrind of the lead rougher concentrate. Beattie Consulting Ltd, Report Titled Flotation Testing of Samples from the Tulsequah Chief Deposit, 1993 - 5300.2.25.1;
- 1994 - 1995 - Brenda Process Technology - drill core representing the overall upper workings was used to create three composites: base, high and low lead content. The test work focused on producing a copper-lead bulk rougher concentrate followed by separation in subsequent flotation stages and a zinc concentrate. Gravity, pyrite flotation and grindability tests were conducted. Wardrop Technical Report to Redfern March 2001;
- 1996 - Redfern Resources - Six samples from the 5,200 level were combined to make a composite at the predicted head grade. Samples of the composite were sent to IMEI and G&T Metallurgical Services to conduct flotation test work and Hazen Research Inc. for comminution testing;
- 2006 - Redfern Resources - Dense Media Separation, (DMS) Flotation test work was performed for Redfern Resources on the 2006 drill core. The test work included bulk copper and lead followed by zinc flotation. Wardrop Technical Report 2007 - no reference with respect to who performed the test work;
- 2011 - Burnie Laboratory (ALS Metallurgy), Australia - Three composites were provided by Chieftain Metals: Chieftain Composite 1 - upper zone and Chieftain Composite 2 (high As)/3 (low As) -lower zone. The test work included 59 bench scale tests and six locked cycle tests that focused on sequential flotation of copper, lead, zinc and pyrite. ALS Burnie Reports T0662-1 and T0662-2;

- 2011 - Gary McArthur, MODA - Mineralogy test work on Chieftain Composites 1 and 2/3 or ALS Composites 1 and 4. ALS Composite 1 was an upper zone composite and ALS Composite 4 lower zone composite. The mineralogy focused on Tennantite in copper cleaner and gold concentrates;.
- 2013 - Gary McArthur, MODA - Mineralogy was carried out on ALS Project T0662 T56, T58 and T59 rougher, cleaner and tailings samples;.
- 2014 - Gary McArthur, MODA - Mineralogy and mineragraphy was carried out on the upper and lower zone samples used for the 2014 ALS test campaign;.
- 2014 - Hazen Research Inc. - Three samples, targeting an average head grade, from the 5400 level were provided by Chieftain for Jk drop weight tests. In addition, two of the samples were used for Bond rod mill work index testing. Hazen Projects 11936 and 11936-01;
- 2014 - Gary McArthur, MODA - Mineralogy and mineragraphy was carried out on the upper and lower zone samples used for the 2014 ALS test campaign; and
- 2014 - ALS Metallurgy Australia, (formerly Burnie Laboratory)- Composites from 2011 continued flotation test work to investigate flowsheet options to producing two copper concentrates. Due to oxidation and age of the samples, the results were poor. Chieftain composites 1 and 2/3, 2011 drill core from site, were sent to provide a fresher sample. The results improved but not to the extent of those seen in the 2011 test work. The samples were used to attest at alternative flowsheets and reagent schemes. A total of 40 bench scale tests were completed and 1 locked cycle test. A formal report has not been issued.

Trevor Watters and Ken Sangster have managed the 2011 ALS T0662 and 2014 ALS T0897 test programs completed at ALS metallurgical laboratories in Burnie, Australia. The program in 2011 to 2012 completed 59 bench scale tests and 6 locked cycle tests on Chieftain composites 1 through 3. The test work from 2011 to 2012 was used to support the metallurgical design criteria developed for this stage of the study. The 2014 test work was done on available sub sample of remaining core from the 2001/2012 program. It focused on producing a split copper concentrate as well as providing samples for backfill test work; one with an arsenic level below 0.5% (Chalcopyrite, Cp) and one above 0.5% (Tennantite, Tn). Due to the age of the samples used in the ALS T0897 test work most of the results have not been used for design criteria and flowsheet development.

13.1 Historical Context

The historical and previous test work write-up was submitted as part of the 2012 Feasibility Study and published in Section 13 as published in the NI 43-101 report.

13.2 Previous Operations

There is a history of mining and processing of the ore from the upper levels of the mine above the main access adit. Ore below the adit level was not mined as the potential costs of shaft sinking and dewatering deterred exploitation at the time. The recoveries and concentrate grades reported at the time (Jan - Nov 1953) are as shown.

Table 13.1: Metallurgical Test Results Jan -Nov 1953

Product	Grade	Recovery
Gravity conc	38.8oz/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%.

Source: 2012 FS Appendix 5.2, Wardrop Technical Report to Redfern March 2001 reference The Canadian Mining and Metallurgical Bulletin, Sept. 1953

13.2.1 Previous Test work

1992 to 2012 Test work

Previous test work is summarized in the table below.

Table 13.2: Test work Summary 1992 to 2013

Description	Date	Test work	Additional Test work	Gold Recovery	Silver Recovery	Copper Grade - Recovery	Lead Grade - Recovery	Zinc Grade - Recovery	Comments
Beattie Consulting Ltd.	1992 - 1993	Pb/Cu/Zn Sequential Flotation		83.3	77.2	25.7/87.3	64.4/91.7 with regrind	60/90.5	Based on LC test work, Bench Scale results were much lower
Brenda Process Technology	1994 - 1995	Cu/Pb Bulk Conc./Zn Flotation	Comminution BMWi= 12.6 kWh/t at 52 microns	80.5, 34% Gravity	73.7	20.4/92.2	16.6/89.8	60/89.4	Focused on base, low and high lead composites. 1.2% As, 0.5% Sb in Cu
IME (International Metallurgical and Environmental)	1996	Bulk Cu/Pb Conc./ Zn - 18 bench scale and 3 LC	Comminution BMWi= 10.8 kWh/t at 59 microns	81, 32.4% Gravity	84.3	21.0/91.3	17.0/95.4	56.5/80.6	LC#10 best results. Focus on Pb/Cu split, Zn recovery and Py Flotation.1.2% As, 0.5% Sb in Cu
G&T Metallurgical Services	1996	Cu/Pb/Zn Sequential Flotation	Mineralogy/Mineragraphy/Microprobe Cu – 75% Cp, 22% Tn, Ag associated with Tn, 1/3 gold free, 80% Cu/Zn liberation at P80=70 microns	68, 36% Gravity	64	27.8/78	49.5/59	57.6/84	1.5% As, 0.5% Sb in Cu. 75 ppm in Zn
Hazen Research	1996	Grindability	Comminution SAGWi=7.7 kWh/t, RMWi=9.3kWh/t, BMWi=9.8-10.2 kWh/t at 150 mesh, Bond Impact Wi = 7.7 kWh/t, Abrasion Index = 0.0461 g						
	2006	Bulk flotation, DMS to pre-concentrate mill feed							Test work focused on replicating results from 1996. Poor flotation results
McArthur Ore Deposite Assessments Pty Ltd	2011 - July	Preliminary Mineragraphy - Chieftain ALS Comp. 1	Mineralogy/Mineragraphy/Microprobe Sp largest grain size with 58% liberated at 53 µm, Sp binaries with Ga and Gn, Gn binaries with sp, Cp binaries with Ga and Py, Tn binaries with Sp, Cp, and Ga						Chieftain Composite 1 Upper zone exceeding average grade of overall test work samples
	2011 - Aug	ALS Comp.1 Mineralogy	Mineralogy/Mineragraphy/Microprobe Mineralogy of 5 size fractions. Overall Sp highest at 75% liberated. At 53 µm Sp and Cp 79 and 87% liberated. Gn and Tn 36%. All above 85% at 20 µm.						Chieftain Composite 1 Minerals observed were Py, Sp, Gn, Bn, Cp, Tn, Ga (muscovite, barite,quartz)
	2011- Nov	ALS Comp. 4 Mineralogy	Mineralogy/Mineragraphy/Microprobe Mineralogy of 5 size fractions. Overall Sp highest at 73% liberated. At 53 µm Sp and Gn 76 and 75% liberated. Cp and Tn 82 and 85%. All above 86% at 20 µm except Cp 72%.						Chieftain Composite 2 Compared to Composite 1 Sp is less associated with Gn, Lower content of Gn and Gn is less associated with Sp more with Ga. Cp content is higher but less liberated and more associated with Sp. Tn is more liberated and less associated with Ga.
	2011 - Dec	ALS Comp. 4 Mineragraphy	Mineralogy/Mineragraphy/Microprobe Cp and Tn similar grain size 33% liberated at 53 µ, Gn Finer at 22% liberated. All sulphides are associated with Ga.						Chieftain Composite 2. Minerals observed were Py, Sp, Gn, Bn, Cp, Tn, Ga (muscovite, barite,quartz)
	2011 - Dec	Tennantite Mineralogy	Mineralogy/Mineragraphy/Microprobe Tn in Cu cleaner 2 is well liberated. Tn in Au is less liberated binaries with Sp and Py.						Analysis of Cu cleaner 2 and gold rougher concentrates. No reference to the source of the concentrates.
K. Goemann	2011 - Dec	Microprobing – Tn variation in ALS Comp. 1 and 4	Mineralogy/Mineragraphy/Microprobe Comp 1 Tn high Zn with variable As. Comp. 4 high Zn low Sb with higher As than Comp. 1						Chieftain Composites 1 (upper zone) and 2 (lower zone)
Burnie Laboratoy (ALS Metallurgy)	2011 - Oct	Report T0662-1 – T01 to T22 Cu/Pb/Zn/Py sequential flotation at a P80= 50 µ. LC01 Completed	Comminution The JK test medium soft to medium with respect to the impact breakage ALS – 662001 – BMWi= 12.9 kWh/t at 75 µm ALS – T12216 – Abrasion Index = 0.0743 (Test A13775 BBWI Full Report)			21/88	Pb results poor and variable. Additional liberation of the fine grain material would be required.	Zn best results at 60% grade and a recovery of 85%	Chieftain Composite 1 (upper zone)..
	2011 - Oct	LC tests 01 – Cu/Pb/Zn/Py sequential flotation similar to bench scale tests		55% recovered to Cu conc.	53% recovered to Cu conc.	25.5/82.2	61.0/65.9	61.1/91.0 (3 stages of cleaning)	Chieftain Composite 1 (upper zone)
	2011 - Oct	Report T0662-2 – T26 to T46 Cu/Pb/Zn/Py sequential flotation with tests to target gold recovery and arsenic removal		T22/32 approximately 30% gold recovered through gravity. Up to 90% can be recovered by gravity, Cu conc. and Pb conc.		24.1/80.3	51.6/74.0	61.9/81.7	Chieftain Composite 1 (upper zone) except T41 and 42 Chieftain Composite 2 (lower zone high As) Overall results from T26-28
Consep (Metcon)	2012 - Feb	Test work and Modeling on GRG		Test work shows the maximum recovery of GRG at 53.7 % simulations indicated gold recovery of 41.3%					ALS Composite 4 test work done by Metcon

Source: FS 2012

2012 ALS Project T0662- T26-28

From the test work performed by ALS, batch flotation Test 26-28 became the basis for reagents, flotation conditions and flowsheet in subsequent test work. Test conditions and results were reported as follows.

Figure 13.1: Flowsheet

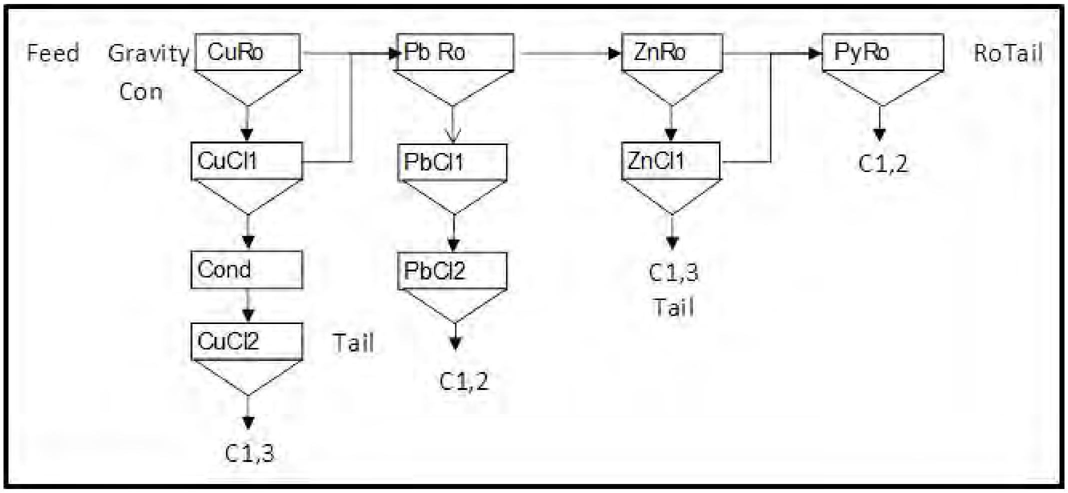


Table 13.3: Reagent Consumption

T(26,27,28)	Lime	SMBS	9810	NaCN	ZnSO4	7021	CuSO4	3418A	H2SO4	PAX	MIBC
	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t
REAGENT TOTALS (g/t)	789	1604	6	346	280	16	509	8	2928	76	153

Table 13.4: Flotation Test Results

CUM PRODUCTS	CUM	WT	Cu	CUM	Pb	CUM	Zn	CUM	Fe	CUM	Ag	CUM	As	CUM	S	CUM	Au	CUM
	Wt	%	%		%		%		%		ppm		ppm		%		ppm	
Gravity Conc	57.8	1.0	1.31	0.97	1.10	1.28	6.59	0.96	11.7	0.92	95.0	1.05	702	1.08	19.8	0.98	42.7	15.9
CuCl2C3	260.4	4.4	24.1	80.3	1.23	6.4	10.2	6.6	24.4	8.7	933	46.5	4952	34.4	35.1	7.9	23.8	40.0
Cu Ro Con	619.8	10.5	12.0	94.9	4.12	51.3	16.1	25.0	19.0	16.1	721	85.5	5269	87.2	31.8	17.0	17.3	69.3
PbCl2C2	71.4	1.2	0.27	0.2	51.6	74.0	6.61	1.2	10.4	1.0	454	6.2	713	1.4	24.3	1.5	16.2	7.5
Pb Ro Conc	133.1	2.3	0.39	0.7	29.5	79.0	9.02	3.0	12.6	2.3	336	8.6	755	2.7	23.7	2.7	13.5	11.6
ZnCl3C3	525.5	8.9	0.69	4.7	0.31	3.2	61.9	81.7	1.95	1.4	97	9.8	520	7.3	31.9	14.4	0.89	3.0
Zn Ro Conc	893.1	15.2	0.48	5.5	0.21	3.8	36.7	82.3	1.43	1.7	66	11.3	414	9.9	19.2	14.8	0.62	3.6
PyRoC3	1637.5	27.8	0.20	4.2	0.11	3.5	0.40	1.6	33.9	75.8	32	10.1	343	15.0	41.1	57.9	0.56	5.9
FEED	5890.8	100.0	1.33	100.0	0.84	100.0	6.76	100.0	12.4	100.0	89	100.0	636	100.0	19.7	100.0	2.63	100.0

Source: ALS T0662 T26-28

2012 ALS Locked Cycle Test work – ALS Project T0662-2 Report

ALS Metallurgy in Burnie, Australia performed six locked cycle tests for Chieftain Metals. A description of the samples is listed below.

Table 13.5: Sample Description

Chieftain Composite	Description	ALS Composite/Sample Number	Ore Zones	Predicted Assays %Cu	%Pb	%Zn	ppm As	Locked Cycle Tests Conducted
1	Upper Zone	1	67% G1 and G2, remainder equal % H3, H4 and Py	1.35	1.2	6.91	595	LC01, LC02. LC05
	half core total	2						
		3						
2	Lower Zone	4	67% H2 and H3, 19% G1, remainder equal % H4 and H7	1.15	1.32	6.03	840	LC03, LC04
	half core total	(Hi Arsenic: lower ore body average)						
3	Lower Zone	5	67% H2 and H3, 19% G1,remainder equal % H4 and H7	0.97	1.32	5.64	482	LC06
	quarter core	(Low Arsenic – excluding						
		DDH 1170)						

Source: 2012 FS Section 13 and Appendix 5.4 and 5.5

Locked cycle test results are shown below.

Table 13.6: ALS Project T0662 – LC01 to 06 Testwork Results

Description	Gold Recovery	Silver Recovery	Copper Grade/ Recovery	Lead Grade/ Recovery	Zinc Grade/ Recovery	Comments
LC01	64.0	60.5	25.5/83.8	61.0/65.9	61.1/91.0	2 stages of cleaning in Cu/Pb, 3 stages in Zn. No gravity gold or rougher.
LC02	83.9	84.4	27.8/75.8	58.4/29.4	62.4/82.6	2 stages of cleaning in Cu/Pb, 3 stages Zn. Gold rougher before Zn flotation.
LC03	79.2	73.0	23.6/79.8	53.8/56.5	62.1/80.6	3 stages of cleaning in Cu with Tn split, 2 stages Pb, 3 stages Zn. Gold rougher before Zn flotation.
LC04	88.5	79.7	24.6/82.2	63.0/65.8	64.0/86.3	2 stages of cleaning in Cu with Cp and Tn conc., 2 stages Pb, 3 stages Zn. Gold rougher before Zn flotation.
LC05	85.0	82.5	25.8/81.8	56.9/64.1	63.1/88.3	2 stages of cleaning in Cu with Cp and Tn conc., 2 stages Pb, 3 stages Zn. Gold rougher before Zn flotation.
LC06	89.3	78.3	23.8/85.7	65.4/65.9	60.2/87.1	2 stages of cleaning in Cu with Cp and Tn conc., 2 stages Pb, 3 stages Zn. Gold rougher before Zn flotation.

Source: ALS T0662-LC01 to LC06

The table below lists the reagent schemes for each locked cycle tests.

Table 13.7: Locked Cycle Test Reagent Schemes

Reagent (gm/t)	LC01	LC02	LC03	LC04	LC05	LC06
Lime	804	1,160	1,186	993	957	1,012
SMBS	1,573	2,300	2,495	2,124	2,125	2,129
9810	6	8	10	8	8	8
NaCN	340	348	354	355	145	356
ZnSO4	275	511	624	825	825	626
3418A	7	8	10	10	10	10
7021	16	24	19	20	20	20
CuSO4.5H2O	499	511	499	500	500	501
H2SO4	2,785	102	0	0	0	0
PAX	75	10	10	10	10	10
MIBC	127	147	198	172	173	184

Source: ALS T0662 – LC01 to LC06

13.2.2 Settling Rates

Settling tests were conducted on three samples of final tailings and one sample of copper and zinc concentrates from Locked Cycle LC01 (ALS, Burnie, September 2011 as reported in 2012 FS Appendix 5.19). The zinc and copper concentrates settled within the first five minutes to achieve 60% solids. The tests indicate it was difficult to reach 50% solids in the tailings settling tests.

13.2.3 Recent Test work - March 2013 to September 2014

Hazen Research Inc. - Comminution Test work

Project 11936

In June of 2014 Chieftain Metals provided Hazen Research Inc. with three Chieftain samples taken from the upper zone 5400 level of A52 for JKTech full drop-weight testing. The results are listed below.

Table 13.8: Summary - JKTech Drop Weight Tests

HRI – Hazen Sample No.	SG	A (max. breakage)	b (relation between energy and impact breakage	A x b (overall AG-SAG hardness)	Ta (abrasion parameter)	Resistance to Impact Breakage	Resistance to Abrasion Breakage
53967-001	3.85	76.7	1.99	152.6	1.67	Very soft	Very soft
53967-002	4.27	73.5	1.44	105.8	1	Soft	Soft
53967-003	4.53	69.1	1.85	127.8	1.76	Very Soft	Very soft

Source: Hazen Projects 11936

Compared to the JK database, the Chieftain Composites that were sent to Hazen are rated as soft to very soft.

Project 11936-01

Retained samples from the JKTech test work were subjected to Bond rod mill work index (RWi) testing. The test results are summarized below.

Table 13.9: Summary - Bond Rod Mill Work Index

HRI – Hazen Sample No.	RWi (kWh/t)
53967-002	8.8
53967-003	7.3

Source: Hazen Projects 11936-01

The results indicate a soft ore with a low RWi which is consistent with the JKTech Drop Weight Tests. Previously a Bond rod mill work index, 9.3 kWh/t, test was completed by Hazen in 1996.

McArthur Ore Deposit Assessments Pty Ltd - Mineralogy/Mineragraphy/Microprobe Test work

The following reports were prepared by McArthur Ore Deposit Assessments Pty Ltd. In Burnie, Australia.

March 2013 Test work Concentrate Mineralogy

ALS Metallurgy had 21 test work samples that included Cu cleaner 1, Cu cleaner 2 and Zn rougher concentrates and tailings from Test 56, 58 and 59 sent for mineralogical assessment. The tests were conducted to get a better understanding of the liberation of the different minerals in each stream. The results indicate the Cp lost in the tailings streams is finely disseminated and associated with pyrite and gangue. There is poor liberation of the Cp in Cleaner 1 Tailings suggesting additional regrind of the rougher concentrate may improve recovery.

June 2014 ALS Composite 8 Mineralogy

ALS Composite 8 Mineralogy (drill core from 2011 test program) was submitted for mineralogy assessment focusing on arsenic content and associations. The minerals identified include Py, Ga, Sp, Cp, Tn, Gn, Me, and PyMa. The composite is not as well liberated as Composite 1 reported in previous mineralogical tests. Arsenic deportment was as follows: "Only rare arsenopyrtie was observed, so virtually all the arsenic is resident in tennantite”.

July 2014 Lower Tulsequah Mineragraphy

The Chieftain samples submitted for test work are a combination of drill core from G1, G2, H4 and COM from both the upper and lower zones. The drill holes that correspond to the upper zone are TCU11158, 160, 167 and 192 and the lower zone are TCU11175, 77, 80 and 86. The mineralogical analysis performed was on a combination of both upper and lower zones not just the lower zone drill core. Gangue minerals identified are Sericitic Muscovite, Quartz, Barite and Carbonate.

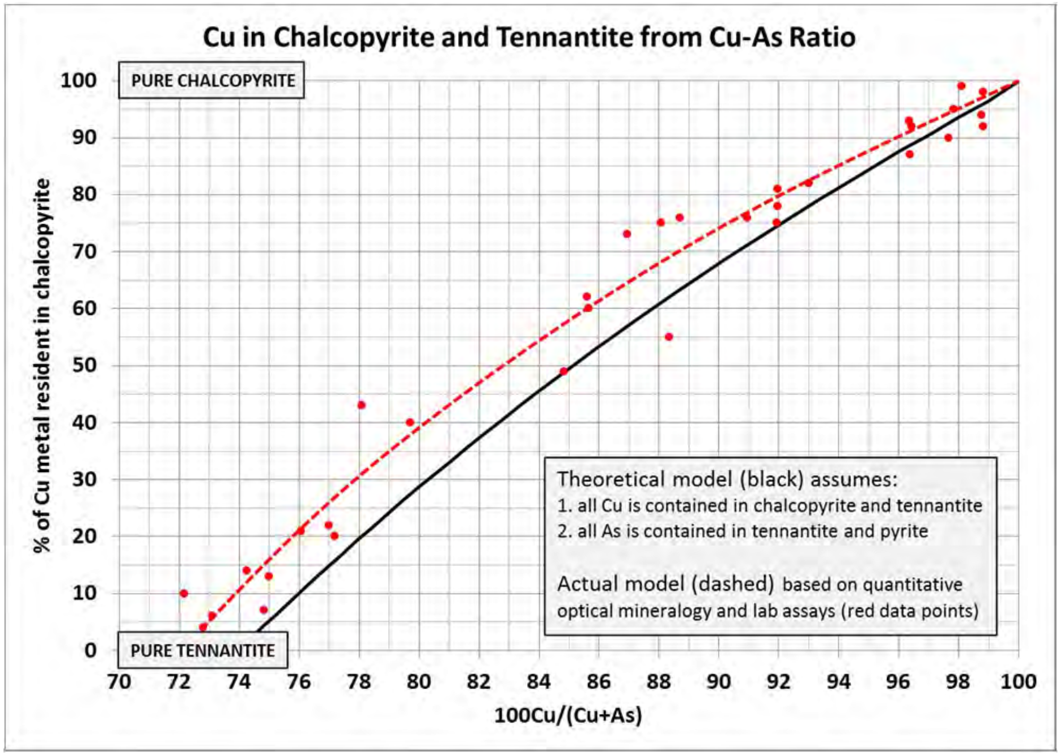
- The results indicate that the liberation grainsize is estimated to vary between 24 to 39 µm, with Sp the coarsest and Tn the finest;
- Cp and Tn have a moderate association with Ga and Py indicating a copper concentrate would contain a significant amount of Py and Ga composites;
- Sp and Gn are moderately associated with Ga but lower with Py. A Zn concentrate made from these samples would be expected to contain from Sp-Ga binaries;
- Pb concentrates from these samples would contain Pb-Ga binaries; and
- Cu metal overall would have 71% Cp and 29% Tn.

The examination determined that the mineralization displays simple mineralogy and is texturally homogenous which will translate into mill feed consistency. The Sp, Gn and Cp present are generally of a medium to coarse grain-size which should be fully liberated at approximately 30 to 35 µm grind. As is associated with Tn with no other species identified.

August 2014 ALS Cu Test work Mineralogy

ALS submitted 11 streams from the copper flotation circuit for analysis. The samples were from Test 13 and 15 on ALS Composite 7, lower zone ore. "Actual data from the quantitative optical mineralogy done on these samples conforms well to a theoretical model based on a stoichiometric chalcopyrite-tennantite mix (see plot below). This has potential to allow prediction of Cu residency from just the Cu and As assays."

Figure 13.2: Cp, Tn and As Relationship



Source: McArthur Ore Deposit Assessments Pty Ltd - August 2014 ALS Cu Test work Mineralogy

13.2.4 2014 ALS Project T0896

June to September 2014 ALS Metallurgy

ALS Metallurgy, Burnie, Australia started a new test program Project T0897 in June 2014 for Chieftain. The program focused mainly on the separation of the low and high arsenic copper concentrates. The test work was performed using two flowsheets: Cp, Tn and ZnScav con in the copper cleaner stage and sequential flotation of Cp rougher followed by Tn rougher with a Zn Scavenger in the Tn cleaner circuit. The program to date has provided test data for T01 to T40 and a summary can be found in Appendix B. The final report has not been issued at the time of writing.

The initial 15 tests were performed on ALS Composites 7 and 8 using drill core stored at ALS from the 2011 test program. The two flowsheet options discussed above were used to produce separate copper concentrates. From the results for T01 - T15 very poor recoveries were reported possibly due to the age of the samples. Drill core samples from site, Chieftain Composite 4, were sent to ALS in June 2014 and two ALS composites were created. ALS Composite A, upper zone and ALS Composite B, lower zone. The available drill core was limited leaving only samples from G1, G2, H4 and Cominco B-E52 lenses from the upper and lower zones of the ore body. The new composites were used for the remainder of the test program.

Test results from T17 indicate that successful separation of the chalcopyrite and tennantite is feasible by using the different flotation rates of the two minerals. Cp with As below 0.5% will be pulled first to produce Copper Concentrate 1 and the slower floating Tn with silver and higher arsenic values will be pulled second as Copper Concentrate 2. The tailings from the Cp and Tn cleaner circuit were conditioned and a Zinc Scavenger Concentrate produced from the next stage of cleaning. The results improved over the first set of tests, T01 – T15, but showed lower recoveries compared to the 2011 test program.

Test T17 results show the performance of ALS Composite A (upper zone) and test T19 Composite B (lower zone). The upper zone composites, with respect to copper, tend to perform better than the lower zone composite. The trend follows the mineralogy results that indicate better liberation of copper in the upper zone composites.

T30 and T31 tests were completed on ALS Composites A and B each mixed with a sample of drill core to increase the copper and zinc grades. The results are within the range of previous bench scale test work and locked cycle tests reported in the 2012 FS.

Tests T32 and T33 were carried out to produce an alternate reagent scheme to eliminate the need for acid (2012 FS) in the pyrite flotation circuit. The results indicate pyrite can be recovered using PAX and MIBC at a pH of 9.

For the ALS Project T0897 Composite A and B grade recovery curves for copper and zinc, see Figures 13.3 and 13.4 below. Note Composite C is a 50/50 blend of A and B.

Figure 13.3: Copper Grade vs. Recovery

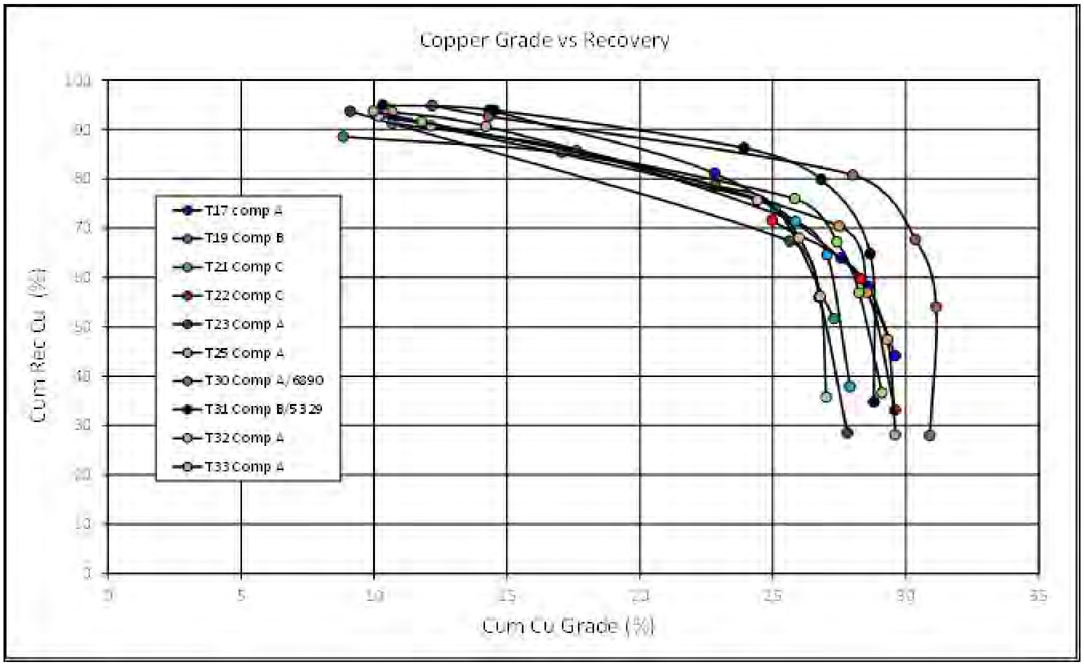
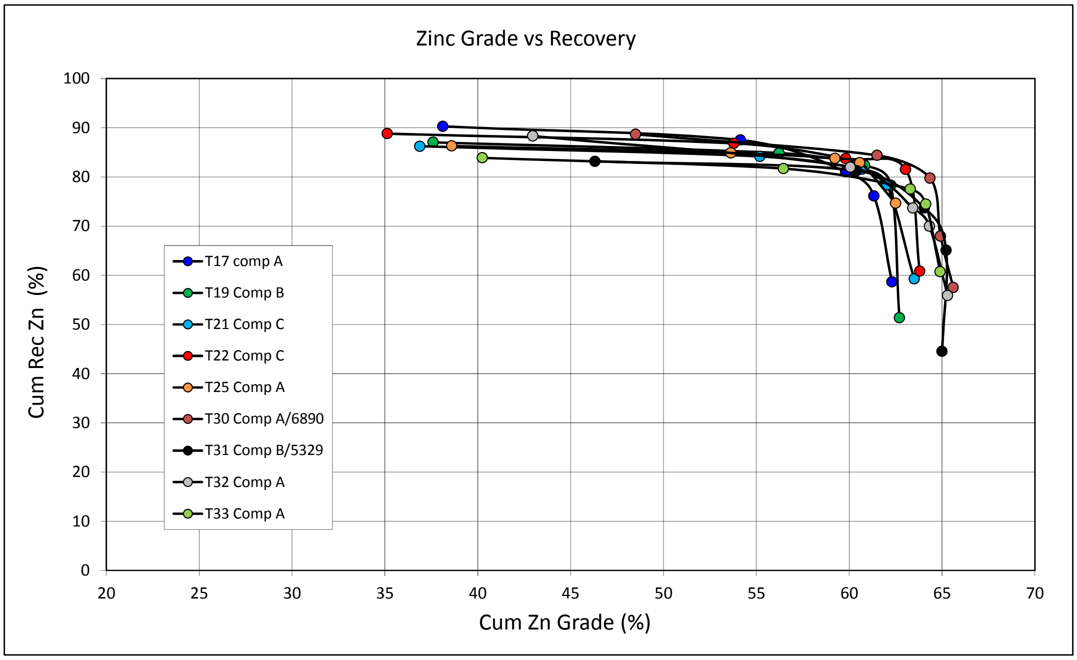


Figure 13.4: Zinc Grade vs. Recovery



Source: ALS Project T0897

13.3 Process Design

13.3.1 Introduction

"VMS ores are universally treated by flotation, but often include a gravity circuit if recovery of gold is warranted. Given that the Tulsequah ore was successfully treated by flotation in the 1950's and test programs in the 1990's all utilized flotation, this was the obvious and only 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" 2012 FS. The flowsheet includes crushing, grinding, gravity, sequential flotation, dewatering and filtration unit operations.

13.3.2 Comminution Circuit

Based on the ore hardness seen from grinding comminution test work, it was assumed a jaw crusher would reduce the underground ore from 80% passing 500 mm to 100 mm in one stage. The SAG mill and ball mills were sized using the Bond rod mill work index, 8.8 kWh/t, Hazen 2014 test work and the Bond ball mill work index, 12.9 kWh/t from a sample of upper zone Chieftain composite 1 ore, in conjunction with the Bond equation and efficiency factors. The power requirements were calculated using average LOM daily tonnage with an assumed plant availability of 90%. The diameter, length and motor size for the mills are based on standard sizes from Vendor database. The chosen Vendor will confirm mill sizing.

Table 13.10: SAG and Ball Mill Design Criteria and Sizing Parameters

Mill Process Design Parameters	**Units**	
Operating Parameters		
Daily Dry Tonnage	tpd	**1,097**
Availability	%	**90**
Dy Hourly (Instantaneous) Throughput	t/h	51
Ore Specific Gravity	-	3.55
Rod Mill Work Index	kWh/t	8.8
Ball Mill Work Index	kWh/t	12.9
Abrasion Index	-	0.0743
Feed Size	µm	**100,000**
Final Grind Size	µm	45
SAG Mill		
Final Grind Size	µm	**425**
SAG Efficiency Factor	-	1.80
Transmission Loss Factor	-	1.10
Power Required	kWh/t	7.90
Unit Power Consumption	kW	401
Power Requirement	HP	538
Installed Power	HP	600
% Power Utilized	%	89.7
Ball Mill No. 1		
Discharge Size P80	µm	**95**
EF1 - Dry/Wet Grind	-	**1.00**
EF2 - Open/Closed Circuit Grinding Factor	-	**1.00**
EF3 - Diameter Efficiency Factor	-	0.914
EF4 - Oversized Feed Factor	-	1.00
EF5 - Fine Grinding Factor	-	1.00
EF6 - N/A - Rod Mill Only	-	1.00
EF7 - Low Ratio of Reduction Factor	-	1.04
EF8 - N/A - Rod Mill Only	-	1.00
Transmission Loss Factor	-	1.10
Power Requirement	kWh/t	7.31
Power Required	kW	371.
Power Required	hp	497
Installed Power	hp	600
Power Utilized	%	82.9
Ball Mill No. 2		
Feed Size P80	µm	**95**
Final Product Size	µm	**45**
EF1 - Dry/Wet Grind	-	1.00
EF2 - Open/Closed Circuit Grinding Factor	-	1.00
EF3 - Diameter Efficiency Factor	-	0.914
EF4 - Oversized Feed Factor	-	1.00
EF5 - Fine Grinding Factor	-	1.07
EF6 - N/A - Rod Mill Only	-	1.00
EF7 - Low Ratio of Reduction Factor	-	1.17

EF8 - N/A - Rod Mill Only	-	1.00
Transmission Loss Factor	-	1.10
Power Requirement	kWh/t	7.57
Power Required	kW	385
Power Required	hp	516
Total Installed Power	hp	600
Mill Efficiency Factor	%	85.9

Source: JDS 2014

13.3.3 Gold Recovery

Free gold is planned to be recovered in the grinding circuit using two gravity concentrators. The concentrators were sized by FL Smidth Knelson, Project No. 140708_MS-1845 Section 5.1 Appendix A, using the test work and modelling from the 2012 Consep report referenced in the 2012 FS. The gravity gold recovery is expected to be approximately 41% of the remaining 59% of the gold approximately 85% is recoverable by flotation to the copper and lead concentrates, 2012 FS - Appendix 5.20. ALS Project T0662 T32 and expected gravity recovery, as discussed in 2012 FS, were used to estimate the proportions of gold reporting to gravity, copper and lead concentrates for a total recovery of approximately 91%.

The gravity recovery circuit equipment was sized by FL Smidth Knelson to include concentrators, Acacia Dissolution Module, electrowinning and smelting to produce the final dore. Design criteria are based on the performance and requirements of the recommended equipment.

13.3.4 Silver Recovery

"The gold is almost all 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" 2012 FS. The majority of the silver is associated with tennantite recovery. Using the mineralogical relationship, McArthur Report August 2014, between copper and arsenic the Tn recovery was calculated. The Ag and Tn recoveries were assumed to be equal. The relationship was used to estimate the Ag reporting to the copper concentrate with the remainder reporting to lead concentrate for a total of approximately 84%.

13.3.5 Flotation

The flotation circuit design criteria are based on ALS Project T26-28 flowsheet, reagent dosages, masspulls and flotation times with the exception of the pyrite rougher circuit. The flotation circuit feed size used for design is P80 = 45 microns. For lower zone ore the mineralogy indicates a finer grind size between 24 to 39 microns may be required to achieve liberation of the metals and met the target recoveries. The flowsheet includes sequential flotation of copper, lead, zinc and pyrite to produce 3 saleable concentrates. In the copper and lead circuits, there will be one bank of four rougher cells that feed two stages of four cleaner cells. The zinc circuit consists of two banks of roughers and cleaners with the same cell configuration as in the copper and lead circuits. The pyrite circuit consists of one bank of 4 rougher cells. The pyrite circuit design criteria are based on a compilation of results from ALS Projects T0662 - LC04 to 06 and T0897 - T32/33.

The copper circuit has two additional stages of cleaning included in the layout for future opportunities to split the Cp and Tn as well as produce a Zn Scavenger. The cells have been sized to meet the lip loading requirements.

There is opportunity to enhance the flotation performance by assessing the number and size of the flotation cells in each circuit. Flotation cell technology will be key in ensuring the recoveries are achievable. Vendors with proven equipment performance results should be considered in the next stage of engineering.

13.3.6 Dewatering and Filtration

Dewatering

The flotation concentrates and final tailings are envisioned to be dewatered in high rate thickeners. Standard settling tests have been carried out by ALS on the copper and zinc concentrates and pyrite tailings. ALS test work indicated the copper and zinc concentrate settle quickly and results show a 60% thickener underflow density can be achieved. The pyrite tailings were more difficult to settle and reached a maximum underflow density of approximately 50% in one hour. Vendor recommendations for settling rates of similar concentrates have been used to size the thickeners. Reagent requirements are from the 2012 FS. Two copper thickeners have been included in the design to allow separate handling of the low and high arsenic concentrates.

Filtration

Standard filtration test work has not been completed on the concentrates given the lack of sample size. The design criteria has been compiled from vendor recommendations using filtration rates of similar concentrates of similar grind size and the performance of their equipment. The moisture content of the concentrates has been assumed to be approximately 8%.

13.3.7 Metallurgical Predictions

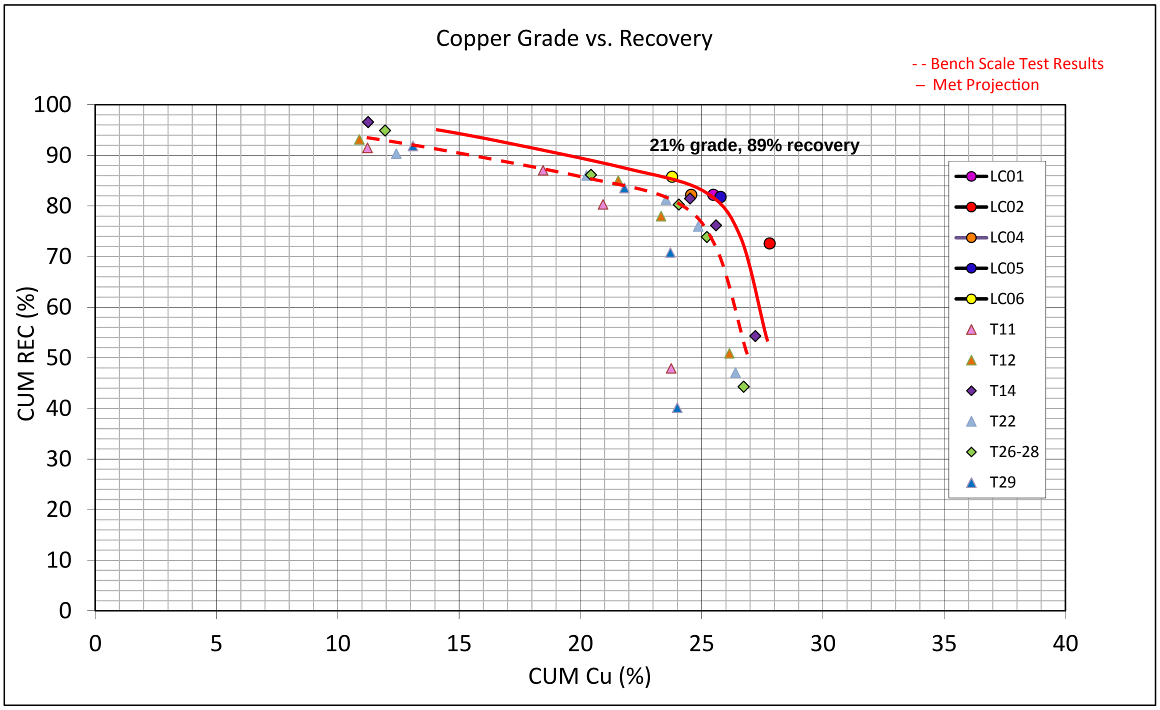
Two recent ALS Metallurgy, Burnie metallurgical test programs- ALS Project T0662 and T0897 were based on Chieftain mineralization. The projects examined five composite samples, designated as follows:

- Chieftain Composite 1 Upper Zone;
- Chieftain Composite 2 Lower Zone high As;
- Chieftain Composite 3 Lower Zone low As;
- Chieftain Composite 4 ALS Composites A upper zone; and.
- Chieftain Composite 4 ALS Composites B lower zone.

Most bench scale flotation tests from ALS Project T0662 focused on Chieftain Composite 1 and six Locked Cycle Tests on Chieftain Composites 1, 2 and 3. ALS Project T0897 examined the production of two copper concentrates; low and high As content. The initial bench scale flotation tests were performed using the remaining Chieftain Composite 2 and 3 from the 2011 test program and later drill core was sent to ALS as Composite 4 for the later part of the program. Due the age of the samples the expected recoveries were not achieved and a reasonable recovery of the two copper concentrates, Cp low As and Tn high As, was not realized. The decision was made to go forward with the one copper concentrate.

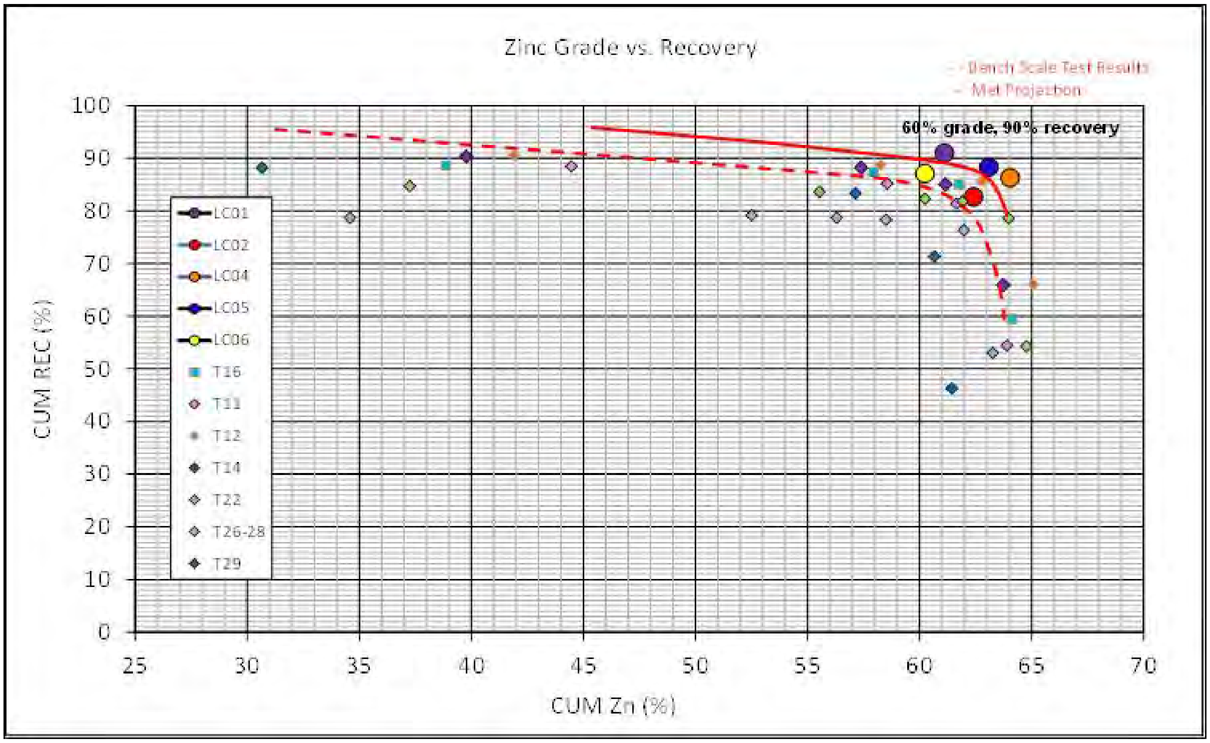
The results from ALS Project T0662, upper zone Chieftain composite 1 samples were used to provide the shape of the projected grade recovery curves for copper and zinc concentrates. The metallurgical projections have been factored up to results achieved in locked cycle tests performed on Chieftain Composites 1 to 3. While the locked cycle tests were performed under varying flotation test conditions, on different composites of varying grades, the data provide indications of trends that can be used to project metallurgical performance in an operating plant. Lead recovery is difficult to predict since the recovery is dependent on the performance of the previous circuit and resulting curves did not show a consistent trend. The lead grade and recovery was chosen within the range of the locked cycle test data.

Figure 13.5: Copper Grade vs. Recovery



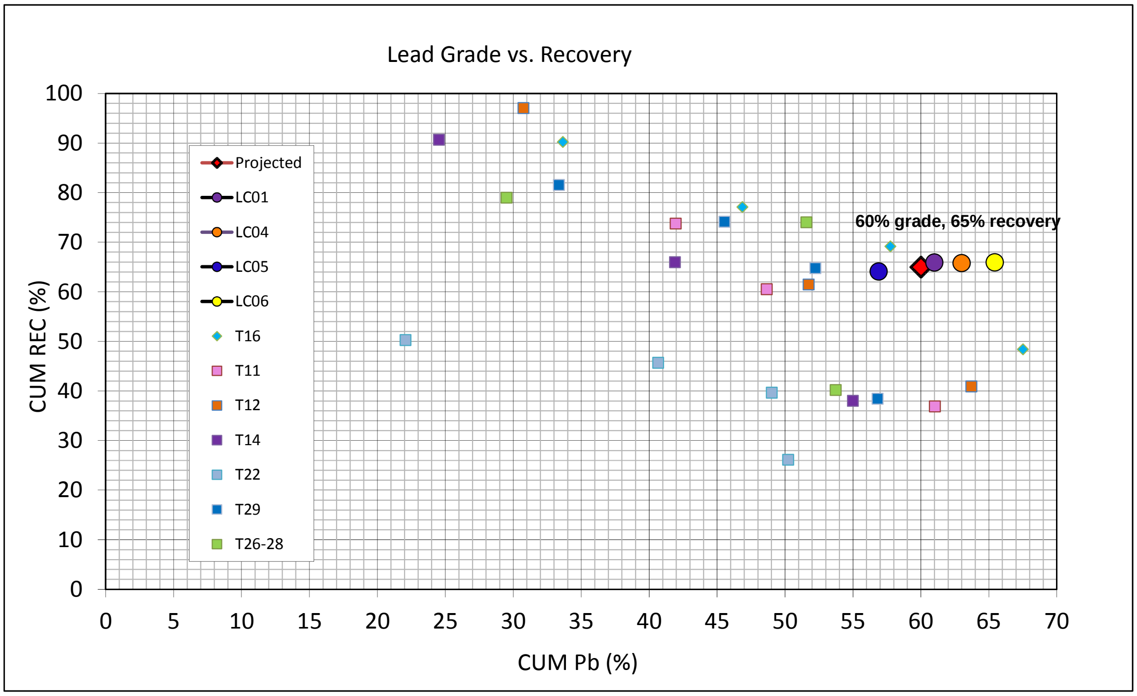
Source: ALS Project T0662

Figure 13.6: Zinc Grade vs. Recovery



Source: ALS Project T0662

Figure 13.7: Lead Grade vs. Recovery



Source: ALS Project T0662

The following Table 13.11 provides the metallurgical projection for the products produced in the process plant.

Table 13.11: Metallurgical Projection

Product	Wt (t)	Concentrate Assay Estimates					Recovery Estimates (%)				
		Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Cu	Pb	Zn	Ag	Au
Copper Conc	6.2	21	2.8	5.1	1300	22	89	13	4.5	78	47
Lead Conc	1.4	0.3	60	7.1	467	5.6	0.3	65	1.4	6.3	2.8
Zinc Conc	10.4	0.7	0.4	60	80	0.8	5	3.4	90	8	2.9
Pyrite Conc	33	0.2	0.3	0.6	22	0.3	3.6	8.5	2.9	6.9	3.6
Tailings	48.8	0.1	0.2	0.1	1.6	0.2	2	9	1	0.8	2.7
Feed	100	1.46	1.29	6.95	103.72	2.85	100	100	100	100	100
Gravity Concentrate	0.2	0.5	3	3	224	522	0.1	0.5	0.1	0.5	41

Source: JDS 2014

- The average recoveries from LC tests 01 to 06 (excluding LC03)were used to determine the approximate percent of individual metals in each of the products;
- Overall gold recovery is based on results from T(26, 28, 29) and T32 as discussed on page 89 of the IMC report, Consep test work and modelling report and gold deportment report. Gold in the copper and lead concentrate accounts for approximately 85% of the remaining gold available for recovery as discussed in 2012 FS;
- The silver grade in the gravity circuit was calculated using the electrum composition of 30% silver to 70% gold;
- Silver recovery was calculated based on the mineralogy results. From the mineralogy 95% of the silver is associated with the Tennantite (Tn), in the copper metal. The mineralogy was used to model the relationship between Tn and Chalcopyrite (Cp), based on Arsenic (As), content and copper head grade. Using the head grade of 1.46% and As content of 893 ppm in the feed material the copper metal in Cp was calculated to be approximately 83% and 17% in Tn. From this relationship the Tn recovery was calculated to be approximately 79% assuming 91% Cp recovery. The calculation is shown below:
- Ag Recovery = 0.95 x 79 x 103.7/100 = 77.6%
- The locked cycle tests and Project T0897 T32/33 were used as a guide to determine the pyrite concentrate grades and recoveries; and

  Antimony (Sb) recovery was assumed to be equal to As and Tn recoveries. Additional test work is recommended to confirm the relationship between Sb, As and Tn.
- Sb = % Tn x Sb grade / wt Cu

  As = % Tn x As grade / wt Cu

Source: 2012 FS Appendix 5.17 and 5.18 ALS Project T0662-1 and, T0662-2

13.3.8 Product Quality Predictions

Table 13.12: Typical Concentrate Analyses - 2012 FS

Element	Unit	Concentrate		
		Copper	Lead	Zinc
Zn	%	8.5	8.5	59.9
Cu	%	24.7	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
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.

Source:2012 FS

14. MINERAL RESOURCE ESTIMATE

14.1 Introduction

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

The resource estimation work was supervised by Dr. Gilles Arseneau, P.Geo (APEGBC # 23474) of SRK Consulting, who is an appropriate "independent qualified person" as defined in NI 43-101. The effective date of the resource statement is October 20, 2014.

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 volcanogenic massive sulphide 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 modeled 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 6.6 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 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; and
- Preparation of the Mineral Resource Statement.

14.3 Tulsequah Chief Mineral Resource

14.3.1 Tulsequah Chief Introduction

The mineral resource model prepared by SRK considers 818 core boreholes drilled by Cominco, Redfern and Chieftain during the period of 1940 to 2011.

14.3.2 Resource Database

The assay database for Tulsequah comprises 18,679 samples, 2,771 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, 10 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 by Cominco at closure had different statistical characteristics, all mineralized material above the 5200 level was assigned a temporary lens code COM (Cominco). From the statistical analysis of temporary lens COM, it was determined that the character of the COM lens was not significantly different from the other mineralized lenses, so the samples from lens COM were reclassified to their appropriate lenses before block modeling, either lens H3 or H4.

Figure 14.1 to 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 75th and 25th percentile, the median is indicated with the line across the box. The whiskers show the 95th and 5th percentile and the symbols indicate the maximum values (red circle), minimum values (blue dash), and the mean value is indicated by the green square.

Table 14.1: Tulsequah Chief Descriptive Statistics of Assay Data within the Mineralized Zones

Statistical Parameters	Cu %	Pb %	Zn %	Au g/t	Ag g/t
Valid cases	2,771	2,771	2,771	2,767	2,767
Mean	1.51	1.42	7.8	2.6	98.69
Std. error of mean	0.04	0.04	0.14	0.07	2.31
Variance	3.43	3.82	56.66	13.61	14,759.21
Std. Deviation	1.85	1.96	7.53	3.69	121.49
Variation Coefficient	1.22	1.38	0.97	1.42	1.23
rel. V. coefficient (%)	2.33	2.62	1.83	2.7	2.34
Skew	3.02	3.86	1.33	6.39	4.52
Minimum	0	0	0	0	0
Maximum	16.7	34.3	45.65	59.9	1,827.66
1st percentile	0	0	0	0	0.34
5th percentile	0.02	0	0.03	0.03	4.8
10th percentile	0.12	0.01	0.24	0.24	11
25th percentile	0.44	0.1	1.9	0.69	30.86
Median	0.92	0.79	5.5	1.71	65.15
75th percentile	1.83	1.91	11.7	3.09	122
90th percentile	3.43	3.7	18.4	5.49	212.6
95th percentile	5.23	5.15	22.6	7.86	305.18
99th percentile	9.62	8.53	32.61	16.42	580.49

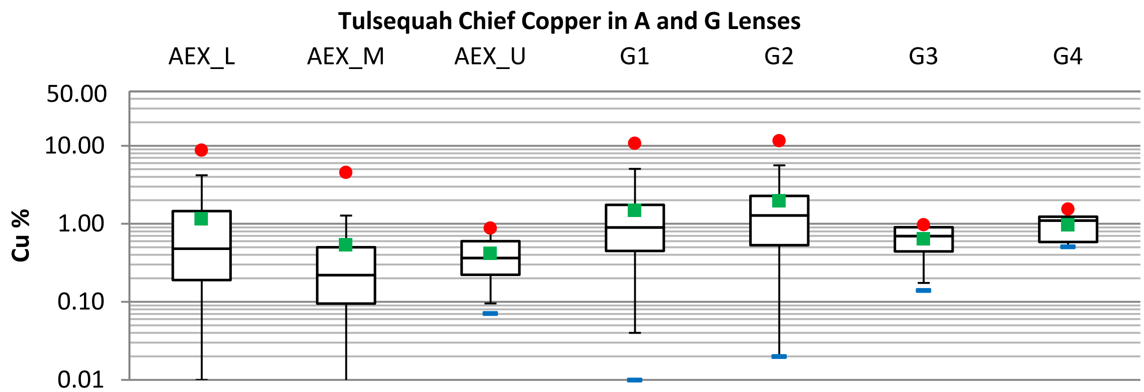
Source: Chieftain 2014

Table 14.2: Tulsequah Chief 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

Source: Chieftain 2014

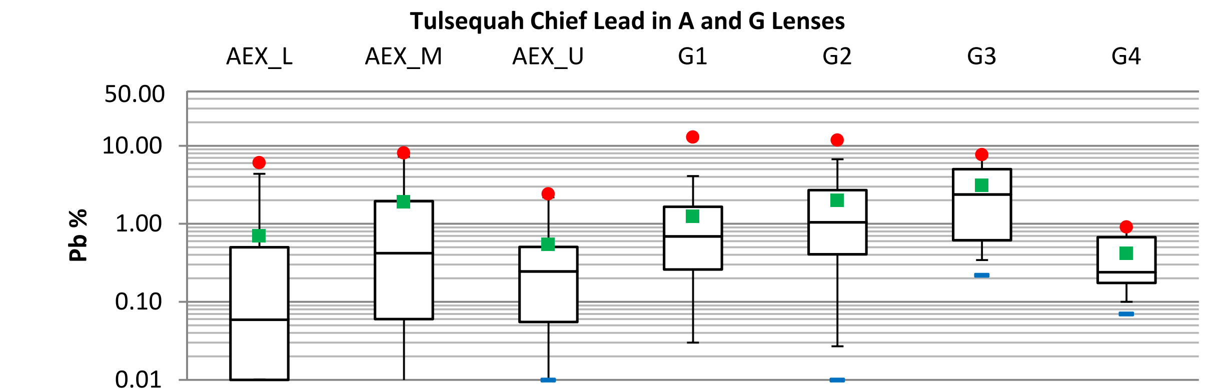
Figure 14.1: Box & Whisker Plot for Copper Assays in A & G Lenses



Cu %	AEX_L	AEX_M	AEX_U	G1	G2	G3	G4
Number of Data	104	20	18	241	38	8	7
Mean	1.16	0.54	0.42	1.48	1.97	0.64	0.97
Maximum	8.76	4.55	0.88	10.72	11.60	0.97	1.54
Upper quartile	1.46	0.50	0.60	1.75	2.28	0.90	1.24
Median	0.48	0.22	0.37	0.90	1.28	0.70	1.10
Lower quartile	0.19	0.10	0.22	0.45	0.53	0.44	0.59
Minimum	0.00	0.00	0.07	0.01	0.02	0.14	0.51
Variance	2.66	0.94	0.06	3.16	5.95	0.09	0.15
Std. Dev.	1.63	0.97	0.25	1.78	2.44	0.30	0.38
CV	1.41	1.81	0.60	1.20	1.24	0.47	0.39

Source: Chieftain 2014

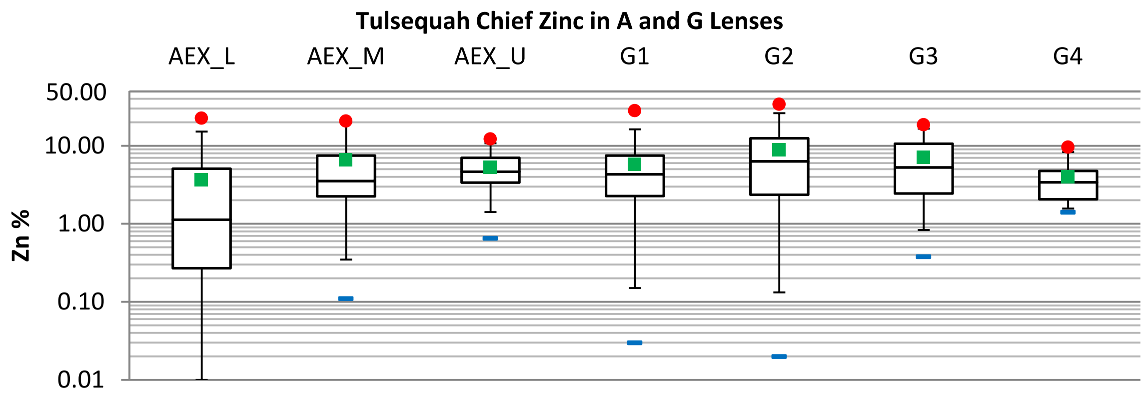
Figure 14.2: Box & Whisker Plot for Lead Assays in A & G Lenses



Pb %	AEX_L	AEX_M	AEX_U	G1	G2	G3	G4
Number of Data	104	20	18	241	38	8	7
Mean	0.70	1.91	0.54	1.25	1.99	3.10	0.42
Maximum	6.05	8.10	2.41	12.90	11.80	7.68	0.91
Upper quartile	0.50	1.95	0.51	1.65	2.70	5.01	0.68
Median	0.06	0.42	0.25	0.69	1.05	2.38	0.24
Lower quartile	0.01	0.06	0.06	0.26	0.41	0.62	0.18
Minimum	0.00	0.00	0.01	0.01	0.01	0.22	0.07
Variance	2.04	7.63	0.56	2.59	7.16	7.48	0.10
Std. Dev.	1.43	2.76	0.75	1.61	2.68	2.74	0.31
CV	2.05	1.45	1.37	1.29	1.34	0.88	0.75

Source: Chieftain 2014

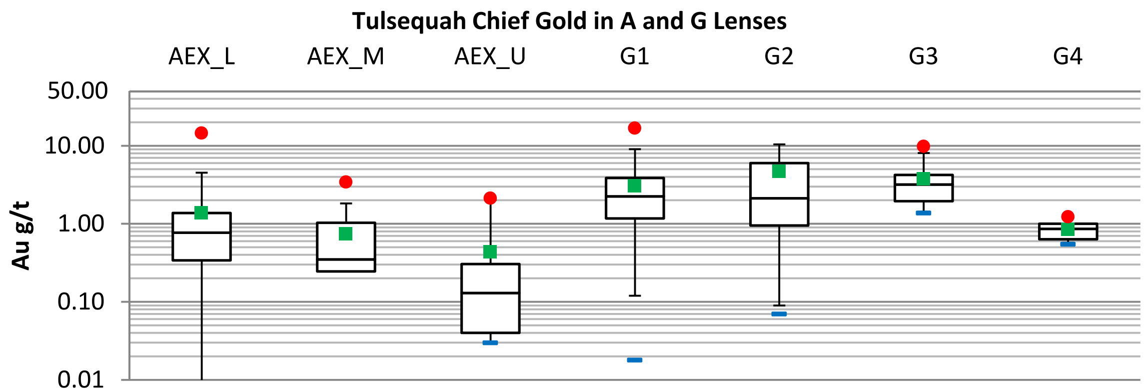
Figure 14.3: Box & Whisker Plot for Zinc Assays in A & G Lenses



Zn %	AEX_L	AEX_M	AEX_U	G1	G2	G3	G4
Number of Data	104	20	18	241	38	8	7
Mean	3.62	6.54	5.32	5.75	8.90	7.07	4.01
Maximum	22.50	20.70	12.20	28.10	34.00	18.60	9.58
Upper quartile	5.09	7.49	7.00	7.49	12.48	10.62	4.77
Median	1.13	3.54	4.64	4.30	6.33	5.27	3.40
Lower quartile	0.27	2.25	3.36	2.27	2.36	2.45	2.07
Minimum	0.00	0.11	0.65	0.03	0.02	0.38	1.41
Variance	26.13	44.82	8.83	26.71	68.40	34.99	6.80
Std. Dev.	5.11	6.69	2.97	5.17	8.27	5.92	2.61
CV	1.41	1.02	0.56	0.90	0.93	0.84	0.65

Source: Chieftain 2014

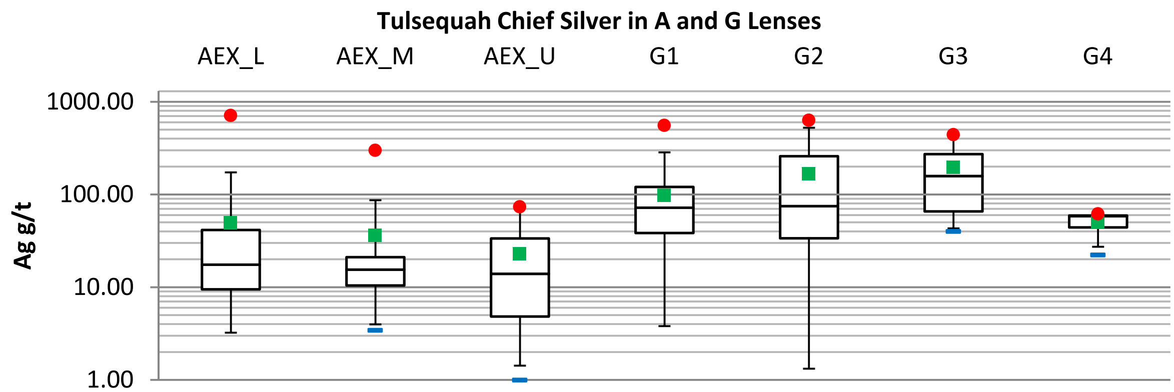
Figure 14.4: Box & Whisker Plot for Gold Assays in A & G Lenses



Au g/t	AEX_L	AEX_M	AEX_U	G1	G2	G3	G4
Number of Data	104	20	18	241	38	8	7
Mean	1.37	0.74	0.44	3.08	4.71	3.75	0.84
Maximum	14.50	3.43	2.13	16.80	57.50	9.81	1.23
Upper quartile	1.38	1.03	0.31	3.87	5.98	4.23	1.00
Median	0.77	0.35	0.13	2.25	2.12	3.18	0.86
Lower quartile	0.34	0.25	0.04	1.17	0.95	1.95	0.64
Minimum	0.00	0.00	0.03	0.02	0.07	1.38	0.55
Variance	4.06	0.65	0.43	8.72	85.60	6.55	0.06
Std. Dev.	2.01	0.80	0.66	2.95	9.25	2.56	0.24
CV	1.47	1.09	1.51	0.96	1.96	0.68	0.28

Source: Chieftain 2014

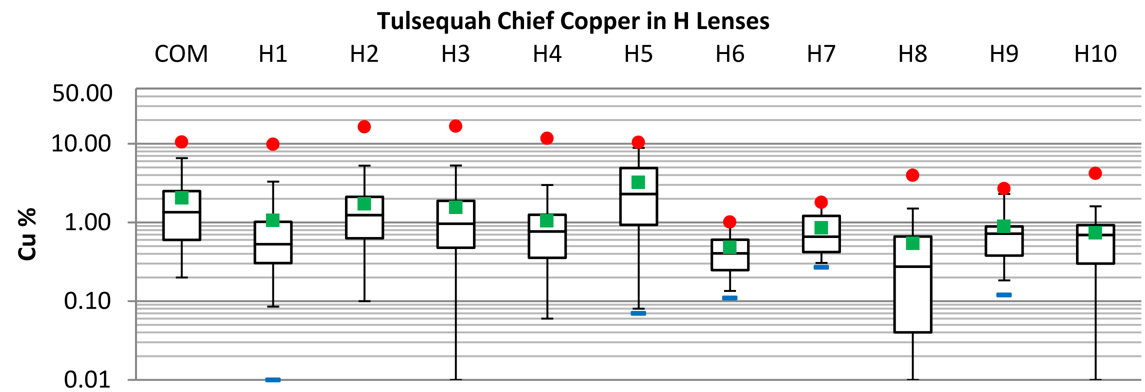
Figure 14.5: Box & Whisker Plot for Silver Assays in A & G Lenses



Ag g/t	AEX_L	AEX_M	AEX_U	G1	G2	G3	G4
Number of Data	104	20	18	241	38	8	7
Mean	49.49	36.08	22.77	97.20	167.28	195.79	49.86
Maximum	710.00	298.32	73.50	555.16	631.48	442.00	61.71
Upper quartile	41.41	21.08	33.58	120.34	258.70	272.00	59.15
Median	17.48	15.44	13.95	72.00	74.70	158.00	58.29
Lower quartile	9.48	10.45	4.83	38.40	33.70	65.74	44.19
Minimum	0.00	3.43	1.00	0.60	0.20	40.00	22.33
Variance	10076	4070.3	511.4	8455.2	33443	22580	181.2
Std. Dev.	100.38	63.80	22.62	91.95	182.87	150.27	13.46
CV	2.03	1.77	0.99	0.95	1.09	0.77	0.27

Source: Chieftain 2014

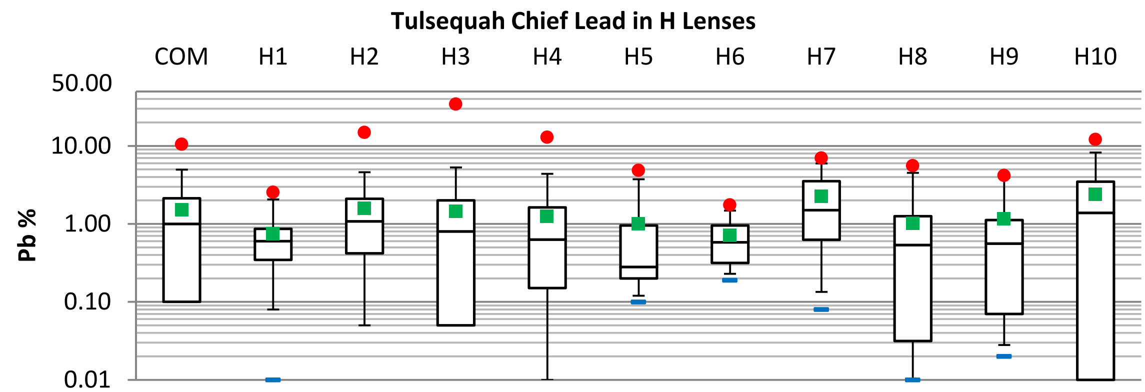
Figure 14.6: Box & Whisker Plot for Copper Assays in H Lenses



Cu %	COM	H1	H2	H3	H4	H5	H6	H7	H8	H9	H10
Number of Data	188	31	441	1364	188	11	8	19	36	9	40
Mean	2.04	1.06	1.72	1.56	1.05	3.24	0.48	0.84	0.54	0.90	0.74
Maximum	10.50	9.85	16.40	16.70	11.70	10.40	1.01	1.80	3.96	2.69	4.20
Upper quartile	2.50	1.02	2.11	1.88	1.25	4.90	0.60	1.21	0.66	0.89	0.93
Median	1.35	0.53	1.24	0.96	0.77	2.30	0.41	0.66	0.28	0.72	0.70
Lower quartile	0.60	0.31	0.63	0.48	0.36	0.93	0.25	0.42	0.04	0.38	0.30
Minimum	0.00	0.01	0.00	0.00	0.00	0.07	0.11	0.27	0.00	0.12	0.00
Variance	4.02	3.33	3.26	3.79	1.59	10.43	0.10	0.25	0.57	0.59	0.47
Std. Dev.	2.00	1.82	1.81	1.95	1.26	3.23	0.32	0.50	0.76	0.77	0.69
CV	0.98	1.73	1.05	1.25	1.20	1.00	0.67	0.59	1.40	0.86	0.93

Source: Chieftain 2014

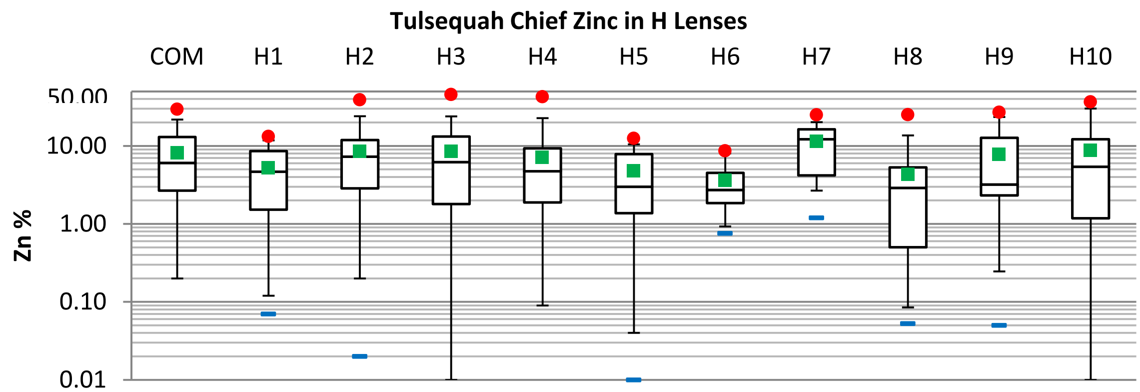
Figure 14.7: Box & Whisker Plot for Lead Assays in H Lenses



Pb %	COM	H1	H2	H3	H4	H5	H6	H7	H8	H9	H10
Number of Data	188	31	441	1364	188	11	8	19	36	9	40
Mean	1.50	0.75	1.58	1.44	1.24	1.00	0.71	2.26	1.02	1.16	2.41
Maximum	10.50	2.54	14.90	34.30	12.95	4.85	1.75	7.00	5.57	4.17	12.09
Upper quartile	2.13	0.87	2.10	2.00	1.63	0.96	0.96	3.53	1.26	1.12	3.48
Median	1.00	0.60	1.08	0.80	0.63	0.28	0.58	1.50	0.54	0.56	1.39
Lower quartile	0.10	0.35	0.42	0.05	0.15	0.20	0.32	0.63	0.03	0.07	0.01
Minimum	0.00	0.01	0.00	0.00	0.00	0.10	0.19	0.08	0.01	0.02	0.00
Variance	2.87	0.38	3.12	4.32	3.28	1.98	0.23	3.78	2.03	2.17	10.41
Std. Dev.	1.69	0.62	1.77	2.08	1.81	1.41	0.48	1.94	1.42	1.47	3.23
CV	1.13	0.83	1.12	1.44	1.46	1.41	0.68	0.86	1.39	1.27	1.34

Source: Chieftain 2014

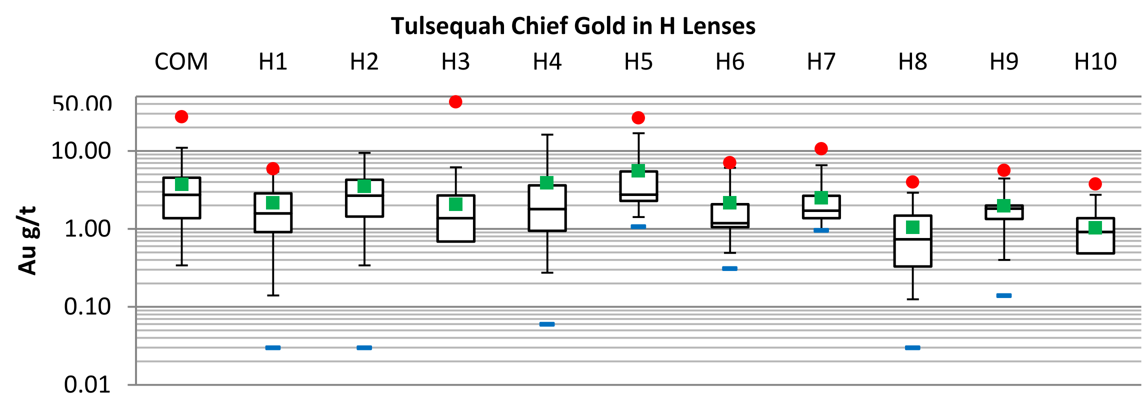
Figure 14.8: Box & Whisker Plot for Zinc Assays in H Lenses



Zn %	COM	H1	H2	H3	H4	H5	H6	H7	H8	H9	H10
Number of Data	188	31	441	1364	188	11	8	19	36	9	40
Mean	8.11	5.23	8.57	8.44	7.08	4.78	3.59	11.47	4.29	7.76	8.75
Maximum	29.60	13.25	39.00	45.65	42.70	12.50	8.65	25.09	25.10	26.97	36.70
Upper quartile	13.03	8.57	11.90	13.21	9.33	7.83	4.53	16.33	5.30	12.70	12.20
Median	6.05	4.67	7.27	6.21	4.73	3.00	2.73	12.18	2.89	3.20	5.40
Lower quartile	2.68	1.52	2.86	1.80	1.89	1.37	1.85	4.18	0.50	2.32	1.18
Minimum	0.00	0.07	0.02	0.00	0.01	0.01	0.76	1.20	0.05	0.05	0.00
Variance	48.07	16.22	53.32	64.59	56.93	15.74	6.78	44.20	32.38	78.95	94.99
Std. Dev.	6.93	4.03	7.30	8.04	7.55	3.97	2.60	6.65	5.69	8.89	9.75
CV	0.85	0.77	0.85	0.95	1.07	0.83	0.72	0.58	1.33	1.15	1.11

Source: Chieftain 2014

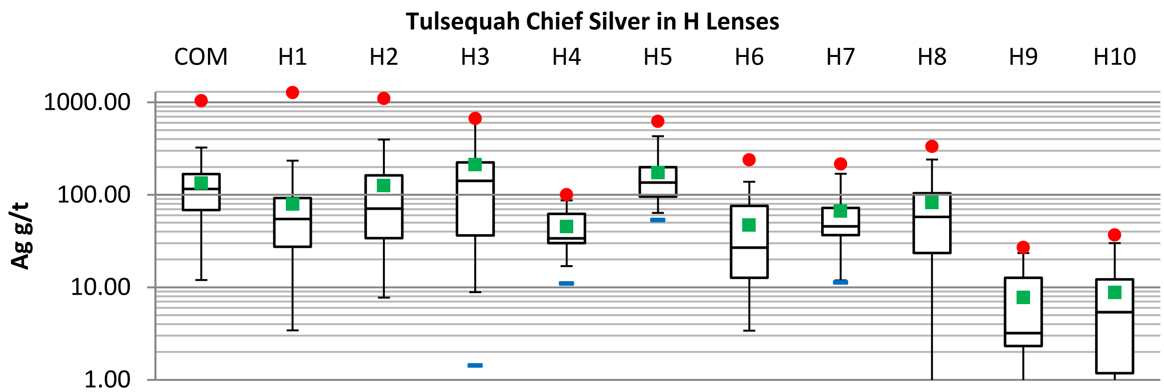
Figure 14.9: Box & Whisker Plot for Gold Assays in H Lenses



Au g/t	COM	H1	H2	H3	H4	H5	H6	H7	H8	H9	H10
Number of Data	188	31	441	1360	188	11	8	19	36	9	40
Mean	3.70	2.15	3.53	2.06	3.90	5.53	2.16	2.48	1.04	1.98	1.03
Maximum	27.43	5.90	59.90	42.52	50.27	26.50	7.07	10.62	3.99	5.62	3.77
Upper quartile	4.55	2.86	4.26	2.69	3.62	5.45	2.08	2.66	1.48	1.99	1.37
Median	2.74	1.58	2.67	1.37	1.80	2.75	1.19	1.71	0.74	1.82	0.92
Lower quartile	1.37	0.91	1.44	0.69	0.95	2.29	1.06	1.37	0.33	1.34	0.49
Minimum	0.00	0.03	0.03	0.00	0.06	1.07	0.31	0.96	0.03	0.14	0.00
Variance	14.55	2.64	16.67	7.39	46.14	47.61	4.64	4.99	0.91	2.12	0.77
Std. Dev.	3.81	1.63	4.08	2.72	6.79	6.90	2.15	2.23	0.96	1.46	0.88
CV	1.03	0.76	1.16	1.32	1.74	1.25	1.00	0.90	0.92	0.74	0.85

Source: Chieftain 2014

Figure 14.10: Box & Whisker Plot for Silver Assays in H Lenses



Ag g/t	COM	H1	H2	H3	H4	H5	H6	H7	H8	H9	H10
Number of Data	441	1360	188	13	8	19	36	9	40	9	40
Mean	132.60	78.97	125.12	210.94	45.28	173.98	47.25	66.72	82.14	7.76	8.75
Maximum	1040.00	1272.16	1100.00	668.00	100.00	620.73	238.00	214.63	332.61	26.97	36.70
Upper quartile	167.45	92.39	162.25	224.00	62.17	199.04	75.98	72.01	104.15	12.70	12.20
Median	116.00	54.86	71.06	142.00	33.92	136.00	26.90	45.60	57.74	3.20	5.40
Lower quartile	68.57	27.43	33.97	36.40	30.03	95.79	12.68	36.69	23.49	2.32	1.18
Minimum	0.50	0.00	0.20	1.43	11.00	53.33	0.20	11.32	0.34	0.05	0.00
Variance	12310	9889	24822	51886	690	17109	2416.1	3454.1	6466.7	79.0	95.0
Std. Dev.	110.95	99.44	157.55	227.78	26.27	130.80	49.15	58.77	80.42	8.89	9.75
CV	0.84	1.26	1.26	1.08	0.58	0.75	1.04	0.88	0.98	1.15	1.11

Source: Chieftain 2014

Zone COM (Cominco) represents all the drill samples found within the mine workings. These samples were reclassified to their appropriate zone for modeling, either H3 or H4. Zone H5 appears to demonstrate a higher copper and silver concentration and zone H7 appears to have higher zinc content than all the other zones. However, these slight differences are attributed to a small sample population comprising zones H5 and H7.

14.3.3 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 volcanogenic massive sulphide deposits, Pb-Zn, Pb-Ag, and Au-Ag display good positive correlations. At Tulsequah Chief, Pb and Zn representing Sphalerite and Galena co-occurring correlate well together though the relationship becomes poor at higher grades, which is usually expected. Ag and Pb representing 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 of Sphalerite and Chalcopyrite. This probably reflects the typical zoning patterns often associated with VMS deposits.

Table 14.3: Tulsequah Chief Assay Correlation Coefficient Matrix for Tulsequah Deposit

Metal	Cu %	Pb %	Zn %	Au g/t	Ag g/t
Cu %	1	-0.078	0.025	0.192	0.318
Pb %	-0.078	1	0.564	0.168	0.347
Zn %	0.025	0.564	1	0.119	0.231
Au g/t	0.192	0.168	0.119	1	0.487
Ag g/t	0.318	0.347	0.231	0.487	1

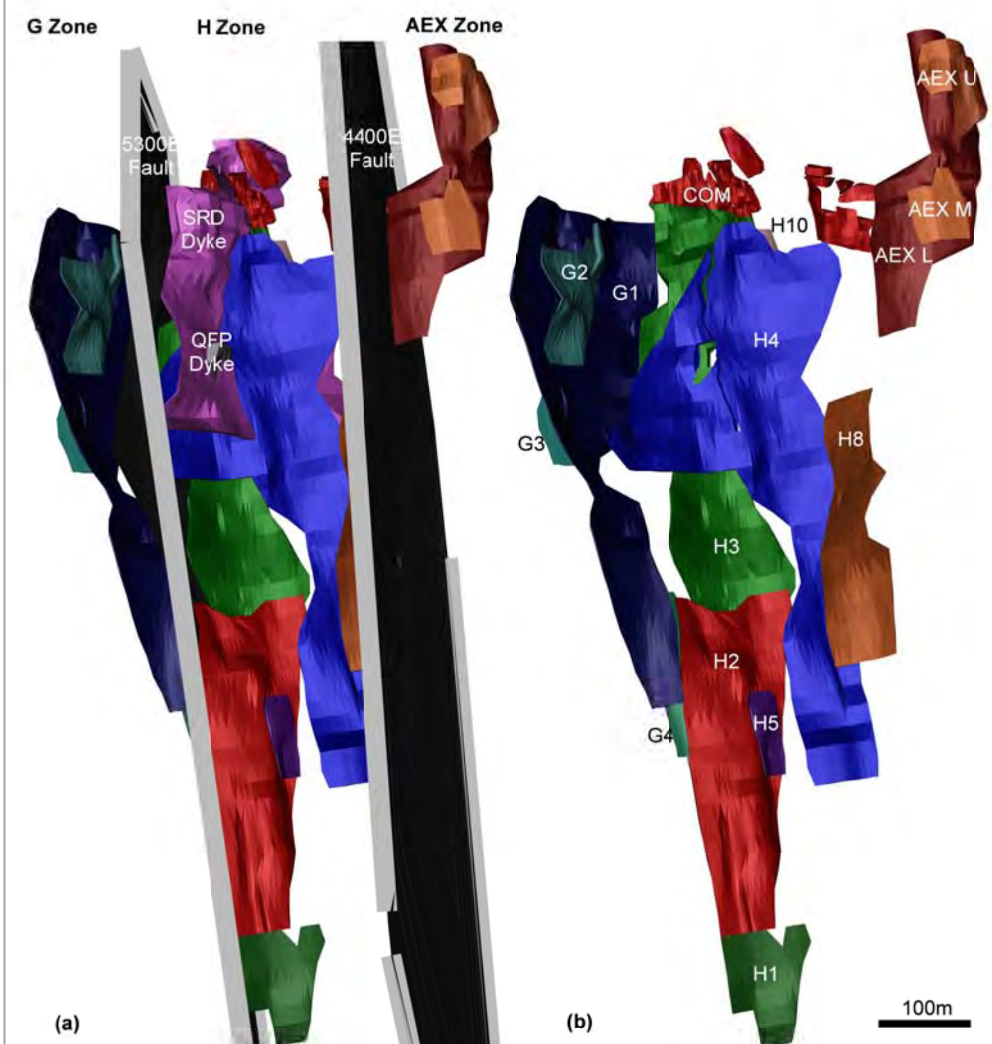
Source: SRK

14.3.4 Solid Body Modeling

The Tulsequah Chief deposit comprises 17 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 5,300 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 the H-syncline upright fold having a 55° to 65° plunge to the northwest (~325° azimuth). The A-Extension Zone consists of three sub-parallel zones simply named AEX_L, AEX_M and AEX_U. The zones occur west of the 4,400E 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: Tulsequah Chief 3D Perspective View Looking Southeast of the Mineralized Lenses



Note: Fig (a) shows the G, H, and AEX zones separated by the 5300E and 4400E main faults, and late intrusive Skoko Rhyolite Dyke (SRD) and Quartz Feldspar Porphyry Dyke (QFP). Fig (b) shows the massive sulphide lenses, with the H6, H7 and H9 lenses not shown - in the footwall hidden behind the shown Lenses. Approximate 100m scale bar is shown.

Source: Chieftain 2014

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.

14.3.5 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 down hole 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. Twenty five composites 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 with the composite assay interval length calculation extended to 1 m and were used during estimation. The statistical properties of the composited metal data by lens are summarized in Table 14.4.

Table 14.4: Tulsequah Chief Statistical Data for 2 m Capped Composites by Lens

Element	Lens	Mean	Q25	Q50	Q75	Max	No. Of Comps
Cu	AEX_L	0.89	0.17	0.46	1.1	6.91	73
	AEX_M	0.4	0.02	0.19	0.48	2.7	18
	AEX_U	0.42	0.28	0.4	0.5	0.82	12
	G1	1.38	0.49	0.88	1.66	8.83	111
	G2	1.75	0.79	0.97	2.18	6.75	16
	G3	0.68	0.51	0.52	0.88	0.97	5
	G4	0.67	0.41	0.47	0.84	1.21	3
	H1	0.76	0.28	0.42	0.72	4.91	17
	H2	1.74	0.78	1.3	2.21	8.09	227
	H3 & COM	1.28	0.41	0.85	1.57	10	1,069
	H4 & COM	1.49	0.52	0.92	1.94	9.91	221
	H5	3.09	1.12	3.2	4.56	6.98	7
	H6	0.5	0.21	0.43	0.66	1.01	5
	H7	0.49	0	0.42	0.93	1.32	14
	H8	0.54	0.12	0.39	0.54	2.59	17
	H9	0.92	0.7	0.92	1.14	1.36	3
	H10	0.73	0.42	0.7	0.95	2.29	28
Pb	AEX_L	0.49	0.01	0.05	0.36	5.8	73
	AEX_M	1.85	0.02	0.45	3.11	7.2	18
	AEX_U	0.46	0.13	0.21	0.46	2	12
	G1	1.1	0.38	0.76	1.58	4.73	111
	G2	1.85	0.81	1.53	2.31	7.95	16
	G3	2.07	0.63	0.98	3.99	4.55	5
	G4	0.44	0.39	0.43	0.48	0.52	3
	H1	0.68	0.41	0.53	0.85	1.57	17
	H2	1.51	0.61	1.27	1.98	7.35	227
	H3 & COM	1.3	0.12	0.84	1.9	8.79	1,069
	H4 & COM	1.36	0.34	0.96	1.95	10	221
	H5	1.07	0.27	0.76	1.28	3.45	7
	H6	0.64	0.33	0.52	0.81	1.23	5
	H7	1.16	0	1.18	2.06	2.54	14
	H8	0.83	0.13	0.46	1.01	3.91	17
	H9	1.03	0.36	0.37	1.36	2.36	3
	H10	2.08	0.01	1.84	3.48	7.8	28
Zn	AEX_L	2.56	0.13	1.03	4.01	16.37	73
	AEX_M	5.93	0.69	3.18	10.6	20.58	18
	AEX_U	5.36	3.52	5.18	6.9	9.79	12
	G1	5.35	2.95	4.42	7.12	24.7	111
	G2	8.98	3.85	6.23	12.99	27	16
	G3	5.12	2.7	3.42	8.49	9.33	5
	G4	3.48	2.66	3.52	4.32	5.12	3
	H1	4.66	2.44	3.74	5.26	11.48	17
	H2	8.51	3.9	7.65	11.78	30	227
	H3 & COM	7.72	2.13	6.2	11.83	30	1,069
	H4 & COM	7.28	2.86	5.74	10.19	26.76	221
	H5	5.55	3.61	5.22	7.68	10.94	7
	H6	3.26	1.86	2.17	3.56	7.05	5
	H7	5.95	0	3.58	10.69	17.51	14
	H8	4.13	1.08	3.16	5.27	19.23	17
	H9	6.44	3.77	4.65	8.21	11.77	3
	H10	7.64	1.41	7.64	11.9	24.17	28
Au	AEX_L	1.02	0.08	0.69	1.26	7.45	73
	AEX_M	0.66	0.14	0.29	0.96	3.43	18
	AEX_U	0.43	0.06	0.12	0.46	2.04	12
	G1	2.95	1.26	2.21	3.93	10.79	111
	G2	3.69	1.34	1.81	5.94	14.59	16
	G3	3.11	1.51	2.09	3.95	6.63	5
	G4	0.79	0.71	0.82	0.89	0.96	3
	H1	2.02	1.16	1.96	2.95	5.18	17
	H2	3.42	1.83	2.82	4.19	14.68	227
	H3 & COM	1.77	0.59	1.29	2.28	18.41	1,069
	H4 & COM	3.55	1.26	2.54	4.11	25	221
	H5	4.63	2.82	3.33	6.39	9.26	7
	H6	2.52	1.18	1.3	2.44	7.07	5
	H7	1.54	0	1.32	2.26	5.45	14
	H8	0.97	0.4	0.72	1.55	2.77	17
	H9	1.98	1.33	1.38	2.33	3.28	3
	H10	1.08	0.54	0.94	1.56	3.77	28
Ag	AEX_L	37.75	5.3	13.72	31.5	600	73
	AEX_M	35.73	6.01	12.98	22.08	298.32	18
	AEX_U	22.08	6.9	16.03	38.29	56.84	12
	G1	91.22	41.34	72.18	118.65	365.57	111
	G2	161.65	51.59	74.33	220.26	523.07	16
	G3	138.14	49.03	71.31	239.17	291.2	5
	G4	40.96	32.4	32.95	45.52	58.09	3
	H1	52.46	36.85	51.86	67.07	134.04	17
	H2	130.63	81.56	111.11	163.51	496.94	227
	H3 & COM	67.99	27.43	51.9	82.34	600	1,069
	H4	133.54	49.67	103.48	184.82	600	221
	H5	238.65	100.27	157.1	351.58	572.7	7
	H6	43.95	32.27	33.3	61.99	74.3	5
	H7	95.01	0	100.21	169.48	239.46	14
	H8	45.15	12.87	25.38	77.75	139.05	17
	H9	65.32	39.55	41.37	79.12	116.86	3
	H10	80.43	22.05	59.56	94.38	332.61	28

Source: Chieftain 2014

14.3.6 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: Tulsequah Chief Capping Levels

Metal	Cap level	No capped	CoV uncapped	CoV Capped	Metal loss (%)
Cu	10%	20	1.22	1.17	0.68%
Pb	10%	18	1.38	1.28	0.59%
Zn	30%	43	0.97	0.94	0.85%
Au	25 g/t	10	1.42	1.22	1.93%
Ag	600 g/t	26	1.23	1.08	1.01%

Source: Chieftain 2014

14.3.7 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.

14.3.8 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 three 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: Tulsequah Chief 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.	
H2,H3 &COM	ID2	1	Ellipse	-80	-60	0	40	30	20	5	8	2
A Extn	ID2	2	Ellipse	-35	-50	-30	70	60	40	3	8	2
G	ID2	2	Ellipse	-35	-50	-30	70	60	40	3	8	2
H8	ID2	2	Ellipse	-40	-50	50	40	60	20	3	8	2
H1-7, 9&10	ID2	2	Ellipse	-80	-60	0	60	45	30	3	8	2
All Zones	ID2	3	Sphere	0	0	0	120	120	120	2	12	No restriction

Source: Chieftain 2014

The first pass required that at least three drill holes and five composites be available within the search ellipse to estimate a grade within a block in H2 and H3/Com areas where there is high density drilling information, to be considered for measured classification. 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 drill holes and three composites be available within the search ellipse to estimate a grade within a block, to be considered for indicated classification, and again a maximum of eight composites were used to interpolate a grade value. The Third 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 pass 2 and pass 3. 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 H4/COM, H9, and AEX_M lenses, 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.

14.3.9 Model Validation 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: Tulsequah Chief Swath Plots of Copper Composites & Copper Block Grades

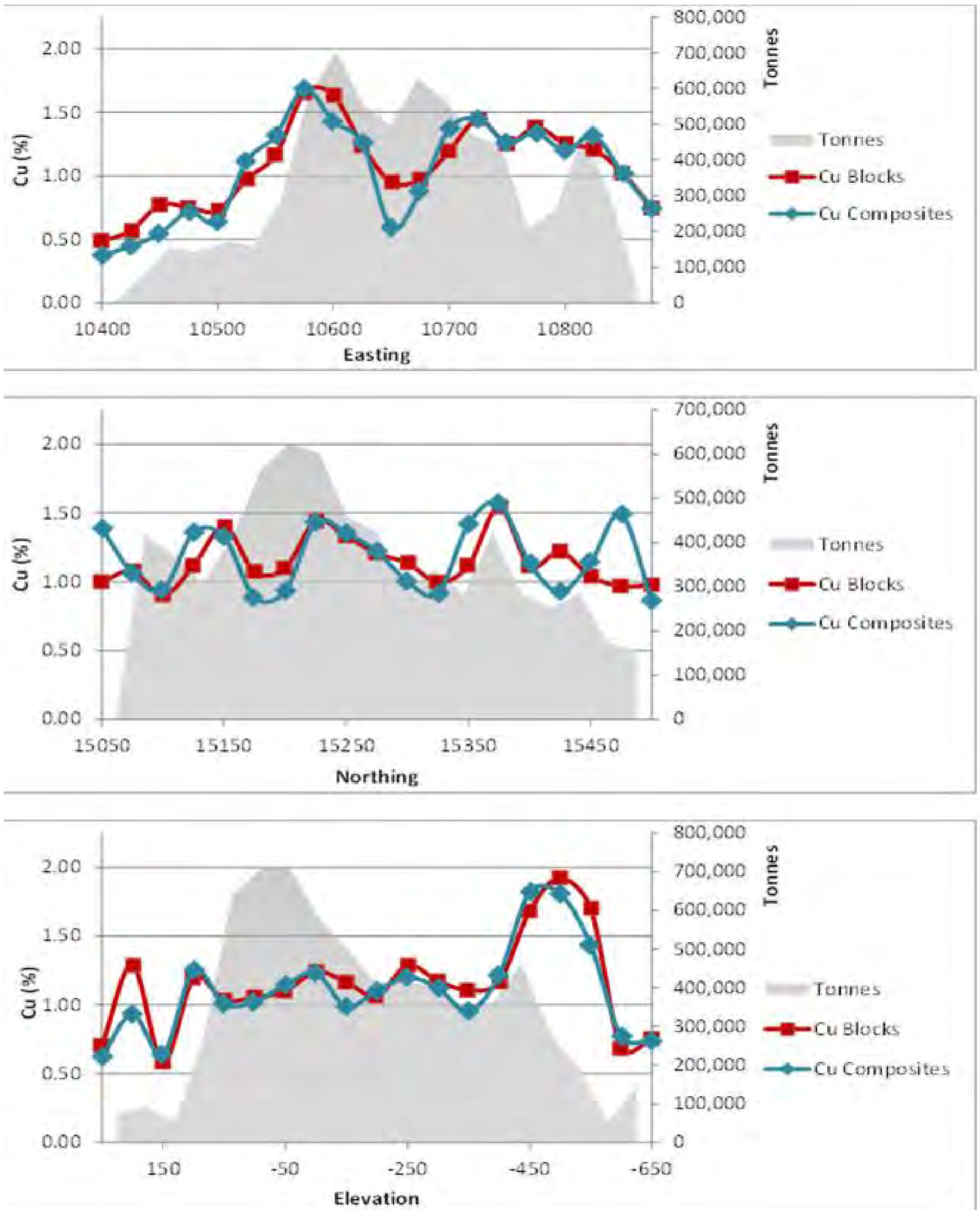


Figure 14.13: Tulsequah Chief Swath Plots of Lead Composites & Lead Block Grades

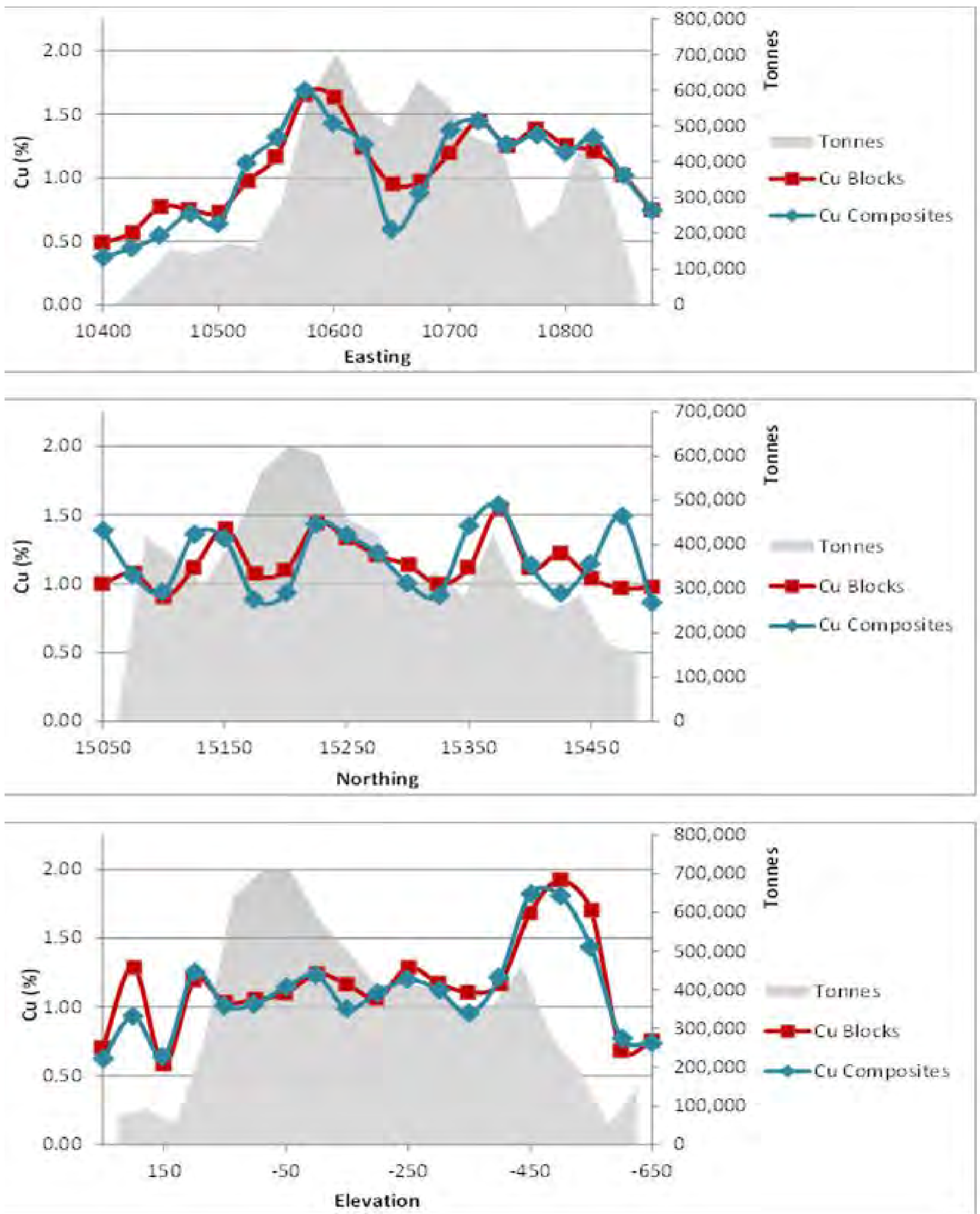


Figure 14.14: Tulsequah Chief Swath Plots of Zinc Composites & Zinc Block Grades

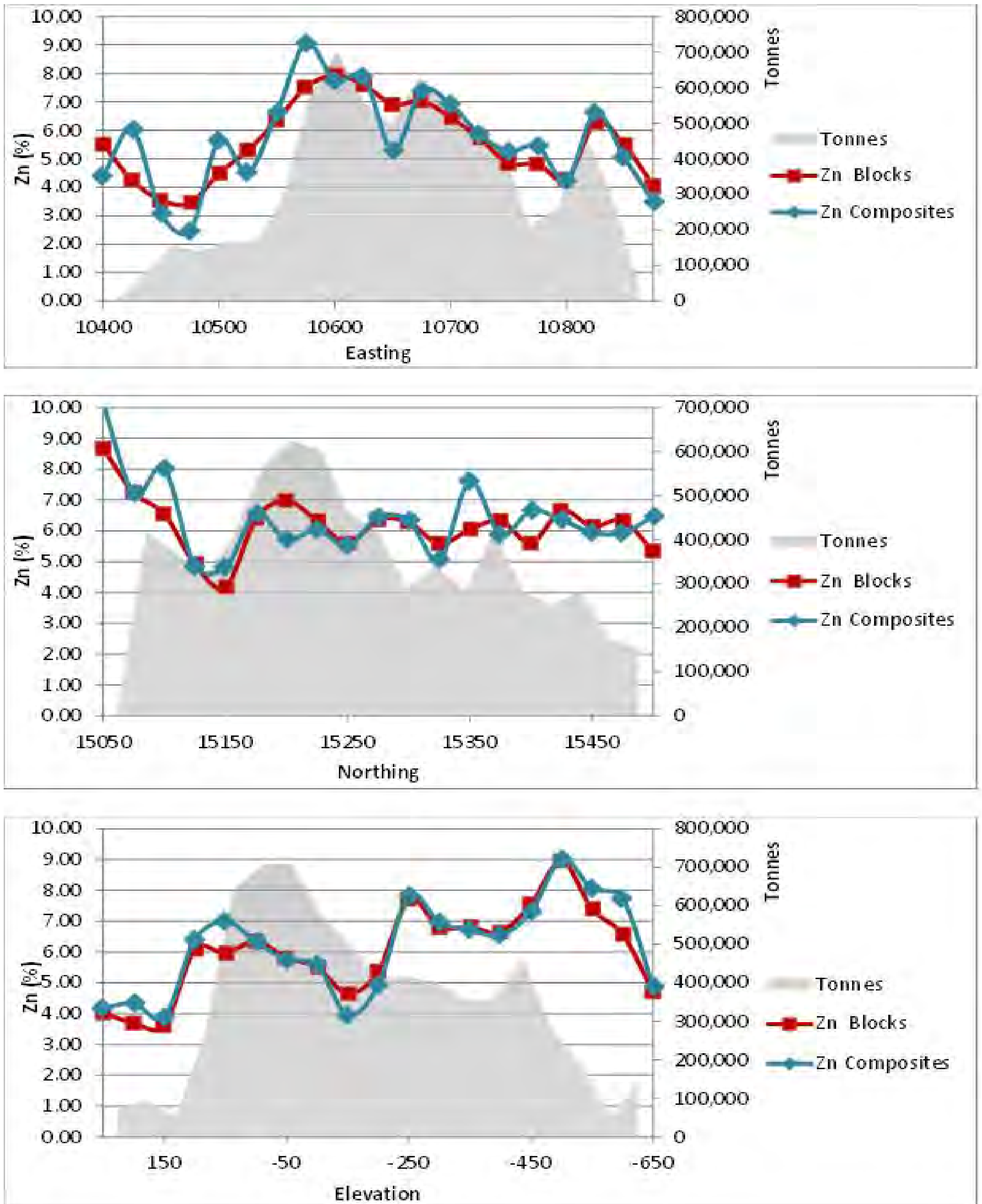


Figure 14.15: Tulsequah Chief Swath Plots of Gold Composites & Gold Block Grades

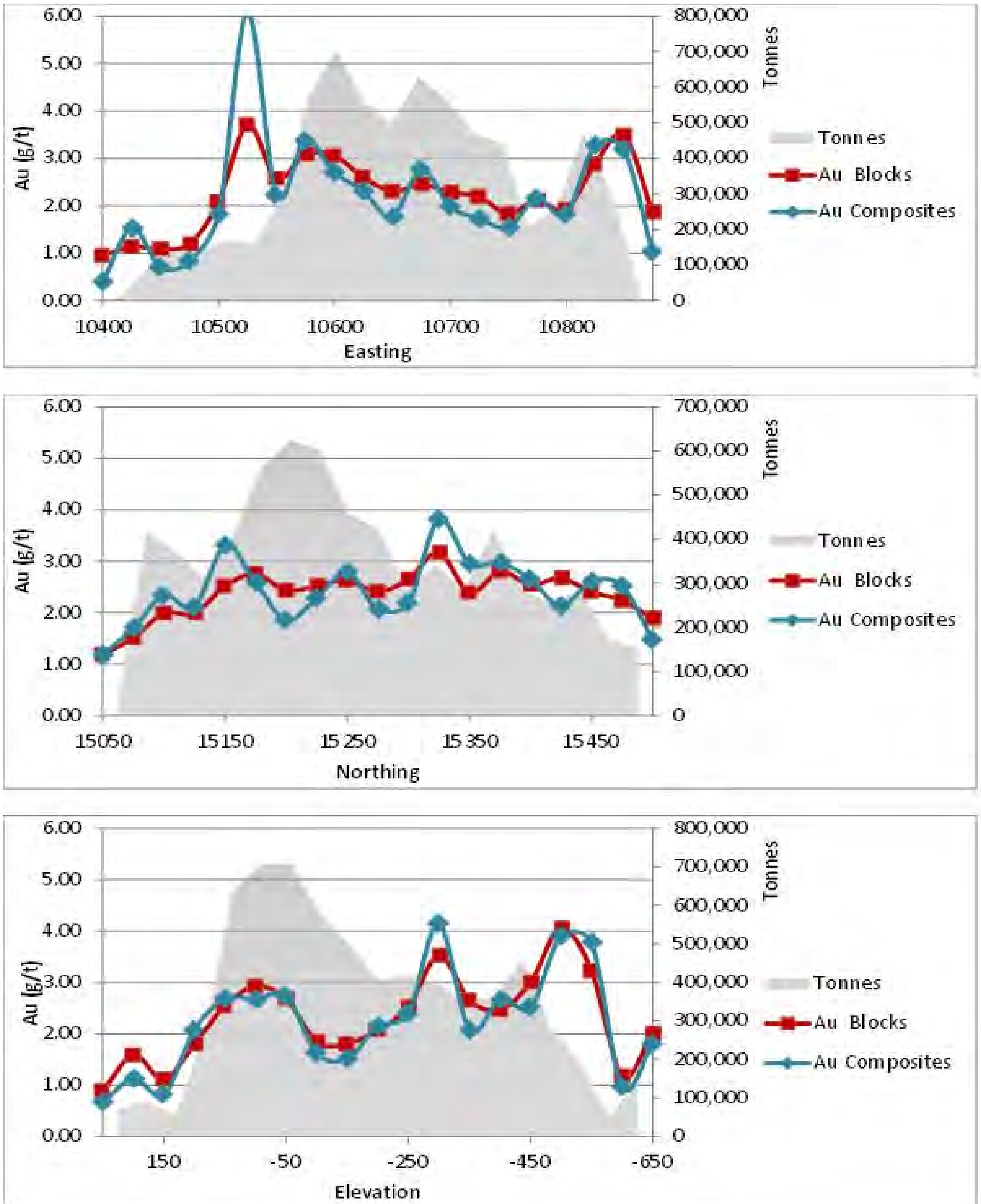
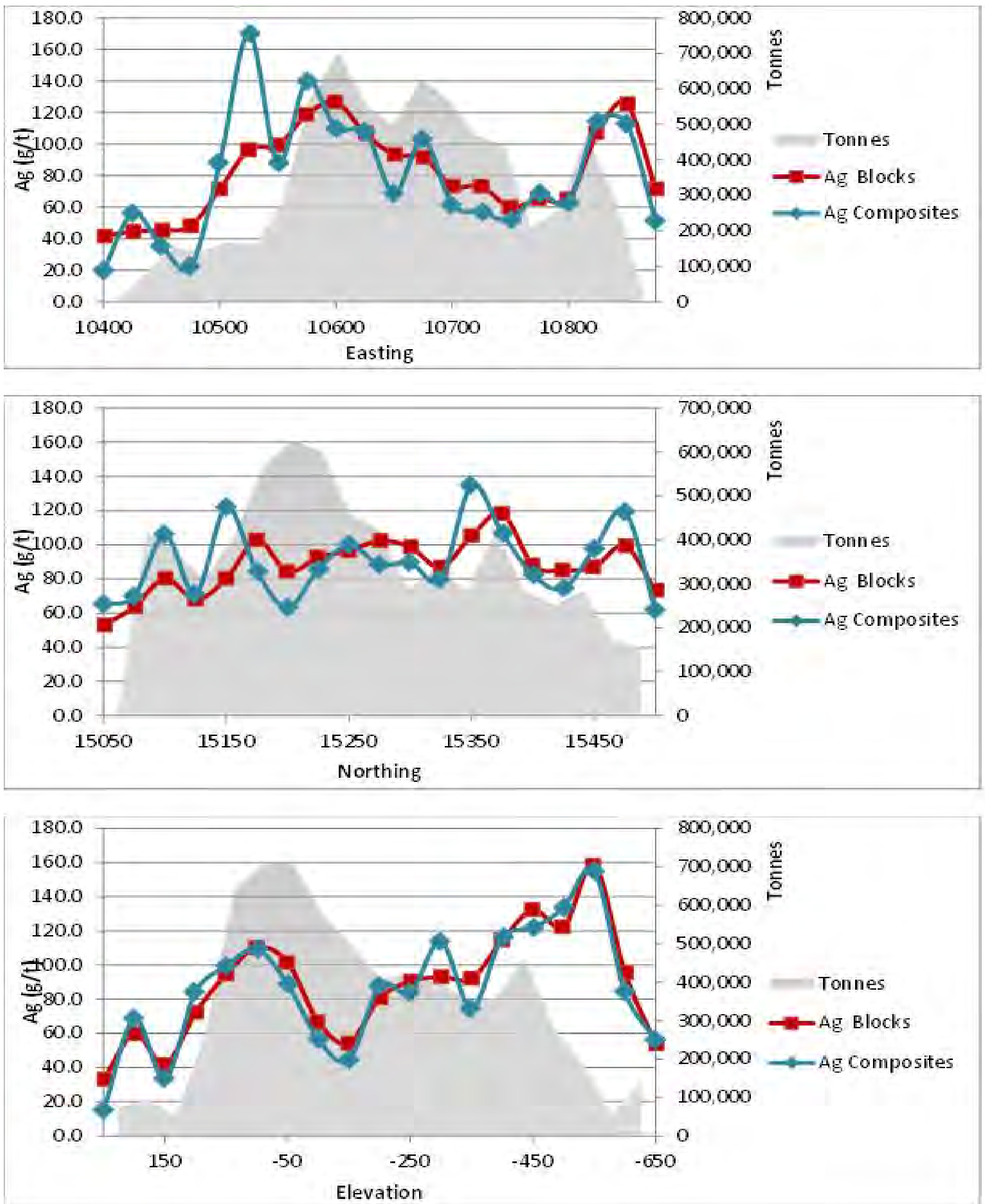


Figure 14.16: Tulsequah Chief Swath Plots of Silver Composites & Silver Block Grades



14.3.10 Model Validation Sensitivity Check for Bias

The model was checked for global bias by comparing the ID2 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 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 ID2 model when compared at a $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 ID2 estimate but the tonnes are slightly lower which yields a very similar overall total metal content, about 1% less metal is reported in the NN model than in the ID2 model at the US$100 Eq cut-off.

Table 14.7: Tulsequah Chief Percent Difference between NN & ID2 Results at US$100Eq

	(NN-ID2)/NN in %	Metal Grade Differences in % (NN-ID2)/NN				
Class	**tonnes**	**Cu %**	**Pb %**	**Zn %**	**Au %**	**Ag %**
Measured	-9.4	3.3	7.9	7.4	4.9	6.3
Indicated	-10.9	11	9.9	10.1	10	9.2
Inferred	-16.6	1.8	-0.8	0.9	11.4	-5
M+I+I	-11.1	9.9	9.2	9.3	9.5	8.3

Source: Chieftain 2014

14.3.11 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 (May 2014) by Dr. Gilles Arseneau, P.Geo. (APEGBC, # 23474), an appropriate independent qualified person for the purpose of NI 43-101.

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 modeling 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 30m.

Mineralization continuity that is confirmed by closely spaced drilling in the upper H3/H4/COM lenses adjacent to the previously producing mine area, mineralization in the lower H2 lens with close spaced sampling and actuate location, and with 10 m of a sample location selected for estimation during the first run, SRK considers can be classified in the measured category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Mineralization exhibiting good geological continuity investigated at an adequate spacing with reliable sampling information accurately located, SRK considers that blocks estimated during the second estimation run can be classified in the indicated category. For the Measured and Indicated classified 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.

Conversely, blocks estimated during the third 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.

14.3.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

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

The "reasonable prospects of eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, 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 eventual economic extraction" from an underground mine, based on parameters summarized in Tables 14.8 and 14.9.

Table 14.8: Net Smelter Return Calculation Data

Metal	Price (US$)	Metallurgical Recovery (%)	NSR Payable Metal (%)	$NSR CAD Multiplication factor
Au	1,250/oz	90.0	80.7	36.69
Ag	19.00/oz	84.5	72.5	0.5013
Cu	2.75/lb	89.0	52.8	36.24
Pb	0.90/lb	66.2	41.8	9.39
Zn	0.90/lb	89.0	45.4	10.20
FX: USD$:CAD$ = 1.00:X	0.93			

*Payable metal (%) including metal prices, metallurgical recoveries, transport, and smelter charges.

Source: Chieftain 2014

Table 14.9: Conceptual Assumptions Considered for Underground Resource Reporting

Parameter	Value	Unit
Exchange rate	0.93	USD$/$CAD
Mining costs	$32.00	CAD$/t mined
Process cost	$18.00	CAD$/t of feed
General and Administrative	$10.00	CAD$/t of feed
Power	$20.00	CAD$/t of feed
Transport	$20.00	CAD$/t of feed
Total Costs	$100.00	CAD$/t of feed
Assumed mining rate	3,000	tpd

Source: Chieftain 2014

SRK considers that the blocks that had total dollar values above $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*, Tulsequah Chief Deposit, Tulsequah Chief Project, British Columbia, SRK Consulting (Canada) Inc. October 20, 2014

Category	M Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (gpt)	Ag (gpt)	Zn (EQ%)
Measured	0.787	1.57	1.50	8.60	2.81	105.5	30.9
Indicated	5.136	1.43	1.28	6.76	2.8	102.1	28.1
Total Measured +Indicated	**5.923**	**1.45**	**1.31**	**7.00**	**2.80**	**102.5**	**28.5**
Total Inferred	0.439	0.79	1.03	5.54	2.33	80.6	21.6

*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 NSR cut-off grade of CAD$100. Cut-off grades are based on a Net Smelter Return of payable metals including metal prices, metallurgical recoveries, transport, and smelter charges; Metal prices: US$1,250/oz of gold, US$19/oz for silver, US$0.90/lb for zinc and lead and US$2.75 for copper, USD:CAD 1:0.93; metallurgical recoveries of 90.0% for gold, 84.5 for silver, 89.0 for copper, 66.2% for lead and 89.0% for zinc; and payable metal: 84.9% gold, 76.3% silver, 55.6% copper, 44.0% lead and 47.8% zinc. Zn EQ% = ((Au g/t*36.69x) + (Ag g/t*0.5013) + (Cu %*36.24) + (Pb %*9.39) + (Zn %*10.2))/10.2

14.3.13 Grade Sensitivity Analysis

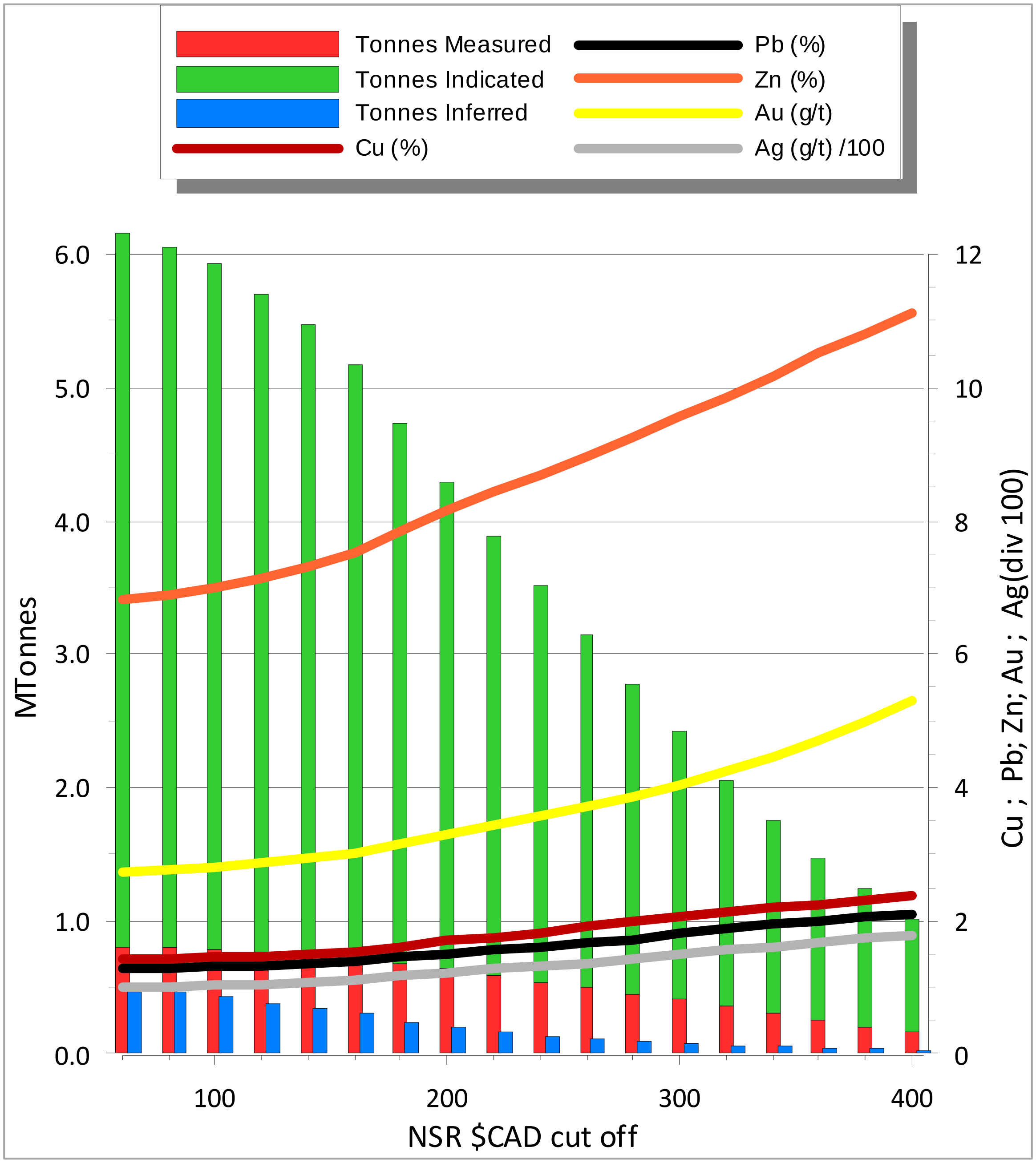
The mineral resources of the Tulsequah Chief project are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the Indicated mineral resource and grade estimates are presented in Table 14.11 at different cut-off grades. The reader is cautioned that the figures presented 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 measured + indicated mineral resource.

Table 14.11: Tulsequah Chief Measured + Indicated class Block Model Quantities & Grade Estimates at Various Cut-off Grades

Cut-off (CAD$)	Tonnage	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn EQ%
>60	6,157,112	1.42	1.27	6.82	2.72	99.4	27.7
>80	6,054,303	1.43	1.29	6.90	2.76	100.8	28.0
>100	5,922,947	1.45	1.31	7.00	2.80	102.5	28.5
>120	5,709,710	1.48	1.33	7.15	2.87	105.1	29.1
>140	5,469,780	1.51	1.37	7.33	2.94	108.0	29.8
>160	5,177,744	1.55	1.40	7.53	3.03	111.3	30.7
>180	4,729,453	1.62	1.46	7.84	3.16	116.2	32.0
>200	4,297,280	1.69	1.51	8.15	3.29	121.6	33.4
>220	3,889,839	1.76	1.56	8.43	3.43	127.0	34.7
>240	3,507,990	1.83	1.61	8.69	3.57	132.3	36.0
>260	3,137,581	1.91	1.67	8.97	3.71	137.3	37.4
>280	2,780,130	1.98	1.73	9.26	3.87	142.5	38.8
>300	2,416,164	2.05	1.80	9.58	4.05	148.2	40.3
>320	2,058,435	2.13	1.89	9.87	4.26	155.2	42.1
>340	1,755,128	2.19	1.95	10.19	4.47	161.6	43.8
>360	1,471,127	2.25	2.01	10.54	4.72	168.0	45.6
>380	1,233,529	2.31	2.05	10.81	5.00	173.9	47.4
>400	1,017,939	2.38	2.10	11.13	5.29	179.3	49.4

Source: Chieftain 2014

Figure 14.17: Tulsequah Chief Grade Tonnage Curves for the Measured + Indicated Mineral Resources



Source: Chieftain 2014

14.4 Big Bull Mineral Resource

14.4.1 Big Bull Introduction

The mineral resource model prepared by SRK considers 313 core boreholes drilled by Cominco, Redfern and Chieftain during the period of 1940 to 2011.

14.4.2 Resource Database

The assay database for Big Bull comprises 4,767 samples, 649 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 14.12).

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 Big Bull is comprised of 10 discreet mineralized lenses: two upper lenses termed U1 and U2, six main lenses termed M1 to M6 and two lower lenses, termed L1 and L2. For coding the block model, the lenses were assigned a corresponding integer code as indicated in (Table 14.13).

Figure 14.18 to Figure 14.22 are box and whisker plots of the lenses for Cu, Pb, Zn, Au and Ag. The box plot displays 75th and 25th percentile, the median is indicated with the line across the box. The whiskers show the 95th and 5th percentile and the symbols indicate the maximum values (red circle), minimum values (blue dash), and the mean value is indicated by the green square.

Table 14.12: Big Bull Descriptive Statistics of Assay Data within the Mineralized Zones

Statistical Parameters	Cu %	Pb %	Zn %	Au g/t	Ag g/t
Valid cases	649	649	649	649	649
Mean	0.46	1.51	4.21	3.12	116.4
Std. error of mean	0.02	0.10	0.24	0.23	8.0
Variance	0.40	7.00	37.63	34.18	41916.3
Std. Deviation	0.64	2.65	6.13	5.85	204.7
Variation Coefficient	1.39	1.76	1.46	1.88	1.8
rel. V. coefficient (%)	5.46	6.90	5.73	7.37	6.9
Skew	3.17	3.44	2.70	7.51	4.3
Minimum	0.00	0.00	0.00	0.00	0.0
Maximum	6.14	19.50	39.60	89.40	2010.0
1st percentile	0.00	0.00	0.00	0.00	0.1
5th percentile	0.00	0.00	0.03	0.03	1.2
10th percentile	0.01	0.00	0.08	0.10	3.4
25th percentile	0.07	0.02	0.56	0.69	13.7
Median	0.24	0.40	1.83	1.37	46.3
75th percentile	0.60	1.75	5.20	3.36	130.3
90th percentile	1.18	4.41	11.04	6.86	301.2
95th percentile	1.60	6.26	17.40	11.05	493.0
99th percentile	3.18	14.33	31.62	24.38	955.7

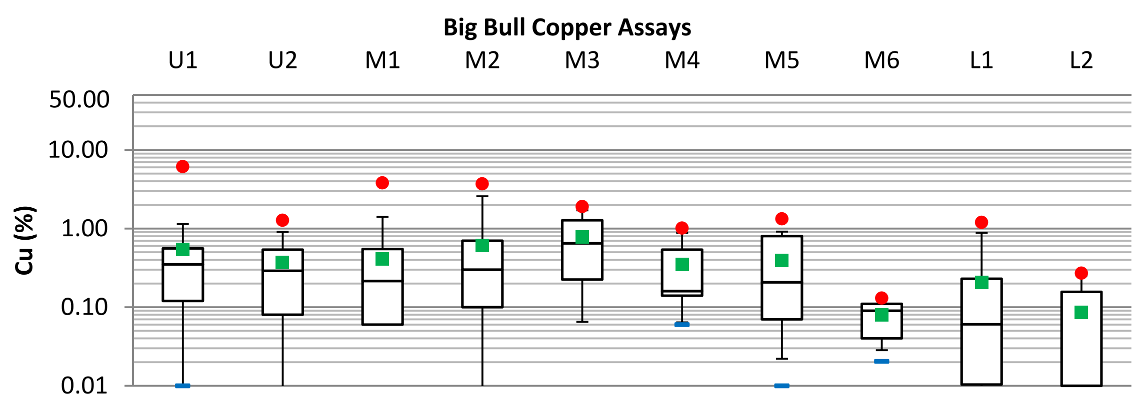
Source: Chieftain 2014

Table 14.13: Big Bull Mineralized Lenses & Corresponding Block Model Codes

Lens Name	Block Model Code
L1	11
L2	12
M1	21
M2	22
M3	23
M4	24
M5	26
M6	26
L1	31
L2	32

Source: Chieftain 2014

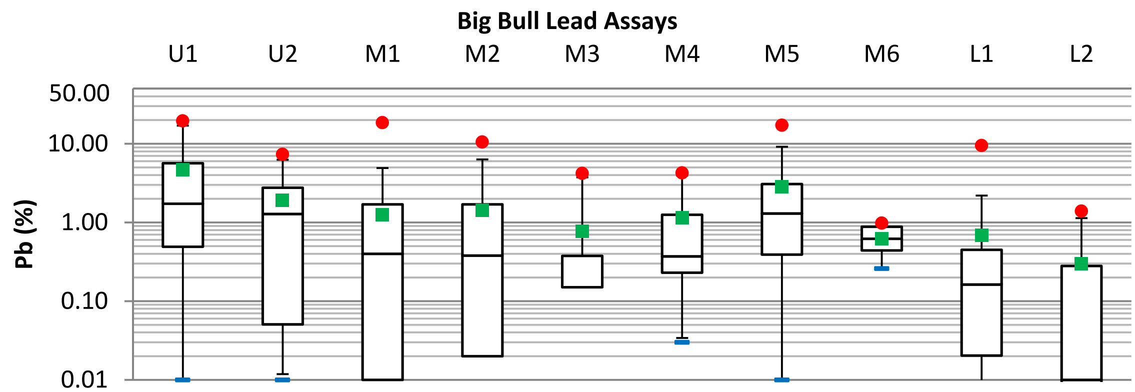
Figure 14.18: Box & Whisker Plot for Copper Assays within Modeled Lenses



Cu (%)	U1	U2	M1	M2	M3	M4	M5	M6	L1	L2
Number of Data	37	13	334	172	14	9	25	9	30	6
Mean	0.54	0.37	0.41	0.61	0.78	0.35	0.39	0.08	0.21	0.08
Maximum	6.14	1.27	3.80	3.70	1.90	1.01	1.33	0.13	1.20	0.27
Upper quartile	0.56	0.54	0.55	0.70	1.28	0.54	0.80	0.11	0.23	0.16
Median	0.35	0.29	0.22	0.30	0.65	0.16	0.21	0.09	0.06	0.01
Lower quartile	0.12	0.08	0.06	0.10	0.23	0.14	0.07	0.04	0.01	0.01
Minimum	0.01	0.00	0.00	0.00	0.00	0.06	0.01	0.02	0.00	0.00
Variance	0.99	0.12	0.27	0.64	0.40	0.10	0.14	0.00	0.10	0.01
Std. Dev.	0.99	0.34	0.52	0.80	0.63	0.31	0.38	0.04	0.31	0.11
CV	1.85	0.93	1.28	1.31	0.81	0.90	0.96	0.50	1.51	1.29

Source: Chieftain 2014

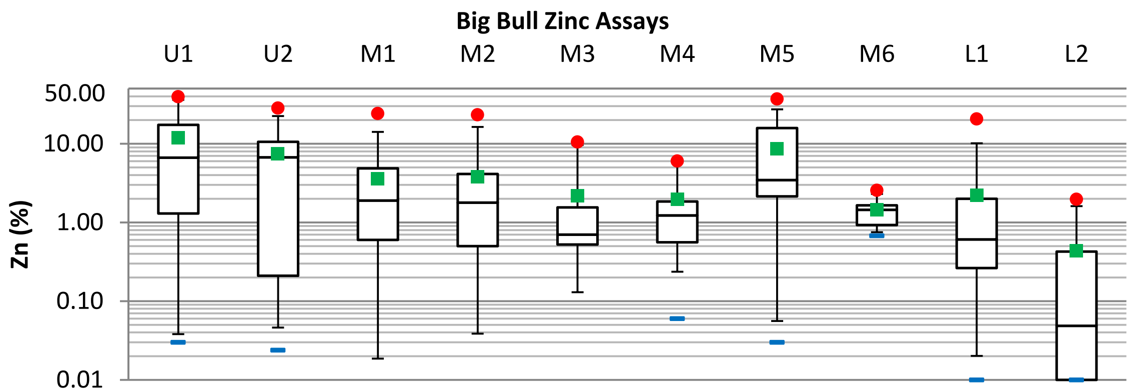
Figure 14.19: Box & Whisker Plot for Lead Assays within Modeled Lenses



Pb (%)	U1	U2	M1	M2	M3	M4	M5	M6	L1	L2
Number of Data	37	13	334	172	14	9	25	9	30	6
Mean	4.65	1.90	1.24	1.42	0.76	1.15	2.84	0.62	0.69	0.30
Maximum	19.50	7.34	18.50	10.50	4.20	4.26	17.20	0.98	9.48	1.39
Upper quartile	5.64	2.76	1.70	1.70	0.38	1.25	3.08	0.88	0.45	0.28
Median	1.73	1.28	0.40	0.38	0.15	0.37	1.30	0.62	0.16	0.01
Lower quartile	0.49	0.05	0.01	0.02	0.00	0.23	0.39	0.44	0.02	0.00
Minimum	0.01	0.01	0.00	0.00	0.00	0.03	0.01	0.26	0.00	0.00
Variance	34.80	5.07	4.04	4.95	1.79	2.24	14.87	0.07	3.01	0.26
Std. Dev.	5.90	2.25	2.01	2.22	1.34	1.50	3.86	0.26	1.73	0.51
CV	1.27	1.18	1.62	1.56	1.75	1.30	1.36	0.42	2.52	1.70

Source: Chieftain 2014

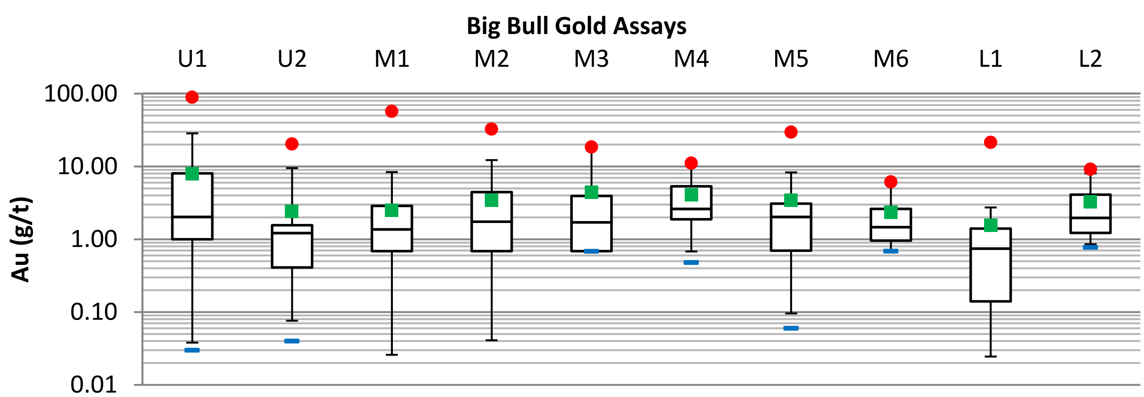
Figure 14.20: Box & Whisker Plot for Zinc Assays within Modeled Lenses



Zn (%)	U1	U2	M1	M2	M3	M4	M5	M6	L1	L2
Number of Data	37	13	334	172	14	9	25	9	30	6
Mean	11.81	7.42	3.60	3.79	2.16	1.98	8.57	1.43	2.19	0.44
Maximum	39.60	28.40	24.20	23.30	10.50	6.03	37.00	2.56	20.60	1.97
Upper quartile	17.40	10.60	4.88	4.14	1.55	1.85	15.80	1.65	2.00	0.43
Median	6.65	6.72	1.90	1.79	0.70	1.23	3.45	1.45	0.61	0.05
Lower quartile	1.30	0.21	0.60	0.50	0.53	0.56	2.14	0.93	0.26	0.01
Minimum	0.03	0.02	0.00	0.00	0.00	0.06	0.03	0.68	0.01	0.01
Variance	165.42	67.00	19.40	25.24	10.42	4.28	103.91	0.30	17.87	0.51
Std. Dev.	12.86	8.19	4.40	5.02	3.23	2.07	10.19	0.55	4.23	0.71
CV	1.09	1.10	1.22	1.32	1.50	1.04	1.19	0.38	1.93	1.62

Source: Chieftain 2014

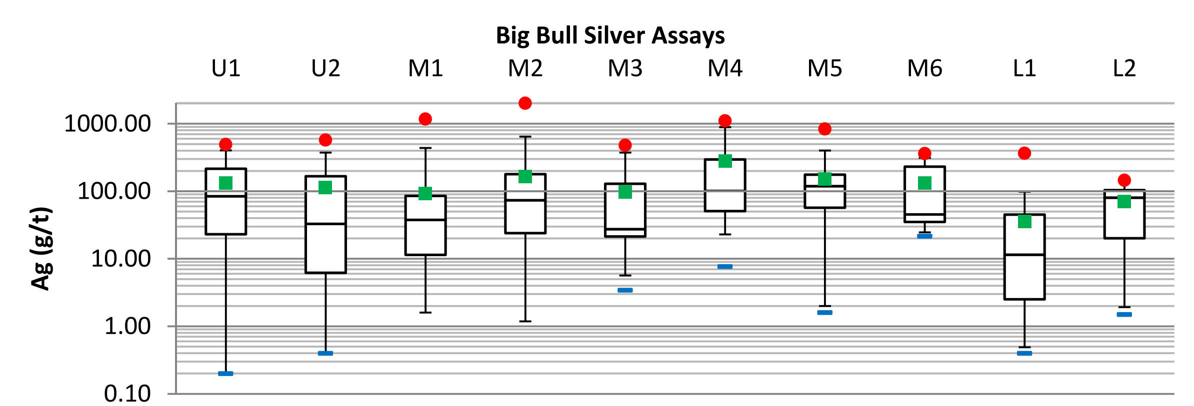
Figure 14.21: Box & Whisker Plot for Gold Assays within Modeled Lenses



Au (g/t)	U1	U2	M1	M2	M3	M4	M5	M6	L1	L2
Number of Data	37	13	334	172	14	9	25	9	30	6
Mean	7.91	2.43	2.51	3.43	4.38	4.10	3.45	2.34	1.55	3.28
Maximum	89.40	20.40	57.40	32.57	18.51	11.10	29.70	6.14	21.40	9.16
Upper quartile	8.04	1.56	2.89	4.46	3.94	5.35	3.09	2.61	1.40	4.11
Median	2.03	1.22	1.37	1.74	1.71	2.62	2.03	1.47	0.75	1.97
Lower quartile	1.00	0.41	0.69	0.69	0.69	1.88	0.70	0.96	0.14	1.22
Minimum	0.03	0.04	0.00	0.00	0.69	0.48	0.06	0.69	0.00	0.78
Variance	246.81	27.38	18.49	20.21	33.08	13.18	33.54	3.73	14.26	8.57
Std. Dev.	15.71	5.23	4.30	4.50	5.75	3.63	5.79	1.93	3.78	2.93
CV	1.98	2.15	1.72	1.31	1.31	0.89	1.68	0.83	2.43	0.89

Source: Chieftain 2014

Figure 14.22: Box & Whisker Plot for Silver Assays within Modeled Lenses



Ag (g/t)	U1	U2	M1	M2	M3	M4	M5	M6	L1	L2
Number of Data	37	13	334	172	14	9	25	9	30	6
Mean	132.66	112.21	91.39	164.95	96.49	277.26	151.84	131.12	35.60	69.85
Maximum	492.00	573.00	1168.12	2010.00	480.00	1100.00	833.00	361.03	365.00	145.00
Upper quartile	216.00	167.00	85.38	178.74	129.43	296.00	176.00	231.43	45.00	103.70
Median	84.60	32.91	37.71	73.72	27.43	101.00	119.00	45.26	11.48	80.60
Lower quartile	23.10	6.20	11.42	24.00	21.43	50.80	57.00	35.20	2.51	20.15
Minimum	0.20	0.40	0.00	0.00	3.43	7.70	1.60	21.70	0.40	1.50
Variance	18077.9	24142.1	26140.6	81820.9	18228.5	113901	30164.1	13813.5	4548.8	2774.6
Std. Dev.	134.45	155.38	161.68	286.04	135.01	337.49	173.68	117.53	67.44	52.67
CV	1.01	1.38	1.77	1.73	1.40	1.22	1.14	0.90	1.89	0.75

Source: Chieftain 2014

Zone U1 appears to demonstrate a higher lead, zinc and gold concentration and zones L1 and L2 appear to have lower base metal content but higher gold concentrations. However, SRK cautions that the observations may be skewed by the small sample population in zone L2.

14.4.3 Grade Correlation

Pb-Zn, Zn-Cu, Pb-Ag, Au-Ag, Au-Cu, and Cu-Ag relationships were evaluated; results are summarized in Table 14.14. Usually in volcanogenic massive sulphide deposits, Pb-Zn, Pb-Ag, and Au-Ag display good positive correlations. At Big Bull, Pb and Zn representing Sphalerite and Galena co-occurring correlate very well together. 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 Big Bull show weak correlations of Sphalerite and Chalcopyrite. This probably reflects the typical zoning patterns often associated with VMS deposits.

Table 14.14: Big Bull Assay Correlation Coefficient Matrix for Big Bull Deposit

Metal	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Cu %	1	0.313	0.318	0.448	0.493
Pb %	0.313	1	0.859	0.438	0.540
Zn %	0.318	0.859	1	0.394	0.506
Au g/t	0.448	0.438	0.394	1	0.442
Ag g/t	0.493	0.540	0.506	0.442	1

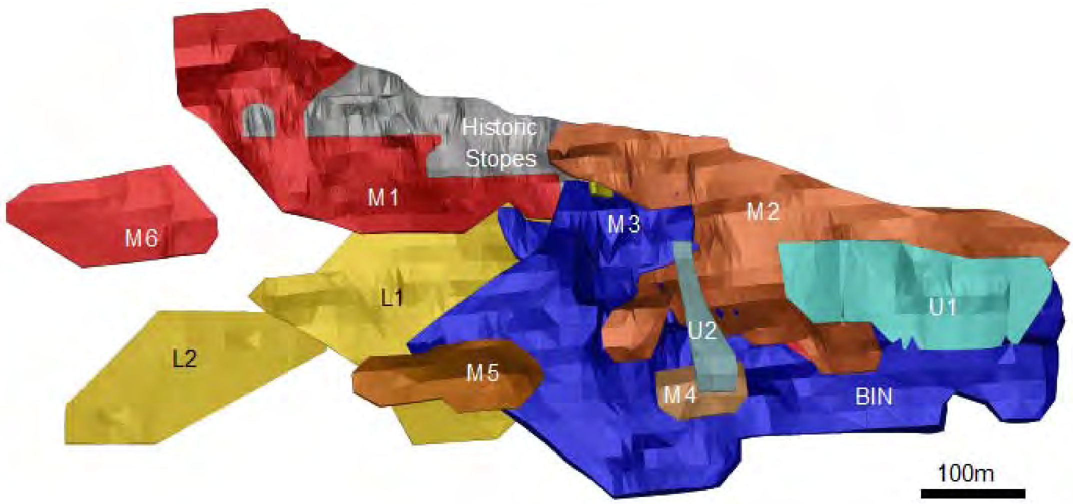
Source: Chieftain 2014

14.4.4 Solid Body Modeling

The Big Bull deposit comprises ten distinct semi-massive to massive sulphide lenses termed U1, U2, M1 to M6 and L1, L2. The U "upper" lens series are stratigraphically higher with the U1 lens representing the previously named 60-62 high grade zone. The M1 lens represent the Big Bull Main mineralization trend, the M2 lens is slightly above the M1 lens by 10-20m and the M3 lens a discrete lens between the M1 and M2 lenses in the upper area of the old Big Bull Mine. The M2 and M4 and are basal extensions of the M1 lens that have been displaced by the barren basalt intrusive, which also disrupts the bottom edge of the M1 and M2 lenses. The M6 lens is lateral extension of the M1 lenses at the north-western extent with deceasing grade. The L1 and L2 lenses represent a stratigraphically lower alteration zone with elevated precious metal concentrations and low grade disseminated base metals. The lenses all strike between 130° and 150° dipping 60°-70° to the south west, except the U1 lens that dips 90°.

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

Figure 14.23: Big Bull 3D Perspective View Looking Northeast of the Mineralized Lenses



Note: The Three Big Bull stratigraphic zones the Upper U1, U2; Main M1-M6 and Lower L1, L2 lenses, the basalt Intrusive (BIN) and historic mined out stopes are also shown. Approximate 100m scale bar is shown.

Source: Chieftain 2014

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 M1 and M2 lenses.

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.

14.4.5 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 one 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 0.98m. For the purpose of resource estimation, all assay intervals within the mineralized units 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.

Thirty seven composites 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 with the composite assay interval length calculation extended to 1 m and were used during estimation. The statistical properties of the composited metal data by lens are summarized in Table 14.15.

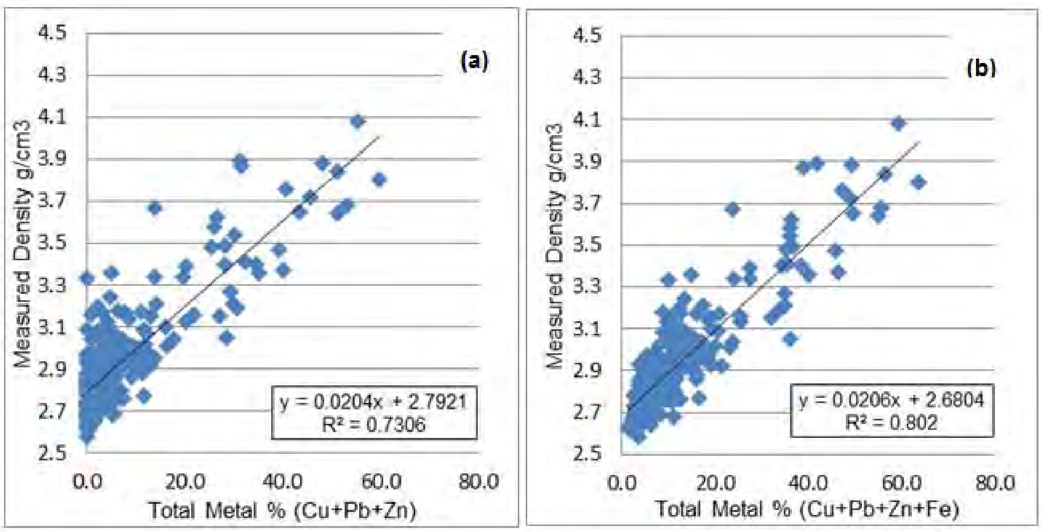
Table 14.15: Big Bull Statistical Data for 2 m Capped Composites by Lens

Element	Lens	Mean	Q25	Q50	Q75	Max	No. Of Comps
Cu	U1	0.32	0.04	0.22	0.43	1.82	22
	U2	0.34	0.26	0.29	0.48	0.51	5
	M1	0.25	0	0.11	0.35	2.57	255
	M2	0.42	0.1	0.24	0.5	2.76	95
	M3	0.51	0	0.21	0.99	1.6	17
	M4	0.3	0.14	0.23	0.48	0.5	5
	M5	0.37	0.19	0.31	0.51	0.89	12
	M6	0.07	0.03	0.08	0.1	0.13	5
	L1	0.14	0	0.04	0.16	1.2	28
	L2	0.07	0.01	0.01	0.08	0.27	6
Pb	U1	3.06	0.2	1.33	5.52	10	22
	U2	1.76	1.05	1.57	2.34	3.07	5
	M1	0.72	0	0.27	0.95	5.66	255
	M2	0.93	0.05	0.51	1.27	7.02	95
	M3	0.61	0	0	0.23	4.2	17
	M4	1.11	0.24	1.01	1.1	3	5
	M5	2.34	0.74	1.51	2.84	6.83	12
	M6	0.52	0.49	0.53	0.65	0.94	5
	L1	0.33	0.01	0.05	0.29	2.66	28
	L2	0.19	0	0.01	0.28	0.72	6
Zn	U1	9.14	0.56	4.3	17.07	30	22
	U2	6.58	5.15	6.32	6.87	10.6	5
	M1	2.22	0.05	1.19	3.46	16.35	255
	M2	2.48	0.5	1.6	2.83	18.29	95
	M3	1.64	0	0.53	0.96	10.5	17
	M4	1.91	1.34	1.59	1.65	4.69	5
	M5	7.61	2.57	4.26	12.06	22.95	12
	M6	1.19	1.08	1.13	1.77	1.96	5
	L1	1.14	0.04	0.44	1.52	9.71	28
	L2	0.29	0.01	0.05	0.43	1.07	6
Au	U1	4.99	0.41	1.01	8.73	19.55	22
	U2	1.73	0.76	1.54	1.67	4.11	5
	M1	1.52	0.05	0.88	2.24	16.36	255
	M2	2.46	0.71	1.42	3.31	14.21	95
	M3	3.18	0.52	1.04	3.09	17.14	17
	M4	4.07	2.37	2.81	4.05	9.83	5
	M5	2.95	1.29	2.22	3.14	12.21	12
	M6	1.94	0.76	1.36	1.98	5.58	5
	L1	0.88	0.06	0.3	1.25	6.05	28
	L2	2.91	1.22	1.9	2.51	9.16	6
Ag	U1	110.56	14.47	35.06	234.33	404	22
	U2	100.7	54.55	109.33	133.94	183	5
	M1	49.23	2.05	21.27	53.54	596.01	255
	M2	95.43	18.51	54.6	132.36	600	95
	M3	62.68	3.43	18.72	45.32	400.18	17
	M4	224.37	160.5	177.11	226.69	523.8	5
	M5	149.12	84.63	122	218.5	296	12
	M6	116.57	28.66	52.26	205.24	296.69	5
	L1	20.66	0.95	5.9	23.6	102.45	28
	L2	60.05	21.63	78.6	87.73	108.2	6

14.4.6 Bulk Density

Specific gravity ("SG") was determined for 2,600 samples in the database, 240 of which fall within the modeled zones. A plot of total metal content (Zn + Cu + Pb grades, Figure 14.24a; or Zn + Cu + Pb + Fe grades, Figure 14.24b) against SG indicates that SG is directly proportional to total metal content. The specific gravity was modeled for the 417 historical assays within the modeled zones without SG determinations using the formula for the slope of the trend line fitted to curve of SG against total metal content greater than 2% for Zn + Cu + Pb, or greater than 0% for Zn + Cu + Pb + Fe. The mean density of all 2,600 SG results for samples with 0% total metal content was 2.72 g/cm^3.

Figure 14.24: Big Bull Measured Specific Gravity vs. Total Metal Content



14.4.7 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 14.16.

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 7%, with the majority of the influence from the gold and silver capped values.

Table 14.16: Big Bull Capping Levels

Metal	Cap level	No. capped	CoV uncapped	CoV Capped	Metal loss (%)
Cu	10%	0	1.39	1.39	0.13
Pb	10%	11	1.76	1.55	6.66
Zn	30%	7	1.46	1.4	2.38
Au	25 g/t	7	1.88	1.43	6.92
Ag	600 g/t	21	1.76	1.39	10.86

Source: Chieftain 2014

14.4.8 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 modeled lenses; southeast trending with a moderately steep dip or plunge to the southwest. Ranges were 40 m along strike, 40 m down dip or plunge, and 10 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.

14.4.9 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 U1 and other lenses. Table 14.17 summarizes the search parameters used to interpolate the block model.

Table 14.17: Big Bull 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.	
U1	ID2	1	Ellipse	0	-90	0	40	60	20	3	8	2
U2, M1-6, L1-2	ID2	1	Ellipse	0	-60	-20	40	60	20	3	8	2
U1	ID2	2	Ellipse	0	-90	0	100	120	40	3	8	2
U2, M1-6, L1-2	ID2	2	Ellipse	0	-60	-20	100	120	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 also required that at least two drill holes and three composites be available within the search ellipse to estimate a grade within a block with a maximum of eight composites were used to interpolate a grade value.

Bulk density values were estimated into the resource model by inverse distance weighting to the second power, from the measured SG values or the determined SG values calculated from the SG vs. total metal linear relationship. Search parameters used were the same as those used for grade interpolation for pass 1 and pass 2.

14.4.10 Model Validation 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 14.24 to Figure 14.28 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 appear to be lower than the composite grades, in particular for Pb, Zn and Au at the 584,450 easting, which is the location of the high grade U1 lens.

Figure 14.25: Big Bull Swath Plots of Copper Composites & Copper Block Grades

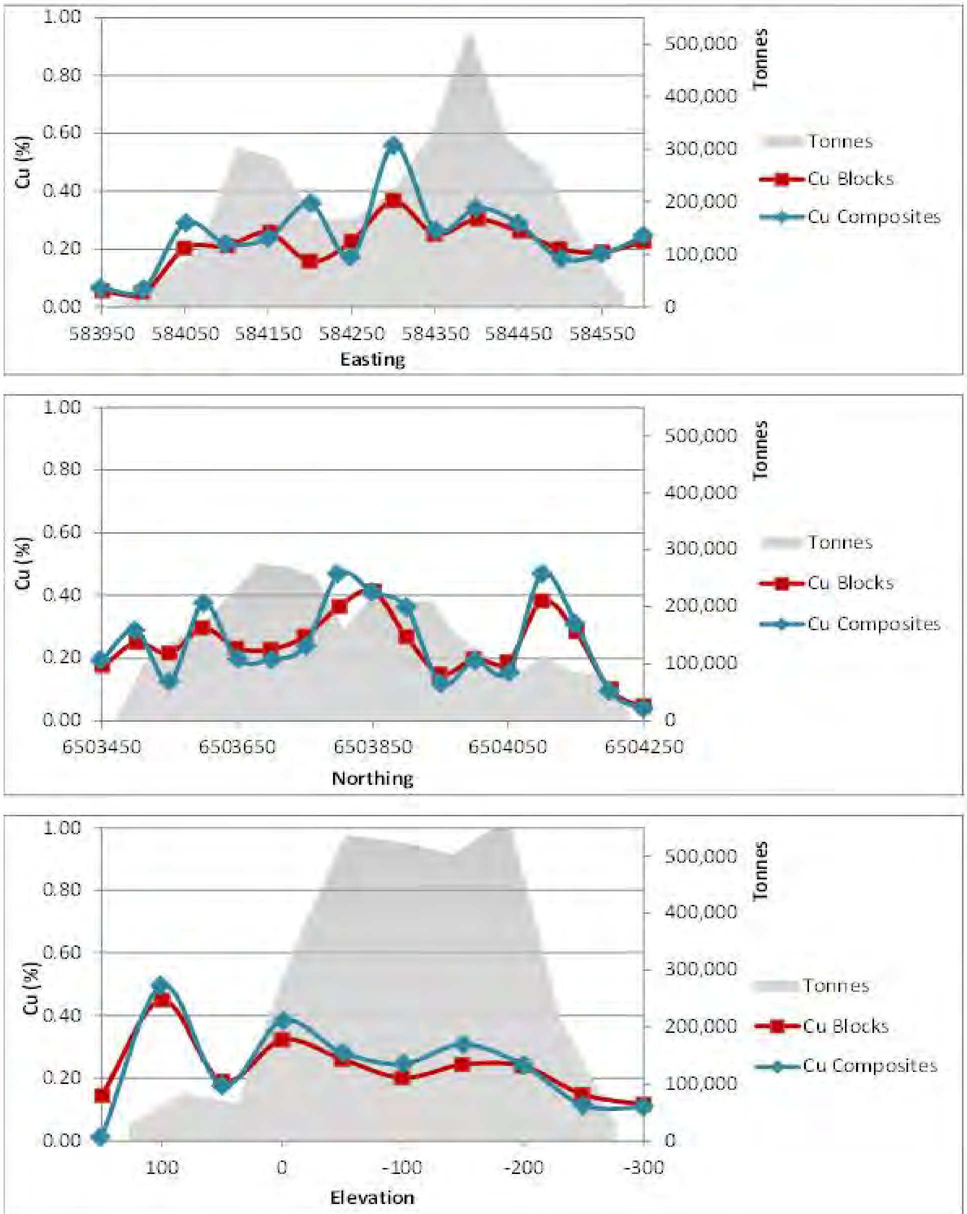


Figure 14.26: Big Bull Swath Plots of Lead Composites & Lead Block Grades

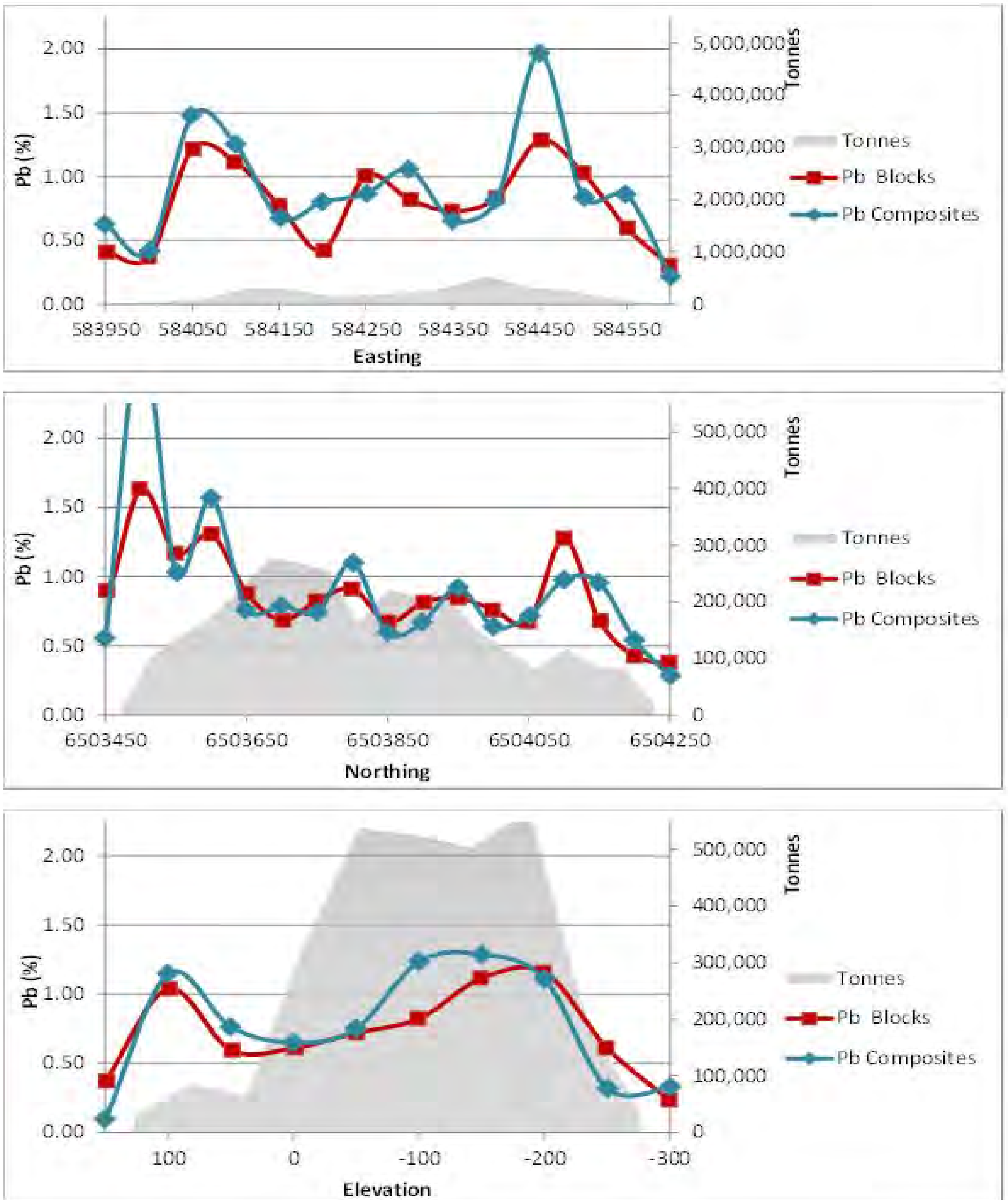


Figure 14.27: Big Bull Swath Plots of Zinc Composites & Zinc Block Grades

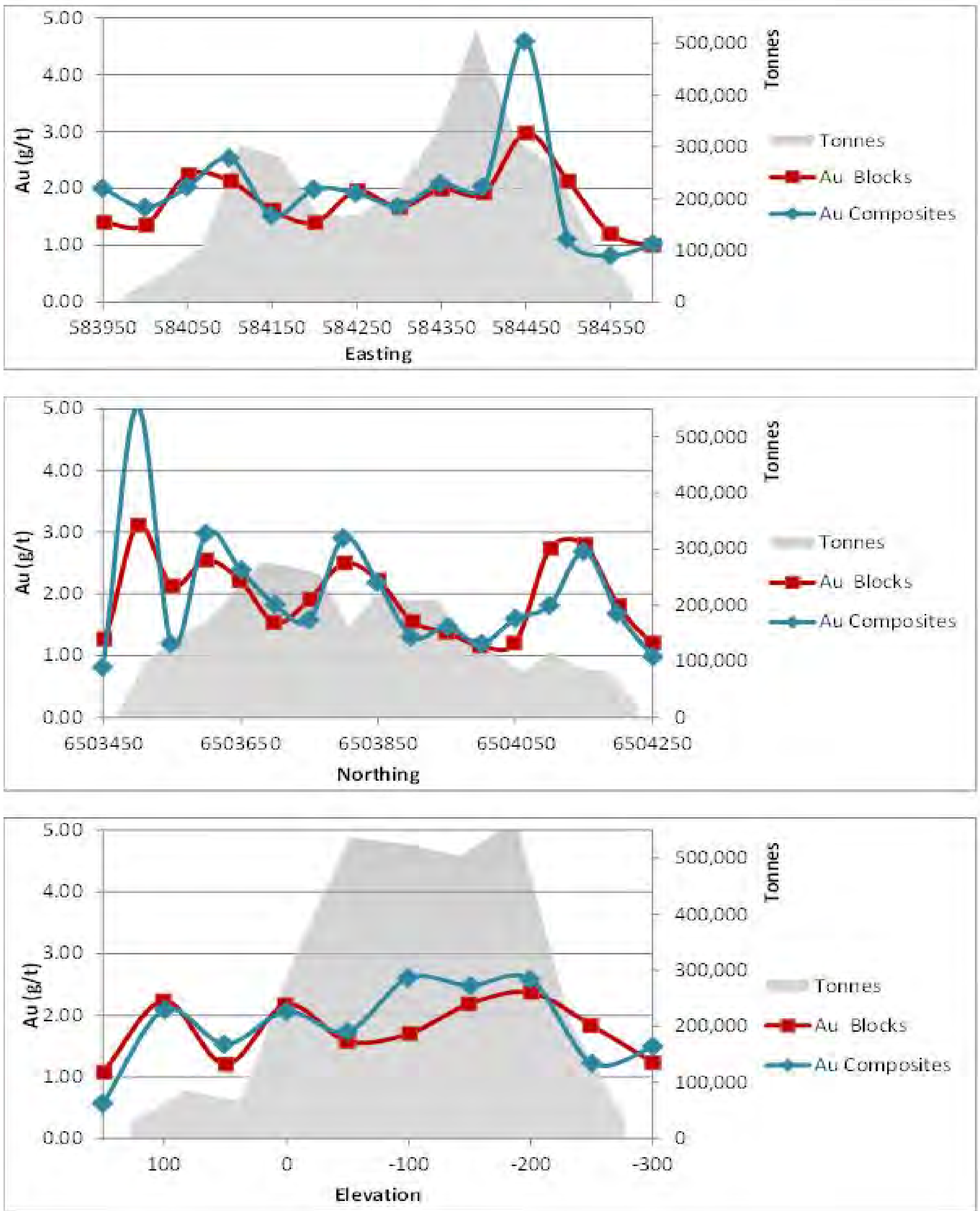


Figure 14.28: Big Bull Swath Plots of Gold Composites & Gold Block Grades

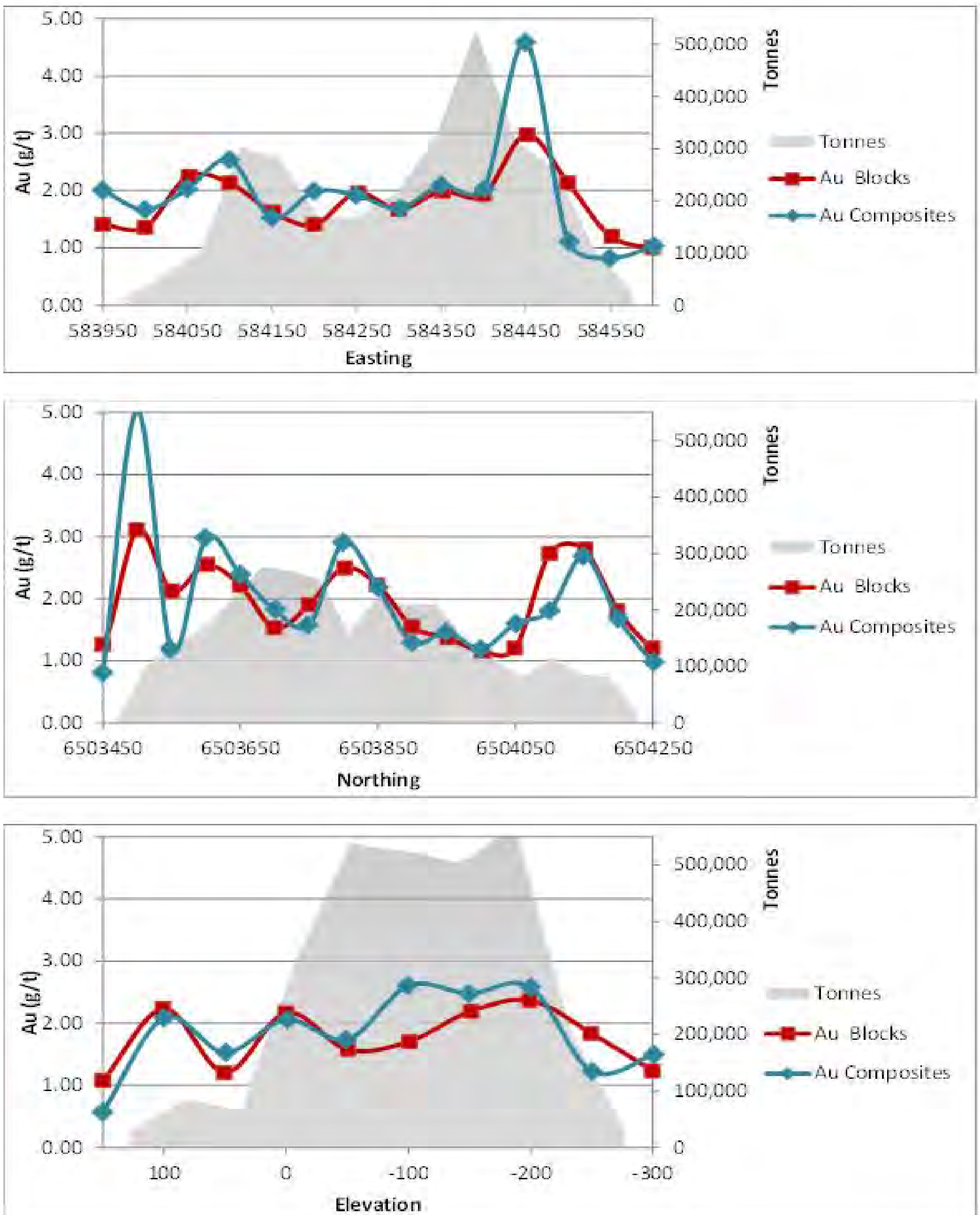
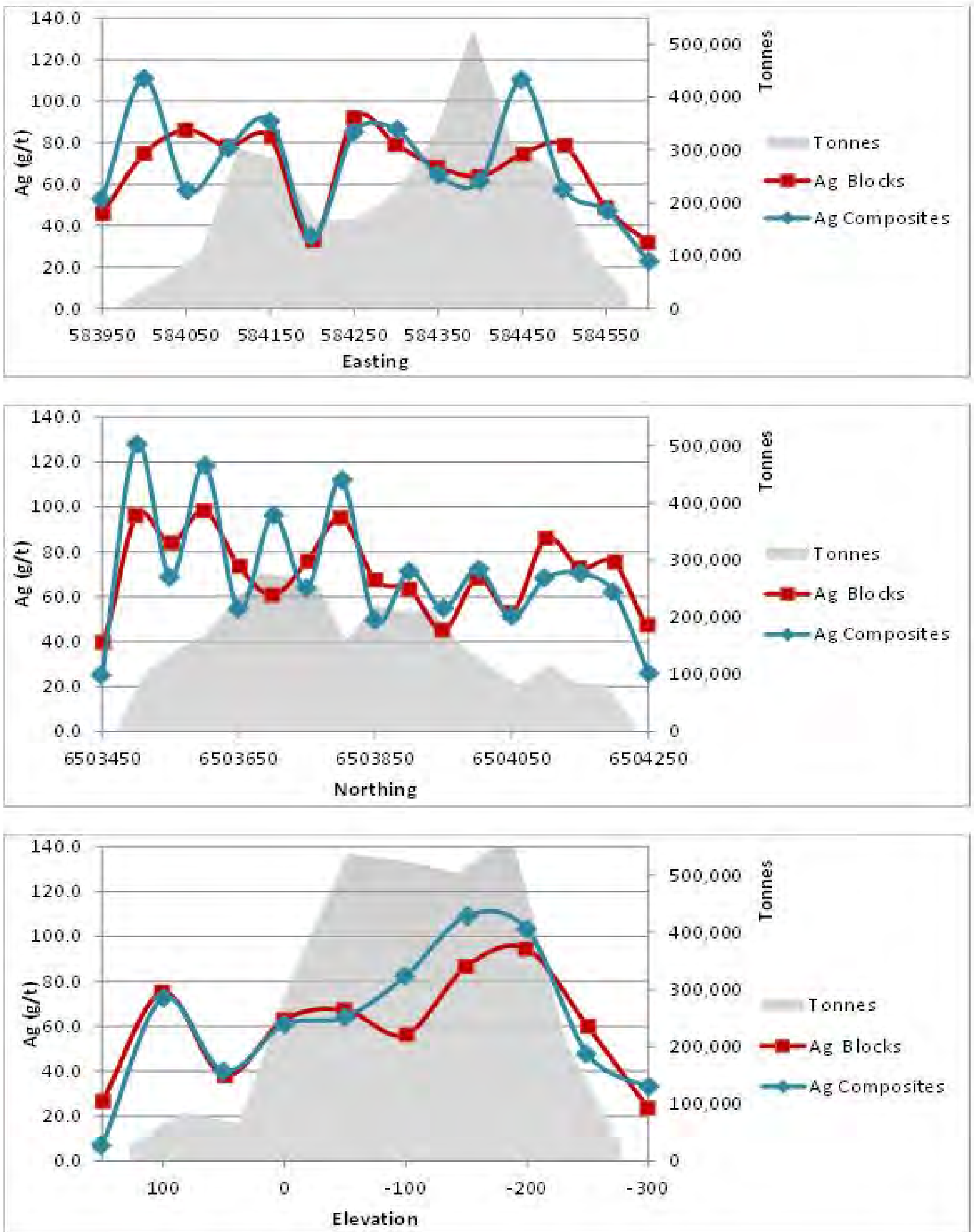


Figure 14.29: Big Bull Swath Plots of Silver Composites & Silver Block Grades



14.4.11 Model Validation Sensitivity Check for Bias

The model was checked for global bias by comparing the ID2 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 ID2 model when compared at a 0.0 cut-off and when compared at the US$100 Eq cut-off the two models returned similar overall values (Table 14.18). The NN grades are higher than for the ID2 estimate and the tonnes are lower which yields a similar overall total metal content, about 4% more metal is reported in the NN model than in the ID2 model at the US$100 Eq cut-off.

Table 14.18: Big Bull Percent Difference between NN & ID2 Results at US$100Eq Cut-off

Class	(NN-ID2)/NN in % tonnes	Metal Grade Differences in % (NN-ID2)/NN				
		Cu	Pb	Zn	Au	Ag
Indicated	-12.49	19.60	21.02	17.41	21.10	20.40
Inferred	-20.46	16.88	18.89	18.98	19.12	16.78
Ind+Inf	-17.87	17.64	19.79	18.47	19.98	18.37

14.4.12 Mineral Resource Classification

Block model quantities and grade estimates for the Big Bull project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Dr. Gilles Arseneau, P. Geo. (APEGBC, # 23474), an appropriate independent qualified person for the purpose of NI 43-101.

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 30 to 40 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 (May 2014). For those blocks, SRK considers that the level of geologic 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. Subsequent to this classification the lenses above historic workings with no post 1956 drilling were re-classified as inferred, further work is required to confirm that the 1956 Cominco stope plans are accurate and these M1, M2 and M3 lens areas remain before they can be included as indicated resources available for mine planning.

Blocks estimated during the second pass are best 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.

14.4.13 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

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

The “reasonable prospects 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 Big Bull deposit are amenable for underground mining.

The block model tonnes and grade estimates were reviewed to determine the portions of the Big Bull deposit having “reasonable prospects for economic extraction” from an underground mine, based on parameters summarized in Tables 14.19 and 14.20.

The classification of the Big Bull deposit as a mineral resource is contingent on the Tulsequah Chief deposit being developed.

The mineral resources do not have a reasonable prospect of economic extraction because of the small size of the deposit and remote location. The deposit, however, does have a reasonable prospect of economic extraction as a satellite deposit to the Tulsequah Chief deposit.

Table 14.19: Net Smelter Return Calculation Data

Metal	Price (US$)	Metallurgical Recovery (%)	NSR Payable Metal (%)	$NSR CAD Multiplication Factor
Au	1,250/oz	90	80.7	36.69
Ag	19.00/oz	84.5	72.5	0.5013
Cu	2.75/lb	89	52.8	36.24
Pb	0.90/lb	66.2	41.8	9.39
Zn	0.90/lb	89	45.4	10.2
FX: USD$:CAD$ = 1.00:X	0.93			

*Payable metal (%) including metal prices, metallurgical recoveries, transport, and smelter charges.

Source: Chieftain 2014

Table 14.20: Conceptual Assumptions Considered for Underground Resource Reporting

Parameter	Value	Unit
Exchange rate	0:93	USD$/$CAD
Mining costs	$32.00	CAD$/t mined
Process cost	$18.00	CAD$/t of feed
General and Administrative	$10.00	CAD$/t of feed
Power	$20.00	CAD$/t of feed
Transport	$20.00	CAD$/t of feed
Total Costs	$100.00	CAD$/t of feed
Assumed mining rate	3,000	tpd

Source: Chieftain 2014

SRK considers that the blocks that had total dollar values above $100.00 satisfied the “reasonable prospects for economic extraction” and could be reported as a mineral resource as summarized in Table 14.21.

Table 14.21: Mineral Resource Statement*, Big Bull Deposit, Tulsequah Chief Project, British Columbia, SRK Consulting (Canada) Inc. October 20, 2014

Category	M Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (gpt)	Ag (gpt)	Zn (EQ%)
Indicated	0.653	0.34	1.54	4.11	3.03	125.0	23.8
Inferred	1.453	0.37	1.37	4.15	2.67	103.9	21.4

*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 NSR cut-off grade of CAD$100. Cut-off grades are based on a Net Smelter Return of payable metals including metal prices, metallurgical recoveries, transport, and smelter charges; Metal prices: US$1,250/oz of gold, US$19/oz for silver, US$0.90/lb for zinc and lead and US$2.75 for copper, USD:CAD 1:0.93; metallurgical recoveries of 90.0% for gold, 84.5 for silver, 89.0 for copper, 66.2% for lead and 89.0% for zinc; and payable metal: 84.9% gold, 76.3% silver, 55.6% copper, 44.0% lead and 47.8% zinc. Zn EQ% = ((Au g/t*36.69x) + (Ag g/t*0.5013) + (Cu %*36.24) + (Pb %*9.39) + (Zn %*10.2))/10.2

Source: Chieftain 2014

14.4.14 Grade Sensitivity Analysis

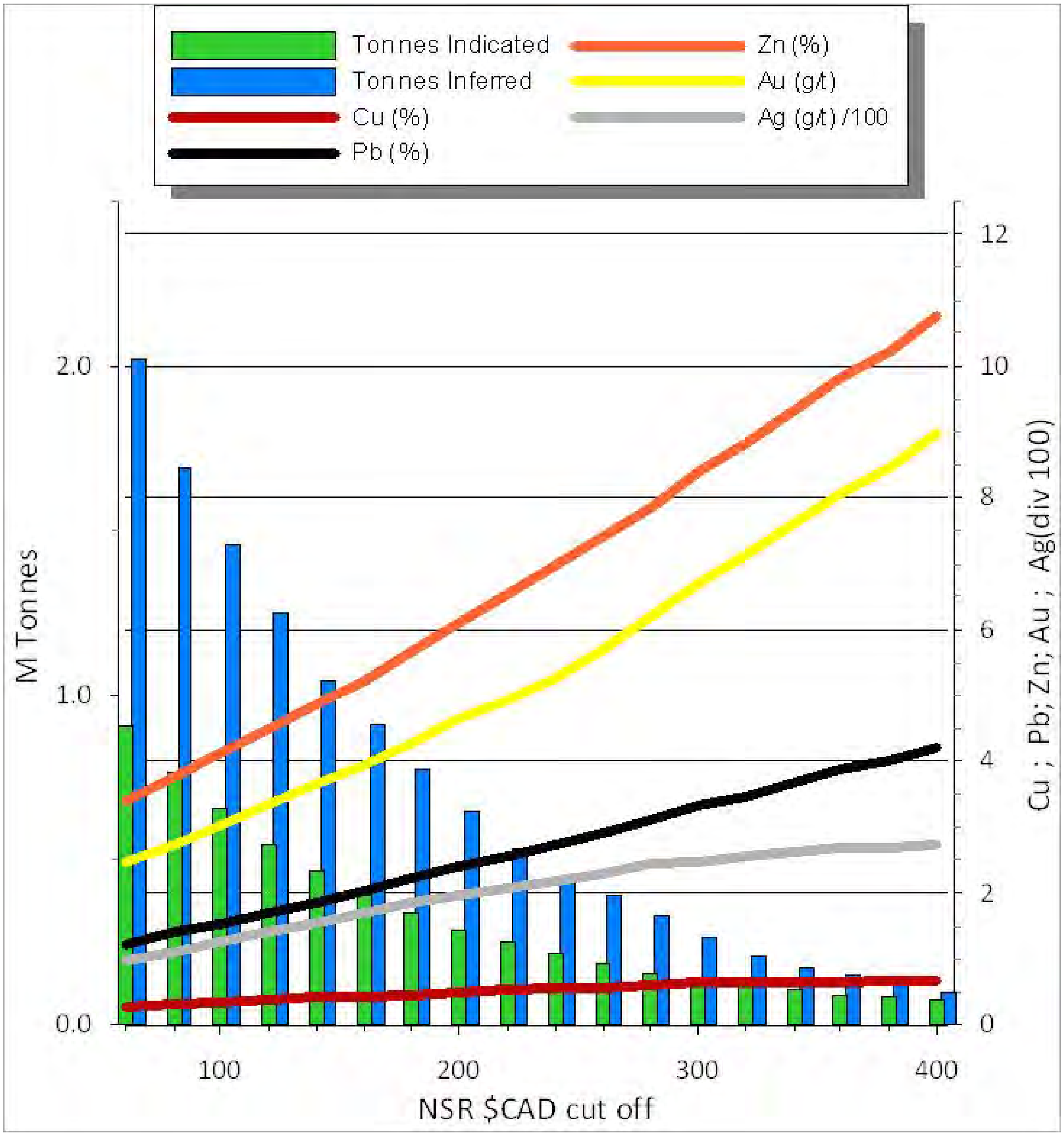
The mineral resources of the Big Bull deposit are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the Indicated mineral resource and grade estimates are presented in Table 14.22 at different cut-off grades. The reader is cautioned that the figures presented 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 14.30 presents this sensitivity as grade tonnage curves for the indicated mineral resource.

Table 14.22: Big Bull Indicated class Block Model Quantities & Grade Estimates at Various Cut-off Grades

Cut-off (CAD$)	Tonnage	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn (EQ%)
>60	908,242	0.29	1.23	3.38	2.47	98.3	19.3
>80	765,462	0.32	1.4	3.78	2.75	111.6	21.6
>100	**652,864**	**0.34**	**1.54**	**4.11**	**3.03**	**125**	**23.8**
>120	543,777	0.37	1.7	4.51	3.34	141.2	26.4
>140	465,214	0.4	1.85	4.86	3.63	155.1	28.7
>160	394,872	0.42	2.03	5.22	3.94	170.8	31.2
>180	337,513	0.45	2.2	5.65	4.26	183.8	33.6
>200	286,499	0.49	2.39	6.12	4.63	195.9	36.3
>220	249,654	0.52	2.55	6.53	4.94	206.9	38.6
>240	216,223	0.55	2.72	6.98	5.27	218.7	41.1
>260	184,134	0.58	2.92	7.45	5.7	230.6	44
>280	157,544	0.6	3.1	7.84	6.19	242.6	47
>300	135,521	0.62	3.31	8.39	6.71	249.3	50
>320	119,900	0.64	3.48	8.84	7.14	256.2	52.6
>340	106,031	0.65	3.69	9.33	7.6	262.3	55.2
>360	94,294	0.65	3.88	9.81	8.06	267.8	57.9
>380	86,034	0.66	4.02	10.21	8.45	270.1	59.9
>400	76,341	0.67	4.22	10.77	8.95	274.6	62.7

Source: Chieftain 2014

Figure 14.30: Big Bull Grade Tonnage Curves for the Indicated Mineral Resources



Source: Chieftain 2014

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 NSR Cut-off Criteria

Mining reserve values were calculated from block model tonnes and grades to define a net smelter return (NSR) cut-off 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 to 15.5.

Table 15.1: NSR Calculation Metal Prices

Commodity	Unit	Price (US$)
Copper Price	US$/lb	2.75
Lead Price	US$/lb	0.90
Zinc Price	US$/lb	0.90
Gold Price	US$/oz	1,250
Silver Price	US$/oz	19.00
Exchange Rate	US$:C$	0.93

Source: JDS 2014

Table 15.2: NSR Copper Concentrate Smelter Terms

NSR Assumptions	Unit	Cu Concentrate
Recoveries		
Cu	%	89.0
Au	%	47.0
Ag	%	77.6
Concentrate Grade	**%**	**21.0**
Moisture Content	%	8.0
Smelter Payables		
Cu Payable	%	96.50
Au Payable	%	95.00
Ag Payable	%	90.00
Minimum Deduction in Concentrate	%	1.0
Au Minimum Deduction	g/t	0.0
Ag Minimum Deduction	g/t	30.0
TC/RCs		
Treatment Charge	US$/dmt concentrate	150.00
Refining Charge		
Cu	US $/lb	0.15
Au	US $/oz	6.00
Ag	US $/oz	0.50
Deleterious Element Penalties		
As	US $/dmt concentrate	41.20
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

Source: JDS 2014

Table 15.3: NSR Lead Concentrate Smelter Terms

NSR Parameters	Unit	Pb Concentrate
Recoveries		
Pb	%	65.0
Au	%	2.8
Ag	%	6.3
Concentrate Grade	**%**	**60.0**
Moisture Content	%	8.0
Smelter Payables		
Pb Payable	%	95.0
Au Payable	%	95.0
Ag Payable	%	95.0
Minimum Deduction in Conc	%	3.0
Au Minimum Deduction	g/t	1.5
Ag Minimum Deduction	g/t	50.0
TC/RCs		
Treatment Charge	$/dmt concentrate	100.00
Refining Charge		
Au	US $/oz	25.00
Ag	US $/oz	1.50
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	**129.46**

Source: JDS 2014

Table 15.4: NSR Zinc Concentrate Smelter Terms

NSR Parameters	Unit	Zn Concentrate
Recoveries		
Zn	%	90.0
Au	%	0.0
Ag	%	0.0
Concentrate Grade	**%**	**60.0**
Moisture Content	%	8.0
Smelter Payables		
Zn Payable	%	85.0
Minimum Deduction in Conc	%	8.0
Au Minimum Deduction	g/t	1.0
Ag Minimum Deduction	g/t	93.0
TC/RCs		
Treatment Charge	$/dmt concentrate	165
Refining Charge		
Au	US $/oz	0.00
Ag	US $/oz	0.00
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

Source: JDS 2014

Table 15.5: NSR Gravity Concentrate Smelter Terms

NSR Assumptions	Unit	Gravity Concentrate
Recoveries		
Au	%	41.0
Ag	%	0.5
Smelter Payables		
Au Payable	%	99.9
Ag Payable	%	99.0
Refining Charge		
Au	US $/oz	0.65
Ag	US $/oz	0.65
Shipping Cost	US$/payable oz	1.15

Source: Chieftain 2014

Mineable blocks, stopes and drifts were defined based on NSR values greater than US$200.00/t and a minimum mining width (MMW) of 3 m. Some lower value or incremental material, greater than US$125.00/t is also included in the mining reserve. The incremental material is predominately development ore that had to be taken to mine the stope in its vicinity.

15.2 Dilution

Two types of dilution were applied to the stope designs:

- External wall dilution – waste that falls into the stope from the geometry of the stope shape; and
- Fill dilution – paste fill expected to fall into the stope being mined from adjacent stopes and/or inadvertently scraped off the stope floors during mucking.

The modes of dilution were estimated by mining method and stope type, based on the stope design tonnages, and are summarized in Table 15.6.

Table 15.6: Dilution by Mining Type

Mining Type	Wall Dilution %	Fill Dilution %
Cut & Fill	0.0	1.0
Longitudinal	15.0	2.0
Transverse Primary	5.0	1.0
Transverse Secondary	12.0	4.0

Source: JDS 2014

The average width, as measured perpendicular to strike of the stopes in the mine plan is approximately 12 m. Based on this width, and the designed stope sizes the dilution percentages in Table 15.6 equate to the following dimensions:

- Longitudinal Stope: 0.7 m of dilution on each wall and approximately 0.6 m of fill dilution;
- Transverse Primary: 0.6 m of dilution on each wall and approximately 0.3 m of fill dilution; and
- Transverse Secondary: 1.35 m of dilution on each wall and approximately 0.4 m of fill dilution.

Both the quality and condition of the walls and longhole drilling deviation are considered as key to minimizing wall and adjacent stope dilution. The dilution is within the sulphide envelope and was assumed to carry the grades shown in Table 15.7. External dilution waste grades adjacent to the planned stopes were calculated from separate waste hanging wall and waste foot wall block models. The foot wall and hanging wall block models were calculated as separate domains from drill hole assay composites located inside a 2 m envelope extending beyond the mineral resource wireframes, the same estimation parameters as the mineral resource were used. The classification of indicated or inferred for the waste blocks was assigned from the classification of the adjacent mineral resource blocks, this is to ensure that the classification is consistent between the waste and mineral resource domains. The hanging wall and foot wall waste dilution grade was calculated for each individual stope from the adjacent hanging wall and foot wall blocks extending into the 2 m wireframe envelope, any inferred blocks were assigned zero waste grade. The individual stope dilution grade is then applied to the designed external dilution tonnes and combined with the mineral resource tonnes and grade for the final stope grade.

Table 15.7: Dilution Grade Values

Metal	Dilution Grade
Au	0.38 g/t
Ag	10.23 g/t
Cu	0.12%
Pb	0.10%
Zn	0.46%

Source: Chieftain 2014

Fill dilution is assumed to carry zero metal grades.

Additional sources of dilution are planned or internal dilution and inferred resource dilution. Planned dilution is comprised of material that may be below the NSR cut-off that is unavoidable in the stope design shape. Planned dilution carries the metal grades of the assigned blocks within the stope shape. Any inferred resource class tonnage within the mining reserve stope shapes have been treated as waste and have been assigned zero metal grades. Planned and inferred dilution comprises approximately 2.7% and 0.7% of the total reserve respectively.

The total external fill, planned and inferred dilution is approximately 17.6% of the total mining reserve.

15.3 Mining Ore Recovery

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

The longhole mining method is largely dependent on 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. An ore recovery of 100% of the material blasted 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 15.8.

Table 15.8: Mineral Reserve Estimate

Category	Tonnes	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Zn Eq (%)
Proven	683,963	1.48	1.36	7.84	2.71	100.59	29.4
Probable	3,751,657	1.45	1.28	6.78	2.88	104.39	28.9
Total P + P	**4,435,619**	**1.46**	**1.29**	**6.95**	**2.85**	**103.72**	**29.0**

1. Underground mineral reserves are reported at a NSR cut-off of US$200/tonne.
2. Cut-off grades are based on a price of US$1,250/oz of gold, US$19/oz for silver, US$0.90/lb for zinc and lead and US$2.75 for copper and recoveries of 90% for gold, 84.5% for silver, 87.8% for copper, 65.1% for lead and 89.3% for zinc.
3. Reserve: Zn EQ% = ((Au g/t*36.64x)+ (Ag g/t*0.4991)+ (Cu %*36.73)+ (Pb %*8.81)+ (Zn %*10.04))/10.04

Source: JDS 2014

The mineral reserves identified in Table 15.8 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

The economic viability of the project is presented in Sections 21 and 22. The proven and 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 October 20, 2014, as detailed in Section 14 of this report. The mine design and plan considers measured and Indicated mineral resources of the Tulsequah Chief deposit only. Inferred resources have been excluded from mine planning for this study. Where inferred resources fall within the stope designs they have been assigned a zero waste grade. 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 12 months, with some development ore mined and processed in Q4 2016 for processing during commissioning in Q1 2017 and ramping to production full production in 2018;
- Full mine production is achieved in Q1 2018;
- Underground mining and maintenance carried out by Owner;
- Contract Alimak raise mining will be utilized;
- Conventional, trackless diesel-electric mining equipment will be utilized; and
- Mined voids will be filled with paste fill 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	Four weeks in/Two week out	4x2
Nominal Ore Mining Rate	tpd	1,100
Annual Ore Mining Rate	tpa	~408,700
Ore Density	t/m^3	variable, from block model. 3.55 average
Waste Density	t/m^3	2.70
Swell Factor		1.35

Cut-off NSR value, dilution and mining 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); and
- TetraTech (2012).

This data has 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).

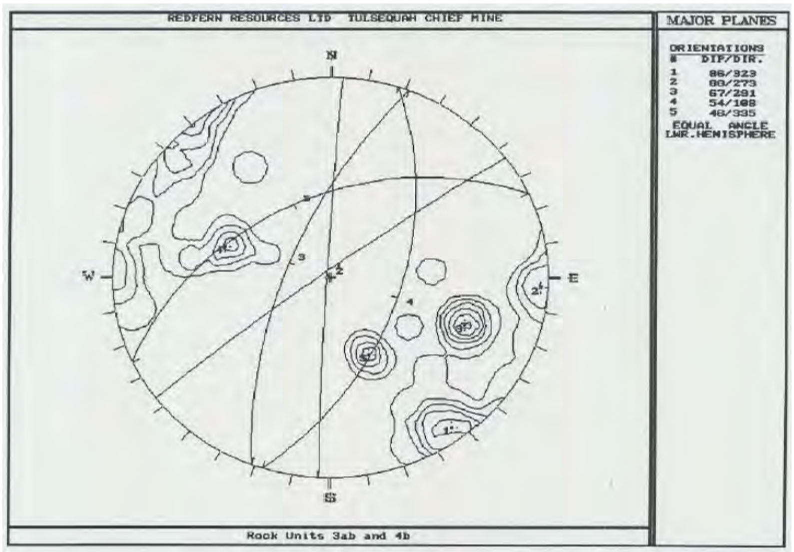
1. Hangingwall Volcanics (Figure 16.1):

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

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

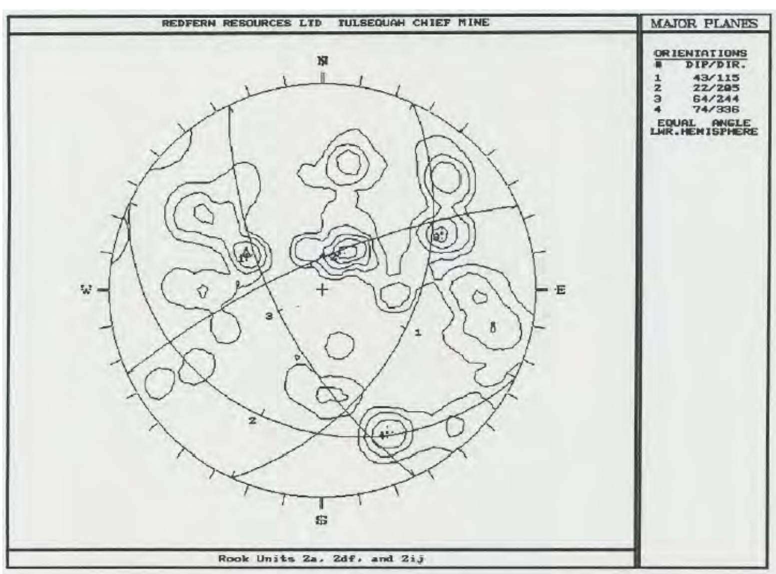
- Joint Set 1 (major) - 43°/115°;
- Joint Set 2 (major) - 22°/206°;
- Joint Set 3 (minor) - 64°/244°; and
- Joint Set 4 (minor) - 74°/336°.

Figure 16.1: Tulsequah Chief Mine – Hangingwall Volcanics



Source: West 2012

Figure 16.2: Tulsequah Chief Mine – Felsic Volcanics & Massive Sulphides

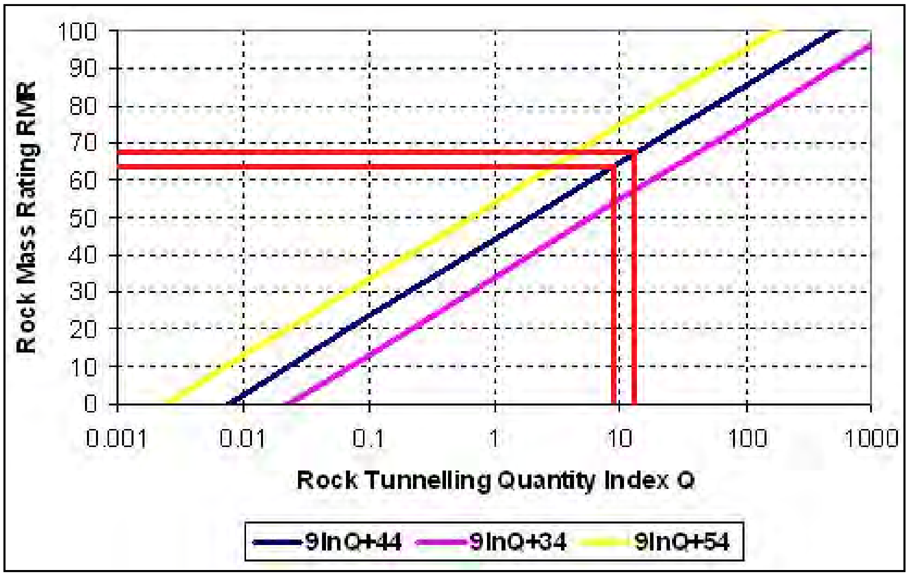


Source: West 2012

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



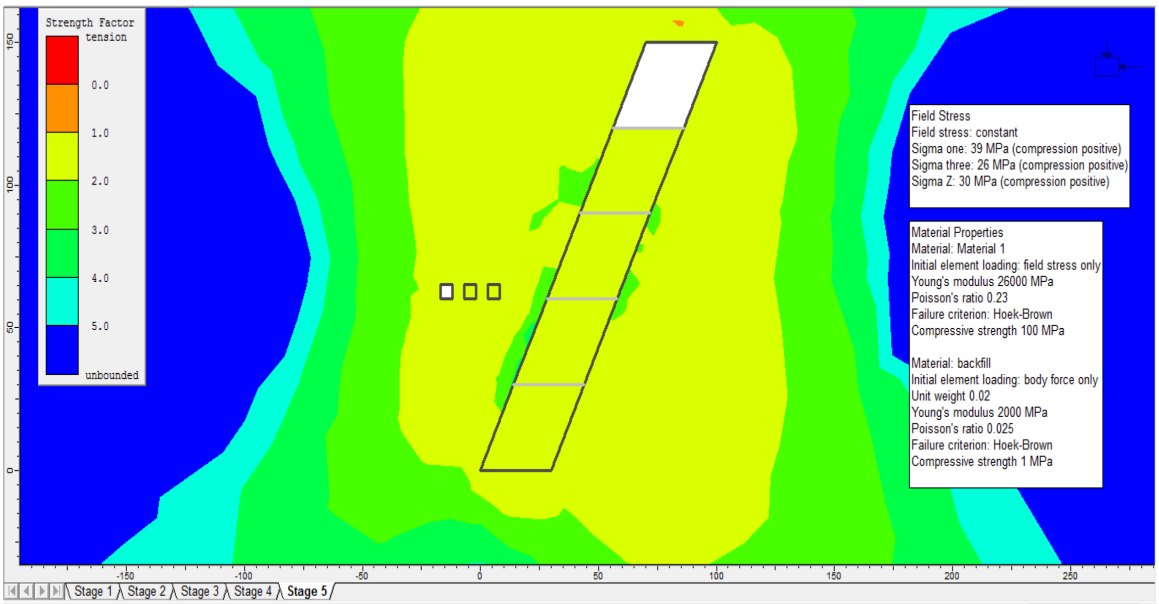
Source: Chieftain 2014

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



Source: West 2012

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.										

Source: West 2012

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.					

Source: West 2012

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.

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	5.0	5.0	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	9.0	13.0	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	10.0		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
Switchroom	5.0	5.3	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
Level/X-cut	4.6	4.6	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
Ore Drift	5.0	5.0	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	5.0	4.0	1) 2.4 m long 16 mm diameter mechanical rock bolts and wire mesh.	2.4	1.2 x 1.5	1.8	
Electrical Sub/Powder Magazine	5.0	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
Cap Mag	4.0	4.0	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
C&F Access Drift	5	4.0	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

Source: West 2014

A 5% shotcrete contingency has been included for supplementary support in ramp and waste drifts and a 10% stope cable bolt contingency has been added to account for 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; and
- 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). Longhole stoping is further subdivided into longitudinal and transverse primary & secondary stoping. 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 97% of the total mining resource will be mined with LH stoping (including ore sublevels) and the remaining 3% 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 paste fill and/or development waste.

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

Where mineralized zone widths perpendicular to strike are greater than 20 m, multiple transverse stopes will be mined in primary-secondary mining sequence. Primary & secondary stopes are sized at 10 and 20 m wide along strike respectively. 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 central ore drift up to mineralization thickness to a maximum 5 m by 4 m high access drift central to the stope.

A slot raise will be developed at one end 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.

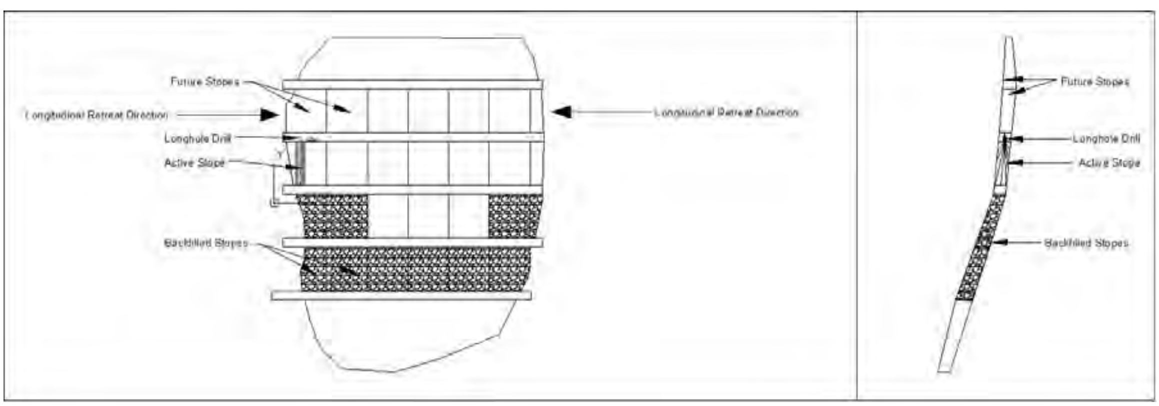
Vertical rings of drill holes will be blasted into the open stope and mineralized material will be mucked from the bottom of the stope by load-haul-dump (LHD) with remote control.

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. Sill pillars have been designed based on a required backfill strength of 780 kpa. 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.

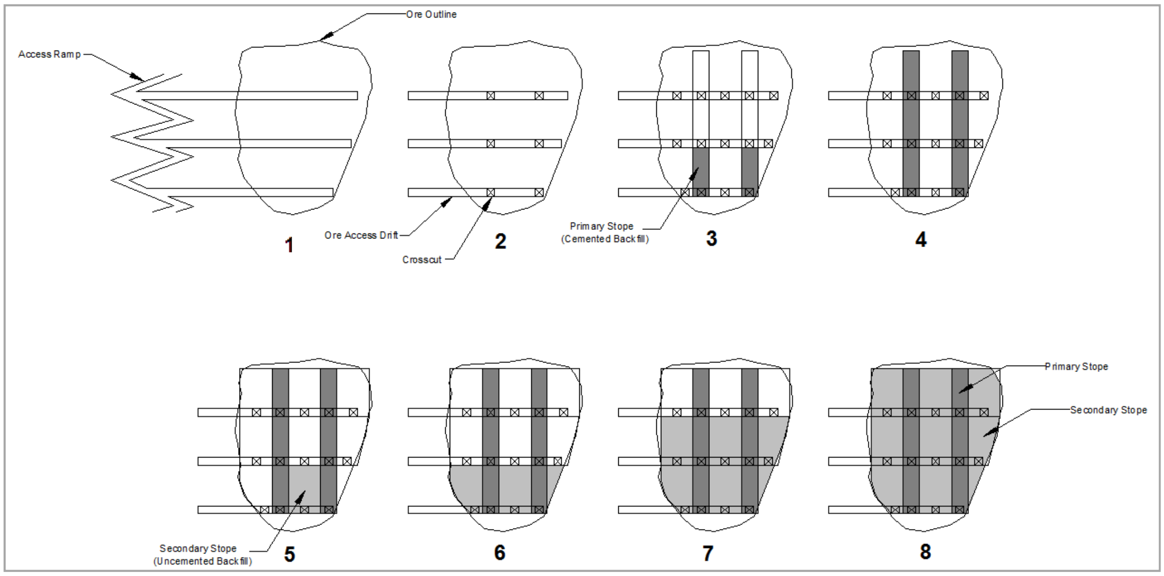
Illustrative, sublevel stoping diagrams are shown in Figures 16.5 and 16.6.

Figure 16.5: Longitudinal Longhole Stoping



Source: JDS 2014

Figure 16.6: Transverse Longhole Stoping



Source: JDS 2014

16.4.2 Mechanized Cut & Fill

Mechanized cut-and-fill mining will be utilized in shallower dipping areas, less than 55^o 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.

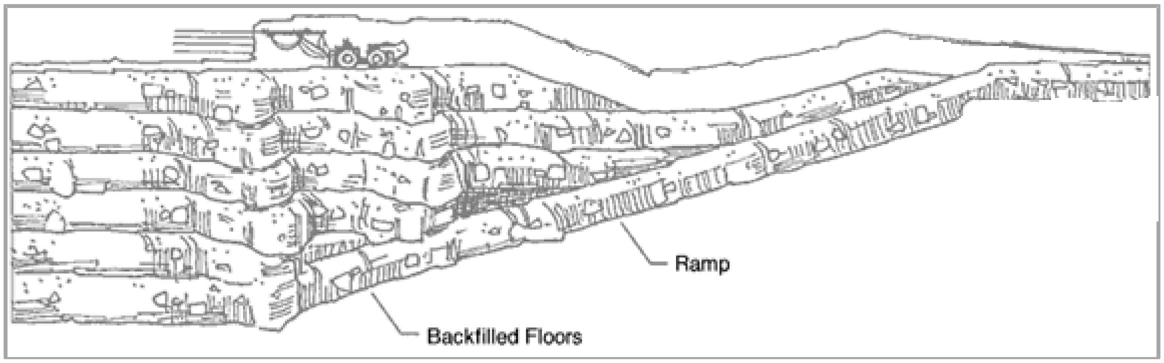
Each MCF mining block is accessed by an 18% to 20% access ramp and mined in 5 m high lifts (MCF stopes). Stopes are developed on the lowest level first, and each subsequent stope or 5 m lift is developed above the depleted and backfilled stope.

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 5 m high stope is then filled with paste fill and/or development waste. The next 5 m lift will then commence on top of the hardened fill of the previous lift.

An illustration of MCF mining is shown in Figure 16.7.

Figure 16.7: Mechanized Cut & Fill Stoping



Source: JDS 2014

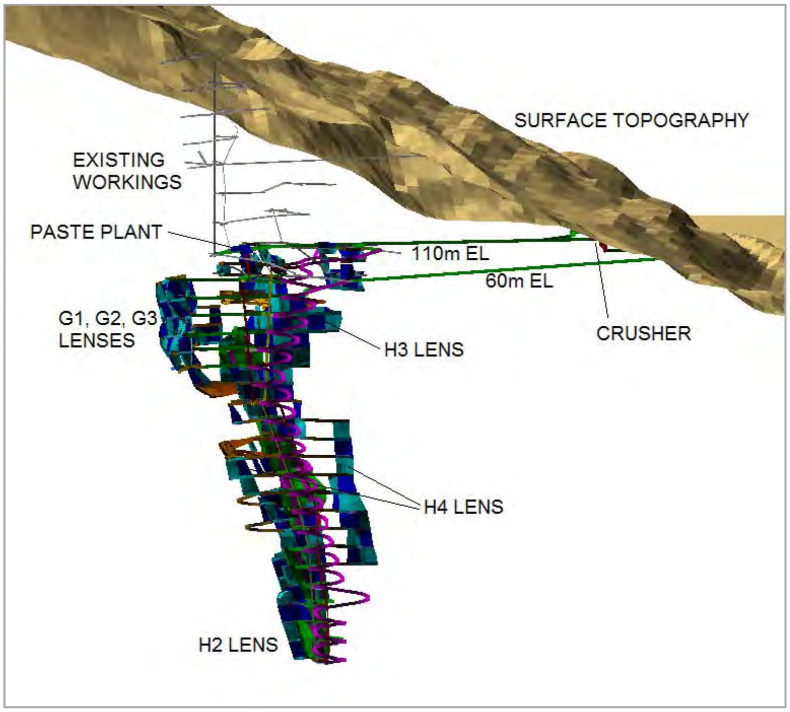
16.5 Mine Design

The Tulsequah deposit will be accessed via the 120 m (former 5400 level) and 60 m (5200) 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 5.0 m to accommodate the trackless equipment fleet. The main mine access will be via the 60 m (5200) level and connect to the main ramp that will access the mining levels. The main ramp is 5.0 m x 5.0 m in section and inclined to 17%. The main ramp will access sublevels 30 m apart vertically.

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.

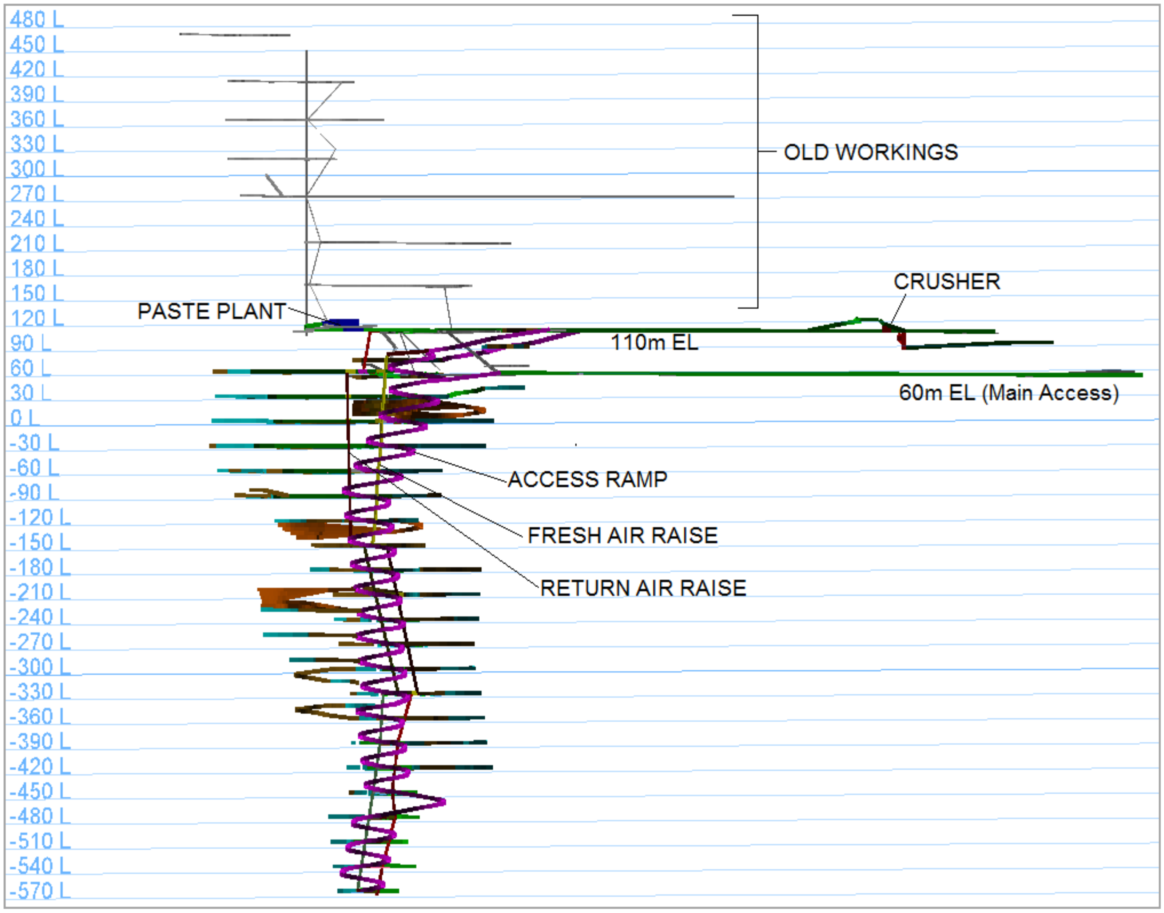
Mining levels (4.6 m x 4.6 m) will be located at 30 m vertical intervals. Truck loading will be done on each mining level to minimize LHD haulage distances. The deepest mining level, -570 m, will be located 630 m below the 60 m level. Three-dimensional views of the mine design are shown in Figures 16.8 and 16.9.

Figure 16.8: Mine Design – 3D View



Source: Chieftain 2014

Figure 16.9: Mine Design – Long Section (Without Stopes)



Source: Chieftain 2014

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 4.0 m wide x 4.0 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 4.5 m high x 15 m long.

Water collection sumps are located on every level. Sumps have been sized at 4.0 m high x 5.0 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 for treatment.

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

Electric power centres will be located on each level in drifts 5.0 m high x 5.3 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.

Level plans are in Appendix C.

16.6 Mine Ventilation

The design basis of the ventilation system at Tulsequah Chief underground operation is 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 289,000 ft^3/min (136 $m^{3/}$s) air volume required.

Table 16.5: Diesel Equipment Ventilation Requirements

Equipment	Units	HP per Unit	Total (hp)	Availability (%)	Utilization (%)	Utilized (hp)	Air Volume (ft^3/min)
Jumbo 2 Boom	1	147	294	85	25	62	6,200
Jumbo 1 Boom	1	83	83	85	25	18	1,800
LH Drill	1	99	99	85	25	21	2,100
LH Drill Narrow	1	99	99	85	15	13	1,300
Mechanized Bolter	1	99	99	85	25	21	2,100
40 t Truck	3	503	1,509	85	90	1,154	115,400
7.0 m^3 LHD	2	325	650	85	90	497	49,700
3.0 m^3 LHD	1	201	201	85	50	85	8,500
Scissor Lift	2	147	294	90	30	79	7,900
ANFO Loader	1	147	147	90	30	40	4,000
Fuel/Lube Truck	1	147	147	90	50	66	6,600
Utility Truck	1	147	147	90	40	53	5,300
Supervisors/Mechanics Vehicles	6	127	635	90	30	171	20,600
Crusher							9,300
Losses (20%)							48,160
	Total						**288,960**

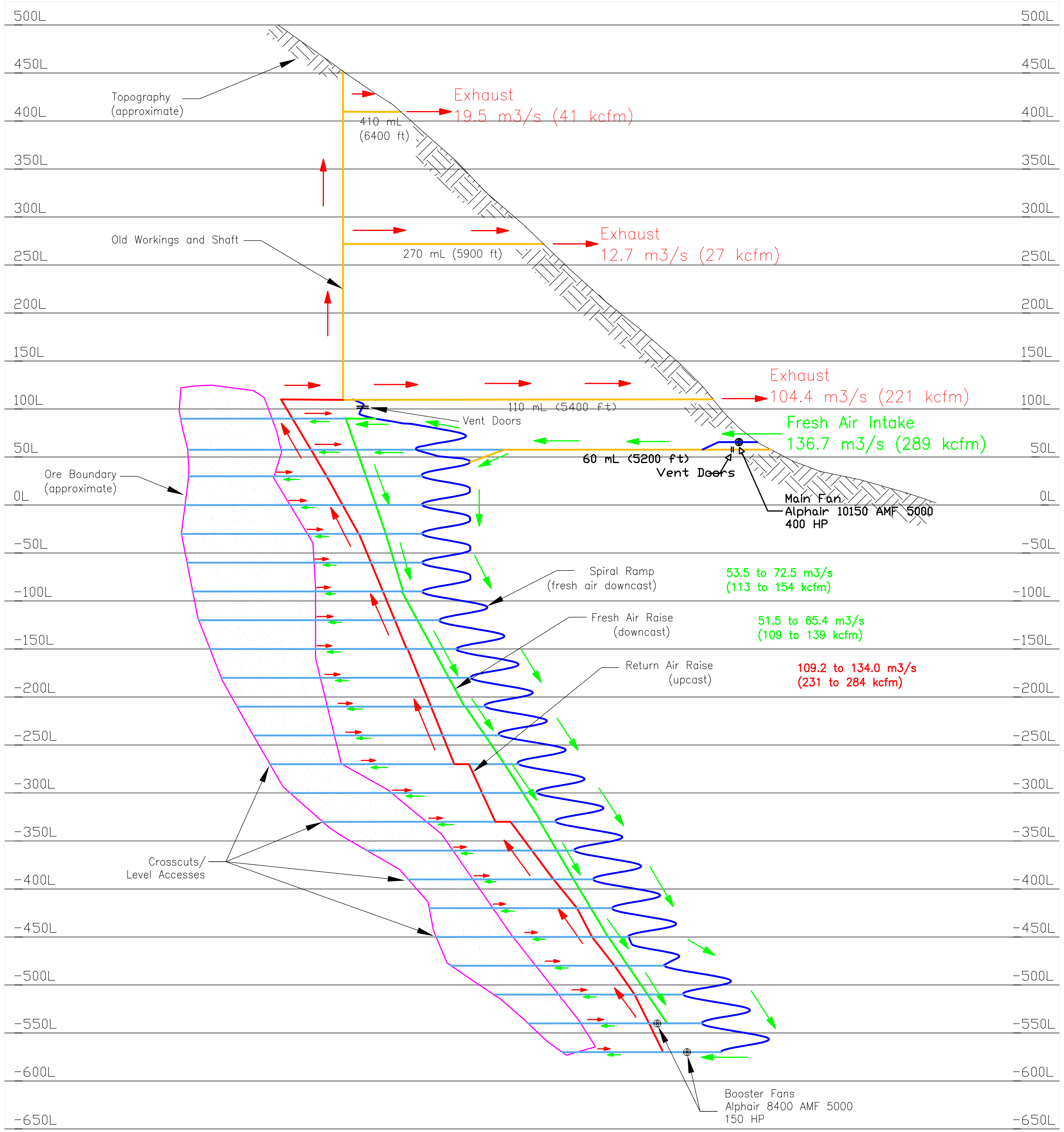
The primary ventilation system utilizes an axial vane fan as the prime mover located on the 60 m level. Fresh air is directed down the haulage ramp and fresh air raise (FAR) from the 60 m (5200 level) adit.

Ventilation modeling indicated high pressure (~2,200 Pa) at the main fan during the second half of the mine life. If not reduced, this high pressure will cause extensive leakage over estimated quantities at ventilation doors and bulkheads. This high pressure in the years 6 or 7 of mine life will be mitigated by adding 2 booster fans at the bottom of the exhaust raises. These two fans will pull the air from the ramp and the fresh air raise and push air directly to the exhaust raise.

Two sets of double ventilation doors to form an air lock will be installed underground near the main fan installation. The first set will be used for separation of the fresh air side from exhaust air. These doors will be installed in the drift connecting new part of the mine with old workings. The second set of the vent doors will be installed at the access decline portal to eliminate short circuiting of the fresh air at the access decline portal. The main fan and booster fans will be equipped with variable frequency drive (VFD) starters.

Use of the VFD type starters will significantly reduce power consumption and will allow for smooth operation of the ventilation system.

The primary ventilation system is shown in Figure 16.10.



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TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
MINE VENTILATION SCHEMATIC
GENERAL ARRANGEMENT
SECTION

PROJECT No. | DRAWING No. 1540-10-001 | REV No. F1

16.6.1 Ventilation Fan Selection

The main air fan located on the 60 m level will be an Alphair model 10150 AMF 5000 Arr. #4. The fan will have a 500 hp (373 kW) electric motor and will run at 710 rpm. The fan will deliver 136 m^3/s (289,000 cfm) at the pressure of 1,620 Pa (6.6" w.g.).

Two additional booster fans are required in years 6 and 7. The fans selected for this duty are Alphair model 8400 AMF 5000 Arr. #4. Each fan will have 150 hp (112 kW) motor operating at 710 rpm and will deliver 68 m^3/s (144,500 cfm) of the air.

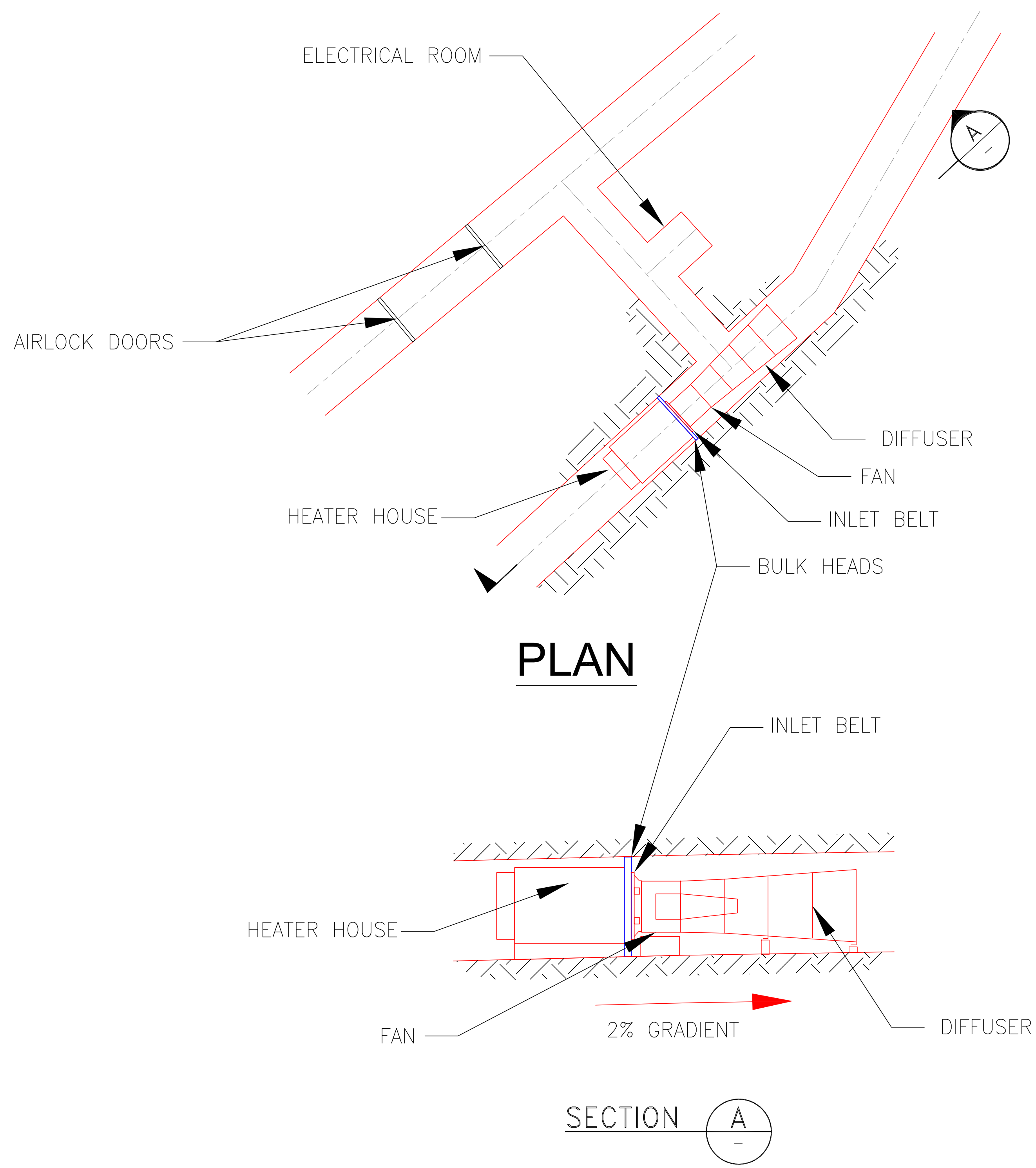
Auxiliary ventilation for ramp, production and level development will be done with 75 kW (100 hp) fans and single or twin 1.22 m (48 inches) and 1.07 m (42inches) diameter flexible ducting.

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 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 fan & waste heater infrastructure. The main fan installation is shown in Figure 16.6.

16.6.3 Emergency Stench System

A stench gas system utilizing Ethyl Mercaptan will be installed on 60 m 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 VFD or remote access.



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MAIN FAN
PLAN AND SECTION

PROJECT No.
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16.7 Underground Mine Services

16.7.1 Mine Power

From the main underground 4160V sub, the primary underground feed voltage at the Chieftain mine will be 5 kV from surface through a 1x3C 350MCM (Teck 5kV, Cu) power cable.

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

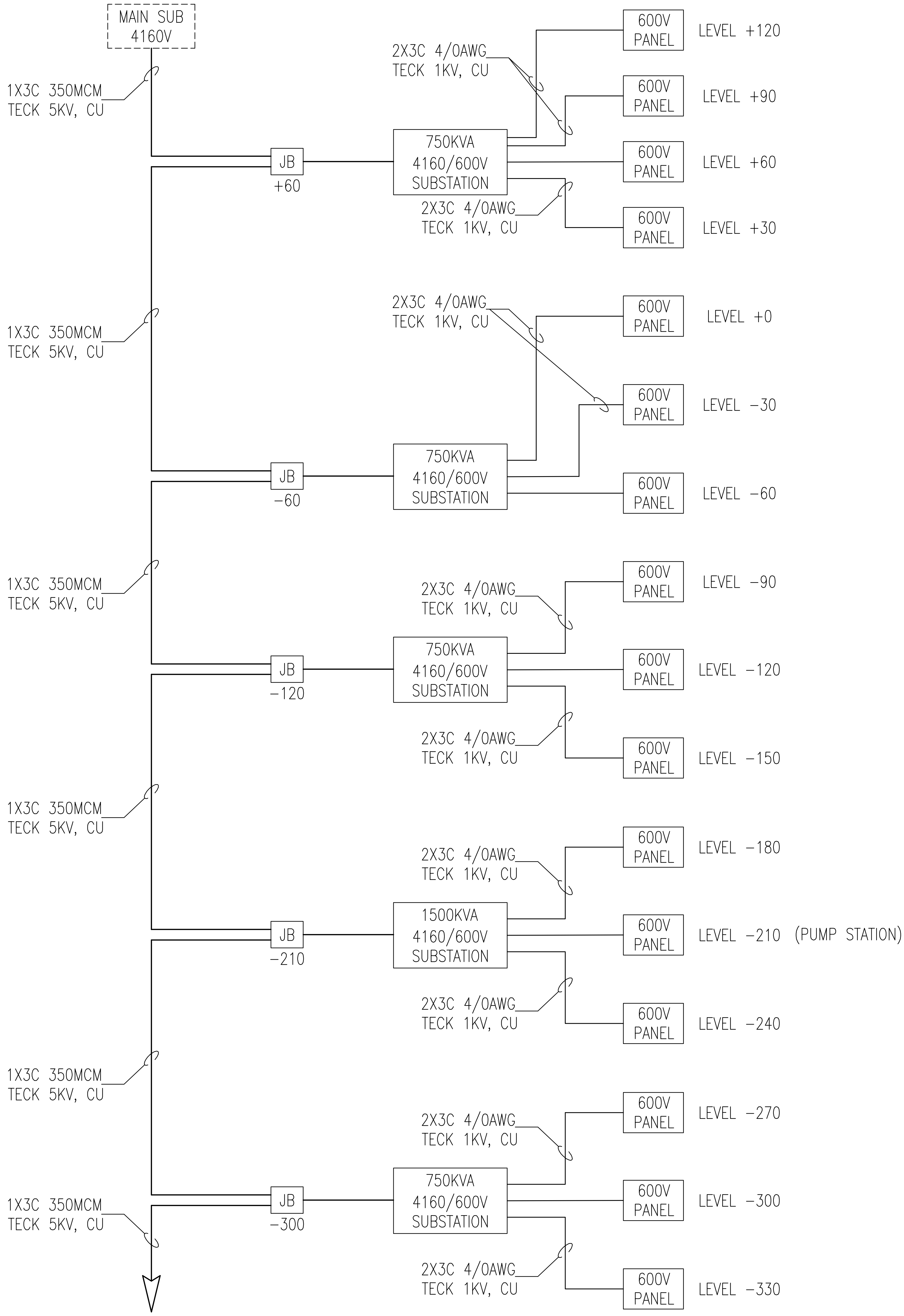
- Mine ventilation;
- Underground crushing plant and conveyor system;
- Underground paste plant;
- Underground dewatering;
- Underground mobile equipment;
- Compressed air;
- Mine lighting; and
- Refuge stations.
-

The underground crusher & conveyor systems and paste plant will be fed from surface from the process plant feeder. Eight main 4.16KV-600V transformer substations will be used to supply energy to 23 levels and sublevels. Each level will have one 600V, 400Amp portable distribution cabinet including fan and pump starters with three mobile equipment plugs. Control cabinet to move data to and from surface for fan controls, gas monitoring and air flow sensors as well as pump status or other is also part of the unit. Trailing cables are installed to supply power to work areas. One of three levels will be equipped with a 4.16kV - 600V, portable substation. It will supply power to the level as well as upper and lower levels.

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. A simplified electrical single-line drawing is shown in Figures 16.12 and 16.13.

Key power statistics are summarized below:

- Average underground power load – 1,501 kW; and
- Annual average consumption – 13,157,359 kWh.



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VÉRIFIÉ PAR/CHECKED BY	S. LACHARITÉ	2014-09-15

CLIENT/CUSTOMER
CHIEFTAIN MINERAL INC.
TULSEQUAH PROJECT

TITRE/TITLE
EVALUATION UNDERGROUND DISTRIBUTION
SINGLE LINE LEVEL +120 TO -300

NO. ING2014005-001-01 REV.
SÉRIE/SERIAL ÉCH./SCALE N/A 0A

LOGO
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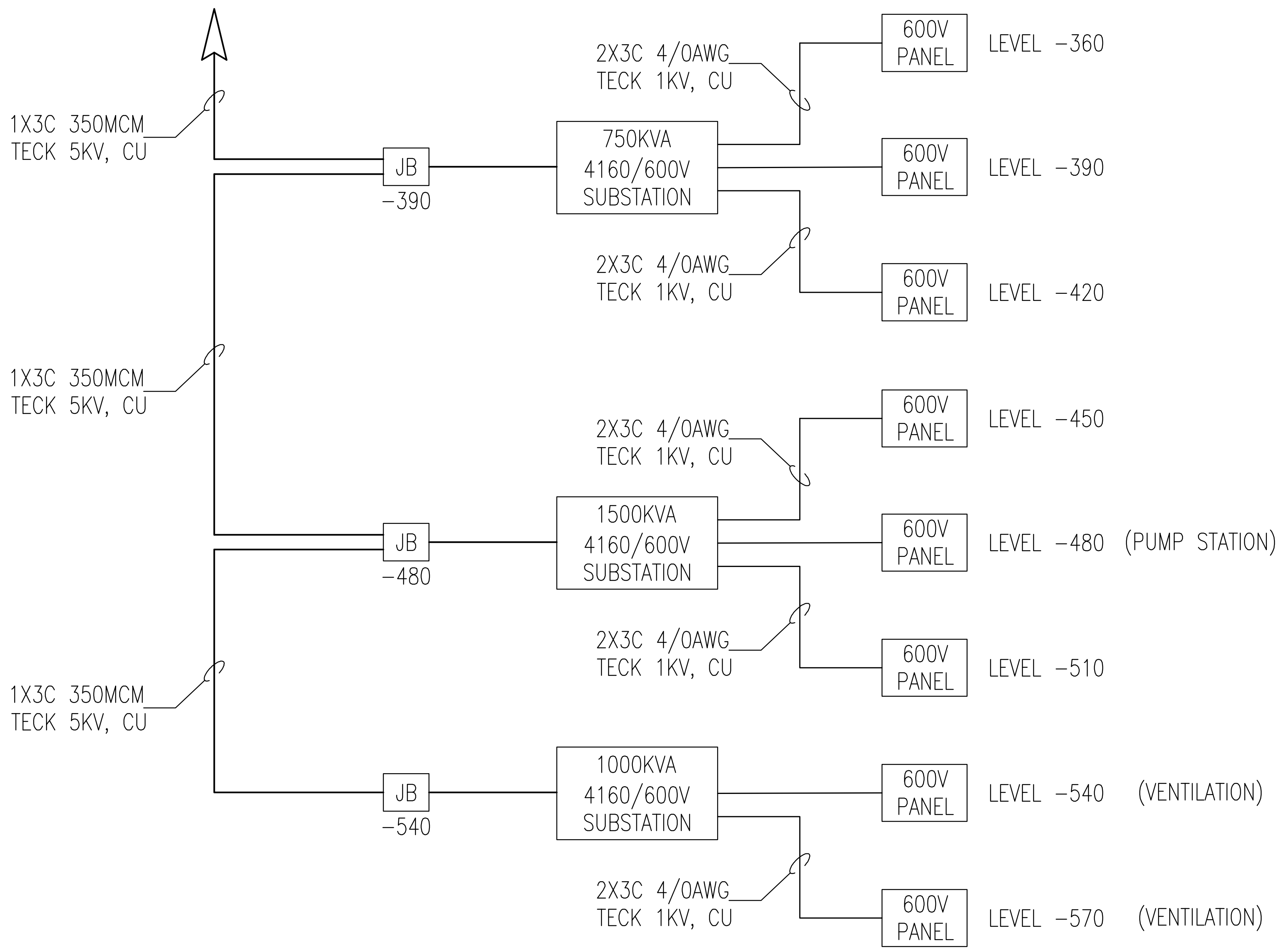

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VÉRIFIÉ PAR/CHECKED BY	S. LACHARITÉ	2014-09-15

CLIENT/CUSTOMER
CHIEFTAIN MINERAL INC.
TULSEQUAH PROJECT

TITRE/TITLE
EVALUATION UNDERGROUND DISTRIBUTION
SINGLE LINE LEVEL -360 TO -570

NO. ING2014005-001-02 REV.
SÉRIE/SERIAL ÉCH./SCALE N/A 0A

LOGO
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16.7.2 Communications

A leaky feeder communication system will connect the mine with surface operations. Telephones will be located at key infrastructure locations such as the crusher, paste backfill plant, electrical substations, refuge stations, and main sump. All personnel 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 60 m 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 eight stages as the main ramp advances to the bottom of the mine at the -570 m level. The mine dewatering system involves two main pump stations, seven secondary pump stations and 23 level sumps.

The secondary pump station on 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 the -570 m level to service the mine bottom.

The dewatering system is detailed in Figures 16.14 to 16.16.

16.7.6 Explosives and Detonator Storage

Explosives will be stored underground in permanent magazines, while detonation supplies (NONEL, electrical caps, detonating cords, etc.) will be stored in a separate 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.

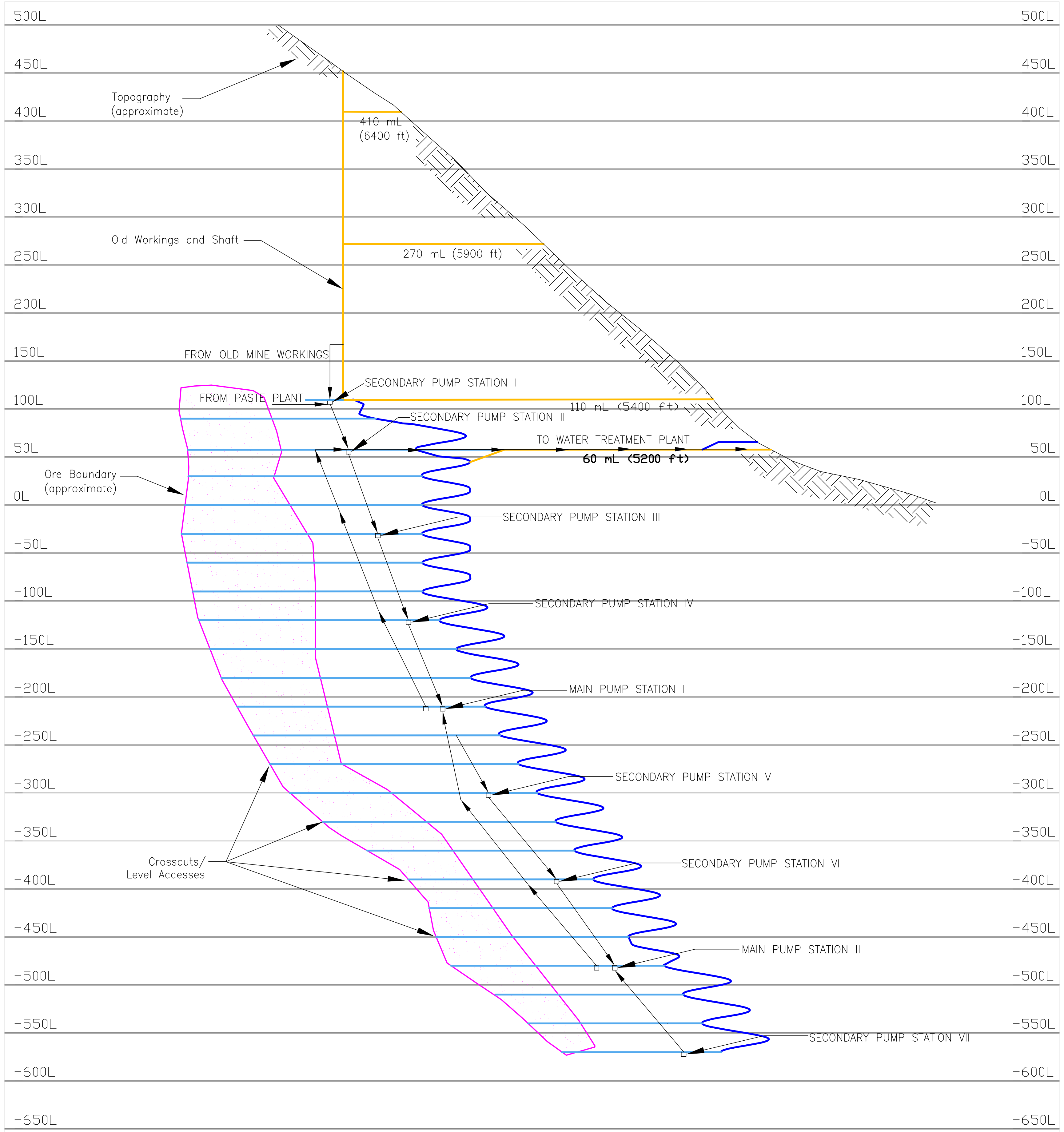
A mixture of ANFO will be used as the major explosive for mine development and stoping. Packaged emulsion will be used as a primer and for loading lifter holes in the development headings and for wet longholes. Smooth blasting techniques may be used as required main access development headings, with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

A mobile equipment fueling & lube station will be located near the 5200 level portal to provide fuel for the underground mobile equipment fleet.

There will be two portable fueling stations, or fuel-lube-Sats 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.



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TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

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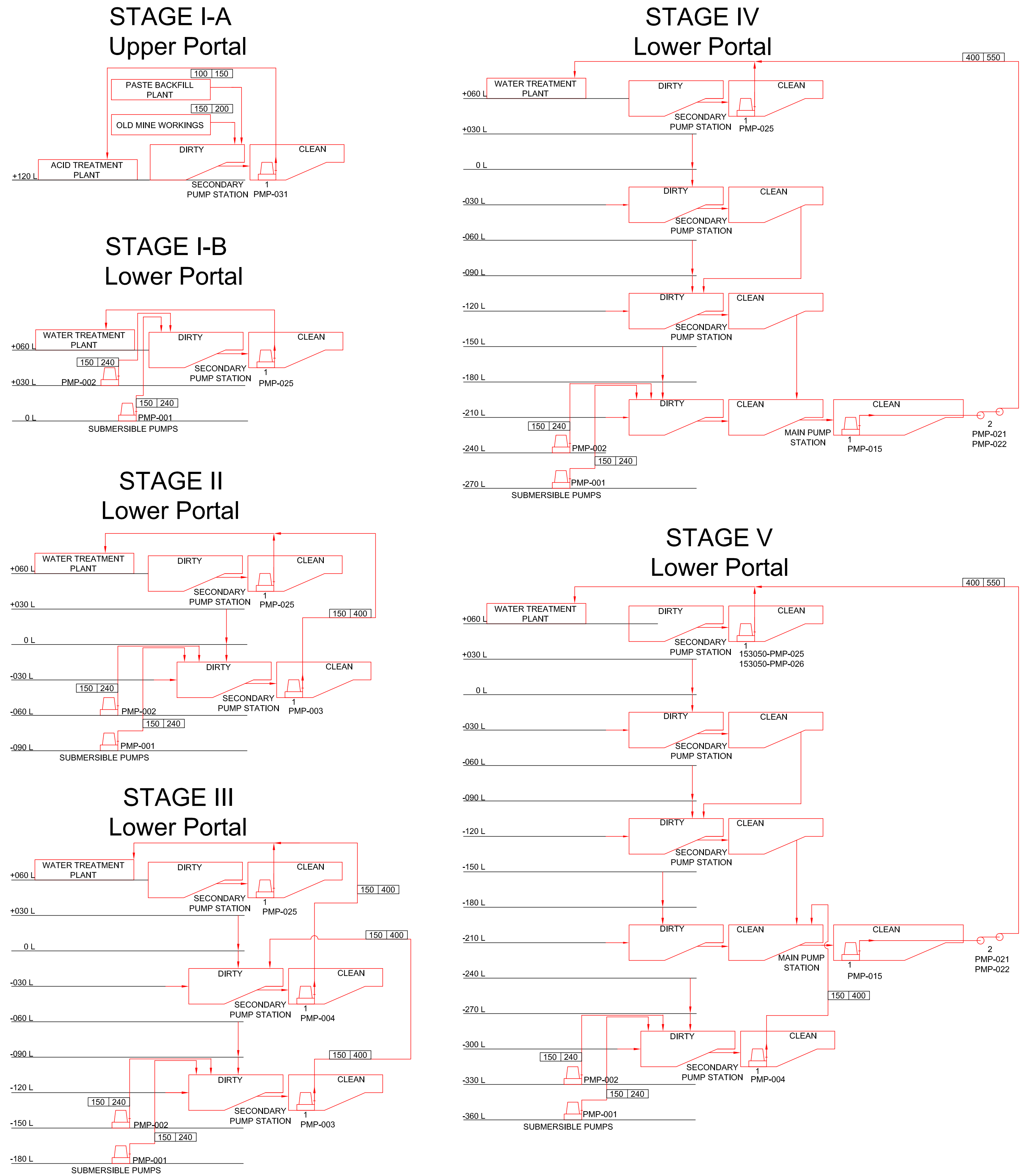
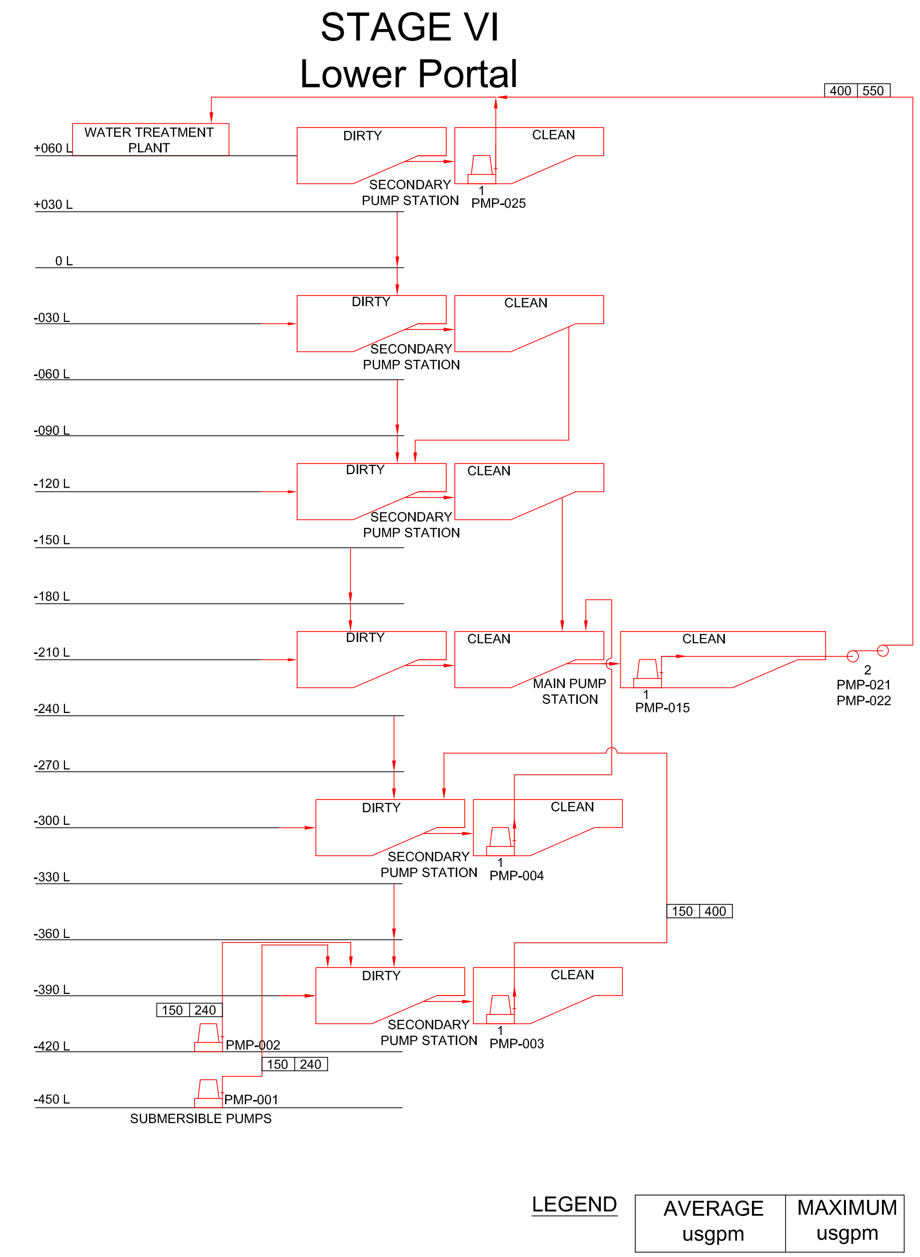
MINE DEWATERING SCHEMATIC
GENERAL ARRANGEMENT
SECTION

PROJECT No.
DRAWING No. 1530-10-001
REV No. F1





LEVEL	QUANTITY	PUMP	MANUFACTURER	MODEL	HP	EQUIPMENT NUMBER
0 -090 -180 -270 -360 -450 -540	1	Tertiary Level Pump	Toyo	DXL-30H	30	PMP-001
+030 -060 -150 -240 -330 -420 -510	1	Tertiary Level Pump	Toyo	DXL-15	15	PMP-002
-090 -390	1	Secondary Lift Pump	Toyo	DXL-100H	100	PMP-003
0 -300 -570	1	Secondary Lift Pump	Toyo	DXL-100H	100	PMP-004
-480	1	Main Charge Pump	Toyo	DL-15	15	PMP-016
-210	1	Main Charge Pump	Toyo	DL-15	15	PMP-015
-480	2	Main Booster Pump	Toyo	DBH-125/100MD	200	PMP-017 PMP-018
-210	2	Main Booster Pump	Toyo	DBH-125/100MD	200	PMP-021 PMP-022
+060	1	Tertiary Level Pump	Toyo	DL-20	20	PMP-025
+120	1	Tertiary Level Pump	Toyo	DL-20	20	PMP-031

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MINE DEWATERING
PROCESS FLOW DIAGRAM
STAGE I–VI

PROJECT No.

DRAWING No. 1530–10–002

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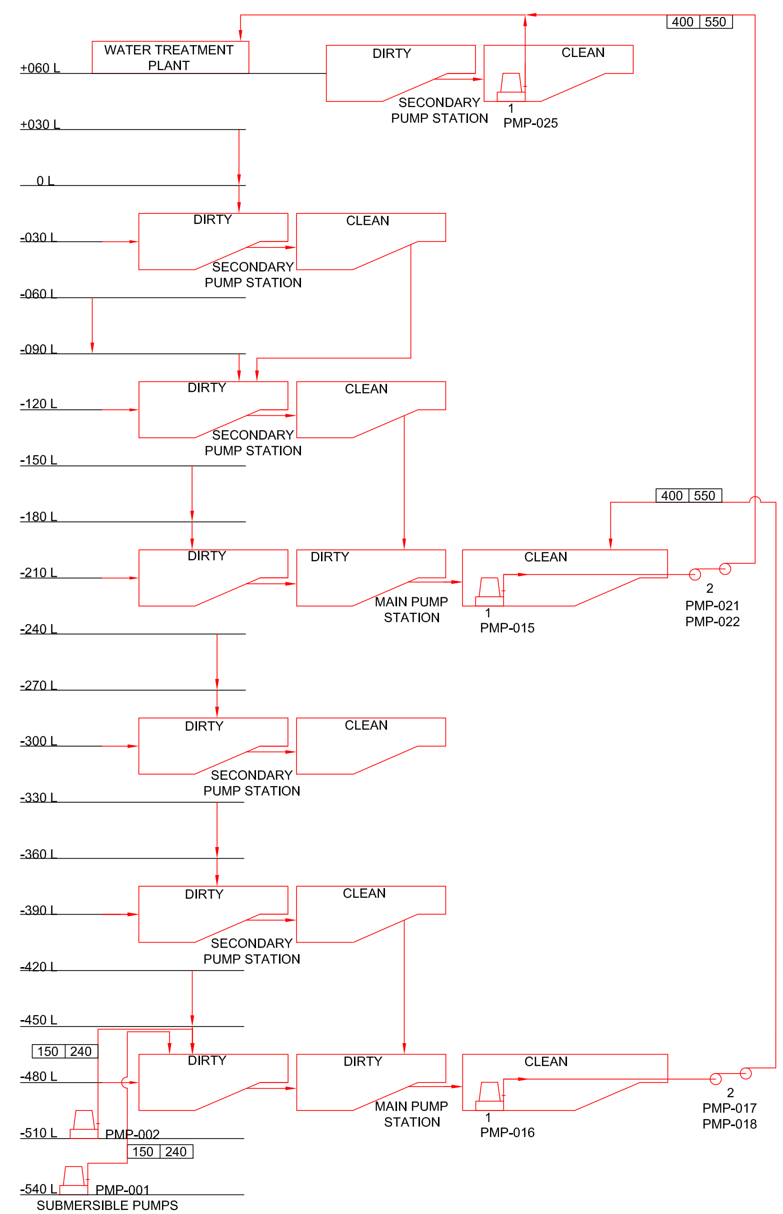
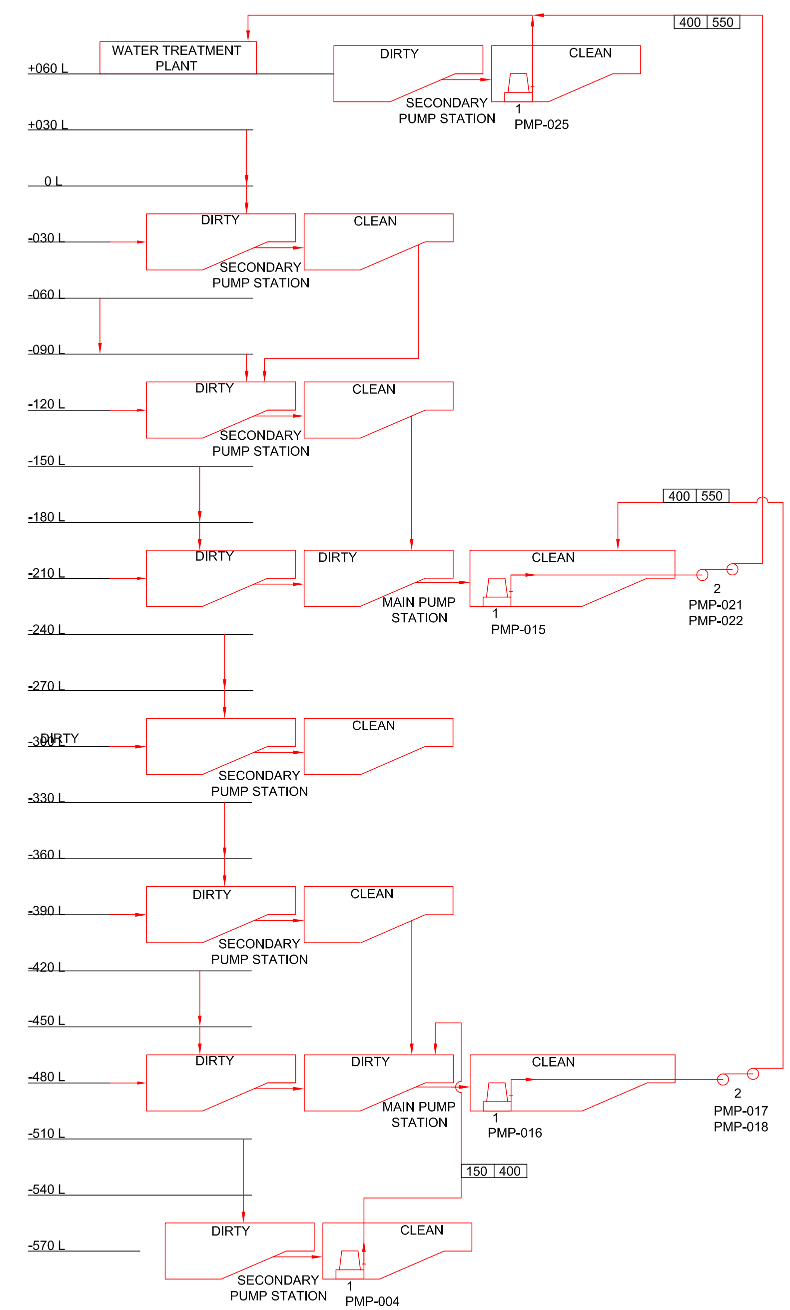
STAGE VII
Lower Portal



STAGE VIII
Lower Portal



LEVEL	QUANTITY	PUMP	MANUFACTURER	MODEL	HP	EQUIPMENT NUMBER
0 -090 -180 -270 -360 -450 -540	1	Tertiary Level Pump	Toyo	DXL-30H	30	PMP-001
+030 -060 -150 -240 -330 -420 -510	1	Tertiary Level Pump	Toyo	DXL-15	15	PMP-002
-090 -390	1	Secondary Lift Pump	Toyo	DXL-100H	100	PMP-003
0 -300 -570	1	Secondary Lift Pump	Toyo	DXL-100H	100	PMP-004
-480	1	Main Charge Pump	Toyo	DL-15	15	PMP-016
-210	1	Main Charge Pump	Toyo	DL-15	15	PMP-015
-480	2	Main Booster Pump	Toyo	DBH-125/100MD	200	PMP-017 PMP-018
-210	2	Main Booster Pump	Toyo	DBH-125/100MD	200	PMP-021 PMP-022
+060	1	Tertiary Level Pump	Toyo	DL-20	20	PMP-025
+120	1	Tertiary Level Pump	Toyo	DL-20	20	PMP-031

LEGEND

AVERAGE usgpm	MAXIMUM usgpm

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: None | DATE:
DESIGNED: J.D.S. | Sept 28 14
DRAWN: B.Wong | Sept 28 14
CHECKED:
APPROVED:

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
MINE DEWATERING
PROCESS FLOW DIAGRAM
STAGE VII – VIII

PROJECT No. | DRAWING No. 1530-10-003 | REV No. F1

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 60 m 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.

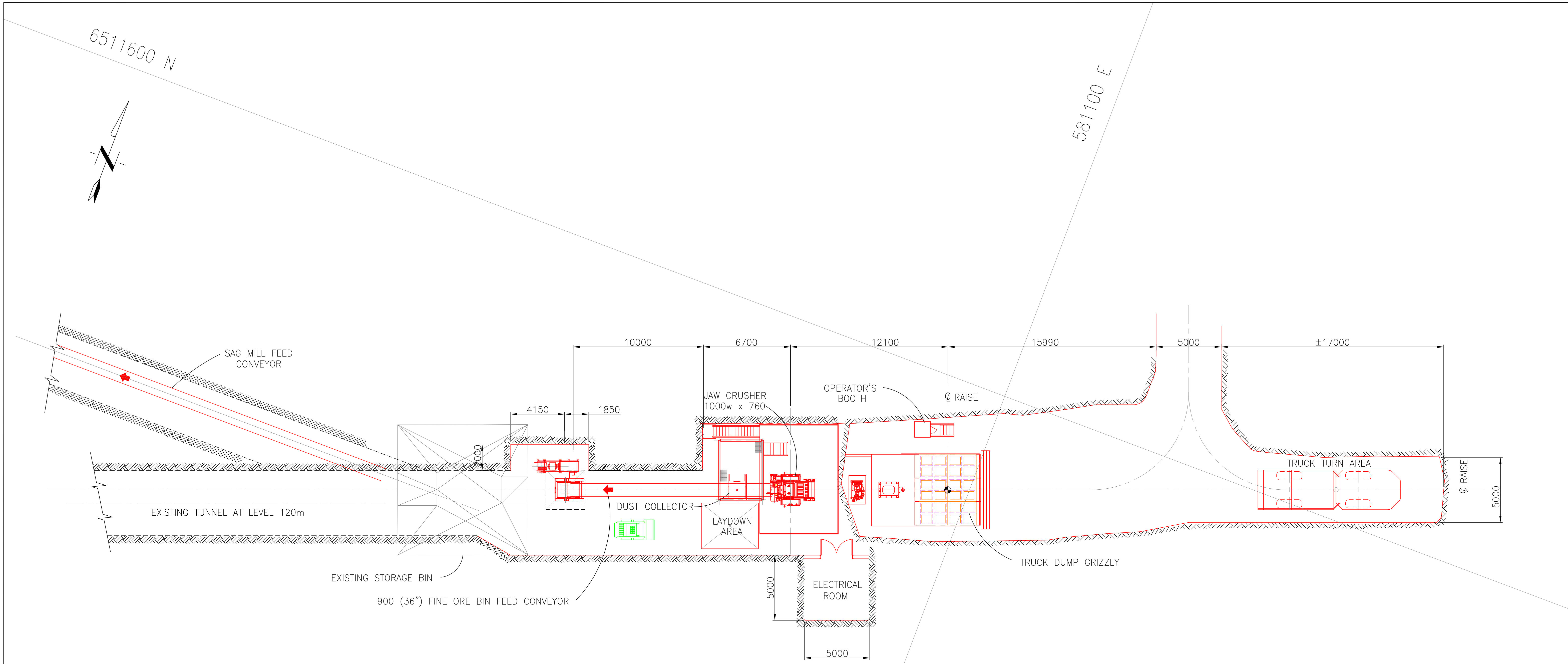
A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

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 Crusher

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 120 m level and will be fed ROM ore through a grizzly and feed raise. Crushed, fine ore will be conveyed to a 2,000 tonne fine ore bin. Ore from the bin is then conveyed to process plant via the new 84 m level portal.

The crusher, dump pocket and reclaim are shown in Figures 16.17 to 16.21.



PLAN – UNDERGROUND CRUSHING

5 0 5 10 15 20m

SCALE

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		20/10/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

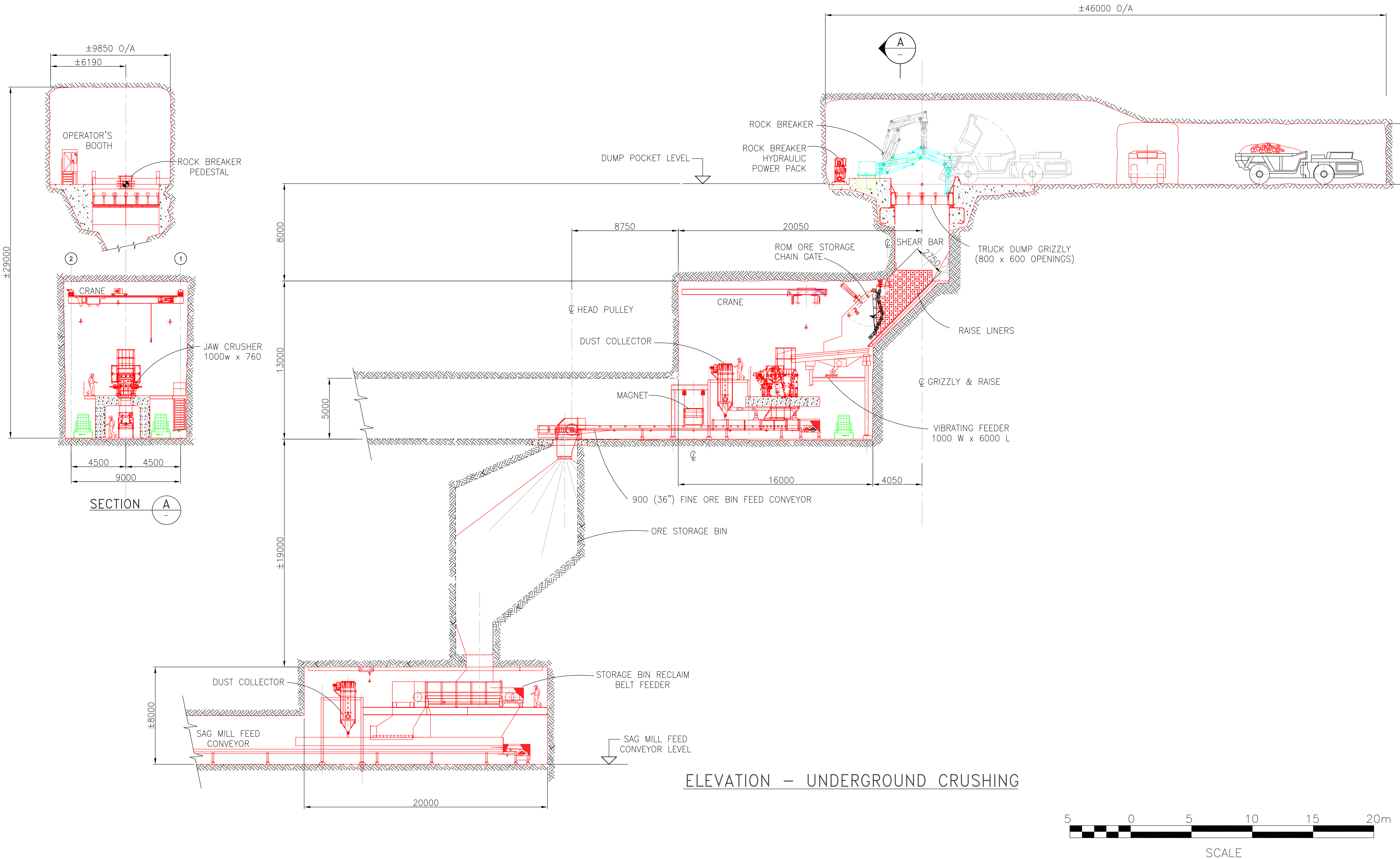
SCALE:		DATE:
DESIGNED:	J.D.S	Aug 23 2014
DRAWN:	B. Wong	Aug 23 2014
CHECKED:		
APPROVED:		

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
UNDERGROUND CRUSHING PLANT
GENERAL ARRANGEMENT
PLAN AND SECTION

PROJECT No. | DRAWING No. 1560-10-001 | REV No. F1



ELEVATION – UNDERGROUND CRUSHING

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		20/10/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14

REFERENCE DRAWING | REF. DWG No.

SCALE:		DATE:
DESIGNED:	J.D.S	Aug 23 2014
DRAWN:	B. Wong	Aug 23 2014
CHECKED:		
APPROVED:		

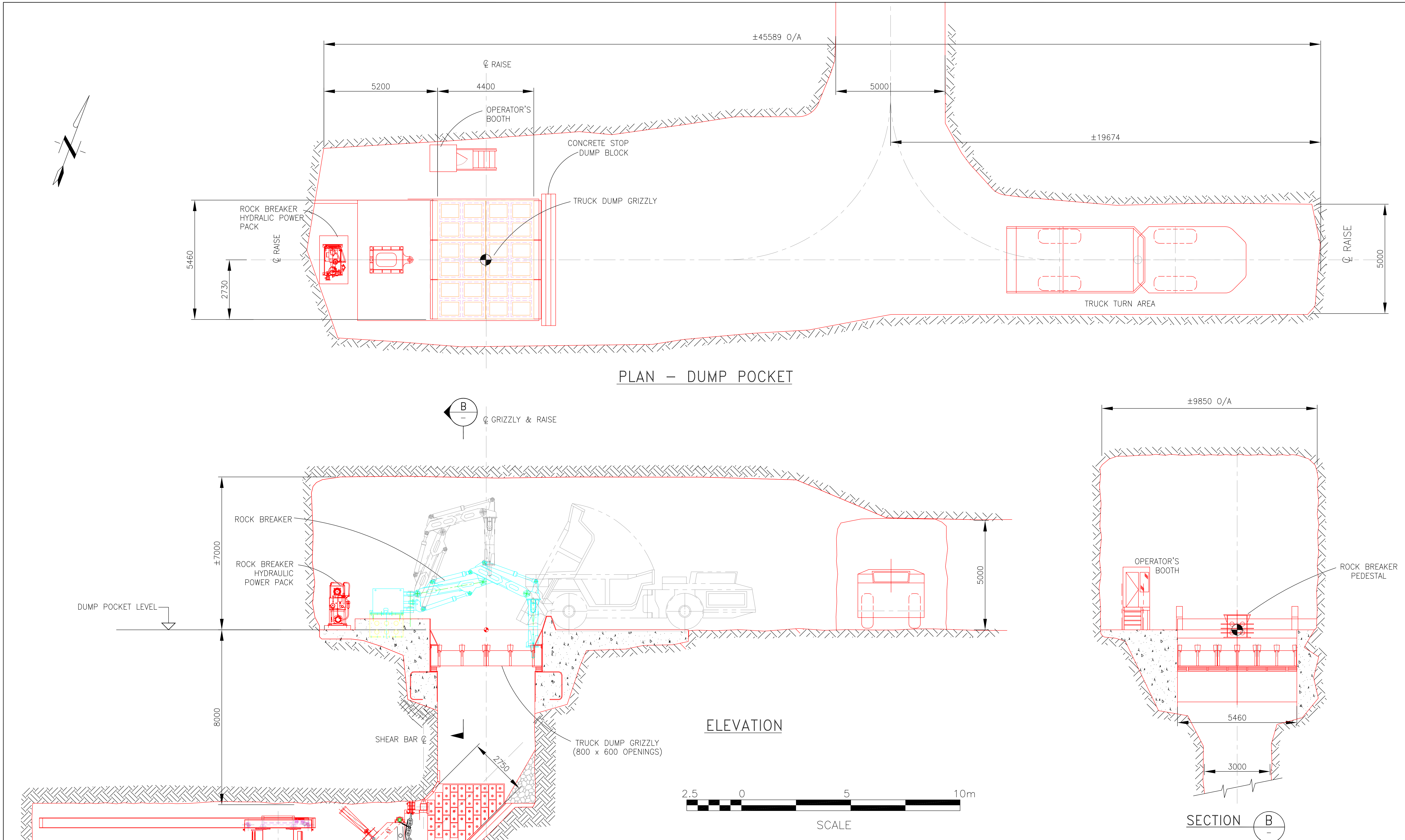
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
UNDERGROUND CRUSHING PLANT
SECTIONS
PLAN AND SECTION

PROJECT No. | DRAWING No. 1560-10-002 | REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		20/10/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

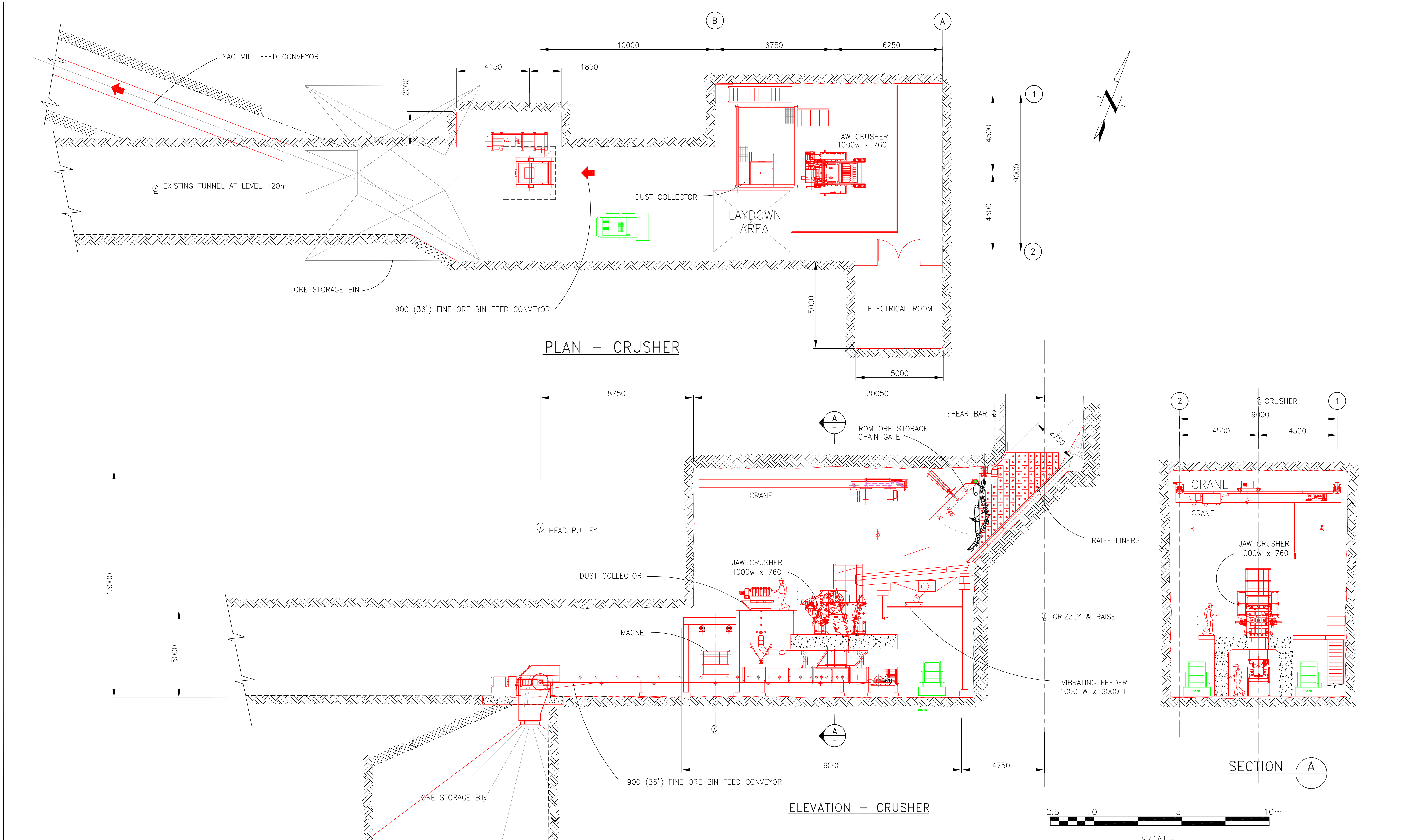
		DATE:
SCALE:		
DESIGNED:	J.D.S	Aug 23 2014
DRAWN:	B. Wong	Aug 23 2014
CHECKED:		
APPROVED:		

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
UNDERGROUND CRUSHING PLANT
DUMP POCKET
PLAN AND SECTION

PROJECT No. | DRAWING No. 1560–10–003 | REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		20/10/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14

REFERENCE DRAWING | REF. DWG No.

SCALE:
DESIGNED: J.D.S — DATE: Aug 23 2014
DRAWN: B. Wong — Aug 23 2014
CHECKED:
APPROVED:

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

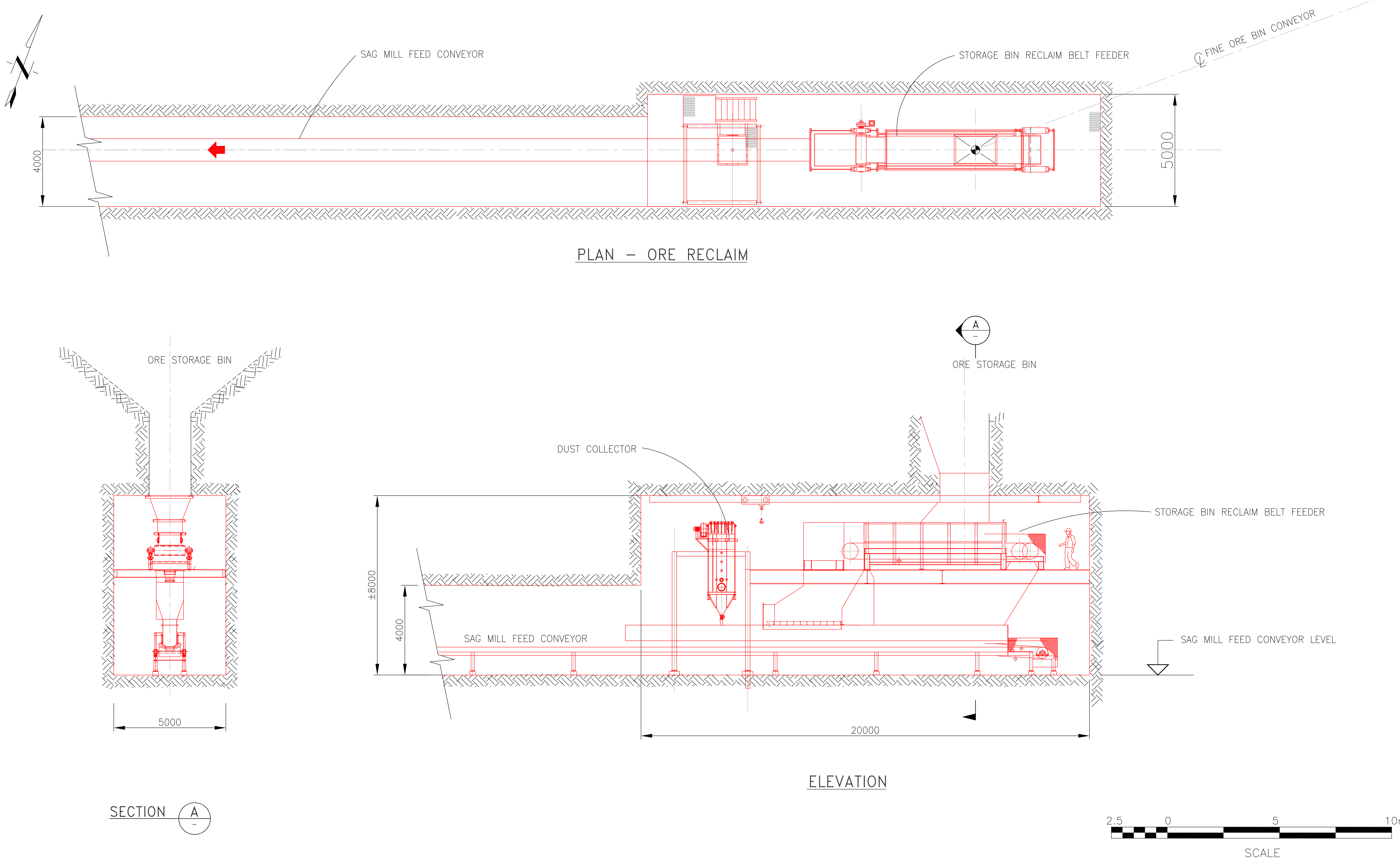
JDS Energy & Mining Inc.

TITLE: UNDERGROUND CRUSHING PLANT PRIMARY CRUSHER PLAN AND SECTION

PROJECT No.

DRAWING No. 1560-10-004

REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		20/10/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14

REFERENCE DRAWING	REF. DWG No.
UNDERGROUND CRUSHING PLANT(SECTIONS)	1560–10–002
UNDERGROUND DUMP POCKET	1560–10–003
UNDERGROUND CRUSHER	1560–10–004
UNDERGROUND ORE RECLAIM	1560–10–005

SCALE:	DATE:
DESIGNED: J.D.S	Aug 23 2014
DRAWN: B. Wong	Aug 23 2014
CHECKED:	
APPROVED:	

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
UNDERGROUND CRUSHING PLANT
ORE RECLAIM
PLAN AND SECTION

PROJECT No.	DRAWING No.	REV No.
	1560–10–005	F1

16.8 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 60 m level adit and main access decline will provide primary access to the underground workings. The 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.

16.9 Paste Backfill

A key driver for the Tulsequah Chief project is to limit the environmental impact by the potential to use all of the sulphides tailings in the underground backfill below 60 m Level (formerly 5200 Level), and using de-pyritized tailings above 60 m Level and disposing of the balance in the Tailings Management Facility. Chieftain's approach is to include a pyrite concentrate circuit to reduce the potential acid rock drainage (ARD) of the TMF. This is meant to address long-term environmental concerns and mitigation strategies for the site, with the intent being for all pyrite concentrate to be utilized in backfill. Additionally, the use of pyrite concentrate in the mine will be influenced by local topography, in order to further limit ARD potential following the closure of the mine. This results in a design where pyrite concentrate use is limited to backfill below the 60 m Level (formerly 5200 Level). Due to these requirements, backfill will often consist of cemented pyrite concentrate. The implications of pyrite concentrations (short and long-term strengths, stability, binder selection) are also key project drivers since the binder makes up approximately 60% of the backfill costs if the total binder cost is considered, or 40% of backfill costs if binder freight is excluded.

Both surface and underground locations were considered for the placement of the paste backfill plant. The underground location was preferred, as cost estimating indicated lower capital and operating costs compared to the surface plant. There are a number of other operational considerations which also favour the underground plant location. These include reduced maintenance and risk for pumping of slurry rather than cemented paste to underground, as well as lower pressures for the paste pumps and distribution system and greater flexibility to deliver lower slump paste to the mine. Should the surface to underground system fail, a pipe full of cemented paste backfill would require redundant pumps to clear the blockage (if possible) or possibly result in hardened paste that would require considerable cost and effort, and lost productivity before resuming backfilling.

The backfill plant has been located underground near the upper 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.

According to the life of mine plan, the Tulsequah Chief Mine will operate at an approximate annual production rate of 408,000 tpy. The milling plan shows that the pyrite concentrate tailings production rate will be approximately 21 tph, while the production rate for de-pyritized tailings will be 26 tph.

The plant capacity was designed primarily around the requirement that the plant and all pumps/pipeline transport systems are sufficiently robust to operate over a wider than normal range of throughput such that all pyrite concentrate reports to underground as backfill while minimizing the costs associated with management of and reclamation from the temporary pyrite concentrate storage facility. As a result, backfilling will be conducted at shorter intervals but at higher throughput, requiring an average of 25% utilization of the operating time of the mill. This low utilization also addresses the lack of redundancy within the plant, allowing ample time for maintenance. Moreover, the plant capacity eliminates the backfill as a bottleneck in the mining cycle.

Over 365 days, the approximate daily average backfill requirement is 360 tonnes of pyrite concentrate and 135 t of de-pyritized tailings as per the mine plan. This translates to the paste plant design capacity of 113 tph of pyrite concentrate and 63 tph of de-pyritized tailings, and operating an average of 5.4 hours per day, or 1.6 days per week.

The paste backfill system includes a surface component within the processing plant and an underground component. The following major components are situated on surface:

- Pyrite concentrate thickener underflow slurry agitated storage tank with four-day retention;
- De-pyritized tailings thickener underflow slurry agitated storage tank (32 hours retention);
- Building enclosure;
- Centrifugal pumps (four in all);
- Live bottom hopper and conveyor to feed reclaimed pyrite concentrate into pyrite concentrate agitated storage tank;
- Clean up pump; and
- Tie-in to medium voltage power, gland water, instrument air, HVAC, building.

Transport of material to the paste plant will require a 5" DR6.3 HDPE pipeline. The expected pressure during pumping is in the range of 15 to 20 bar, while the pipe is rated for 26.2 bar. The pipe is also sized suitably for adequate velocity in the range of 100% pyrite concentrate to 100% de-pyritized tailings. The feed material is received at the plant within an agitated tank which further ensures adequate mixing of the tailings and a more consistent feed for filtration.

The paste backfill plant is proposed to have a single disc vacuum filter system, including a vacuum pump, snap air tank, and filtrate receiver with pump. Instead of an extra installed unit, redundancy is considered in that the utilization (filling) time is roughly 25%, leaving 75% or approximately five of seven days for maintenance.

Instrumentation and automation is included to provide for a complete automated plant to the extent possible (operator is required for QA/QC checks and occasional input), maintain continuous quality control, lower operating labour and control of a final product. Data from the PLC will also be available for tracking, quality control and forecasting.

100% PYRITE CONCENTRATE (3% BINDER)

LINE NO.			1A		2	3	4	5	6	7	8
DESCRIPTION			PYRITE CONCENTRATE	PYRITE CON. MAKEUP	COMBINED TAILINGS	MIXER DILUTION	FILTER FEED	FILTRATE WATER	FILTER CAKE	BINDER	MIXER DISCHARGE
Mass	tph	Solids	20.5	92.5	113.0	20.6	92.5	0.0	92.5	3.5	116.5
	tph	Water	8.8	39.6	48.4	8.8	39.6	24.6	15.1	0.0	23.9
	tph	Slurry	29.3	132.1	161.5	29.4	132.1	24.6	107.5	3.5	140.4
wt% Solids	%		70%	70%	70%	70%	70%	0%	86%	100%	83%
Flow	m^3/hr	Solids	4.6	20.6	25.1	4.6	20.5	0.0	20.5	1.1	26.2
	m^3/hr	Water	8.8	39.6	48.4	8.8	39.6	24.6	15.1	0.0	23.9
	m^3/hr	Slurry	13.4	60.2	73.6	13.4	60.2	24.6	35.6	1.1	50.1
S.G.	t/m^3	Solids	4.50	4.50	4.50	4.50	4.50	4.50	4.50	3.14	4.44
	t/m^3	Water	1.00	1.00	1.00	1.00	1.00	1.0	1.00	1.00	1.00
	t/m^3	Slurry	2.20	2.20	2.20	2.20	2.20	1.0	3.02	3.14	2.80

100% DE-PYRITIZED TAILINGS (3% BINDER)

LINE NO.			1B		2	3	4	5	6	7	8
DESCRIPTION			DE-PYRITIZED TAILINGS	DE-PYRITIZED MAKEUP	COMBINED TAILINGS	MIXER DILUTION	FILTER FEED	FILTRATE WATER	FILTER CAKE	BINDER	MIXER DISCHARGE
Mass	tph	Solids	26.1	36.9	63.0	16.6	46.4	0.0	46.4	1.9	65.0
	tph	Water	17.4	24.6	42.0	11.1	31.0	15.5	15.5	0.0	26.5
	tph	Slurry	43.5	61.5	105.0	27.7	77.4	15.5	61.9	1.9	91.5
wt% Solids	%		60%	60%	60%	60%	60%	0%	75%	100%	71%
Flow	m^3/hr	Solids	9.7	13.7	23.3	6.1	17.2	0.0	17.2	0.6	24.0
	m^3/hr	Water	17.4	24.6	42.0	11.1	31.0	15.5	15.5	0.0	26.5
	m^3/hr	Slurry	27.1	38.3	65.4	17.2	48.2	15.5	32.7	0.6	50.5
S.G.	t/m^3	Solids	2.70	2.70	2.70	2.70	2.70	2.70	2.70	3.14	2.71
	t/m^3	Water	1.00	1.0	1.00	1.00	1.00	1.0	1.00	1.00	1.00
	t/m^3	Slurry	1.61	1.61	1.61	1.61	1.61	1.0	1.89	3.14	1.81

–FOR REVIEW–
–NOT FOR CONSTRUCTION–

REFERENCE DRAWINGS

DRAWING NO.	TITLE

REFERENCE DRAWINGS

DRAWING NO.	TITLE

REVISIONS

NO.	DESCRIPTION	DATE	DRAWN	CHECKED	APPROVED	REVIEWED

REVISIONS

NO.	DESCRIPTION	DATE	DRAWN	CHECKED	APPROVED	REVIEWED
A	ISSUED FOR REVIEW	02-OCT-2014	V.R.	A.H.	-	
B	MODIFIED CEMENT SYSTEM	17-OCT-2014	V.R.	A.H.	F.P.	

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CHIEFTAIN METALS INC

CHIEFTAIN METALS INC. - TULSEQUAH FEASIBILITY STUDY - PASTE PLANT PROCESS FLOWSHEET

KOVIT PROJECT NO.: J14-0026
DRAWING NO.: J14-0026-F-PPP-0002
SCALE: N.T.S.
REV.: B

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

- Agitated receiving slurry (pyrite concentrate, de-pyritized tailings) filter feed tank complete with pumps;
- Underground day-silo cement storage and metering system;
- Cement bag unloading system;
- Process water tank and pumps;
- Disc filter (10 discs x 3.2 m diameter) with vacuum pump 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; and
- High-pressure paste pipeline flush pump.

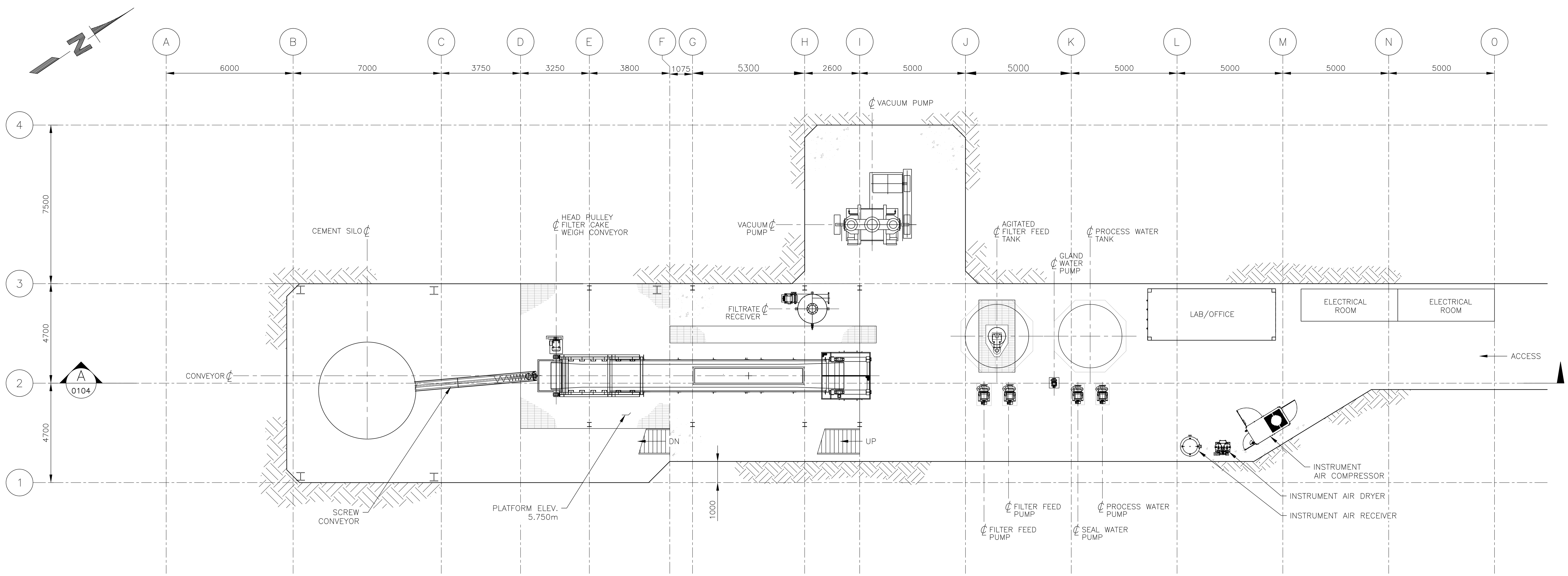
The tailings will be received in an agitated tank to allow for continuous operation for 15 to 20 minutes independently of the surface storage system. Tailings will be pumped into the filters and a slip-stream will be by-passed into the mixer to achieve the target slump.

Binder (90:10 ground granulated iron blast furnace slag to cement) will be added at between 2 wt% and 7.5 wt% of solids, depending on the backfill recipe requirement. Binder will be delivered to the site in 1.8 metric tonne bags and stored on surface. As required, bags will be transported underground and loaded by blower from a bulk bag unloaded system into the underground day silo.

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 operating pressure, wear and maintenance.

Additional UCS testing is recommended to optimize both the tailings management strategy and binder consumption (operating cost); however, it is not expected that any major changes to the plant design or capital cost estimate will be required.

The underground paste plant general arrangement is shown in Figures 16.23 and 16.24.



PLAN VIEW @ EL. 6.350m

—FOR REVIEW—
—NOT FOR CONSTRUCTION—

REFERENCE DRAWINGS	
DRAWING NO.	TITLE

REFERENCE DRAWINGS	
DRAWING NO.	TITLE

REVISIONS						
NO.	DESCRIPTION	DATE	DRAWN	CHECKED	APPROVED	REVIEWED

REVISIONS						
NO.	DESCRIPTION	DATE	DRAWN	CHECKED	APPROVED	REVIEWED
A	ISSUED FOR REVIEW	03-OCT-2014	V.R.	A.H.		
B	MODIFIED CEMENT SYSTEM	17-OCT-2014	V.R.	A.H.	F.P.	

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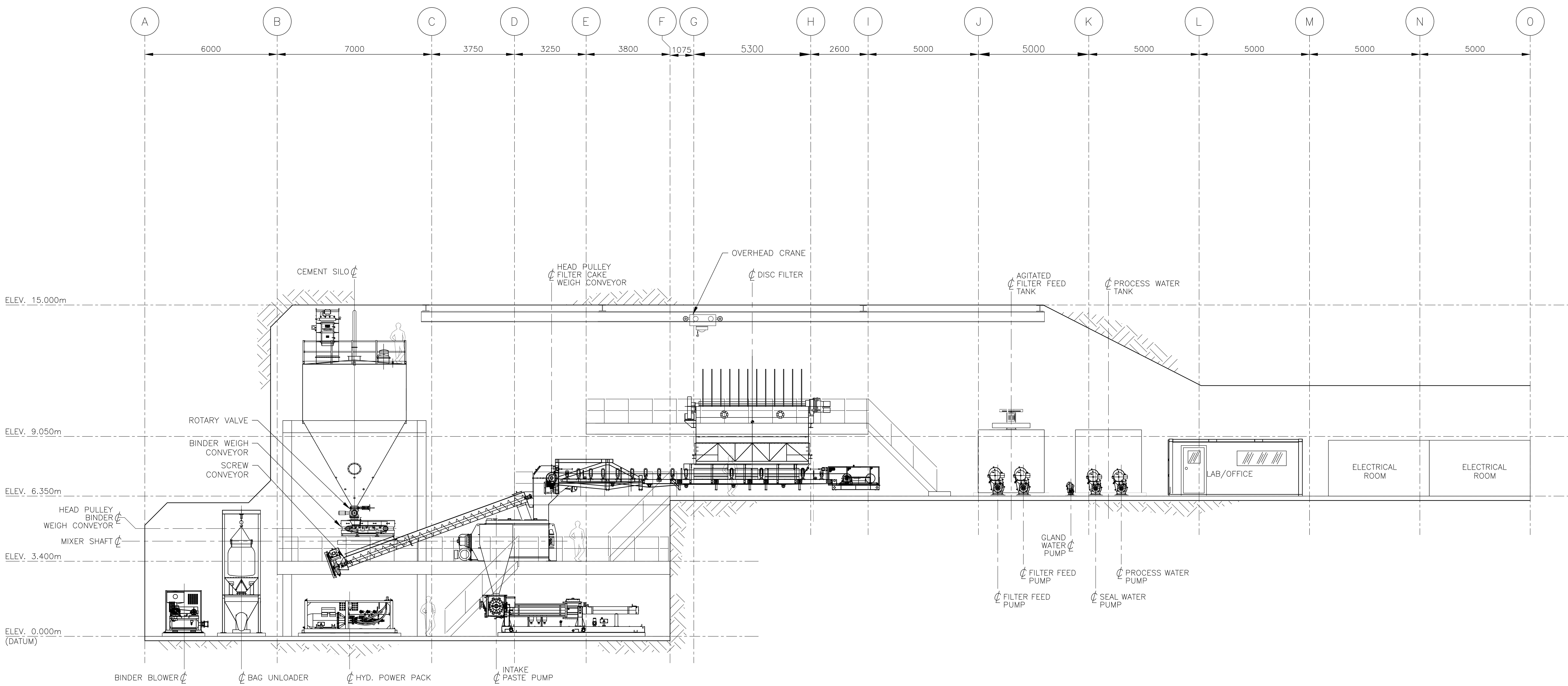
CHIEFTAIN METALS INC. - TULSEQUAH FEASIBILITY STUDY - PASTE PLANT GENERAL ARRANGEMENT PLAN VIEW @ EL. 6.350m

KOVIT PROJECT NO.:	J14-0026
DRAWING NO.:	J14-0026-A-PPP-0102
SCALE: 1:100	REV.: B



A SECTION
0101,0102,0103

—FOR REVIEW—
—NOT FOR CONSTRUCTION—

REFERENCE DRAWINGS

DRAWING NO.	TITLE

REFERENCE DRAWINGS

DRAWING NO.	TITLE

REVISIONS

NO.	DESCRIPTION	DATE	DRAWN	CHECKED	APPROVED	REVIEWED

REVISIONS

NO.	DESCRIPTION	DATE	DRAWN	CHECKED	APPROVED	REVIEWED
A	ISSUED FOR REVIEW	03-OCT-2014	V.R.	A.H.		
B	MODIFIED CEMENT SYSTEM	17-OCT-2014	V.R.	A.H.	F.P.	

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CHIEFTAIN METALS INC. - TULSEQUAH
FEASIBILITY STUDY - PASTE PLANT
GENERAL ARRANGEMENT
SECTION A

KOVIT PROJECT NO.: J14-0026
DRAWING NO.: J14-0026-A-PPP-0104
SCALE: 1:100
REV.: B

16.10 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 12 years, it is assumed that all mobile equipment will be remanufactured to avoid major refurbishment expenditures.

Two boom and single boom diesel/electric jumbos will be used for lateral development and MCF stoping, while production drilling will be completed by diesel/electric LH drills capable of drilling 4” production holes and 2.5” cable bolt holes. 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 t trucks.

The underground equipment fleet is summarized in Table 16.6.

Table 16.6: Mine Equipment Summary

Equipment Type	Quantity
Two Boom Jumbo	1
Single Boom Jumbo	1
Production Drills	2
7 m^3 LHD with Remote	3
3 m^3 LHD with Remote	1
40 tonne Truck	3
Mechanized Bolter	1
Fuel/ Lube Truck	1
Grader	1
Deck and Boom Truck	1
Scissor Lift	2
ANFO Loader	1
Supervisor Vehicles	4
Mechanic Vehicles	2

Source:JDS 2014

16.11 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 four-week-on, two-week-off (4x2) rotation, while technical staff and management will work an eight-day-on, six-day-off (8x6) 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 79 personnel during full production, with 53 on site at one time. 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	8x6	Salary
Mine Captain	1	8x6	Salary
Mine Shift Supervisors	4	4x2	Salary
Production Drillers	3	4x2	Hourly
Jumbo Drillers	3	4x2	Hourly
LHD Operators	6	4x2	Hourly
Truck Drivers	9	4x2	Hourly
Blasters	2	4x2	Hourly
Services	4	4x2	Hourly
Ground Support	3	4x2	Hourly
General and Backfill Labourers	9	4x2	Hourly
Paste Plant Operators	6	4x2	Hourly
Mine Operations Total	**51**		

Source: JDS 2014

Table 16.8: Mine Maintenance Personnel Summary

Position	Quantity	Schedule	Hourly/Salary
Maintenance Shift Supervisors	1	8x6	Salary
Maintenance Foreman	1	8x6	Salary
Maintenance Planner	1	8x6	Salary
Mechanics and Welders	9	4x2	Hourly
Electrician	4	4x2	Hourly
Bit and Lamp Man	2	2x2	Hourly
Mine Maintenance Total	**18**		

Table 16.9: Technical Services Personnel

Position	Quantity	Schedule	Hourly/Salary
Chief Mine Engineer	1	8x6	Salary
Senior Mine Engineer	1	8x6	Salary
Senior Mine Technician	1	8x6	Salary
Surveyor/ Mine Technician	2	2x2	Salary
Chief Geologist	1	8x6	Salary
Mine Geologists	1	8x6	Salary
Geotechnical Technician/Sampler	3	2x2	Salary
Technical Services Total	**10**		

Table 16.10: Total Mine Personnel Summary

Position	Quantity
Mine Operations	51
Mine Maintenance	18
Technical Services	10
Grand Total Mine Personnel	**79**

Source: JDS 2014

16.12 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;
- Stope productivities; and
- Sequence of mining and stope availability.

The underground mine production rate of 1,100 tpd 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 eleven years in addition to the 12 months of preproduction.

16.12.1 Mine Development

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

Preproduction development is scheduled to:

- Development of ore stopes prior to production;
- Provide access for trackless equipment;
- Provide ventilation and emergency egress;
- Establish underground crushing and paste backfill infrastructure; and
- Install mining services.

Two development crews will start working at 60 m level, 120 m level and the new 84 m conveyor portals. Vertical raise development will be done with contract mining crews. During pre-production, the combined Owner and contract mining crews will:

- Slash 60 m (5200) level portal and adit to 5.0 m x 5.0 m size;
- Develop a 60 m level ventilation bypass for ventilation and mine air heating equipment;
- Develop underground infrastructure on 60 m 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 120 m (5400) level portal 120 m level adit to 5.0 m x 5.0 m size;
- Provide sublevel lateral development on the levels between 60 and -60 m; and
- 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 6 of mine production.

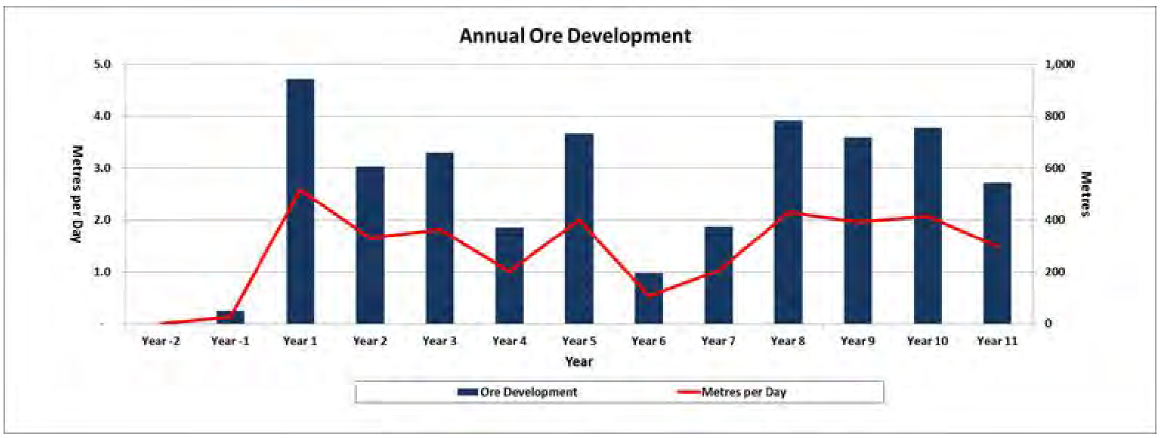
Total underground capital and sustaining lateral waste development is 17,685 m and averages 1,474 m/a or 5.1 m/d over the 11-year project life. Annual waste development is shown in Figure 16.25.

Total ore sublevel development is 6,738 m and averages 608 m/a or 1.7 m/d over the 10-year ore production period. Annual ore development is shown in Figure 16.26.

Figure 16.25: Annual Waste Development

Source: JDS 2014

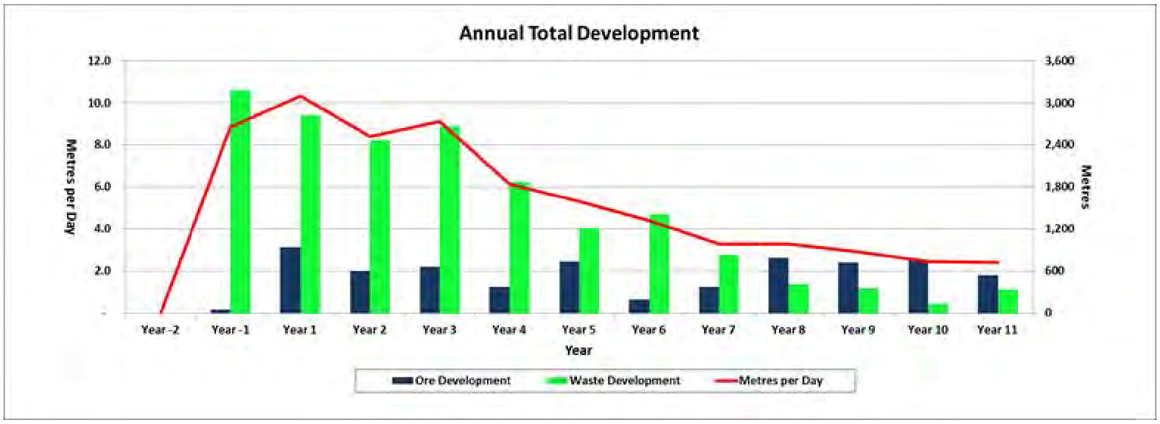
Figure 16.26: Annual Ore Development



Source: JDS 2014

Total ore and waste development is 24,423 m and averages 2,035 m/a or 5.1 m/d over the mine life. Annual total ore and waste development is shown in Figure 16.27.

Figure 16.27: Annual Total Development



Source: JDS 2014

16.12.2 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 408,000 tpa was scheduled, including ore from development and stopes;
- The mine will operate two 10-hour shifts per day, 365 days per year; and
- Provide enough production faces to support a daily mine production rate of 1,100 tpd.

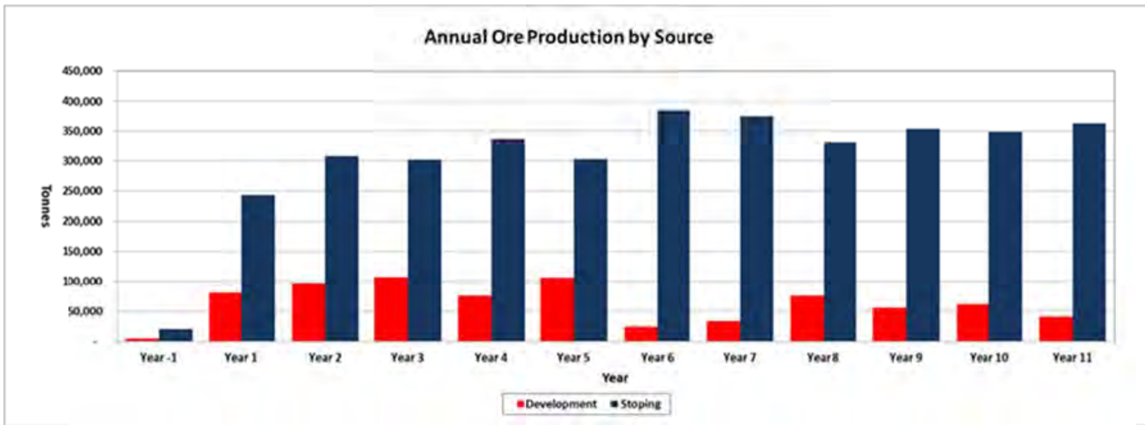
•

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 5.

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 1,100 tpd.

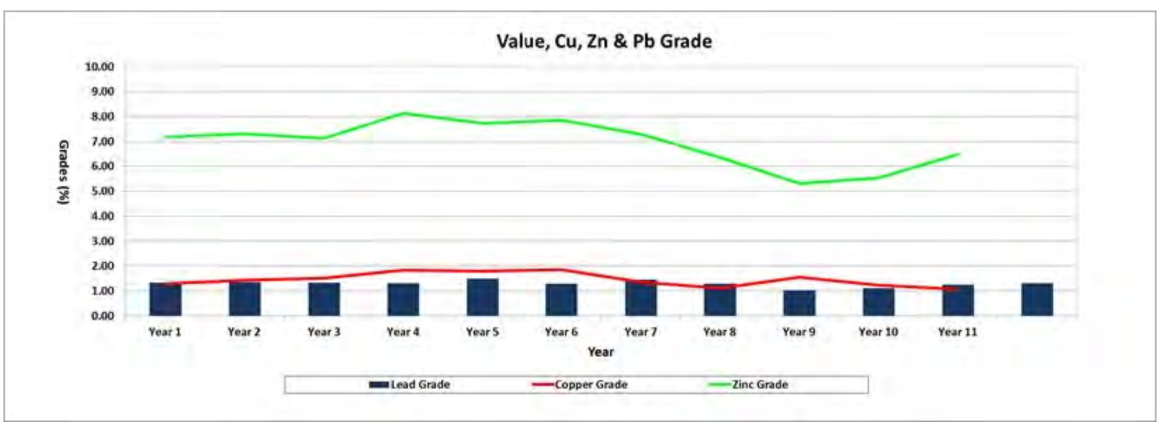
The average mined grades for the eleven-year mine life are 1.46% copper, 6.95% zinc, 1.29% lead, 103.7 g/t silver and 2.85 g/t gold. Annual production by ore source and metal grades are shown in Figures 16.28 to 16.29.

Figure 16.28: Annual Ore Production by Source



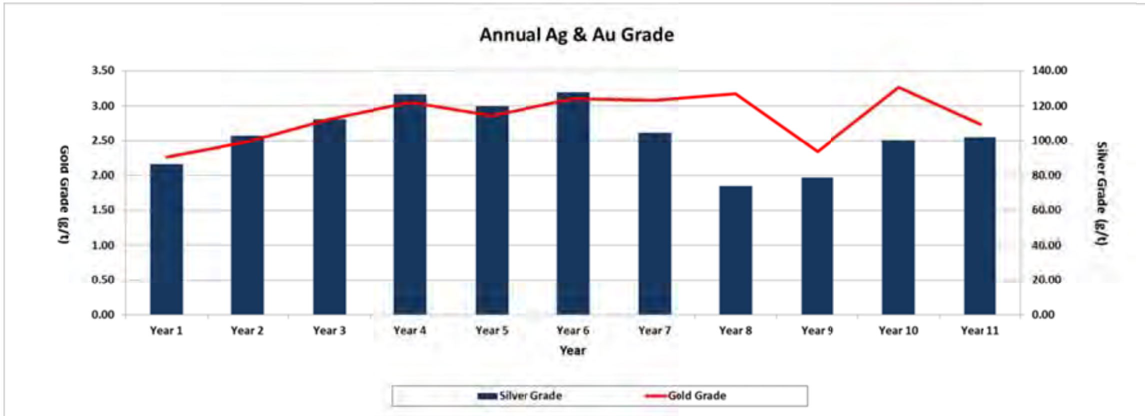
Source: JDS 2014

Figure 16.29: Annual Zinc, Copper & Lead Grades



Source: JDS 2014

Figure 16.30: Annual Gold and Silver Grades



Source: JDS 2014

Detailed mine planning and scheduling has been done monthly for Years -2 to Year 2 but has been summarized annually in this report. 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. Ore, waste and backfill tonnages have been rounded to the nearest thousand.

Table 16.11: Annual Mine Production & Development Schedule

Parameter	Unit	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	Totals
Total Mine Production	**kt**	**24**	**324**	**405**	**409**	**413**	**409**	**409**	**408**	**408**	**410**	**411**	**404**	**4,436**
Daily Production Rate	tpd	-	887	1,111	1,122	1,131	1,118	1,121	1,119	1,117	1,120	1,126	1,107	**1,098**
Gold Grade	g/t	2.91	2.26	2.49	2.82	3.06	2.86	3.11	3.09	3.18	2.34	3.27	2.74	**2.85**
Silver Grade	g/t	106.83	86.68	103.01	112.59	126.84	119.84	128.05	104.60	73.89	78.69	100.44	102.20	**103.72**
Copper Grade	%	1.50	1.28	1.43	1.51	1.83	1.79	1.85	1.36	1.11	1.55	1.22	1.07	**1.46**
Lead Grade	%	1.33	1.35	1.33	1.30	1.48	1.30	1.45	1.28	1.03	1.10	1.25	1.30	**1.29**
Zinc grade	%	9.87	7.18	7.31	7.13	8.12	7.72	7.85	7.27	6.37	5.30	5.54	6.47	**6.95**
Net Smelter Return	$/t	326	257	280	298	337	319	337	300	267	245	281	267	**291**
Total Lateral Development	**m**	3,233	3,772	3,064	3,327	2,242	1,944	1,602	1,204	1,201	1,069	891	877	**24,423**
	m/d	8.9	10.3	8.4	9.1	6.1	5.3	4.4	3.3	3.3	2.9	2.4	2.4	**5.1**
Raise Development	m	20	216	368	348	301	151	150	-	-	-	-	-	**1,554**
Mined Underground Waste	kt	177	171	157	172	115	74	84	50	23	19	8	19	**1,069**
Paste Backfill Placed	kt	-	102	116	209	59	171	132	151	192	189	219	203	**1,743**

17. RECOVERY METHODS

The process design criteria and flowsheets have been developed based on the metallurgical test work results from historical and current test work programs as described in Section 13 using industrial design factors as noted. The test work has shown that the Tulsequah Chief ore can be treated using conventional mineral processing techniques applying differential flotation for the recovery of saleable copper, lead and zinc concentrates.

The plant is envisioned to accept primary crushed ore from an underground storage bin. This feeds to a SAG mill followed by two stages of ball milling to produce a final cyclone overflow product of 80% passing 45 microns at a rate of approximately 1,100 tpd. The plant is planned to operate 24 hours per day for 365 days per year with a plant availability of 90%. The crusher is planned to operate for 16 hours per day.

Designated cyclones from each of the ball mill circuits are equipped with gravity concentrators to recover gravity gold (electrum). The gravity gold concentrates report to intensive cyanide leach and electrowinning circuits to produce doré.

The cyclone overflow from the final stage of grinding is treated in a sequential flotation circuit starting with copper then lead, zinc and finally pyrite.

The copper, lead and zinc concentrates are planned to be dewatered and pressure filtered in designated circuits before storage and transport. The filtered concentrates are handled in 2- tonne bags.

Pyrite concentrate gets stored in the pyrite pond until stopes are available to be filled below the 60 m level. It is sent to the paste plant to be included in the paste fill. Final tailings, pyrite flotation tailings, report to either the underground paste plant or the tailings management facility, TMF.

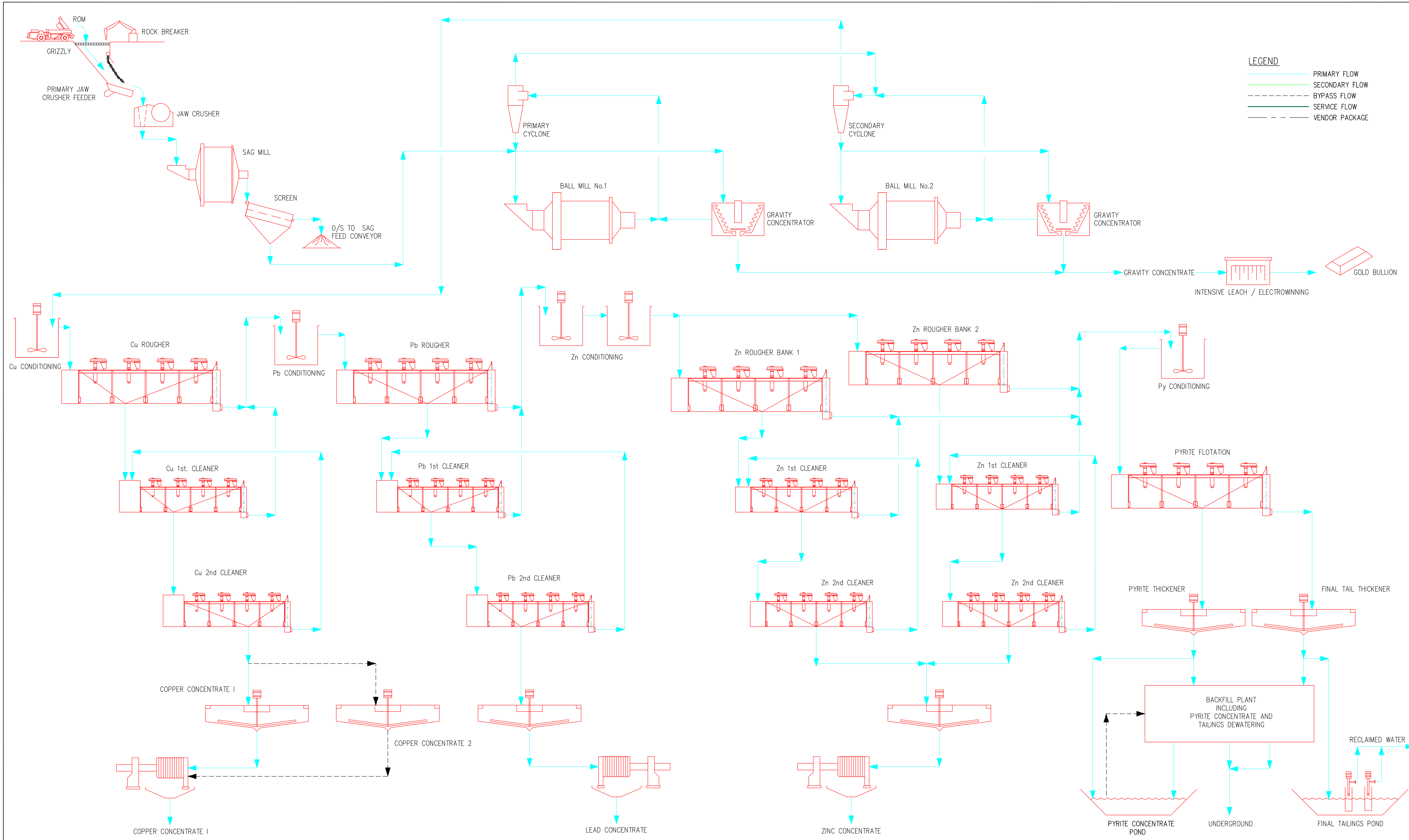
Tailings not required for paste backfill are pumped to the tailings pond. Excess process water gets sent to the effluent treatment plant and then pumped to the plant as fresh make-up water or discharged to the environment. The tailings slurry from the mill will be deposited on the outer perimeter of the TMF.

Reagents are shipped to site in approved containers and properly packed in Department of Transport (DOT) approved packaging. Packaging is in the form of standard drums, totes and bulk bags. The reagents are sent to site on the return barges during the shipping season.

17.1 Introduction

Figure 17.1 (Reference Flowsheet: 1010-09-001) presents a conceptual flowsheet of the processing plant for the Tulsequah Chief Project.

A simplified description of the ore processing at the mine site is summarized in this section with details following in the descriptions of unit operations.



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

SCALE: None	DATE:
DESIGNED: J.D.S.	June 22 2014
DRAWN: B. Wong	June 22 2014
CHECKED:	
APPROVED:	

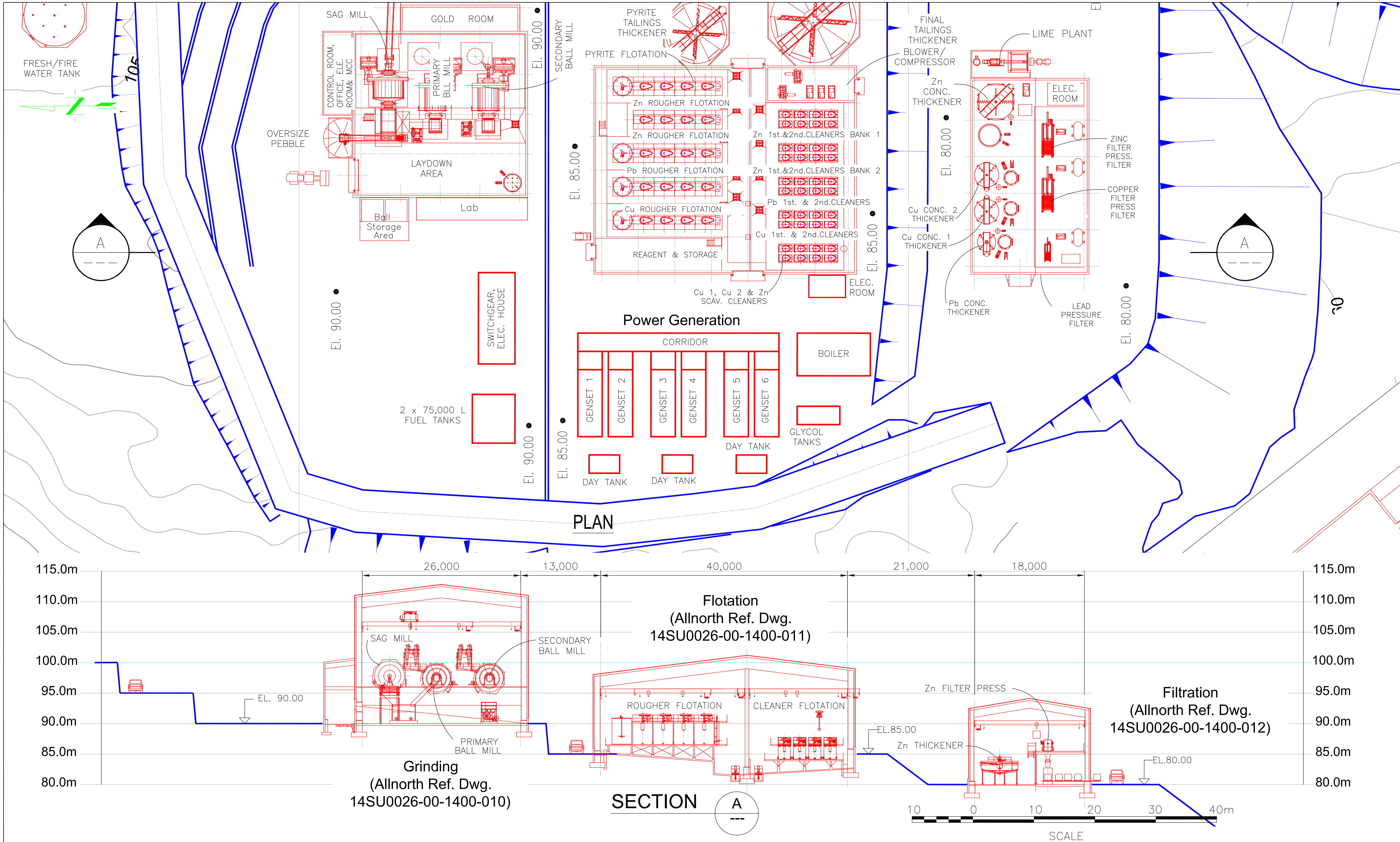
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
SIMPLIFIED FLOW DIAGRAM
PROCESS FLOW DIAGRAM

PROJECT No.	DRAWING No.	REV No.
	1010-09-001	E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D2	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		5/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		27/06/14

REFERENCE DRAWING | REF. DWG No.

TULSEQUAH CHIEF PROJECT

SCALE: | DATE:
DESIGNED: J.D.S. | June 21 2014
DRAWN: B. Wong | June 21 2014
CHECKED:
APPROVED:

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
MIDDLE BENCH (PLANT SITE GENERAL SECTION 1)
GENERAL ARRANGEMENT
PLAN AND SECTION

PROJECT No. | DRAWING No. 1010-10-005 | REV No. F1

17.2 Design Criteria

The process design criteria for the Tulsequah Chief Project has been based on metallurgical test work undertaken by ALS Metallurgy, Burnie Australia, Project T0662 for the 2012 2,000 tpd Feasibility Report conducted by JDS and more recently ALS Project T0897. Representative composites of samples from two ore zones, Upper and Lower, were prepared and subjected to various metallurgical test programs. The average ore head grades and LOM average production rate of 400,405 tpa from the 2014 Mine Plan were used to develop the mass balance and design criteria for the base case scenario. The operating data and ore characteristics are summarized in Table 17.1. The detailed design criteria, mass balance, process equipment list and flowsheets referenced in the following sections can be found in Appendix B.

Table 17.1: Tulsequah Chief Project Mill Feed Grade - Operating Data and Ore Characteristics

Description	Units	Nominal	Design
Crushing Plant Throughput - Nominal	tph	69	
Process Plant Throughput - Nominal	tph	51	
Design Factor – unless otherwise noted	-	1.2	
Ore Solids Density	SG	3.55	
Ore Moisture	% w/w	5	
Head Grade (Average LOM)	%Cu	1.46	
Head Grade (Average LOM)	%Pb	1.29	
Head Grade (Average LOM)	%Zn	6.95	
Head Grade (Average LOM)	g/t Au	2.85	
Head Grade (Average LOM)	g/t Ag	103.72	
Gold - Gravity			
Gold Recovery	g/t	1.17	
	% Au	41	
Silver Recovery	% Ag	0.5	
Copper			
Copper Concentrate Production, hourly	dry tph	3.14	3.77
Copper Concentrate Grade	% Cu	21	
Copper Recovery	% Cu	89	
Gold Recovery	% Au	47	
Silver Recovery	% Ag	78	
Lead			
Lead Concentrate Production, hourly	dry tph	0.71	0.85
Lead Concentrate Grade	% Pb	60	
Lead Recovery	% Pb	65	
Gold Recovery	% Au	3	
Silver Recovery	% Ag	6	
Zinc			
Zinc Concentrate Production, hourly	dry tph	5.29	6.35
Zinc Concentrate Grade	% Zn	60	
Zinc Recovery	% Zn	90	
Total Concentrate Production			
Total Concentrate Production, annually	dry tpa	72,113	86,536
Pyrite			
Pyrite Concentrate Production, hourly	dry tph	16.93	20.31
Pyrite Concentrate Grade	% Py	67	
Pyrite Recovery	% Py	77	
Tailings			
Tailings Production, hourly	dry tph	24.53	29.43

Source: 2014 FS Design Criteria and Mass balance

17.3 Process plant Description

The design criteria, flowsheet and process description for each unit operation is discussed in the following sections.

17.3.1 Primary Crushing and Ore Storage and Reclaim

Reference Flowsheet: 1010-09-002

Table 17.2: Crushing and Ore Storage Design Criteria

Description	Units	Nominal	Design
Maximum Feed Lump Size, Grizzly	mm	500	
Reclaim Rate	t/h	69	82
Primary Crusher			
Crusher type	-	Jaw	
Circuit Configuration	-	Open	
Size		30" x 40"	
Installed Power	kW	110	
Closed Side Setting	mm	70 - 75	
Estimated Feed F80	mm	500	
Estimated Product P80	mm	100	
Fine Ore Surge Bin, Live	t	1,250	

Source: 2014 FS Design Criteria and Mass balance

This area is planned to consist of a grizzly, a 120 t dump pocket, a jaw crusher, a belt feeder and dust collection system. A dump pocket capable of receiving ore from underground trucks is located on the top level of the crushing area. A vibrating grizzly feeder (156010-FDR-013) draws ore from the dump pocket and provides a constant feed of material to the jaw crusher (156010-CRU-001). Crushed ore discharges to the storage bin feed conveyor (156010-CNV-001) located directly below the crusher.

A dust collector (156010-COL-001) collects dust generated in the crushing area. The dust is collected and discharged onto the storage bin feed conveyor through a rotary valve and screw conveyor.

The crushed ore is reclaimed by belt feeders (156020-FDR-001) that feed the SAG mill feed conveyor (251010-CNV-005). A belt scale (251010-SCB-002) on the SAG mill feed conveyor controls the speed of the belt feeder and tonnage to the SAG mill.

17.3.2 Grinding

The grinding circuit, SAG/ball/ball mill, has been adopted from the 2012 FS.

Table 17.3: Grinding Design Parameters

Description	Units	Nominal
Rod Mill Work Index, Wi	kWh/t	8.8
Bond Ball Mill Work Index, Wi	kWh/t	12.9
Bond Abrasion Index, Ai	g	0.0743

SAG MILL

Reference Flowsheet: 1010-09-002

Table 17.4: SAG Mill Grinding Circuit Design Criteria

Description	Units	Nominal
Primary Grinding Circuit - SAG Mill		
Primary Grinding Feeder	-	Belt Conveyor, VSD
Circuit Configuration	-	Open
Mill Type	-	SAG Mill
Number of Mills	#	1
Mill Size – Diameter x Length	m	2.4 x 4.6
Installed Power	kW	448
Product Size, P80	µm	425
SAG Mill Discharge Screen		
Vibrating SAG Discharge Screen	m	2.4 x 6.1
Aperture Size	mm	10

Source: 2014 FS Design Criteria and Mass balance

Reclaimed ore feeds a 448 kW, 4.6 m diameter by 2.4 m long SAG mill (251010-MIL-001) driven by a variable speed motor. The variable speed motor enables the SAG mill to vary power draw for circuit optimization under varying feed material conditions. SAG mill discharge feeds a 2.4 m x 6.1m vibrating screen (251010-SCN-002) with a deck aperture of 10 mm. The screen undersize discharges into the primary cyclones feed pumpbox and SAG discharge screen oversize will feed a stockpile via the SAG mill coarse material discharge conveyor (251010-CNV-006). The oversize material is reclaimed from the stockpile and feeds back onto the SAG mill feed chute. Process water is added directly to the SAG mill feed chute and as wash water to the SAG mill vibrating screen oversize to maintain a target slurry density in the SAG mill.

Primary and Secondary Ball Mills No. 1 and 2

Reference Flowsheet: 1010-09-003

Table 17.5: Ball Mill Grinding Circuits Design Criteria

Description	Units	Nominal	Design
Ball Mill No. 1			
Type	-	Ball Mill	
Circuit Configuration	-	Closed	
Number of mills	#	1	
Mill Size – Diameter x Length	m x m	2.9 x 4.9	
Installed Power	kW	448	
Product Size, P80	µm	95	
Ball Mill No. 2			
Mill Type	-	Ball Mill	
Circuit Configuration	-	Closed	
Number of mills	#	1	
Mill Size – Diameter x Length r	m x m	2.9 x 4.9	
Total Power Installed	kW	448	
Product Size, P80	µm	45	

Source: 2014 FS Design Criteria and Mass balance

SAG discharge screen undersize slurry is collected in the primary cyclones feed pump box (251010-PBX-001) with the primary ball mill discharge and gravity concentrator No. 1 (252040-CNC-001) tailings and then pumped to the primary cyclones (251010-CYC-001) for size classification. The primary cyclopac consists of eight, five operating, 250 mm diameter cyclones. The underflow from the cyclopac is fed to ball mill No. 1(251010-MIL-002) and gravity concentrator No. 1.Cyclone overflow reports to the final grinding circuit for further size reduction.

The overflow from the primary cyclopac flows by gravity to the secondary cyclones feed pump box (251010-PBX-002) and combined with secondary ball mill No. 2 (251010-MIL-003) discharge and the gravity concentrator No. 2 (252040-CNC-002) tailings. Slurry from the secondary cyclones feed pump box are pumped to the secondary cyclopac of eight, five operating, 250 mm diameter cyclones, with designated cyclones feeding ball mill No. 2 and gravity concentrator No. 2.

The cyclone overflow flows by gravity to the copper flotation circuit. The target particle size P80 of the secondary cyclopac overflow at 30% solids will be 45 microns.

Process water addition to the primary and secondary cyclones feed pump box is controlled to maintain pump box level and/or cyclone feed density to achieve a consistent grind and slurry density to optimize mill operations.

Gravity Gold Circuit

Reference Flowsheet: 1010-09-003

Table 17.6: Gravity Gold Recovery Design Criteria

Description	Units	Nominal	Design
General – Concentrator No. 1 and No. 2			
Percent of New Feed to Gravity	%	100	
Vibrating Trash Screen Size	m x m	0.915 x 2.439	
Aperture Size	mm	2	
Concentrator Type	-	Centrifugal	
Feed	-	Cyclone Underflow	
Concentrate Mass (20 min/batch)	kg / batch	19	
	dmt / day	1.4	
	kg/h	57	
Recovery	%	20.5	
Intensive Leaching and Refining			
Cyanide Solution Strength	%w/w	25-30	
Intensive Leaching Batch	batch/day	1 (16 hours/batch)	
Intensive Leaching Rate	t/batch	2.7	
Gold Recovery From Pregnant Solution, PLS	-	Electrowinning	
Smelting Process Batch	batch/week	2	
Smelting Process Rate	kg/batch	4.5	

Source: FLSmidth Knelson modelling and design, 2014 FS Design Criteria and Mass balance

Slurry from designated cyclones flows by gravity over the concentrator trash screens, (252040-SCN-003/4) with the oversize reporting to the cyclone feed pump box and the undersize to a 20 inch gravity concentrators, (252040-CNC-001/2). The gravity concentrators each receive 100% new feed to achieve the combined target of 41% gold recovery.

The gravity gold concentrate discharges to a secure area containing an intensive leach reactor. The pregnant solution for the leach circuit is pumped to an electrowinning circuit to produce a doré. The smelter is expected to run 2 batches per week producing 4.54 kg/batch.

The residue from the leach circuit is pumped to the grinding circuit. Barren solution is used in the cyanide reagent mixing tank as make-up solution and then added as flotation or leach reagent.

17.4 Flotation

The flotation circuits are sized based on test work from ALS Project T0662 data. The laboratory retention time required for effective rougher and cleaner flotation was scaled up by two and a half, and four times respectively. The launders for the flotation cell design was based on lip loading of 200 kg/m/hr for the roughers and 100 kg/m/hr for the cleaners.

17.4.1 Copper Processing

This section describes the copper processing circuit which includes rougher and cleaner flotation, and concentrate dewatering and handling. The cleaner circuit includes two stages of cleaning to produce one concentrate. Two additional banks of four cleaner flotation cells and an additional copper thickener have been included in the layout and capital cost. The equipment will allow the flexibility to produce low and high arsenic concentrates.

Copper Rougher Flotation

Reference Flowsheet: 1010-09-004

Table 17.7: Copper Rougher Flotation Circuit Design Criteria

Description	Units	Nominal	Design
Cell Type	-	Conventional	
Number of Banks	#	1	
Number of Cells per Bank	#	4	4
Cell Volume	m^3	38	38
Concentrate Mass Pull	%	10.5	
Slurry pH	pH	7.2	

Source: ALS Project T0662 Test No. T26-28, 2014 FS Design Criteria and Mass balance

Secondary cyclone overflow flows by gravity to the copper conditioning tank. SMBS, 9810 and MIBC are added prior to rougher flotation to assist the flotation chemistry. The slurry then gravitates to the rougher flotation circuit which consists of one bank of four 38 m^3 cells (252050-CEL-002 to 005). The cells uses a combination of reagents, agitation, and air to recover the copper sulphides for further processing in the cleaner flotation circuit.

Rougher concentrate froth is collected in a common launder which feeds the copper cleaner flotation circuit. The metals content of the rougher feed, rougher concentrate, rougher tailings, second cleaner concentrate, and first cleaner tailings are determined by inline samples that collect samples for metallurgical analysis. Copper rougher tailings and first cleaner tailings are combined and pumped to the lead flotation circuit conditioning tank.

Copper Cleaner Flotation

Reference Flowsheet: 1010-09-004

Table 17.8: Copper Cleaner Circuit Design Criteria

Description		Units	Nominal	Design
Copper 1st Cleaner Flotation				
Cell Type		-	Conventional	
Number of Banks		#	1	
Number of Cells per Bank		#	4	4
Cell Volume		m^3	5.8	5.8
Concentrate Mass Pull		%	8.3	
Copper 2nd Cleaner Flotation				
Cell Type		-	Conventional	
Number of Banks		#	1	
Number of Cells per Bank		#	4	4
Cell Volume		m^3	5.8	5.8
Concentrate Mass Pull		%	6.2	

Source: ALS Project T0662 Test No. T26-28, 2014 FS Design Criteria and Mass balance

The copper cleaner circuit are comprised of four 5.8 m^3 1st cleaner cells (252050-CEL-006 to 009) and four 5.8 m^3 2nd cleaner cells (252050-CEL-010 to 013). Rougher concentrate feeds the 1st cleaner cells. The 1st cleaner concentrate is collected in a common launder that flows by gravity to the 2nd cleaner flotation circuit. The 2nd cleaner concentrate flows by gravity to one of two copper thickeners. The 2nd cleaner flotation tailings are pumped back to the previous stage of flotation and 1st cleaner tailings are directed to the lead flotation circuit. SMBS, $ZnSO_4$ and MIBC are added to aid in flotation.

Copper Concentrate Dewatering and Storage

Reference Flowsheet: 1010-09-008

Table 17.9: Copper Dewatering and Filtration Circuit Design Criteria

Description	Units	Nominal	Design
Thickener Type	-	High Rate	
Number of Thickeners	-	2	
Thickener Underflow Density	%	60	
Thickener Diameter	m	4	
Filter Type	-	Pressure	
Number of Filters	#	1	
Target Concentrate Moisture Content	%	8	

Source: Information is based Vendor data and recommendations

The concentrate dewatering circuit will remove water from the concentrate slurry for shipping of the concentrate as damp filter cake.

The thickening operation concentrates suspended solids by gravity settling. Flocculant is added as a dilute solution to the thickener to agglomerate fine solid particles which assist the settling of the fine particles. Settled solids are raked to the center discharge cone where the thickened slurry is withdrawn using one of two centrifugal pumps (253010-PSL-022/023) for transfer to the concentrate stock tank (253010-TNK-023). The copper concentrate stock tank provides 8 hours of surge capacity between the 4 m diameter copper concentrate 1 thickener (253010-THK-001) and copper concentrate pressure filter (253010-FIL-003). The concentrate stock tank will be agitated to prevent sanding out of solids. Two centrifugal slurry pumps (253010-PSL-040/041) feeds thickened slurry from the concentrate stock tank to the concentrate filter.

The thickener overflow solution is pumped (253010-PSL-024) to the copper cleaner flotation circuit for use as process dilution water and as launder sprays water. Excess water from the concentrate thickeners is collected in the copper concentrate 2 thickener overflow standpipe (253010-TNK-021) and pumped to the effluent treatment plant. An identical thickening and stock tank circuit feeds the 2nd copper concentrate to the common filter and bagging system.

A horizontal pressure filter is used for final concentrate dewatering. A pressure filter is a series of cloth covered plates on a rack. Concentrate is pumped into the chambers between the plates through channels. The plates are then squeezed using a hydraulic piston. The filter then undergoes a blow operation to push out any remaining free water.

The piston then releases and the plates separate allowing concentrate cake to freely fall down through bomb-bay doors to the conveyor (253010-CNV-007) below. The filter then undergoes a wash cycle to remove any remaining solids stuck to the plates.

Filtrate recovered from the squeezing process flows by gravity back to the copper concentrate thickener.

Copper concentrate discharged from the filter dumps onto a conveyor that feeds the bagging system. The bagging system discharges moist concentrate cake into a bagging chute feeding 2-tonne bags. The bags are weighed and tagged before being stored in a designated area near the barge landing.

17.4.2 Lead Processing

This section describes the lead processing circuit and includes rougher and cleaner flotation and concentrate dewatering and handling. Copper flotation tailings are pumped to the lead flotation conditioning tank (252050-TNK-012) followed by a bank of four 38 m^3 rougher cells. The rougher concentrate feeds the first of two stages of cleaning incorporating four 5.6 m^3 cells each. Lead cleaner concentrate flows by gravity to a dedicated thickener, followed by filtering for storage in two tonne bags.

17.4.3 Lead Rougher Flotation

Reference Flowsheet: 1010-09-005

Table 17.10: Lead Rougher Circuit Design Criteria

Description	Units	Nominal	Design
Cell Type	-	Conventional	
Number of Banks	#	1	
Number of Cells per Bank	#	4	4
Cell Volume	m^3	38	38
Concentrate Mass Pull	%	4.9	
Slurry pH	pH	9.5	

Source: ALS Project T0662 Test No. T26-28, 2014 FS Design Criteria and Mass balance

Copper tailings are pumped to the lead conditioning tank. Lime, NaCN, ZnSO4, Cytec 3418a, and MIBC are added prior to rougher flotation. The slurry then gravitates to the rougher flotation circuit which consists of one bank of four 38 m^3 cells. The cells (252050-CEL-022 to 025) use a combination of reagents, agitation, and air to recover the lead sulphides for further processing.

Rougher concentrate froth is collected in a common launder which feeds the lead cleaner circuit. The metals content of the rougher feed (copper tailings), cleaner concentrate, and lead tailings are determined by inline samples that are collected for metallurgical analysis. Lead rougher tailings are delivered to the zinc flotation circuit conditioning tank.

17.4.4 Lead Cleaner Flotation

Reference Flowsheet: 1010-09-005

Table 17.11: Lead Cleaner Circuit Design Criteria

Description	Units	Nominal	Design
Lead 1st Cleaner Flotation			
Cell Type	-	Conventional	
Number of Banks	#	1	
Number of Cells per Bank	#	4	4
Cell Volume	m^3	5.8	5.8
Concentrate Mass Pull	%	3.1	
Lead 2nd Cleaner Flotation			
Cell Type	-	Conventional	
Number of Banks	#	1	
Number of Cells per Bank	#	4	4
Cell Volume	m^3	5.8	5.8
Concentrate Mass Pull	%	1.4	

Source: ALS Project T0662 Test No. T26-28, 2014 FS Design Criteria and Mass balance

The lead cleaner circuit is comprised of four 5.8 m^3 1st cleaner cells (252050-CEL-026 to 029) and four 5.8 m^3 2nd cleaner cells (252050-CEL-030 to 033). Rougher concentrate feeds the 1st cleaner cells. The 1st cleaner concentrate is collected in a common launder that flows by gravity to the 2nd cleaner flotation circuit. The 2nd cleaner concentrate flows by gravity to the lead thickener. The staged cleaner flotation tailings is pumped back to the previous stage of flotation except for the 1st cleaner tailing that is directed to the zinc flotation circuit. Lime, NaCN, ZnSO4, Cytec 3418a, and MIBC are added to enable flotation.

17.4.5 Lead Concentrate Dewatering and Storage

Reference Flowsheet: 1010-09-008

Table 17.12: Lead Dewatering and Filtration Circuits Design Criteria

Description	Units	Nominal	Design
Thickener Type	-	High Rate	
Number of Thickeners	-	1	
Thickener Underflow Density	%	60	
Thickener Diameter	m	3	
Filter Type	-	Pressure	
Number of Filters	#	1	
Target Concentrate Moisture Content	%	8	

Source: Information is based on Vendor data and recommendations

A 3 m diameter thickener is used to thicken lead concentrates to an underflow density of 60% solids. The thickened slurry is withdrawn using one of two centrifugal pumps (253010-PSL-030/031) for transfer to the agitated concentrate stock tank (253010-TNK-025). The lead concentrate stock tank provides eight hours of surge capacity between the 3 m diameter lead concentrate thickener (253010-THK-003) and lead concentrate pressure filter (253010-FIL-004). A series of centrifugal slurry pump (253010-PSL-044/045) feeds thickened slurry from the concentrate stock tank to the concentrate filter. The thickener overflow solution is pumped (253010-PSL-084) to the lead cleaner circuit for process dilution water and as launder spray water. Excess thickener overflow water that is not utilized in the flotation circuit flows to the effluent treatment plant via the copper concentrate 2 thickener overflow standpipe.

A horizontal pressure filter is used for final concentrate dewatering. Lead concentrate cake will drop from the filter onto a conveyor that feeds the bagging system. Lead concentrate filter cake is stored in 2-tonne bags. The lead bagging area is enclosed within the dewatering building with dust control to prevent contamination to other areas.

Filtrate recovered from the squeezing process gravitates to the lead concentrate thickener.

17.4.6 Zinc Processing

Lead circuit tailings are pumped to the first of two zinc-conditioning tanks and then they flow to two banks of four 38 m^3 cells. The rougher concentrate is fed to two cleaning stages incorporating eight 5.8 m^3 cells per bank. Zinc cleaner concentrate gravitates to a dedicated thickener and filter.

Zinc Rougher Flotation

Reference Flowsheet: 1010-09-006

Table 17.13: Zinc Rougher Circuit Design Criteria

Description	Units	Nominal	Design
Cell Type	-	Conventional	
Number of Banks	#	1	
Number of Cells per Bank	#	4	4
Cell Volume	m^3	38	38
Concentrate Mass Pull	%	15.2	
Slurry pH	pH	10.5	

Source: ALS Project T0662 Test No. T26-28, 2014 FS Design Criteria and Mass balance

Lead flotation tailings are pumped to the first of two zinc conditioning tank where lime is added. In the 2nd conditioning tank CuSO4, Cytec 7021 and MIBC is added prior to rougher flotation. Reagents are added in the conditioning tanks to increase and control the pH at 10.5, suppress lead and pyrite, and activate sphalerite. Conditioned slurry feeds the rougher flotation circuit which consists of two banks of four 38 m^3 cells. The cells (252050-CEL-034 to 037 and 038 to 041) use a combination of reagents, agitation, and air to recover the zinc sulphides. Rougher concentrate froth is collected in a common launder which feeds the two banks of first and second cleaner cells.

Zinc Cleaner Flotation

Reference Flowsheet: 1010-09-006

Table 17.14: Zinc Cleaner Circuit Design Criteria

Description		Units	Nominal	Design
Zinc 1st Cleaner Flotation				
Cell Type		-	Conventional	
Number of Banks		#	1	
Number of Cells per Bank		#	4	4
Cell Volume		m^3	5.8	5.8
Concentrate Mass Pull		%	12.8	
Zinc 2nd Cleaner Flotation				
Cell Type		-	Conventional	
Number of Banks		#	1	
Number of Cells per Bank		#	4	4
Cell Volume		m^3	5.8	5.8
Concentrate Mass Pull		%	10.4	

Source: ALS Project T0662 Test No. T26-28: 2014 FS Design Criteria and Mass balance

The cleaner circuit is comprised of two banks of four 5.8 m^3 1st cleaner cells (252050-CEL-042 to 045 and 046 to 049) followed by two banks of four 5.8 m^3 2nd cleaner cells. Concentrate from each bank of the rougher flotation cells feed the 1st cleaner cells. The 1st cleaner concentrates are collected in common launders that gravitate to the zinc 2nd cleaner cells (252050-CEL-050 to 053 and 054 to 057). Concentrate from the 2nd cleaner cells in bank 1 and 2 gravitate to the concentrate thickener (4210-TH-004). The 1st cleaner tailings combine with the rougher tailings as pyrite flotation circuit feed and 2nd cleaner flotation tailings are pumped back to previous stage of flotation.

Zinc Concentrate Dewatering and Storage

Reference Flowsheet: 1010-09-009

Table 17.15: Zinc Dewatering and Filtration Design Criteria

Description	Units	Nominal	Design
Thickener Type	-	High Rate	
Number of Thickeners	-	1	
Thickener Underflow Density	%	60	
Thickener Diameter	m	6.5	
Filter Type	-	Pressure	
Number of Filters	#	1	
Target Concentrate Moisture Content	%	8	

Source: Information is based on recent Vendor data and recommendations

A 6 ½ m diameter thickener is used to thicken zinc concentrates to an underflow density of 60% solids. The thickened slurry is withdrawn using one of two centrifugal pumps (253010-PSL-034/035) for transfer to the eight hour capacity agitated concentrate stock tank (253010-TNK-026). The zinc concentrate stock tank provides surge capacity between the zinc concentrate thickener (253010-THK-004) and zinc concentrate pressure filter (253010-FIL-005). A series of centrifugal slurry pump (253010-PSL-046/047) feeds thickened slurry from the concentrate stock tank to the concentrate filter. The thickener overflow solution is pumped (253010-PSL-032) to the zinc cleaner circuit for process dilution water and as launder spray water. Excess thickener overflow water that is not utilized in the flotation circuit flows to the effluent treatment plant via the copper concentrate 2 thickener overflow standpipe.

A horizontal pressure filter is used for final concentrate dewatering. Concentrate cake from the filter falls down through bomb-bay doors to a conveyor. The conveyor feeds a bagging system for concentrate storage in 2-tonne bags. Filtrate recovered from the squeezing process flows by gravity to the zinc concentrate thickener.

17.5 Concentrate Handling

Copper, Lead and Zinc concentrates are handled in a similar manner; for example, concentrate will have a dedicated thickener and filter and load-out areas. Copper, lead and zinc concentrates are stored in 2-tonne bags. The bags are weighed and tagged before being loaded by forklift onto flatbed trucks for transport to the planned storage area at the barge landing for further concentrate handling. The concentrate bags are loaded into containers and then shipped during the barge season to markets in Asia.

17.6 Pyrite Flotation

Reference Flowsheet: 1010-09-007

Table 17.16: Pyrite Rougher Flotation Circuit Design Criteria

Description	Units	Nominal	Design
Cell Type	-	Conventional	
Number of Banks	#	1	
Number of Cells per Bank	#	4	4
Cell Volume	m^3	38	38
Rougher Concentrate Mass Pull	%	33	
Slurry pH	pH	8.9	

Source: ALS Project T0662 Locked Cycle Tests LC04 to 06 and T0873 Test No. 32/33, 2014 FS Design Criteria and Mass balance

Zinc rougher and zinc first cleaner tailings are fed to the pyrite flotation conditioning tank followed by a bank of four 38 m^3 rougher cells. The rougher concentrate gravitates to the pyrite thickener and the pyrite rougher tailings to the final tailings thickener, PAX and MIBC are added to enable pyrite flotation.

17.6.1 Pyrite Concentrate Dewatering

Reference Flowsheet: 1010-09-009

Table 17.17: Pyrite Dewatering Design Criteria

Description	Units	Nominal
Pyrite Concentrate		
Thickener Type	-	High Rate
Number of Thickeners	-	1
Thickener Underflow Density	%	50
Thickener Diameter	m	12

Source: Information is based on Vendor data and recommendations

A 12 m diameter thickener is used to thicken pyrite concentrate to an underflow density of 50% solids. The thickened slurry is withdrawn using two centrifugal pumps (253010-PSL-036/037) for transfer to the pyrite pond during the first two years of operation. The pyrite concentrate reports to the paste plant after year 2. The thickener overflow solution is pumped (253010-PSL-084) to the pyrite flotation circuit for process dilution water and as launder spray water. Excess thickener overflow water that is not utilized in the flotation circuit will flow to the effluent treatment plant.

17.7 Reagents Handling

Reference Flowsheet: 1010-09-11/12/13

Reagents consumed within the flotation circuits are prepared and distributed via the reagent handling circuits. This facility includes mixing and storage for PAX, Sodium Cyanide, A9810, A7021, A3418, CuSO4, SMBS, Zinc Sulphate, MIBC, Flocculant and lime. All reagent areas are bermed with sump pumps which transfer spills to the pyrite thickener, with the exception of the Flocculant. Flocculant spills are returned back to the storage tank. The reagents are mixed, stored and then delivered to the flotation and dewatering circuits with dosage controlled by flow meters and manual control valves. The storage tanks a sized for a minimum of one day. The reagents are delivered in powder form with the exception of MIBC, A7021, A3418, A9810 and antiscalant which are delivered as solution. In addition to the reagents listed above Caustic, Leach Aid (31% HCL) and NaOCL are used in the gravity recovery circuit at dosages of 11, 2, and 8 tpa, respectively.

The following table presents the estimated annual consumption for each reagent.

Table 17.18: Estimated Annual Reagent Consumption

Reagents	Use	Annual Consumption (tonnes)	Delivered Form
PAX	Pyrite Sulphide Collector	62	850 kg box/bag
MIBC	Frother	54	1 t tote
Sodium Cyanide, NaCN	Zinc Sulphide Depressant, Gold Recovery	192	1 t box
A9810	Copper Sulphide Collector	3	1 t tote
A7021	Zinc Sulphide Collector	6	1 t tote
A3418	Lead Sulphide Collector	3	1 t tote
Copper Sulphate	Zinc Sulphide Activator	204	1 t bags
SMBS	Zinc Sulphide Depressant	643	1 t bags
Lime	pH Modifier	316	1 t bags
Zinc Sulphate,$ZnSO_4.7H_2O$	Iron Sulphide Depressant	112	1 t bags
Flocculant	Fine Particle Agglomeration	70	25 kg bag
Antiscalant	Scale Inhibitor	32	220 kg drum

Source: ALS Project T0662 Locked Cycle Tests LC04 to 06 and T0873 Test No. 32/33, 2014 FS Design Criteria and Mass balance

17.7.1 PAX (Potassium Amyl Xanthate) - Collector

PAX is used as a flotation collector in the pyrite circuit. It is delivered to the plant in the form of 850 kg bags of dry solid product. The bags are lifted using the reagent area hoist onto a hopper (254010-HPR-005). The solids are discharge into an agitated mixing tank (254010-TNK-034). At the agitated tank, solids are mixed with fresh water to a solution of 10% by weight of the dissolved product. From the mixing tank, the solution is discharged by gravity to a 700 mm square storage tank (254010-TNK-035) that stores the solution for distribution in the plant.

At the PAX storage tank outlet, pumps (254010-PMT-016/017) transfer the solution to a supply loop. The supply loop will deliver PAX solution as required directly into the pyrite conditioning tank.

17.7.2 MIBC - Frother

The frother, MIBC, is planned to be used as a flotation froth stabilizer. Frothers strengthen bubbles in flotation cells, enabling them to support the load of the activated mineral particles. The ready to use reagent will be transported to site in 1-tonne totes.

MIBC totes are stacked 2 high with the top tote draining into the bottom tote and pumped by metering pumps (254010-PMT-011 to 015) to a supply each of the flotation circuits. The MIBC is supplied at 100% strength.

17.7.3 Sodium Cyanide, NaCN - Zinc Sulphide Depressant and Gold Recovery

NaCN is used as a flotation reagent in the lead circuit. NaCN helps to depress the sphalerite from floating in the lead flotation circuit. It is delivered to the plant in 850 kg bags of dry pellets. The bags is lifted using a crane onto a hopper (254010-HPR-006). The pellets are discharged into an agitated mixing tank (254010-TNK-036). The pellets are mixed with fresh water to form a solution of 10% by weight of the dissolved product in the agitated tank. From the mixing tank, the solution is discharged by gravity to a 730 mm square storage tank (254010-TNK-037) that stores the solution for distribution throughout the plant.

At the NaCN storage tank outlet, a pump (254010-PMT-016/017) transfers the solution to a supply loop. The supply loop delivers NaCN solution as required directly to lead conditioning tank and first cleaner flotation as well as the gold recovery circuit.

17.7.4 A9810 - Collector

A9810 will also be used as a copper sulphide collector in the copper rougher circuit. It is delivered to the plant in the form of 1 tonne totes. The totes are stacked one on top of the other with the top tote flowing into the bottom tote. The top tote is replaced as required. A9810 will be metered (254010-PMT-031/032) directly to the copper rougher conditioning tank from the bottom tote at 100% solution with no dilution.

17.7.5 SMBS - Depressant

SMBS is also envisioned to be used as a zinc and pyrite depressant in the copper circuit. It is delivered to the plant in the form of 1-tonne bags of dry solid product. The bags are lifted using the reagent area hoist onto a hopper (254010-HPR-011). The solids are discharged into an agitated mixing tank (254010-TNK-038). At the agitated tank, solids are mixed with fresh water to a solution of 20% by weight of the dissolved product. From the mixing tank, the solution gravitates to a 700 mm square storage tank (254010-TNK-039) that stores the solution for distribution to the copper flotation circuits.

17.7.6 A7021 - Collector

A7021 are used as a zinc sulphide collector in the zinc rougher and cleaner circuits. It is delivered to the plant in the form of 1-tonne totes. The totes are stacked one on top of the other with the top tote flowing into the bottom tote. The top tote is replaced as required. A9810 is metered (254010-PMT-024/025) directly to the zinc rougher conditioning tank and the 2nd cleaner flotation circuit from the bottom tote at 100% solution with no dilution.

17.7.7 A3418 - Collector

A3418 is used as a lead sulphide collector in the lead rougher and cleaner circuits. It is delivered to the plant in the form of 1-tonne totes. The totes are stacked one on top of the other with the top tote flowing into the bottom tote. The top tote is replaced as required. A3418 is metered (254010-PMT-027/028) directly to the lead rougher conditioning tank and the lead 1st cleaner flotation circuit from the bottom tote at 100% solution with no dilution.

17.7.8 Copper Sulphate, CuSO4 - Activator

$CuSO_4$ will be used as an activator in the zinc circuit. It is delivered to the plant in the form of 1-tonne bags of dry solid product. The copper sulphate is added to the zinc flotation conditioning tank. The bags are lifted using the reagent area hoist onto a hopper (254010-HPR-004). The solids are discharged into an agitated mixing tank (254010-TNK-029). At the agitated tank, solids are mixed with fresh water to a solution of 10% by weight of dissolved product. From the mixing tank, the solution is discharged by gravity to a 730 mm square storage tank (254010-TNK-036) that stores the solution for distribution throughout the plant.

At the $CuSO_4$ storage tank outlet, a pump (254010-PMT-007/008) transfers the solution to a supply loop. The CuSO4 is delivered to zinc conditioning tank No. 2.

17.7.9 Zinc Sulphate, ZnSO4 - Depressant

$ZnSO_4$ is used as a zinc sulphide depressant in the copper and lead circuits. It is be delivered to the plant in the form of 1-tonne bags of dry solid product. The bags are lifted using the flotation aisle crane onto a hopper (254010-HPR-003). The solids are discharged into an agitated mixing tank (254010-TNK-031). At the agitated tank solids are mixed with fresh water to a solution of 10% by weight of dissolved product. From the mixing tank, the solution are discharged by gravity to a 730 mm square storage tank (254010-TNK-032) that stores the solution for distribution throughout the plant.

At the ZnSO4 storage tank outlet, a pump (254010-PMT-007/008) transfers the solution to a supply loop. The ZnSO4 is delivered to the copper 1st cleaner circuit, lead rougher conditioning tank and lead 1st cleaner circuit.

17.7.10 Flocculant - Agglomeration

Flocculant is received in 25 kg bags and is prepared by using a vendor supplied mixing system. Bags of solid product are loaded into a hopper from which the particles are slowly fed into the system via an educator to generate a concentration of 0.5% into a mix tank (254010-TNK-027) at a concentration of 0.5% by weight. From the mix tank the Flocculant will gravitate to a storage tank (254010-TNK-028). A gravity line delivers the Flocculant to six metering pumps (254010-PMT-001 to006) that discharge into each of the thickeners. Process water is added to dilute Flocculant to the concentrate thickeners at mix strength of 0.25%. The Flocculant dosage has been estimated pending further test work.

17.7.11 Lime - pH Modifier

Lime is used in the lead and zinc circuits for pH control. It is delivered to the plant in the form of 1-tonne bags of dry solid product. The bags are lifted using the reagent area hoists onto a hopper (254010-HPR-011). The solids are discharged into an agitated mixing tank (254010-TNK-048). At the agitated tank, solids are mixed with fresh water to a solution of 20% by weight of dissolved product. From the mixing tank, the solution is pumped (254010-PMT-059/060) to a supply loop. The lime is delivered to the lead conditioning tank, lead 1st and 2nd cleaner circuits, zinc conditioning tank and zinc first cleaner circuit.

17.7.12 Test Reagent

This system is used to prepare and distribute other collectors to test for potential improvements of the mineral recovery in the flotation circuit. It is delivered to the plant in form of bags. The bags are lifted using the flotation aisle crane onto a hopper (4960-HP-018) the solids discharge into an agitated, mixing tank (254010-TNK-029). From the mixing tank the solution gravitates to a storage tank (254010-TNK-030).

At the test reagent storage tank outlet, a pump (254010-PMT-007/008) transfers the solution to a supply loop that delivers the test reagent solution as required to the flotation circuits.

17.7.13 Antiscalant

Antiscalant is shipped to the plant in 220 kg drums. The antiscalant is added at a rate of approximately 50 g per m^3 of fresh water used in the plant.

17.8 Tailings

17.8.1 Final Tailings

Reference Flowsheet: 1010-09-010

Pyrite flotation tailings reports to the final tailings thickener (255010-THK-006). Thickener underflow is pumped (255010-PSL-061) to the paste backfill plant or collected in the final tailings pump box (255010-PBX-005) and pumped (255010-PSL-065/066) to the TMF. The tailings thickener overflow is treated and discharged into nearby streams.

17.8.2 Reclaim and Fresh Water Distribution

Reference Flowsheet: 1010-09-010 and 016

A reclaim barge with two 75 kW pumps located on the TMF is pumings the recovered process water from the TMF to either the fresh/fire water tank (355010-TNK-050), effluent treatment plant or discharge to the Tulsequah River (The reclaim water will be supplying the plant with process make-up water in the grinding circuit and rougher circuit. Make-up water for the cleaner circuits is recycled from the respective thickener overflows with additional fresh water as required.

Fresh water for the plant is sourced from the Tulsequah River. Fresh water is collected in the plant fresh/fire water storage tank in addition to reclaim water or treated water from the effluent treatment plant. The fresh/fire water storage tank supplies clean water for potable water, process make-up water, reagent mixing, pump gland water system and cooling water for the electrical and mill ancillary equipment.

Plant Air Compressors and Blowers

Reference Flowsheet: 1010-09-015

The primary consumers of compressed air are: the primary crushing plant, cleaner flotation and copper, lead and zinc concentrate filters. In addition, minor users of compressed air are: dust collection/suppression, samplers, mill gear lubrication systems, and air hose stations located throughout the plant.

There is one compressed air system for the process plant. The process plant air system consists of three compressors (256020-AIC-005 to 007), one instrument air dryer (256020-DRY-006) and six air receivers. The plant and instrument air receivers is envisioned to be located in the compressor room whereas the remaining are at their respective points of application. The air system is set up such that if a power failure occurs, the instrument air loop does not flow back into any other loop.

Plant Air is drawn to the compressor inlets from within the compressor room. Three plant air compressors each with a capacity of 276 Nm^3/hr, 38 kW meet the plant air requirements. Two compressors operate at any one time with one standby. It is recommended that the compressed air requirements for the plant be re-evaluated in the next stage of engineering once the filter supplier has been identified.

The underground compressed air system will be supplying the crushing circuit. Crusher plant air compressor has a capacity of 172 Nm^3/hr, 15 kW.

Instrument Air

The consumers of the instrument compressed air are control valves, pinch valves, knife gate valves, and instruments supplied with the packaged equipment, SAG mill, ball mills, concentrate filter, gear spray and dust collectors.

Flotation Air

Three blowers (256030-BLO-003 to 005), with a capacity of 180 Am^3/min at 38 kPag, 150 kW, will provide air to the rougher flotation circuits and one blower (256030-BLO-006), with a capacity of 144 Am^3/min at 22 KPag, 112 kW provides air to the cleaner flotation circuits. In the event that the low pressure blower is down for maintenance the standby higher pressure blower provides air to the cleaner circuit.

Sampling

Online sampling and analysis systems are utilized on critical streams in the plant. In the copper circuit multi-stage, integrated in-stream analyzers and metallurgical accounting sampling systems aid in producing the desired copper concentrates. The system provides on- line sampling that utilizes a multi-element probe and slurry density updated every one to two minutes. The other main streams has full stream slurry sampling systems using a multi-stage tank with fixed cutters to provide samples for assay.

Assay Laboratory

The Assay Laboratory consists of a sample preparation/metallurgical module and a wet laboratory module.

The Laboratory performs test work for the underground mine, mill, and environmental group. The mine is expected to generate approximately 30 samples per day that need to be dried, crushed, and prepared for assaying. The mill is planned to assay approximately 40 samples per day. Atomic absorption (AA) machines are used to analyze the ore grade samples for Cu, Pb, Zn, Fe, As, Ag and Au. Samples may also be analyzed for C, SiO_2, S, and SO_4. The concentrates are tested for Cu, Pb, Zn, Au, Ag, As, Sb, Fe, S and Cd using the AA machine and SiO_2 and C using other methods. The high grade concentrates assays are volumetric for Cu, Pb, Zn and external off site fire assay for Au and Ag.

Water samples are tested to detect limits below aquatic life standards prior to discharge into the surrounding streams. Two main tests are performed: water quality and ARD/ML potential. Water samples a analyzed for sulphates, ammonia, nitrates, bi-weekly. Samples are also prepared to be sent for analysis by third party laboratories that meet the standards set in the Metal Mining Effluent Regulations (MMER).

17.9 Process Control

The process control system is planned to be a PLC based system. The PLC's is used to control and monitor all the operations of the plant. The plant is broken into different process areas. Each process area is controlled by a single PLC system. The PLC's are to be tied together to form a plant wide control system through the use of an Ethernet communication system. The motor starter, VFD's as well as some of the field devices are controlled by the PLC via a Devicenet communication system. Local Flex I/O's are distributed around the plant to pick up the signals from instruments and digital devices. Process control and monitoring for the facility is performed in one operator control room utilizing Human Machine Interface (HMI) operator station.

The HMI is planned to contain the graphical representation of the process equipment and will interface to the PLCs via the ethernet network. There will be one master operator control and is located at the Grinding Area.

In addition to the main HMI in the control room, Flotation and Filtration areas also are planned to have local Touch Screen HMI's for monitoring and control respective of their area.

Crusher plant is controlled and monitored in the Crusher Electrical Room.

The process Ethernet communication system is a 1,000/100 based Ethernet system. All PLC's and HMI's are connected to the Ethernet system. There is a plant wide fiber optic backbone interconnecting the PLC and the HMI systems.

An overhead fiber optic line will connect the Crusher and Paste Plant to the system. An Ethernet Fiber will also pick up the signals from the Powerhouse.

The overall plant process control system (PLC configuration) is included in Appendix B.

A radio link between the Mine Ventilation and Filtration Area will be provided at this time and will be reviewed during detailed design.

The business Ethernet communication system shall be 1000/100 based Ethernet system. The business systems, office computers, and telephone (IP) systems will be connected to this system. There will be a site wide fiber optic backbone interconnecting all building together.

18. PROJECT INFRASTRUCTURE

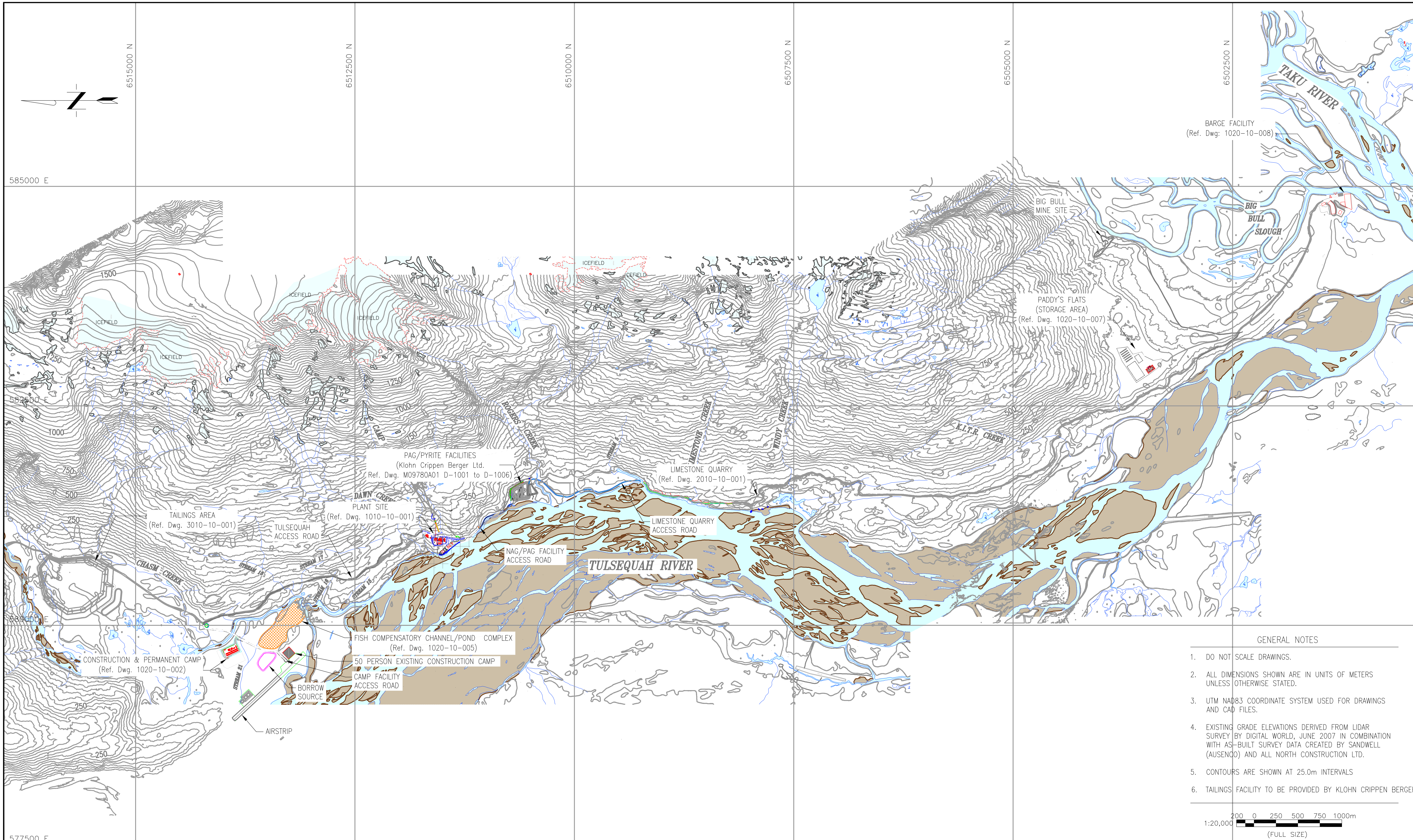
18.1 Summary

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

- Diesel fueled power plant, heat recovery system and power distribution network;
- Bulk fuel storage tanks and containment;
- Construction and permanent camp (total 160 beds) including potable and wastewater;
- Treatment plants and incinerator;
- Mill complex;
- Administration offices, mine dry and maintenance shop;
- Effluent treatment plant;
- Tailings management facility; and
- Temporary PAG waste and pyrite concentrate storage facility.

In addition to the infrastructure listed above, improvements will be made to the existing Acid Water Treatment Plant, barge facility, site road network and airstrip to better accommodate the operation.

Figure 18.1 provides an overview of the project site.



GENERAL NOTES

1. DO NOT SCALE DRAWINGS.
2. ALL DIMENSIONS SHOWN ARE IN UNITS OF METERS UNLESS OTHERWISE STATED.
3. UTM NAD83 COORDINATE SYSTEM USED FOR DRAWINGS AND CAD FILES.
4. EXISTING GRADE ELEVATIONS DERIVED FROM LIDAR SURVEY BY DIGITAL WORLD, JUNE 2007 IN COMBINATION WITH AS-BUILT SURVEY DATA CREATED BY SANDWELL (AUSENCO) AND ALL NORTH CONSTRUCTION LTD.
5. CONTOURS ARE SHOWN AT 25.0m INTERVALS
6. TAILINGS FACILITY TO BE PROVIDED BY KLOHN CRIPPEN BERGER

1:20,000 200 0 250 500 750 1000m
(FULL SIZE)

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D3	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14
D2	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		31/07/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		27/06/14

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:		
DESIGNED:	J.D.S.	June 21 2014
DRAWN:	B. Wong	June 21 2014
CHECKED:		
APPROVED:		

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
PROJECT SITE OVERVIEW
GENERAL ARRANGEMENT
PLAN

PROJECT No.	DRAWING No.	REV. No.
	1000-00-001	F1

CADD FILE No.: S:\

18.2 Barge Landing Facility

The barge landing facility on the Taku River is a critical component of the transportation infrastructure for the Tulsequah Project. The majority of consumables and all concentrate will pass through the barge landing facility as they are shipped to and from site during the annual barging season.

The barge berth will be constructed using gabion baskets placed in the river to form an approximately 110 m long berth face designed to accommodate two barges tied-up at any one time. Locally available, natural rounded stone and/or screened rock will fill the gabion baskets. The space between the baskets and existing shoreline will be backfilled with granulated compacted backfill and crushed rock. Tires attached along the berth face to act as fenders and minimize damage to the barges.

The primary container handling equipment used at the barge facility will be a 165 tonne crawler crane and Taylor TXLC 975 container handler. Support equipment will include wheel loaders, forklifts and tractor-trailer and flat deck truck.

The barge facility will include a small office complex and diesel generator to supply power.

Figure 18.2 shows the general arrangement for the barge facility.

BARGE FACILITY

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D3	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		04/09/14
D2	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		31/07/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		27/06/14

REFERENCE DRAWING	REF. DWG No.

SCALE:	DATE:
DESIGNED: J.D.S.	June 21 2014
DRAWN: B. Wong	June 21 2014
CHECKED:	
APPROVED:	

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
BARGE FACILITY
GENERAL ARRANGEMENT
PLAN

PROJECT No.	DRAWING No.	REV No.
	1020-10-008	F1

18.3 Site Road Network

The existing site road network consists of approximately 18 km of all-season, single lane gravel road that connects the airstrip and camp at the north to the barge facility in the south. The road includes eighteen bridges, between 10 to 40 m in length, crossing streams and creeks that flow into the Tulsequah River. The construction of 2 km of new road is planned to connect the existing road to the tailings management facility.

The Shazah Creek Bridge will be relocated approximately 10 m upstream to permanent abutments. The Roger's Creek Bridge will be replaced with permanent steel abutments and a steel bridge structure already on site. All bridges are inspected on a regular two year schedule and maintenance is on-going.

The site road network is used to transport personnel and materials between the working areas. Traffic will include light vehicles, tractor-trailers and underground mine trucks (between the portal and waste storage areas). The road is approximately 5 m wide and pullouts are located where space is available due to the steep terrain. The maximum grade is designed to be 12% in areas where heavy equipment or tractor-trailers will routinely operate. The road will be radio controlled.

18.4 Airstrip

The airstrip will be used for shuttling workers back and forth from Whitehorse, Yukon and Atlin, BC. The airstrip is approximately 1,050 m long x 50 m wide and is able to accommodate the DHC-5 Buffalo or other short takeoff and landing (STOL) aircraft. This aircraft enables efficient transportation of personnel and supplies to and from the site by air.

The runway will be resurfaced with crushed and screened gravel. No other major works have been taken to upgrade the airstrip to meet NavCan standards. Present operations are restricted to visual take-off and landing.

18.5 Power Supply

18.5.1 Power Generation

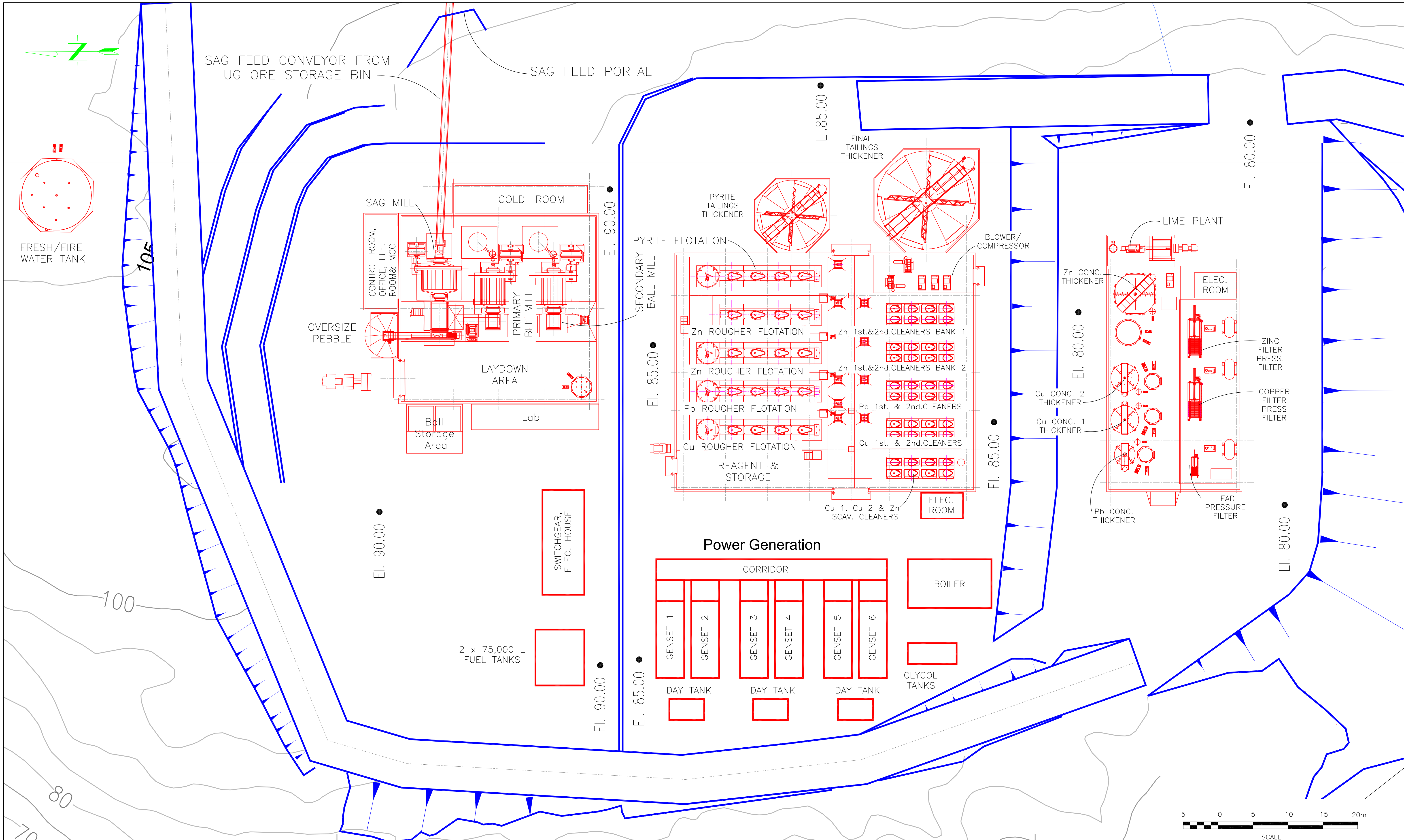
Diesel generators will supply power for the site. The total installed electrical load for the mine, mill complex and nearby infrastructure is 8,800 kW with an average annual demand load of 4,740 kW as summarized in Table 18.1.

Table 18.1: Estimated Power Demand by Area

Area	Average Annual Demand (kW)
Mine	1,400
Process Plant & Ancillary	3,340
Total Main Power Plant	4,740
Camp	310
Reclaim Water Pumps	50

The main power plant will consist of six CAT 3516B generators, four running, one standby and one maintenance for an N+2 arrangement. Each generator has a peak rating of 1,825 kW.

The fully enclosed units will have individual circuit breakers for isolation and will be tied to a common bus. A main breaker feeds a 4160 V switchgear for power distribution. The generators will be supplied with heat recovery packages in order to provide heat to the processing plant and underground mine. Figure 18.3 shows the plant site layout and the relative locations of the main power generators, diesel storage tanks and process facilities.



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D2	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		5/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: | DATE:
DESIGNED: J.D.S. | June 21 2014
DRAWN: B. Wong | June 21 2014
CHECKED:
APPROVED:

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TULSEQUAH CHIEF PROJECT

TITLE
MIDDLE BENCH (PLANT SITE FACILITY)
GENERAL ARRANGEMENT
PLAN

PROJECT No. | DRAWING No. 1010-10-003 | REV No. F1

18.5.2 Power Distribution

The generators feed power to a common switchgear bus. The power is distributed over the property by means of three (3) underground and two (2) overhead power lines at 4160 V. The overall plant single line diagram is included in Appendix D.

The power distribution will primarily consist of:

- FU1 – Grinding Building MCC1;
- FU2 – Flotation Building MCC2;
- FU3 – Filtration Building Substation T3;
- FU4 – Maintenance Shop, Water Treatment Plant, Ventilation Fans and Underground Mine; and
- FU5 – Administration, Underground Crushing and Paste Plant.

Mill Buildings

The mill is comprised of three buildings: grinding, flotation and filtration. Each building has its own electrical room and is individually serviced by a 4160 V, three phase buried line originating from the power house. The lines for grinding and flotation are routed directly to a 4160 V MCC located in the grinding and flotation electrical rooms, respectively. The third underground line is connected to a 4160 V - 600 V transformer located adjacent to the filtration electrical room.

The 4160 V MCC (MCC1) in the grinding building will feed the SAG mill, primary and secondary ball mills at 4160 V level and to a step down 4160 V – 600 V transformer that will be connected to a 600 V PDC for power distribution to drives and other services.

The 4160 V MCC (MCC2) in the flotation electrical room will feed the flotation blowers at 4160 V level and to a step down 4160 V – 600 V transformer that will be connected to a 600 V PDC for power distribution.

The 4160 V – 600 V, 2000 kVA transformer in the filtration building will be connected to a 600 V PDC for power distribution.

Maintenance Shop, Water Treatment Plant, Ventilation Fans and Mine

The underground mine will be serviced by a 4160 V, three phase, overhead power line originating at the power house. Approximately 950 m of 4160 V high-yield-strength cable will be routed to the main substation for the underground mine power distribution.

The power line servicing the underground mine will also feed the two (2) 4160 V – 600 V, 500 kVA transformers feeding the water treatment plant and ventilation fans and one (1) 4160 V – 600 V, 150 kVA transformer feeding the maintenance shop. Each area will have a dedicated 600 V MCC with main circuit breaker connected directly to the transformer for distribution.

Administration, Underground Crushing and Paste Plant

The underground paste plant will be serviced by a 4160 V, three phase overhead power line originating from the power house. The power line will feed a 4160 V MCC (MCC 5) in the paste plant electrical room in order to supply power to the vacuum and paste pumps. The line will also be tied into a step down 4160 V - 600 V transformer that will be connected to a 600 V MCC for power distribution to drives and other services.

The power line servicing the underground paste plant will also feed the two (2) 4160 V – 600 V, 1000 kVA and 75 kVA transformers feeding the underground crushing plant and administration offices, respectively. Each area will have a dedicated 600 V MCC with main circuit breaker connected directly to the transformer for distribution.

Standalone Generators

The camp and reclaim water pumps, located approximately 5 km from the main power plant, will be supplied by independent, standalone generators (750 and 100 kW respectively).

Smaller generators (70 – 90 kVA) are employed to supply power to other remote working areas on site, including: the PAG and pyrite storage area, bulk fuel storage and barge facility.

Emergency Power Supply

The process plant will be provided with a 600 V - 300 kW diesel generator located in the grinding building electric room. Changeover from main to emergency diesel power will be through an auto switching operation.

18.6 Lighting

18.6.1 Lighting Applications

The following list describes the type of lighting used for various applications:

- High Bay Areas - Metal Halide;
- Electrical Room - Fluorescent fixtures;
- Control Room - Fluorescent fixtures;
- Offices - Fluorescent fixtures; and
- Outdoor Lighting- High Pressure Sodium (HPS).

All outdoor lighting will be controlled with photocells so that the lights can automatically be turned off during daylight hours.

Illumination Levels

Lighting levels for various areas of the plant are recorded in Table 18.2.

Table 18.2: Illumination Levels

Area	Lighting Level (Footcandles)
General plant areas	35-40
Control Room	50
Electrical Room	50
Maintenance Shop	50
Offices/ Laboratory	50
Outdoor Lighting	2-3

Emergency Lighting

Emergency lighting will be provided by means of wall mounted emergency lighting packs.

The number and location of the emergency lighting packs or fixtures will be such as to provide for a sufficient illumination level to allow personnel to egress safely in the event of a power outage.

Exit Lighting

The exit lighting will be the installed at all exits. Exit lighting fixtures shall be equipped with standby battery and charger.

18.7 Heat Recovery & Ventilation

The following facilities will be heated by heat recovered from the power plant heat exchangers:

- Underground mine;
- Grinding building;
- Flotation building;
- Filtration building; and
- Maintenance shop.

The following remote facilities will be heated by electric heaters:

- Administration building;
- Camp; and
- Water treatment plant

Waste heat recovered from the power plant will be transferred to a water/glycol solution via heat exchangers and will be distributed by insulated piping loops to heat the process plant and the underground fresh air as required. In normal, full production conditions, sufficient heat will be recovered to maintain the above facilities at design temperature under winter ambient conditions of -9.4°C. Each building will have a make-up air unit to supply conditioned fresh air to the building and two unit heaters to maintain the buildings at the required temperature.

Based upon vendor performance data for the generators and the estimated load levels, in full production condition an estimated 6 MW of heat energy will be rejected through the engines exhaust.

Additionally, emergency backup boilers are included in the site heating system for supplemental heating capacity when the process plant is shut down for maintenance and only one diesel generator is running to support non-process mine site load.

The estimated heat recovery for the mine ventilation air and for the processing plant is shown in Table 18.3.

Table 18.3 Estimated Heat Requirements

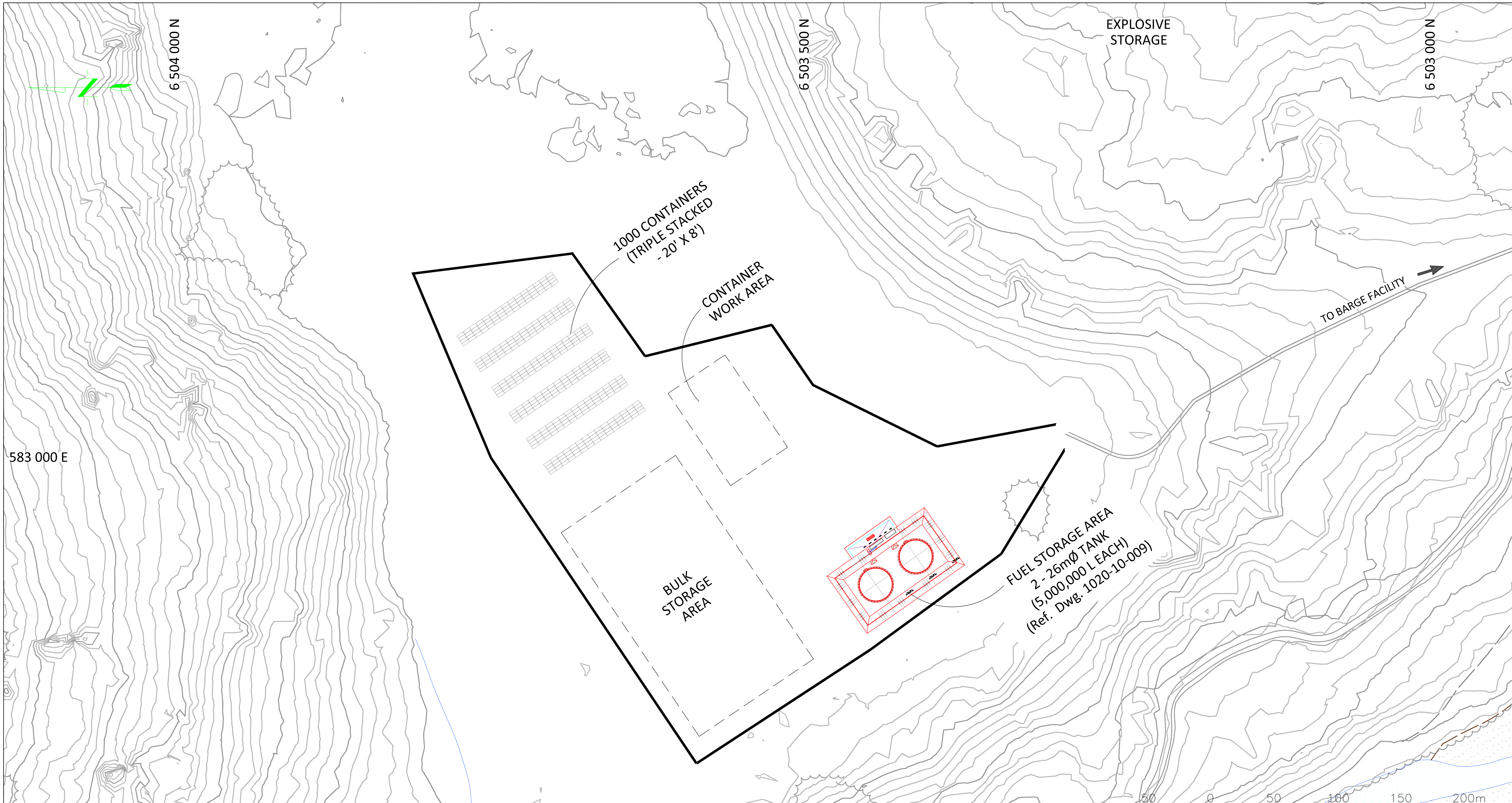
Area	Heat Load (kW)
Mine Ventilation	1,975
Grinding Building	275
Flotation Building	370
Filtration Building	210
Maintenance Shop	200
Total Heating Load	**3,030**

Source: JDS 2014

18.8 Bulk Fuel Storage & Containment

Bulk diesel will be stored in two 5,000,000 l tanks placed within a containment area at Paddy's Flats. Each tank will be 26 m diameter x 10 m tall. The tanks will be constructed using typical methods; pre-fabricated steel panels, shipped to site and assembled in place.

An overview of the storage area (Paddy's Flats) is provided in Figure 18.4 and the general arrangement of the bulk fuel storage is shown in Figure 18.5.



PADDY'S FLATS

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D3	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		04/09/14
D2	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		20/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		27/06/14

REFERENCE DRAWING	REF. DWG No.

	DATE:
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DESIGNED: J.D.S.	June 21 2014
DRAWN: B. Wong	June 21 2014
CHECKED:	
APPROVED:	

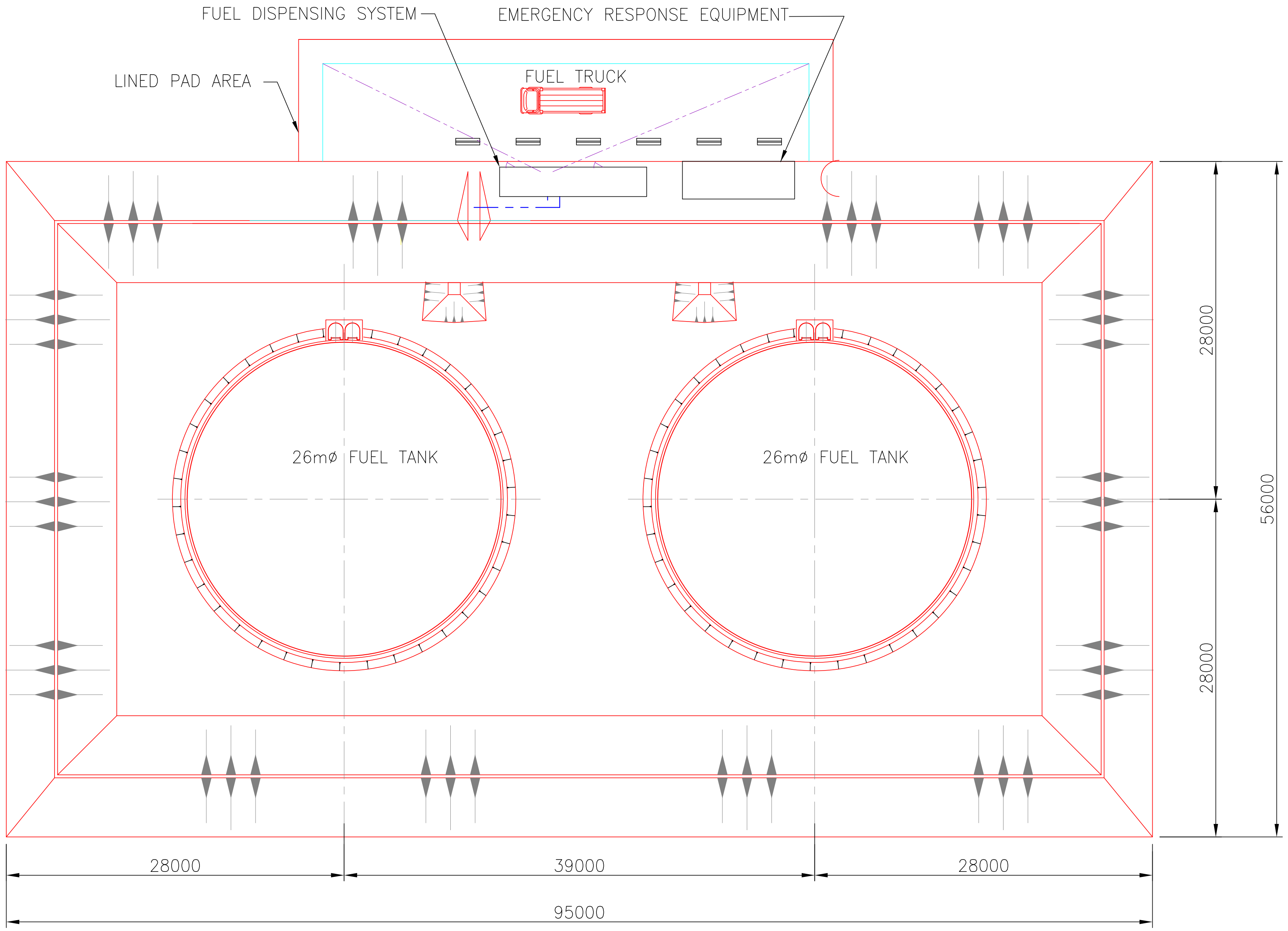
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
PADDY'S FLAT (STORAGE AREA)
GENERAL ARRANGEMENT
PLAN

PROJECT No.	DRAWING No.	REV No.
	1020-10-007	F1



PLAN – BULK FUEL STORAGE
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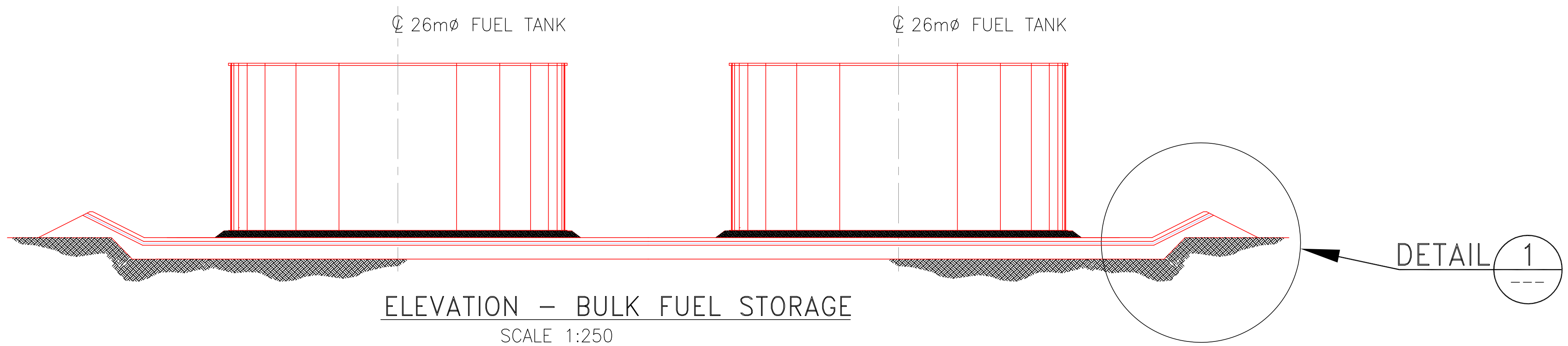

ELEVATION – BULK FUEL STORAGE
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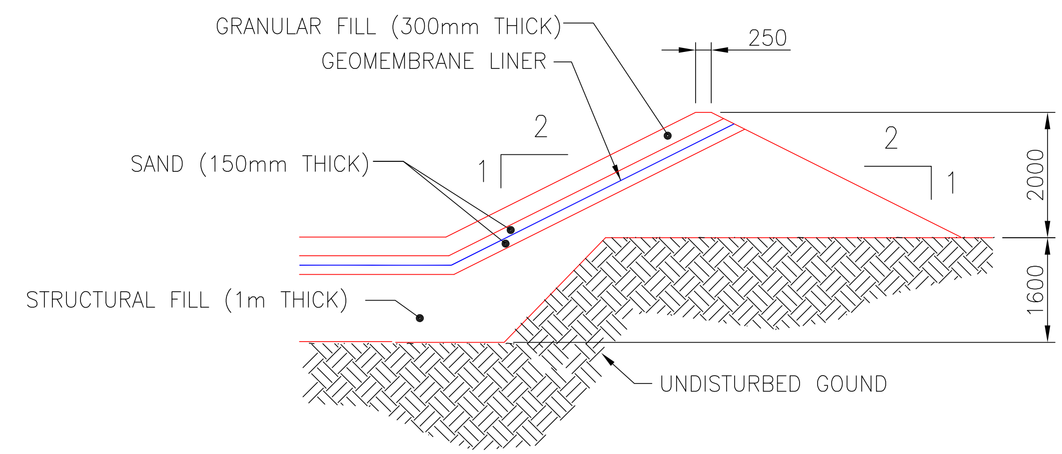

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REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		04/09/14

REFERENCE DRAWING	REF. DWG No.

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DESIGNED:	J.D.S	Sept. 2 2014
DRAWN:	B. Wong	Sept. 2 2014
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TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
BULK FUEL STORAGE
GENERAL ARRANGEMENT
PLAN AND SECTION

PROJECT No.
DRAWING No. 1020-10-009
REV No. F1

The containment area will consist of a lined berm and impoundment sized to contain 120% of the volume of diesel stored in the bulk tanks (110 % of the volume of the first tank and 10% of the volume of each subsequent tank).

Diesel will be transported to site during the annual barging season in ISO 20 ft. 24,000 l fuel containers. The containers will be offloaded by crane onto a truck at the barge facility and transported to Paddy's Flats where the diesel will be transferred to the bulk storage tanks.

An offload and dispensing system will be located at the bulk storage tanks on the outside of the containment berm. The system will consist of two centrifugal pumps with 60 m^3 / hr (260 gpm) capacity housed in a modified sea container. The area around the pump module will be lined and graded to drain to a sump should there be a leak while offloading or dispensing. Power will be supplied by a small diesel generator.

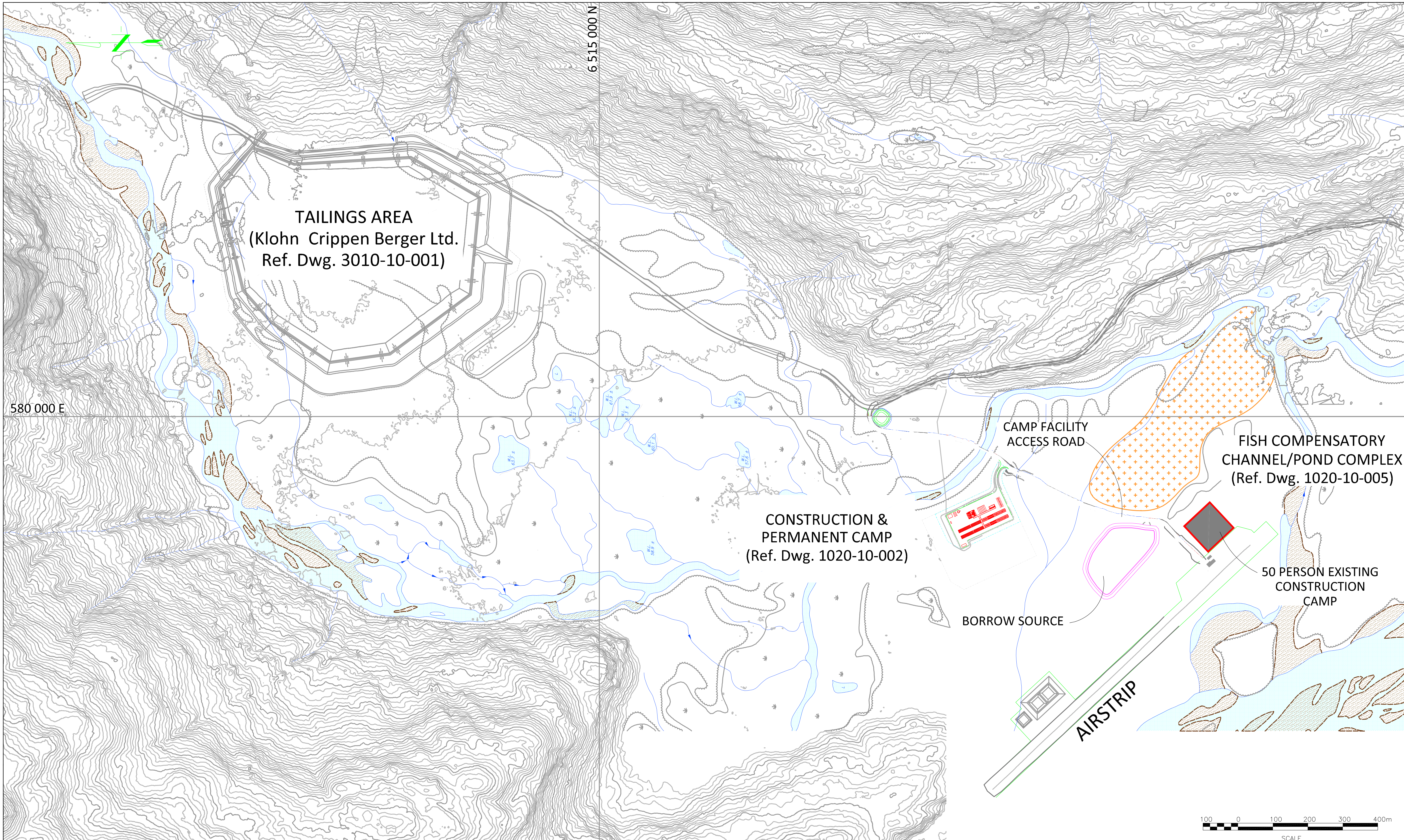
Diesel will be dispensed from the bulk tanks into a conventional 30,000 L tanker trailer for delivery to the main power plant, mobile equipment fuel bay and other generators located around the site.

Two 75,000 l double-walled tanks will be located at the main power plant. These tanks will drain by gravity to the generator day tanks. The camp and reclaim water pump generators and mobile equipment fuel bay will make use of single 75,000 L double-walled tanks. Each area will be equipped with a spill kit to mop up small spills and drips that may occur while transferring fuel from the tanker trailer to the storage tank.

The mobile equipment dispensing system currently on site will be relocated to a position near the 60 level portal. A single dispensing pump will be provided for fueling mobile equipment.

18.9 Construction Camp

A used, 72-person construction camp will be mobilized to site as part of the 2015 barge campaign to supplement the existing 50-person camp. The construction camp will be located on a new pad near the existing camp and airstrip, as highlighted in Figure 18.6.



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D2	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		27/06/14

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DATE:
DESIGNED: J.D.S. June 21 2014
DRAWN: B. Wong June 21 2014
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APPROVED:

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
CAMP, AIRSTRIP AND TAILINGS AREA OVERVIEW
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1020-10-001
REV. No. F1

The camp will be a typical skid-frame style modular facility. It will consist of two, 36-person dormitory wings, each with communal washrooms, shower facilities and laundry rooms.

18.10 Permanent Camp

In addition to the construction camp, a new permanent camp will be mobilized to site during the 2015 barging campaign. The permanent camp includes new kitchen & dining facilities, two 44-person dormitory units, arctic corridors, mine dry and mud-room. The kitchen and dining facility is sized to provide meals for the maximum number of personnel on-site during the construction period. The permanent dormitory units will be skid-frame style modular units. Each dorm unit will have the "jack-and-jill" bathroom arrangement (i.e. two bedrooms share one bathroom) and communal laundry facilities.

A new potable water treatment plant and sewage wastewater treatment plant will be installed as the existing system cannot support additional load. The potable water and sewage treatment systems are sized to handle the maximum number of personnel on-site during the construction period. Fresh water will be supplied from a well located near the potable water treatment plant.

The camp will have electric heating. A 750 kW diesel generator will supply power for the camp and associated facilities. A second 750 kW generator will be used to supply the mine during construction before the main power plant is commissioned. This generator will serve as backup for the camp once the main power plant has been commissioned.

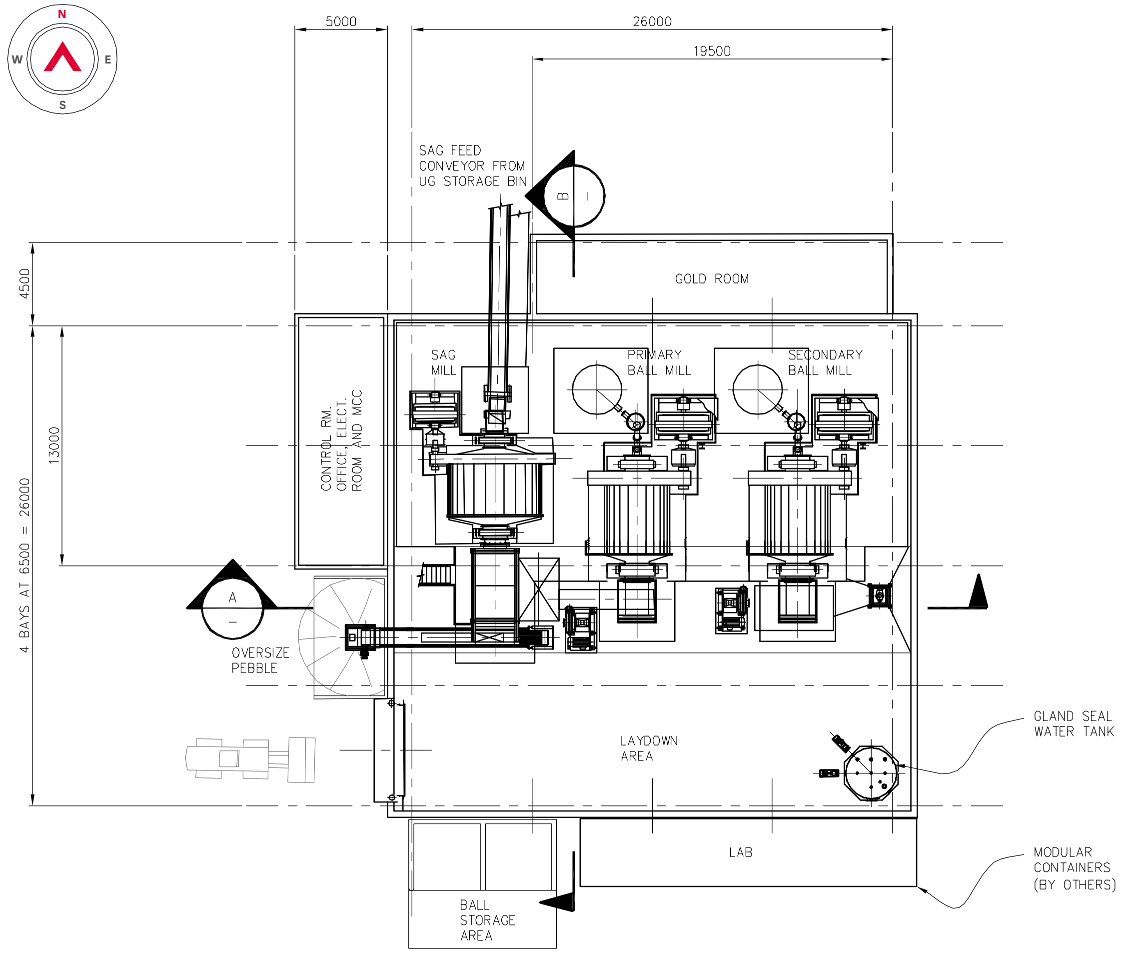
After the construction period, surplus rooms in the construction camp will be winterized and deactivated except for use during plant shutdown maintenance.

18.11 Concentrator Buildings

The process plant is split into three buildings to separate different processes and take advantage of the natural grade of the site: grinding, flotation and filtration. The buildings are independent and connected by pipe racks or utilidors to bring utilities and process material between them.

18.11.1 Grinding Building

The grinding area is housed in a pre-engineered metal building. The SAG and ball mills will be installed on one side of the building and separated from the laydown area by a trench. All slabs will be sloped towards the center trench for ease of clean-up and waste management. The area will be serviced by a 10 t overhead crane that spans the length of the building. The main building is approximately 26 m x 26 m with a lean-to gold room and electrical rooms attached to the main structure. A modular assay lab will be located adjacent to the grinding building but will be an independent structure.



PLAN
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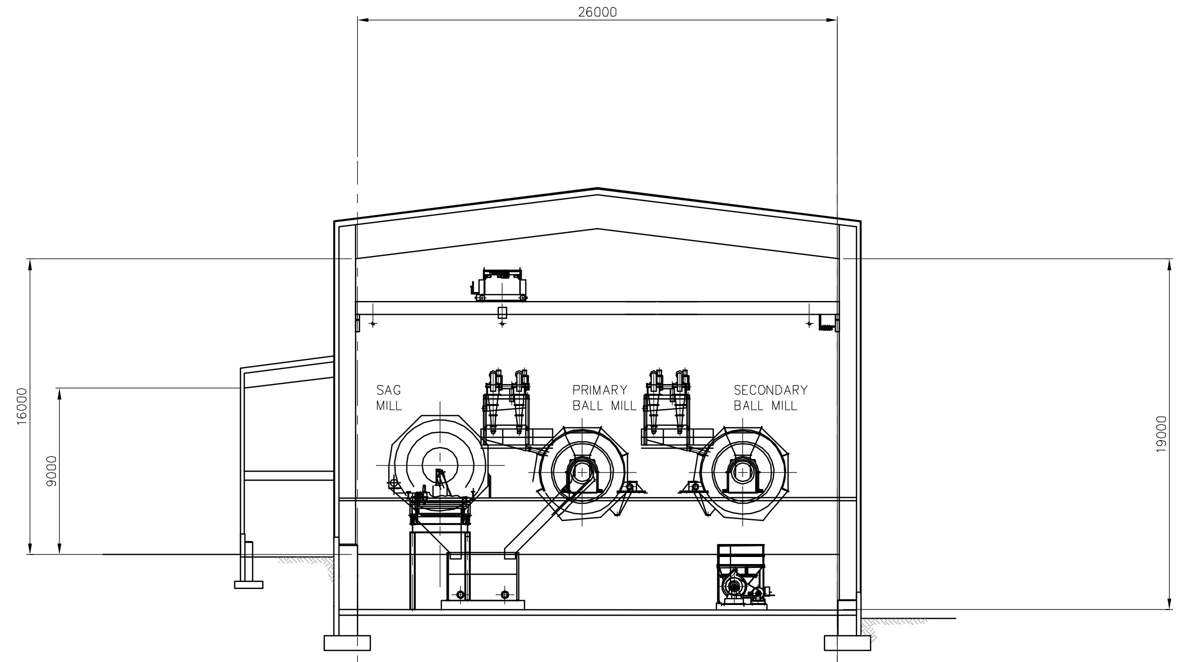

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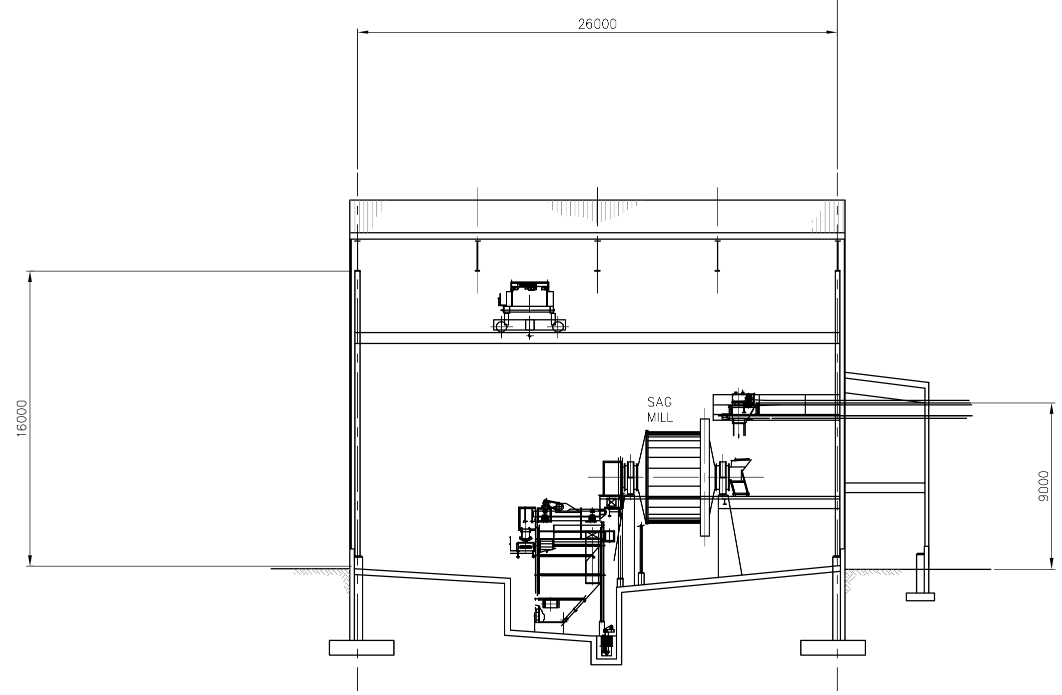

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Date: 2014/08/15

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REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
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A	14/08/07	ISSUED FOR APPROVAL	MC	-

CLIENT:

Allnorth

TITLE: GRINDING AREA

CLIENT NO:	-	DRWN:	MLC	DATE:	14/07/30
PROJECT NO:	14SU0026	DSGN:	-	DATE:	-
DRAWING SIZE:	ANSI "D"	CHKD:	-	DATE:	-
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PROJECT: TULSEQUAH CHIEF FEASIBILITY STUDY INPUT

DWG NO: 14SU0026-00-1400-010 REV: B

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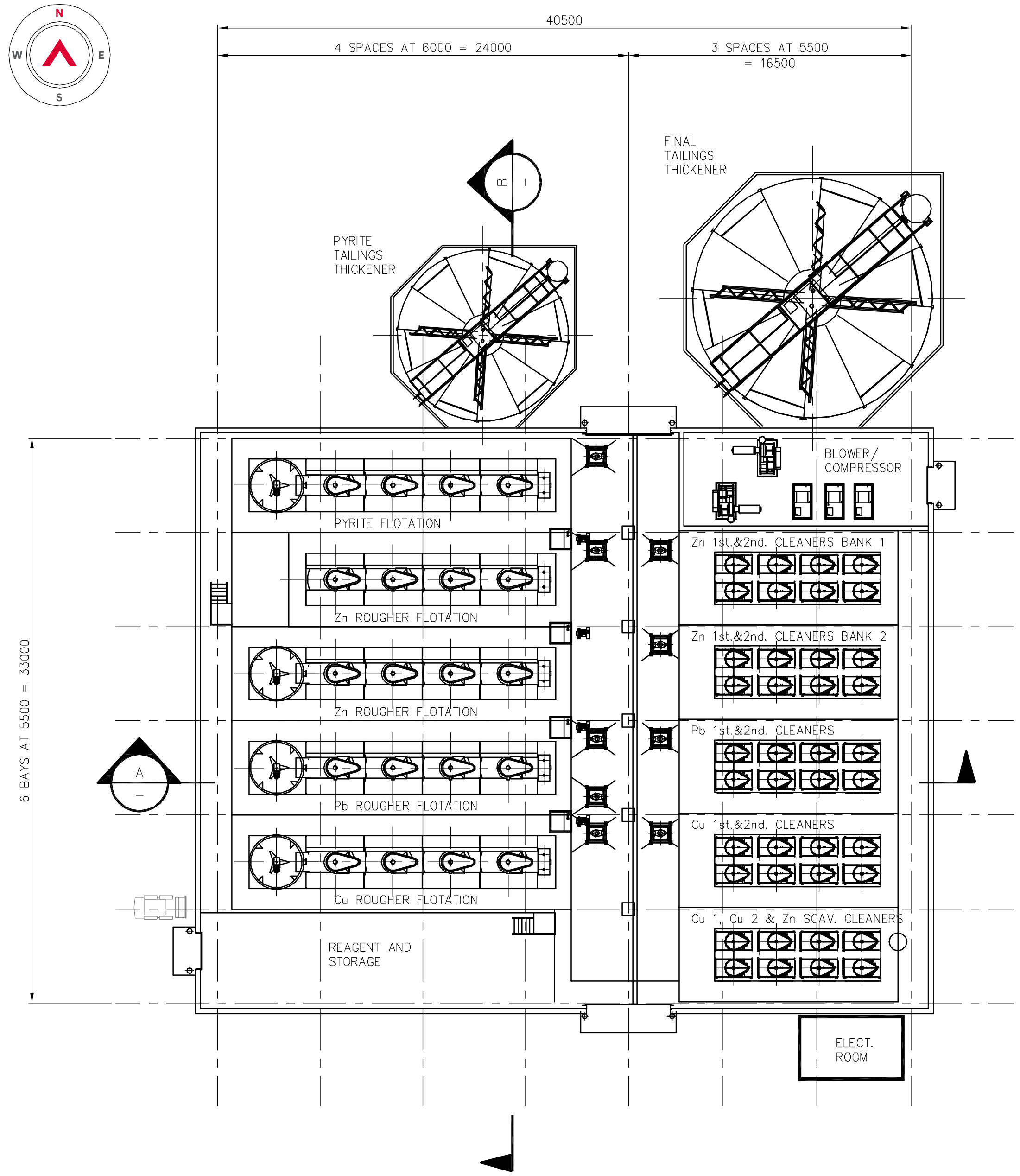
18.11.2 Flotation Building

The flotation equipment will also be installed in a pre-engineered metal building with rougher flotation cells on one side of the building and cleaners on the other. The rougher cells are elevated above the cleaners to allow for gravity flow and the slabs slope towards specific sumps to keep the products separate. The building is approximately 33 m x 40.5 m and has two 2 t overhead cranes to service the roughers and cleaners. Reagent storage and the compressor and blowers are in separate rooms within the main building envelope.



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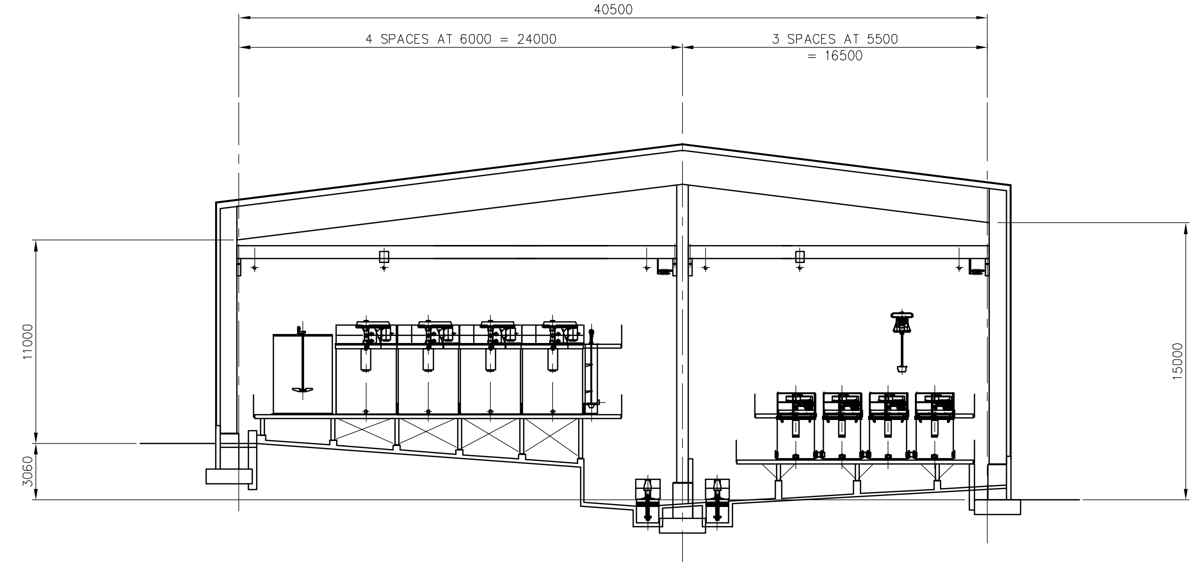

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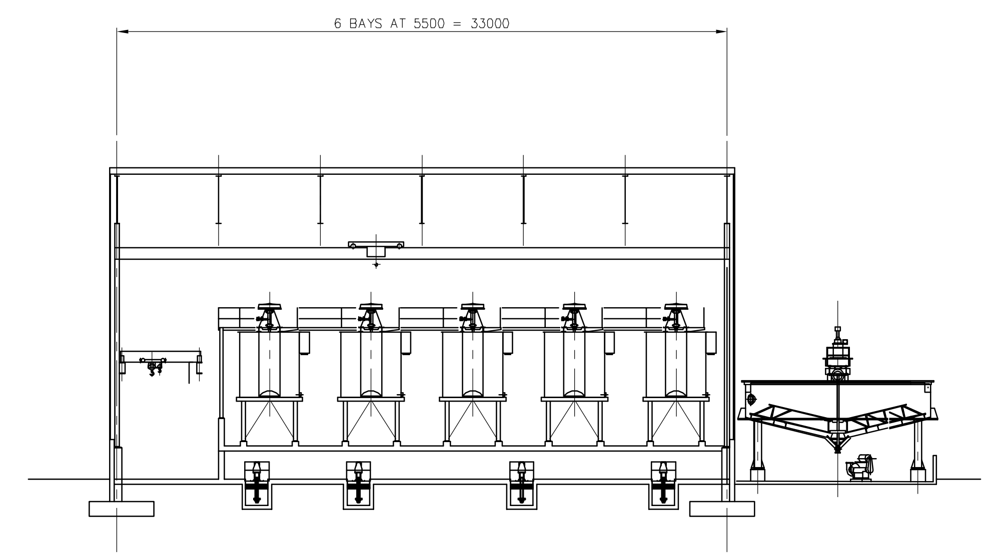

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Date: 2014/08/28

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REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
C	14/08/28	BLOWER ROOM DOOR ADDED	MLC	-
B	14/08/15	ISSUED FOR INFORMATION.	MLC	-
A	14/08/07	ISSUED FOR APPROVAL.	MLC	-

CLIENT:

Allnorth

TITLE: **FLOTATION AREA**

CLIENT NO: -	DRWN: MLC	DATE: 14/07/30
PROJECT NO: 14SU0026	DSGN: -	DATE: -
DRAWING SIZE: ANSI "D"	CHKD: -	DATE: -
SCALE: AS NOTED	APVD: -	DATE: -

PROJECT: **TULSEQUAH CHIEF FEASIBILITY STUDY INPUT**

DWG NO: **14SU0026-00-1400-011** REV: **C**

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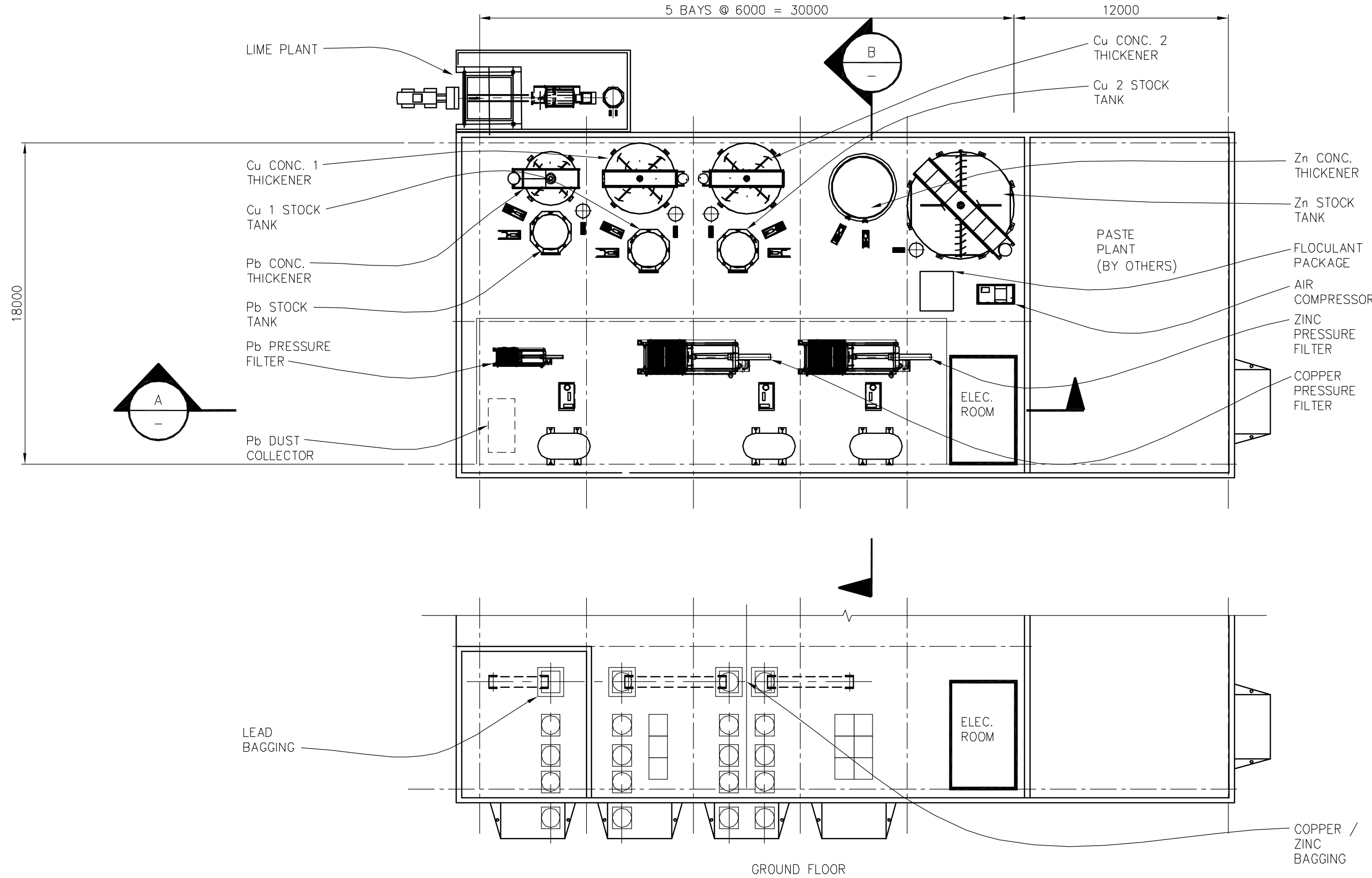
18.11.3 Filtration Building (Concentrate Dewatering & Load-out)

Concentrate dewatering takes place on an elevated level with the bagging underneath for ease of material flow. The building is 18 m x 30 m. Each product is situated in a separate bay with the lead area walled in to prevent potentially hazardous materials from spreading. Each product bay has its own overhead door for ease of access to the bagged product. The building is serviced by a 5 t overhead crane for maintenance on the filter presses and thickeners.



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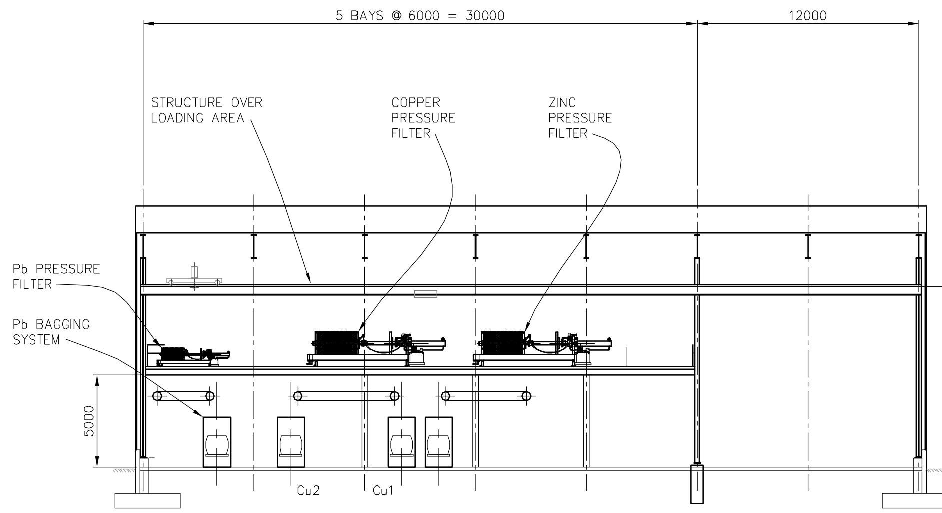

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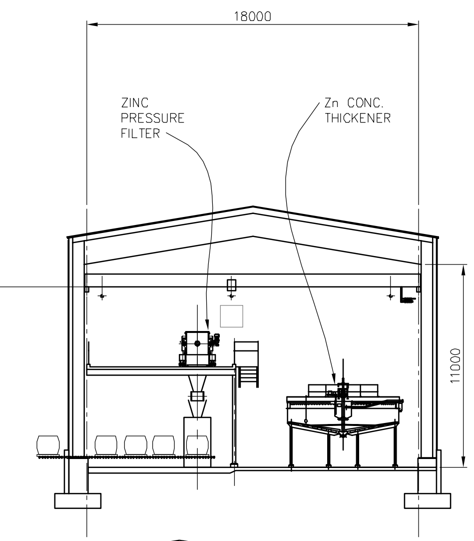

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INFORMATION
Date: 2014/08/28

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REFERENCE DRAWINGS

DRAWING NO	DRAWING DESCRIPTION/TITLE	REF
-	-	1

REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
D	28/08/15	ISSUED FOR INFORMATION	RM	
C	14/08/15	ISSUED FOR INFORMATION	RM	
B	14/08/07	REVISED TO 2t BAGGING	LBR	
A	14/08/06	ISSUED FOR APPROVAL	RM	

CLIENT:

Allnorth

TITLE:
CONCENTRATE DEWATERING AND LOADOUT

CLIENT NO: -	DRWN: LRM	DATE: 14/07/30	
PROJECT NO: 14SU0026	DSGN: -	DATE: -	
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SCALE: AS NOTED	APVD: -	DATE: -	

PROJECT:
TULSEQUAH CHIEF FEASIBILITY STUDY INPUT

DWG NO: **14SU0026-00-1400-012** REV: **D**

18.12 Assay Laboratory

The assay and metallurgical laboratory will be housed in two modified sea containers. The lab will be outfitted to perform atomic adsorption analysis for metals and will handle all mine, mill and water samples on site. Fire assays will be sent offsite to a third party lab for processing.

18.13 Maintenance Shop

The maintenance shop for the mobile equipment fleet will be located on the lower bench of the plant site near the 60 level portal as shown in Figure 18.10. There will be one insulated shop for equipment repairs and six unheated sea containers for temporary warehousing of parts and consumables. The building will be a 22 x 33 m (70 x 105 ft.) fabric “tent” structure on a concrete floor slab. The shop has two 5 m (16 ft.) overhead doors, two man-doors and transparent overhead panels to allow available ambient light into the building. No overhead crane is planned for the shop; instead mobile gantry cranes and the 20 T boom truck will be utilized to support maintenance activities. The shop will include a fabric partition and exhaust system so that one end can be isolated for welding.

18.14 Freight Shipping

Table 18.4 below details both the outbound freight in the form of concentrate and inbound freight mainly supplies, materials and fuel to support mining and processing.

Table 18.4: Production Freight Requirements

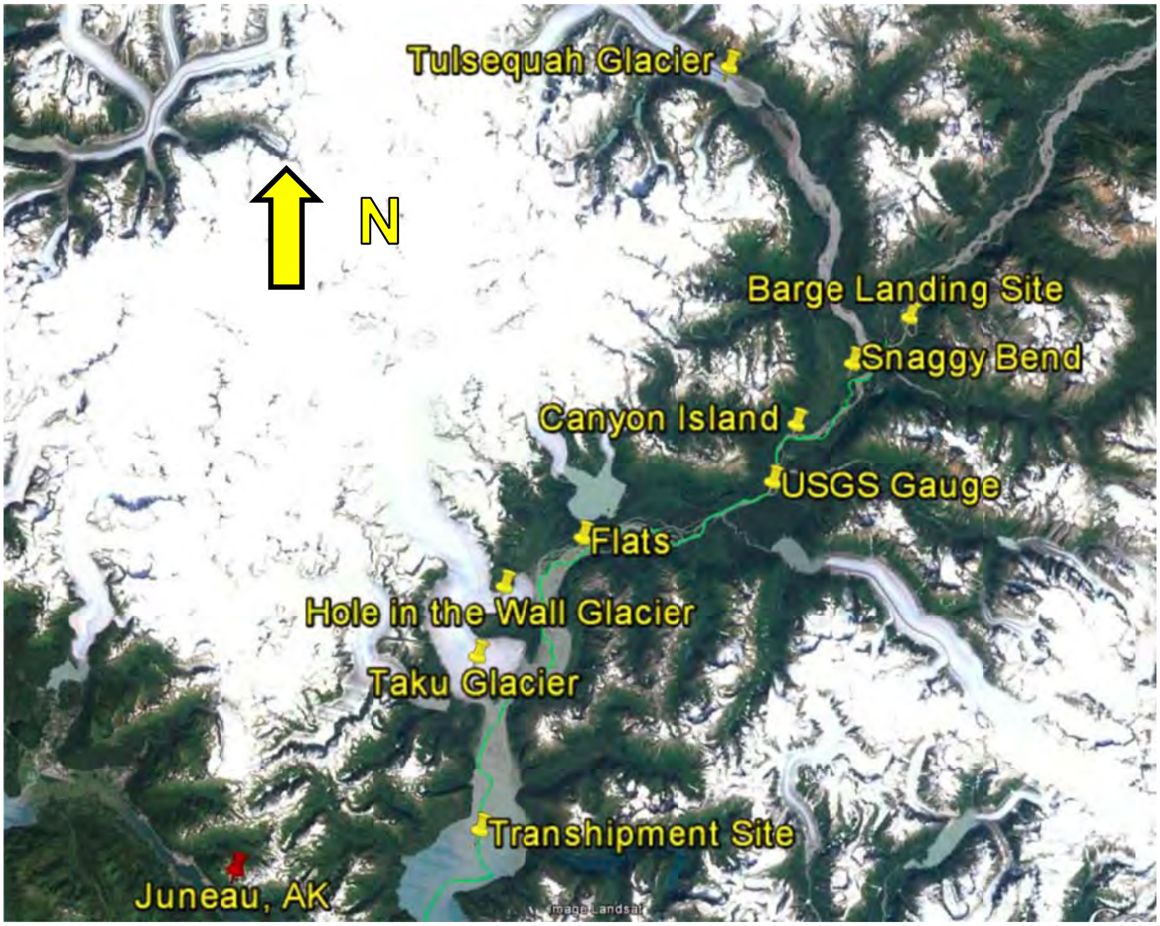
	Mining Freight (tonnes)	Process Freight (tonnes)	Fuel (liters)	Inbound Freight (excl. Fuel)	Inbound Freight (incl. Fuel)	Outbound Freight Concentrate (wmt)
Year 1	6,176	1,927	11,562,624	7,028	16,856	45,879
Year 2	7,238	2,122	12,865,586	9,115	20,051	76,588
Year 3	5,193	2,148	12,737,037	7,247	18,073	81,931
Year 4	3,455	2,148	13,247,380	5,772	17,033	91,871
Year 5	3,843	2,138	13,901,173	6,484	18,300	93,252
Year 6	3,583	2,136	13,904,622	6,173	17,992	93,357
Year 7	4,100	2,132	13,640,887	6,791	18,386	84,865
Year 8	5,091	2,136	13,369,437	7,325	18,689	72,124
Year 9	5,872	2,144	13,254,645	7,722	18,988	69,413
Year 10	5,651	2,129	13,076,258	7,816	18,931	67,477
Year 11	2,639	1,056	6,501,212	3,797	9,323	67,931
Year 12	0	0	0	0	0	22,935
Total	**52,842**	**22,215**	**138,060,862**	**75,271**	**192,622**	**867,624**

Source: JDS 2014

18.14.1 Outbound Concentrate Shipping

The primary method used to transport concentrate from the Mine to its final export destination utilizes seasonal barging on the Taku River. Ausenco Engineering Canada Inc. (Ausenco) was retained to assess, analyze and report on the navigability of the Taku River for transporting concentrate and supply of materials. Figure 18.10 below shows the approximately 62 km long river barge route starting from the barge landing site and ending at the Taku Inlet.

Figure 18.10: Overview of Taku River Barge Route



Source: Ausenco 2014

Ausenco analyzed 26 years of historical river flow data from gauge height readings from the USGS station near Canyon Island. The gauge height data to correlate and determine navigable draft levels in the river. The data indicates that flows in the Taku River start to increase in May and as the freshet proceeds the river level rises. In the months of June and July the gauge height reading exceeds 35ft approximately 85% of the time. The river flows begin to decrease in August and by September is almost completely dissipated with the 35ft mark only being exceeded 22% of the time.

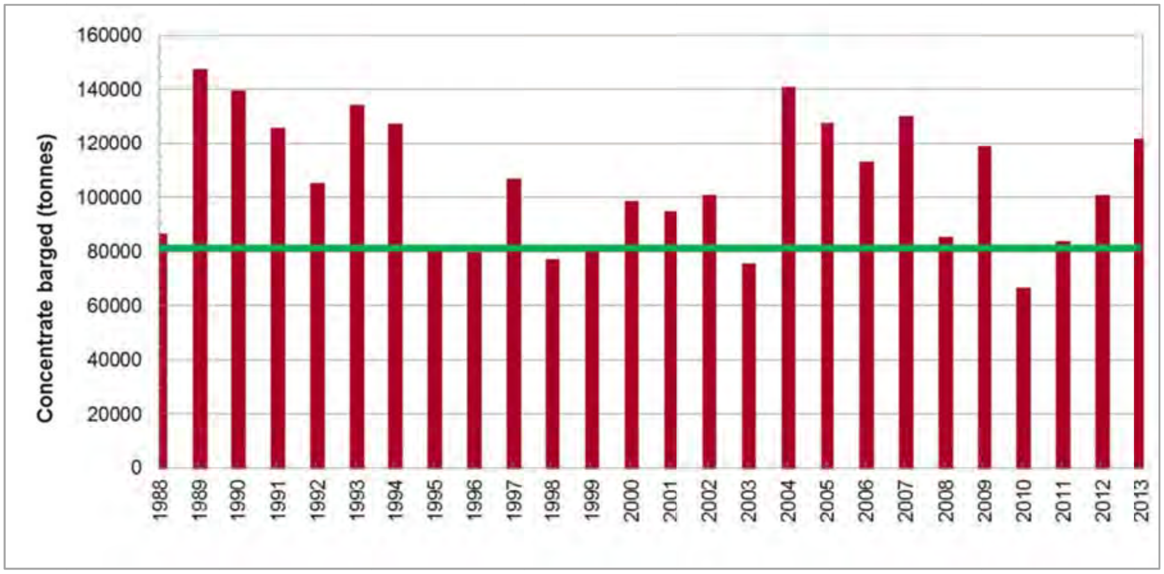
The operating draft of custom built shallow-draft barges and shallow-draft pusher tugs were calculated and used to determine barge capacities. Annual mean, median and standard deviation of barging throughput in each of the previous 26 years (1988 through 2013) were calculated using an assumed barge fleet of 4 barges and tugs. The probability of an annual throughput shortfall was also determined. Table18.5 below summarizes the applicable barging conditions for various gauge heights.

Table 18.5: Barging Statistics for Average Barging Season

USGS Canyon Island Gauge Height (ft)	Frequency of Occurrence During Barging Season	Barging Operation	Barge Efficiency %	Barge Capacity (tonne)
<33.5	20% - too low for barging	No Operations due to risk of grounding	0	0
33.5 to 35	23% - tidal assist required through flats	Barges loaded to 0.6m draft, need tidal assist	60	144
35 to 36.5	31%	Barges loaded to 0.6m draft, no tidal assist needed	100	144
36.5 to 38.5	19%	Barges loaded to 0.9m draft, no tidal assist needed	100	288
38.5 to 40	4%	Barges loaded to 1.2m draft, no tidal assist needed	100	432
>40	3%	No operations as current in Canyon Island area too high	0	0

The annual theoretical historical throughput capability of a four barge fleet is shown in Figure below. The green line in the figure represents the annual targeted concentrate throughput of 81,000 wet metric tonnes (wmt).

Figure 18.11: Annual Barging Throughput - Four Barge Fleet.



Source: Tulsequah Barging Study, Ausenco, October 2014

Ausenco's statistical analysis for a four barge fleet indicated the following statistics:

- Mean annual throughput = 103,600 wmt;
- Median annual throughput = 100,700 wmt;
- Standard Deviation = 23,600 wmt; and
- Probability of a 1 t shortfall in any year = 23.1% (based on 81,000 wmt target).

Based on Ausenco's analysis a custom built fleet of four hopper style barges (42.7 m x 13.1 m) and four pusher tugs are considered optimal and can achieve an annual throughout of 81,000 wmt with an acceptable level of risk of shortfall.

Robert Allan Ltd (RAL), a naval architectural firm based out of Vancouver BC, with extensive experience in shallow-draft tugs and barges, was retained by Ausenco to provide preliminary design, operational input and cost values for a custom fleet of river barges and tugs.

Several options were considered by Ausenco for the containerized shipment of concentrate from the mine to export port facilities. Ausenco's report recommends concentrate be shipped in standard ISO TEU (20 ft equivalent unit) containers carrying two-tonne super sacs of concentrate. Empty containers from an international shipping line in the Seattle, WA area would be transported to a transshipment point at the mouth of the Taku River by ocean going barges. This transshipment site will provide a base for short term storage of empty containers, incoming supplies, fuel and outgoing concentrate containers. Empty containers will be offloaded from the ocean going barges at the transshipment point onto the fleet of river barges and transported to site where they will be loaded with 24 tonnes of concentrate in super sacs.

Loaded containers will then be transported back to the transshipment site for delivery to the International shipping line in Seattle.

Potential Shortfall Effects

According to JDS analysis, although the probability of a shortfall in any given year was calculated to be 23% this represents an insignificant impact to the project's economics. The calculated 23.1% probability represents the likelihood of a shortfall in any given year, regardless of the size of the shortfall. The average shortfall over the 26 year data period was calculated to be 6,000 wmt. Given an 11-year mine life an estimated 15,180 wmt of concentrate would be the probable shortfall amount. This is approximately 1.75% of the total 867,623 wmt of concentrate to be shipped over the life of the mine.

However, the revenue associated with this 15,180 wmt shortfall would in all likelihood represent the least valuable Zinc Concentrate as more valuable Copper and Lead concentrates should be shipped to market first to maximum cash flow.

On a net revenue basis this 15,180 wmt of Zinc concentrate would equate to approximately $13.8M at a net revenue of $912.11/wmt for Zinc concentrate. This represents less than 1% of the total project net revenue of $1,472M. The revenue is not lost but potentially deferred for one year (until the next barging season).

18.14.2 Inbound Material Shipping

Fuel delivery will be by ISO bulk liquid containers with each container holding 24,000 l. Fuel supply will be from Juneau, AK as this is the closest bulk fuel facility. Fuel containers will be hauled via ocean barges to the transshipment site and then transferred to the river barges for transport to site.

Bulk consumables and materials will be shipped to the transshipment in ISO TEU containers site via ocean barges to the transshipment site. The containers will then be transferred to river barges for shipment to site. Containers will be offloaded by crane at the barge facility and trucked to Paddy's Flats for storage.

18.15 Warehousing & Storage

Paddy's Flats has sufficient space to accommodate 1,000 containers arranged in six areas of 4 x 15 x 3 (rows, columns and stack height) as shown in Figure 18.4. A Taylor TXLC 975 container handler will manage containers. Full containers will be moved to a working area on the ground for quick access by personnel and forklift. Once a container has been emptied, it will be re-handled to the bulk storage area and again stacked, ready for shipment offsite during the next barging season.

Consumables will be transported around site by tractor-trailer and the 20 t boom truck.

Six 20 ft sea containers located near the maintenance shop will be used for temporary warehousing of in-process parts and consumables.

18.16 Concentrate Storage & Handling

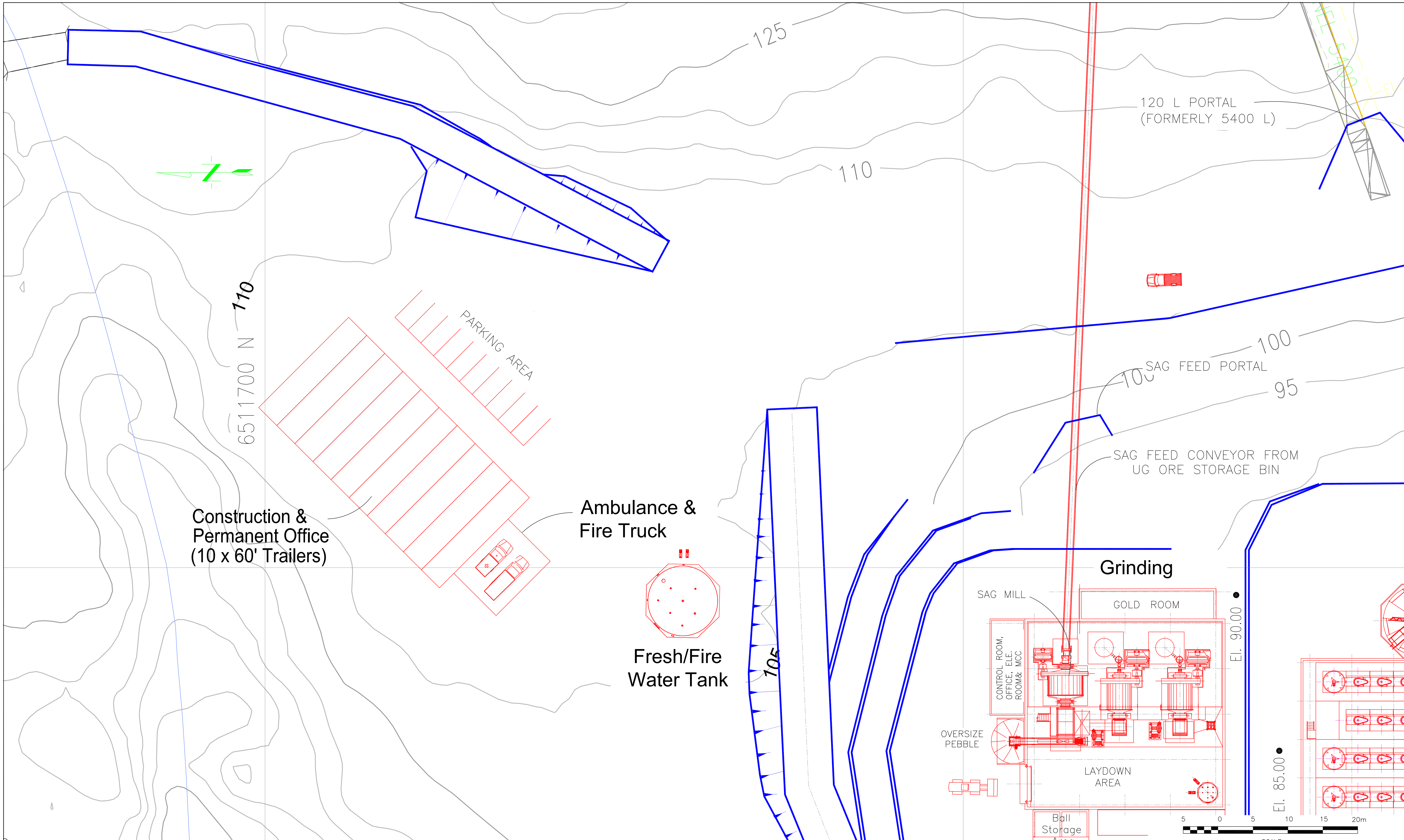
Concentrate will be loaded into 2 t bags at the plant site. The bags will be colour coded to clearly identify the different concentrate products. Bags will be trucked to the barge facility for storage in specially designated areas for each type of concentrate.

Tarps will cover the bags to prevent moisture from penetrating into the concentrate. If additional storage area is required, bags will be placed at Paddy's Flats.

During the barging season, approximately 12 bags will be packed into 20 ft sea containers using a forklift or bobcat. The containers will be loaded onto river barges for final shipment off site.

18.17 Administration Office

The administration offices will be housed in ten 60 ft trailers located on the upper bench of the plant site, as shown in Figure 18.12.



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		06/09/14

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	DATE:
SCALE:	
DESIGNED: J.D.S.	June 21 2014
DRAWN: B. Wong	June 21 2014
CHECKED:	
APPROVED:	

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
UPPER BENCH
GENERAL ARRANGEMENT
PLAN

PROJECT No.

DRAWING No. 1010-10-002

REV No. F1

The administration office will include the first aid room and parking for the fire truck and ambulance.

The trailers will be heated electrically. Power will be supplied from the main power plant.

18.18 Water Supply

18.18.1 Camp Potable Water

As noted by Sanitherm, water quality data for Shazah Well and Creek indicates the potable water treatment system has the potential to be a basic application consisting of multi-media pressure filters, followed by a two-stage cartridge filtration (Sanitherm, 2014). The system would include UV disinfection and chlorination for residual water quality requirements.

Raw water for the camp will be pumped from a well located near the water treatment plant to the fresh / fire water tank and from there to the potable water treatment plant.

The potable water treatment plant will be housed within two 40 ft modified sea containers including a 57,000 L storage tank and distribution pumps.

18.18.2 Sewage Wastewater & Solids Disposal

A modular wastewater treatment plant (WWTP) will be located near the camp facility. Treated effluent from the WWTP will be discharged. Solids generated will be dewatered, bagged and incinerated in the on-site incinerator facility.

18.18.3 Process Water

Raw water for process operations is planned to be drawn from the Tulsequah River or from the Effluent Treatment Plant, 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.

18.19 Fire Protection

The camp and plant site will have independent fire protection systems as detailed in the following section. A mobile fire truck will be provided for the site.

The camp fire protection system will be supplied from a 9 m diameter x 7 m tall combined fresh / fire water tank with a capacity of approximately 425,000 l. The plant site tank will be 10 m diameter x 8 m tall with a capacity of approximately 620,000 l.

The firewater reserve level in either tank will contain sufficient volume to supply the largest calculated fire flow for a minimum period of two hours. The distribution systems 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.

The accommodations complex will have a fire alarm system that will alert the entire complex in the case of a fire. Fire suppression will consist of hydrants, hose reels and chemical extinguishers and a stand-alone suppression system for the kitchen grease hoods.

Plant site firewater distribution is a combination of perimeter hydrants, hose cabinets in the buildings, and fire extinguishers. All occupied areas (staffed 24/7) and 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.

Each process building is self-contained with fire detection/alarm system that provides continuous monitoring to detect the location of any fire. Fire alarm panels in each building are connected to a central panel located in the grinding building, which is staffed 24/7.

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18.20 Incinerator

A 2 t per day batch incinerator for food waste, sewage sludge and other waste products will be located on a concrete pad near the permanent camp. The incinerator is estimated to burn 100 L of diesel per hour of operation depending on the material being incinerated, moisture content, etc.

18.21 Surface Explosives Magazines

Bulk explosives will be stored on surface in one of six secured and monitored magazines located along the road to the Big Bull site. The access road to the magazines will be gated and each magazine will be surrounded by a berm to prevent propagation in the event of an accidental detonation. Each magazine will store 40,000 kg's of explosives and will be separated from other magazines and inhabited areas by minimum distances according to Natural Resources Canada guidelines.

18.22 Communications

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 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.

18.23 Mine & Effluent Water Treatment

18.23.1 Acid Treatment Plant

The acid treatment plant (ATP) was designed to continually treat 40 m^3/h (and up to 100m^3/h for short periods) 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 18.9.

Table 18.6: Water Quality of Portal Discharges

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	0.1	0.034	0.05	0.05
Cadmium	0.51	0.31	0.23	0.18	0.11	0.1
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	74.6	78.7	45.6	27	23.7
SO_4	800	525	758	443	135	127
Ca	113	96.4	279	141	39	36
pH	2.9	3.2	4	4.5	7.1	6.7

Source of data: Redfern archived data in spreadsheet titled “Portal discharge quality 2005.xlsx” and spreadsheet titled “Water Treatment basis design V2.xls”. 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. Historic ATP reagent consumption rates are recorded in Table 18.10.

Table 18.7: 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

Source: Chieftain Metals, ATP Operational Records, May-June 2012

18.23.2 High Density Sludge Process

Improvements are planned to convert the existing water treatment process to a more reliable high density sludge process (HDS). The HDS process design flow is 97 m^3/hr and is expected to produce 30 to 40 kg/h of solids that will be included in the paste backfill mix. The HDS process reduces the unit rate of sludge production due to increased sludge density and improves sludge stability, both chemically and physically (Applied Water Treatment, 2014).

Applied Water Treatment (AWT) summarizes the process as follows: Lime and recycled sludge are added to the lime-sludge mix tank at the head of the process and this becomes the main neutralization agent. This mixture is discharged to the rapid mix tank where it is mixed with influent, thereby achieving neutralization. This mixture is fed to the main lime reactor where a combination of aggressive aeration and high shear agitation ensures optimum process chemistry and clarifier performance. The discharge from the lime reactor is treated with flocculent in the flocculation tank. The clarifier separates the treated effluent from the sludge, a portion of which is recycled to the head of the process. Clarifier overflow will be pumped through the existing polishing filter to ensure total suspended solids meet discharge requirements.

The existing reactor tanks in the ATP will be modified to improve flow and include air sparging. The discharge from the second reactor tank will be transferred to a new conventional clarifier located outside the existing building. A small lime-sludge mix tank will be installed on top of the existing tank.

Theoretical reagents consumption is shown in Table 18.11.

Table 18.8: HDS Process Reagent Consumption

Reagent	Units	Consumption
Hydrated Lime	g/m^3	166 to 180
Flocculent	g/m^3	1 to 1.5

Source of data: Applied Water Treatment report, “Tulsequah Chief Project Water Treatment Plant Feasibility Design” (November 3rd, 2014). Hydrated Lime consumption is based on Option 1 equipment and assuming 50 to 60 % lime consumption.

18.23.3 Effluent Treatment Plant

The effluent treatment plant (ETP) was designed to treat the following water sources produced during mine operations: mill water, tailings reclaim 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 nonmetals. 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. ETP reagent consumption rates are recorded in Table 18.12.

Table 18.9: ETP Reagent Consumption Rates

Reagent	Units	Consumption
Hydrated Lime	g/m^3	110
Flocculent	g/m^3	1 to 1.5

* Dosages have not been laboratory verified.

18.24 Waste Production Schedule

The waste production schedule forecast as part of the mine plan is shown in Table 18.13.

Table 18.10: Waste Production Schedule

Period		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Days	Unit				365	365	366	365	365	365	366	365	365	3,287
Ore Processed	tonnes	24,276	323,910	405,412	409,366	412,981	409,309	409,247	408,475	407,774	409,881	410,825	404,163	**4,435,619**
HPAG Placed Underground	tonnes	-	-	-	-	-	22,000	35,000	33,000	30,000	-	-	-	**120,000**
PAG Waste														
PAG Produced	tonnes	113,818	90,847	56,972	48,464	43,414	21,967	45,900	40,880	22,772	19,266	7,848	11,133	**523,281**
PAG to Surface Waste Dump	tonnes	113,818	21,749	-	-	-	-	-	-	-	-	-	-	**135,567**
PAG Direct to UG	tonnes	-	69,098	56,972	48,464	43,414	21,967	45,900	40,880	22,772	19,266	7,848	11,133	**387,714**
PAG from Surface Dump to UG	tonnes	-	-	56,542	-	79,025	-	-	-	-	-	-	-	**135,567**
PAG to Backfill	tonnes	-	69,098	113,514	48,464	122,439	21,967	45,900	40,880	22,772	19,266	7,848	11,133	**523,281**
PAG Dump Balance	tonnes	113,818	135,567	79,025	79,025	-	-	-	-	-	-	-	-	**407,435**
NAG Waste														
NAG Produced	tonnes	63,400	80,574	99,897	123,281	71,828	51,813	38,529	8,836	-	-	-	7,797	**545,955**
NAG to Surface Waste Dump	tonnes	63,400	80,574	99,897	114,122	18,223	15,901	-	-	-	-	-	-	**392,116**
NAG Direct to UG	tonnes	-	-	-	9,159	53,605	35,912	38,529	8,836	-	-	-	7,797	**153,838**
NAG from Surface Dump to UG	tonnes	-	-	-	-	-	-	-	42,109	21,988	20,000	20,000	20,000	**124,097**
NAG to Backfill	tonnes	-	-	-	9,159	53,605	35,912	38,529	50,945	21,988	20,000	20,000	27,797	**277,935**
NAG Dump Balance	tonnes	63,400	143,974	243,871	357,993	376,216	392,116	392,116	350,008	328,020	308,020	288,020	268,020	
Pyrite Concentrate														
Pyrite Concentrate Produced	tonnes	7,950	106,081	132,772	134,067	135,251	134,049	134,028	133,776	133,546	134,236	134,545	132,363	**1,452,665**
Pyrite Concentrate to Surface Pyrite Surge Pond	tonnes	7,950	3,104	26,500	-	77,000	-	3,000	-	-	-	-	-	**117,555**
Cumulative Pyrite Concentrate Surface Surge Pond	tonnes	11,193	87,239	189,156	-	77,000	42,000	45,000	30,000	-	-	-	-	
Pyrite Concentrate Straight to Paste Backfill	tonnes	-	102,976	106,272	134,067	58,251	134,049	131,028	133,776	133,546	134,236	134,545	132,363	**1,335,111**
Pyrite Concentrate from Surface Pyrite Surge Pond	tonnes	-	3,000	600	33,955	-	35,000	-	15,000	30,000	-	-	-	**117,555**
Total Pyrite Concentrate to Backfill	tonnes	-	105,976	106,872	168,022	58,251	169,049	131,028	148,776	163,546	134,236	134,545	132,363	**1,452,665**
Tailings														
Tailings Produced	tonnes	11,830	157,841	197,557	199,484	201,246	199,456	199,426	199,050	198,708	199,735	200,195	196,949	**2,161,477**
Tailings to Tailings Pond	tonnes	11,830	113,935	147,751	119,313	193,766	190,696	192,020	190,239	164,458	140,005	112,080	121,719	**1,697,809**
Tailings to Paste Backfill	tonnes	-	39,969	44,700	76,048	784	2,171	771	2,237	28,567	54,892	84,242	71,024	**405,406**
Limestone to Tailings Pond	tonnes	409	3,937	5,106	4,123	6,696	6,590	6,636	6,574	5,683	4,838	3,873	4,206	**58,672**
Total Tailings to Tailings Pond	tonnes	12,238	117,872	152,857	123,436	200,462	197,285	198,655	196,813	170,142	144,843	115,953	125,925	**1,756,480**
Backfill														
PAG (operations + Historical)	tonnes	-	69,098	113,514	48,464	122,439	43,967	80,900	73,880	52,772	19,266	7,848	11,133	**643,281**
NAG	tonnes	-	-	-	9,159	53,605	35,912	38,529	50,945	21,988	20,000	20,000	27,797	**277,935**
Total Pyrite Concentrate to Backfill	tonnes	-	105,976	106,872	168,022	58,251	169,049	131,028	148,776	163,546	134,236	134,545	132,363	**1,452,665**
Tailings & Limestone to Paste Backfill above 5200 (old voids)	tonnes	-	24,000	36,000	35,000	-	-	-	-	-	-	-	-	**95,000**
Tailings to Paste Backfill below 5200	tonnes	-	15,969	8,700	41,048	784	2,171	771	2,237	28,567	54,892	84,242	71,024	**310,406**
Cement (included in backfill tonnage)	tonnes	-	4,497	6,085	6,873	1,966	3,765	2,880	3,348	4,182	5,453	5,814	5,019	**49,883**
Total Paste Backfill Required below 5200	**tonnes**	-	101,598	115,573	209,070	59,035	171,220	131,799	151,013	192,113	189,128	218,788	203,387	**1,742,723**
Total Paste Backfill Placed	**tonnes**	-	145,946	151,573	244,070	59,035	171,220	131,799	151,013	192,113	189,128	218,788	203,387	**1,858,071**
Total Backfill	**tonnes**	-	215,044	265,087	301,693	235,079	251,099	251,228	275,837	266,873	228,394	246,636	242,317	**2,779,287**

18.25 Tailings Management Facility

18.25.1 Summary of Facility

Detailed design of the tailings management facility (TMF) is presented in the KCB report entitled, "Tailings Management Facility – Detail Design," (February, 2012). As part of the updated 2014 Feasibility Study, two major revisions were incorporated:

- The option of constructing a Starter Dam to provide storage for 1.5 years was assessed and adopted; and
- The total mine tailings tonnage was reduced from 3 Mt to a revised value of 1.76 Mt, with the option to increase the storage capacity in the future.

The results of the study are presented in the KCB report entitled, "Starter Dam Trade-Off Study" (October 2014). Prior to completion of the initial 1.5 year operating period the tailings facility would be expanded to its design capacity of 1.76 Mt. The Ultimate Dam is designed to the Canadian Dam Association Guidelines (2007) for a "High" consequence structure.

The TMF is located approximately 4 km upstream (north) of the main mine facilities on the east bank of the Shazah Creek. The impoundment will be formed with a homogeneous compacted earthfill dams with a 1.5 mm (60 mil) LLDPE geomembrane liner. The Starter Dam will be 5 m high and cover approximately 10 ha and the Ultimate Dam will be 9 m high and cover 45 ha. The dams will be constructed using material excavated from within the impoundment area.

The embankment will have a 6 m wide crest at El. 75.1 m (up to 9 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).

Water from the TMF will be recycled back to the process plant, and storage is provided for the 1/200 year 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 18.13.

18.25.2 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 “Significant” for the Starter Dam and “High” for the Ultimate Dam.

For a “High” consequence dam, the maximum design earthquake (MDE) has an 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; and
- 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.

18.25.2 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 “Significant” for the Starter Dam and “High” for the Ultimate Dam.

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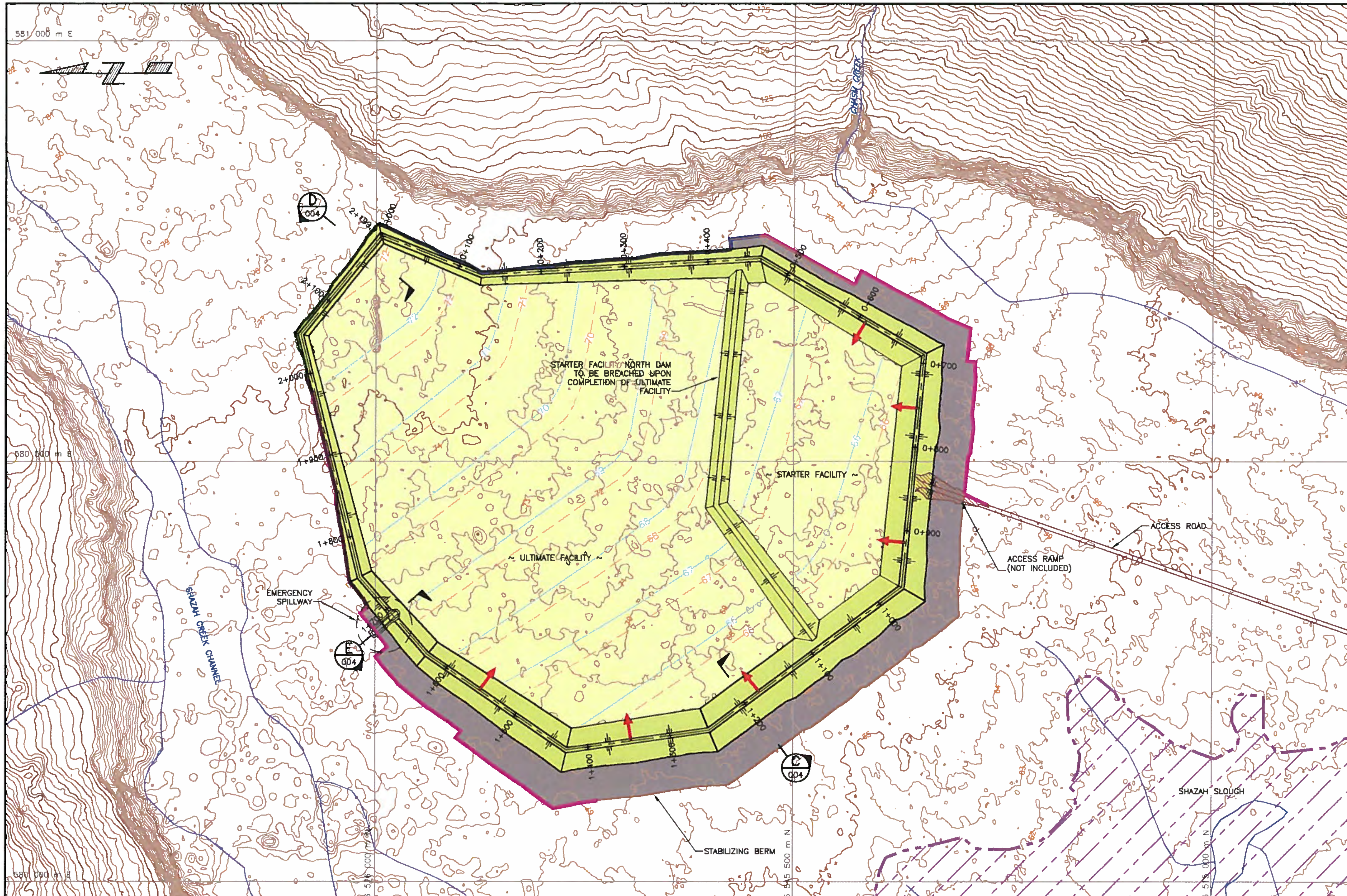
- The inflow design flood (IDF) is 1/3 between 1/1,000 year and the probable maximum flood (PMF);
- The TMF will store the 30-day 1/200 year precipitation event;
- The TMF will be designed with an emergency spillway to route the IDF during operations;
- The TMF will be designed with a permanent closure spillway to route the peak flow from the IDF on closure; and
- A minimum of 1 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.



PLAN VIEW
SCALE A

STABILIZING BERM SIZING

STATION		BERM WIDTH
FROM	TO	(m)
0+000	0+430	0
0+430	0+580	12
0+580	0+700	30
0+700	1+550	40
1+550	1+680	30
1+680	1+720	12
1+720	2+217	0

LAUNCHING APRON RIPRAP SIZING

COLOR	RIPRAP CLASS	DETAIL
(blue)	RIPRAP A	FIGURE 6.3, DETAIL 2
(magenta)	RIPRAP B	FIGURE 6.3, DETAIL 3
(green)	RIPRAP C	FIGURE 6.3, DETAIL 3

LAUNCHING APRON RIPRAP

MATERIAL	LAYER THICKNESS (mm)	D_{50}
RIPRAP A	500	250
RIPRAP B	1000	500
RIPRAP C	1700	850

• FILTER SPECIFICATION NOT SHOWN.

LEGEND

- 69 — INTERPRETED WATER TABLE CONTOURS (NOTE 2)
- 69 — EXCAVATION CONTOURS
- 100 — EXISTING GROUND CONTOURS
- TAILINGS DISCHARGE LOCATION

NOTES:

1. ALL UNITS ARE METRIC, UNLESS OTHERWISE STATED.
2. WATER CONTOURS BASED ON MAX. WATER TABLE ELEVATION AS PER SITE DATA.

SCALE A 0 100 m

NOT FOR CONSTRUCTION

OCTOBER, 2014

CLIENT: CHIEFTAIN METALS CORP

Klohn Crippen Berger

PROJECT: TULSEQUAH TAILINGS MANAGEMENT FACILITY

TITLE: ULTIMATE DAM PLAN

SCALE	PROJECT NO.	DWG. NO.	REV.
–	M09780A02	3010-10-003	–

CANCEL PRINTS BEARING PREVIOUS REVISION

KCB-DWG-D-L

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Xrefs: BM-Tulsequah_Apr08

18.25.3 Site Conditions

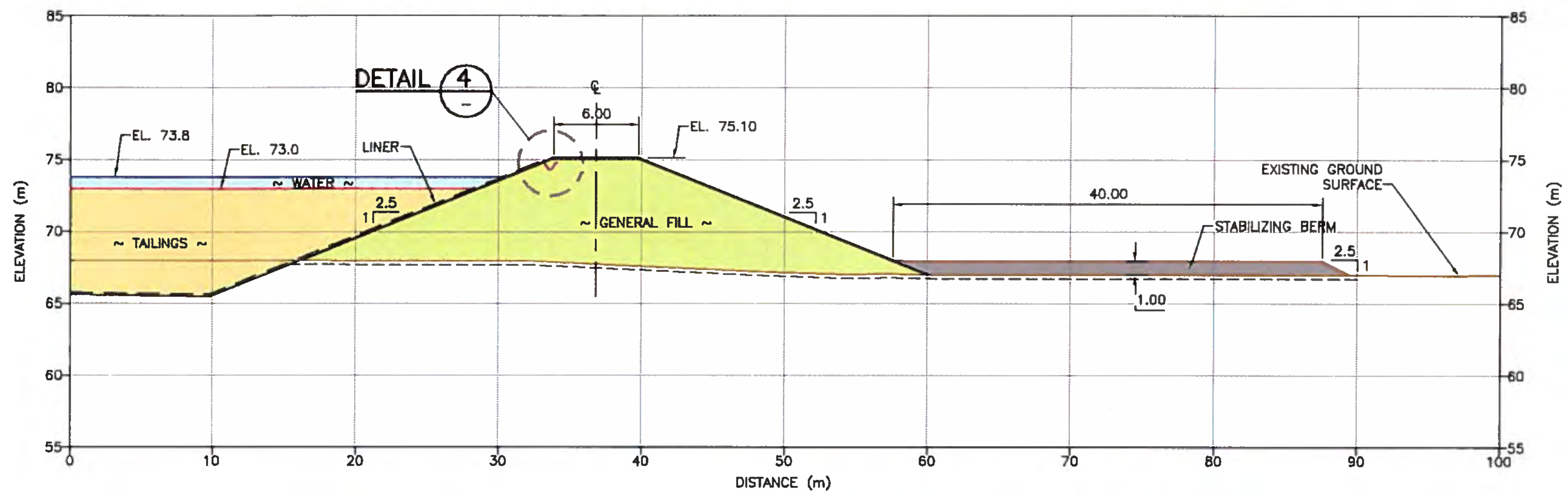
The TMF site is located in mountainous terrain on the northern coast of BC, bordering Alaska. The mountaintops are steep, rugged ridges and peaks, while the valleys have forested, steeply sloping sides with networks of narrow streams and creeks flowing into larger rivers that meander across wide, flat 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 10-5 m/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.

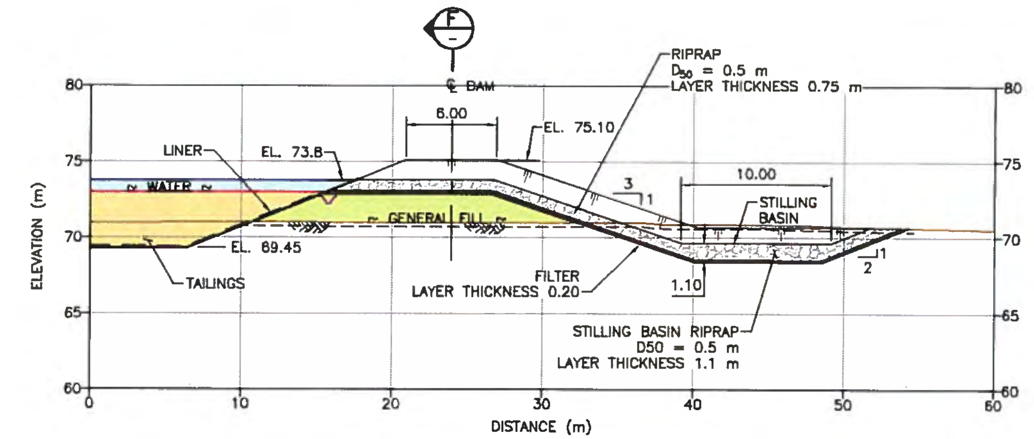
A seismic hazard assessment was carried out to determine the appropriate seismic parameters for the selected seismic design criteria. Probabilistic and deterministic seismic hazard analyses were conducted to derive the MDE parameters for the design return period of 2,475 years. The peak ground accelerations of between 0.06 and 0.20 g were predicted.

18.25.4 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 9 m high (to 75 m elevation), with upstream and downstream slopes of 2.5H:1V. The toe berm will be 1 m thick and from 18 m to 40 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. A 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.



SECTION C
SCALE A
003



SECTION E
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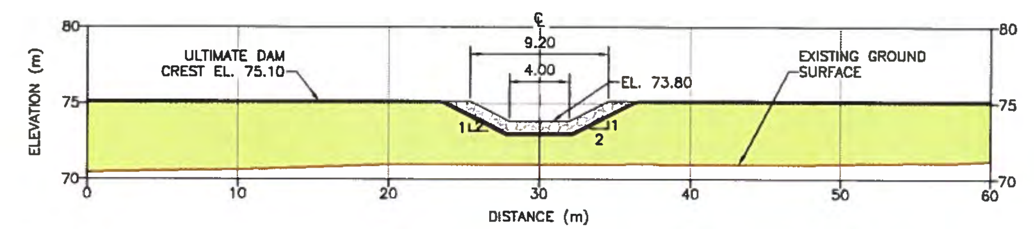

SECTION F
SCALE A
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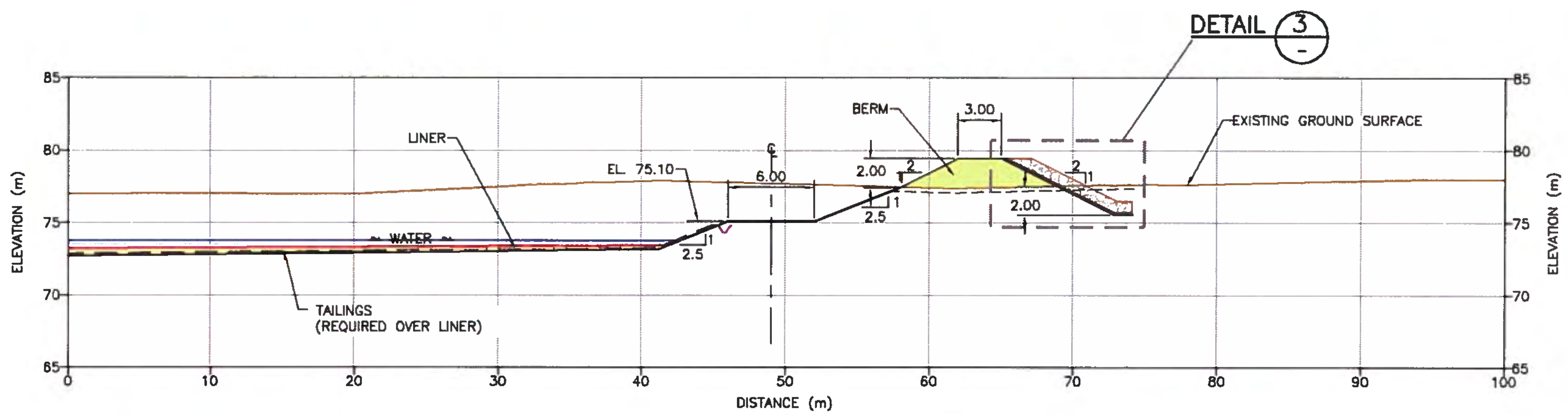
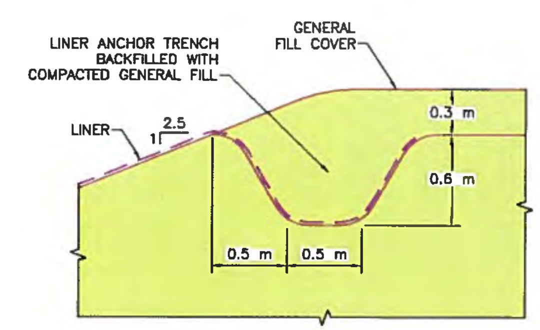


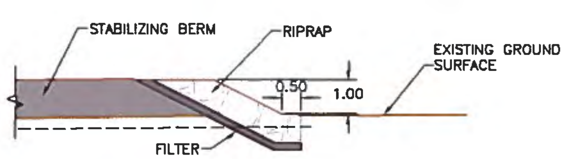
SECTION D
SCALE A
003



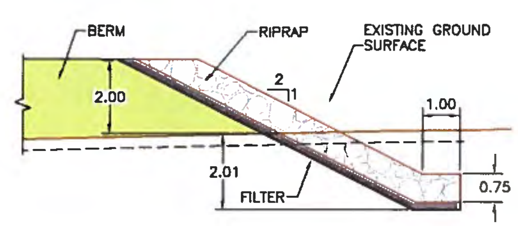
DETAIL 4 – LINER ANCHOR TRENCH (TYPICAL)
NTS
ULTIMATE FACILITY

NOTE:

1. ALL UNITS ARE METRIC, UNLESS OTHERWISE STATED.



DETAIL 2 Fig.6.2 LAUNCHING APRON RIPRAP
NTS
(RIPRAP CLASS A)



DETAIL 3 – LAUNCHING APRON RIPRAP
NTS
(RIPRAP CLASS B AND C)

NOT FOR CONSTRUCTION

OCTOBER, 2014

SCALE A 0 10 m

AS A MUTUAL PROTECTION TO OUR CLIENT, THE PUBLIC AND OURSELVES, ALL REPORTS AND DRAWINGS ARE SUBMITTED FOR THE CONFIDENTIAL INFORMATION OF OUR CLIENT FOR A SPECIFIC PROJECT AND AUTHORIZATION FOR USE AND/OR PUBLICATION OF DATA, STATEMENTS, CONCLUSIONS OR ABSTRACTS FROM OR REGARDING OUR REPORTS AND DRAWINGS IS RESERVED PENDING OUR WRITTEN APPROVAL.

CLIENT: CHIEFTAIN METALS CORP

Klohn Crippen Berger

PROJECT: TULSEQUAH TAILINGS MANAGEMENT FACILITY

TITLE: ULTIMATE DAM SECTIONS AND WATER MANAGEMENT DETAIL

SCALE	PROJECT NO.	DWG. NO.	REV.
–	M09780A02	3010-10-004	–

CANCEL PRINTS BEARING PREVIOUS REVISION

KCB-DWG-D-L

Time: 16:30:52
Date: 02/10/2014
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Drawing File: Z:\M\VCR\M09780A02 - CMI -Tulsequah Tailings FS\400 Drawings\Figures\3010-10-004.dwg (persyem)
Xrefs: BM-Tulsequah_Apr08

The dam stability assessment is based on the assumption that portions of the foundation soils are potentially liquefiable, as indicated 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 strength condition).

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.

18.25.5 Tailings Deposition & Water Management

Based on a total production over the life of mine, approximately 1.76 Mt of tailings will be deposited in the TMF. The average flow rate of tailings will be 438 tpd 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 75.1 m would provide sufficient volume to store 1.76 Mt and provide freeboard for the closure spillway. A water pond will form at the southern side 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 northwest 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 65,000 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 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 73.8 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.

18.25.6 Closure Plan

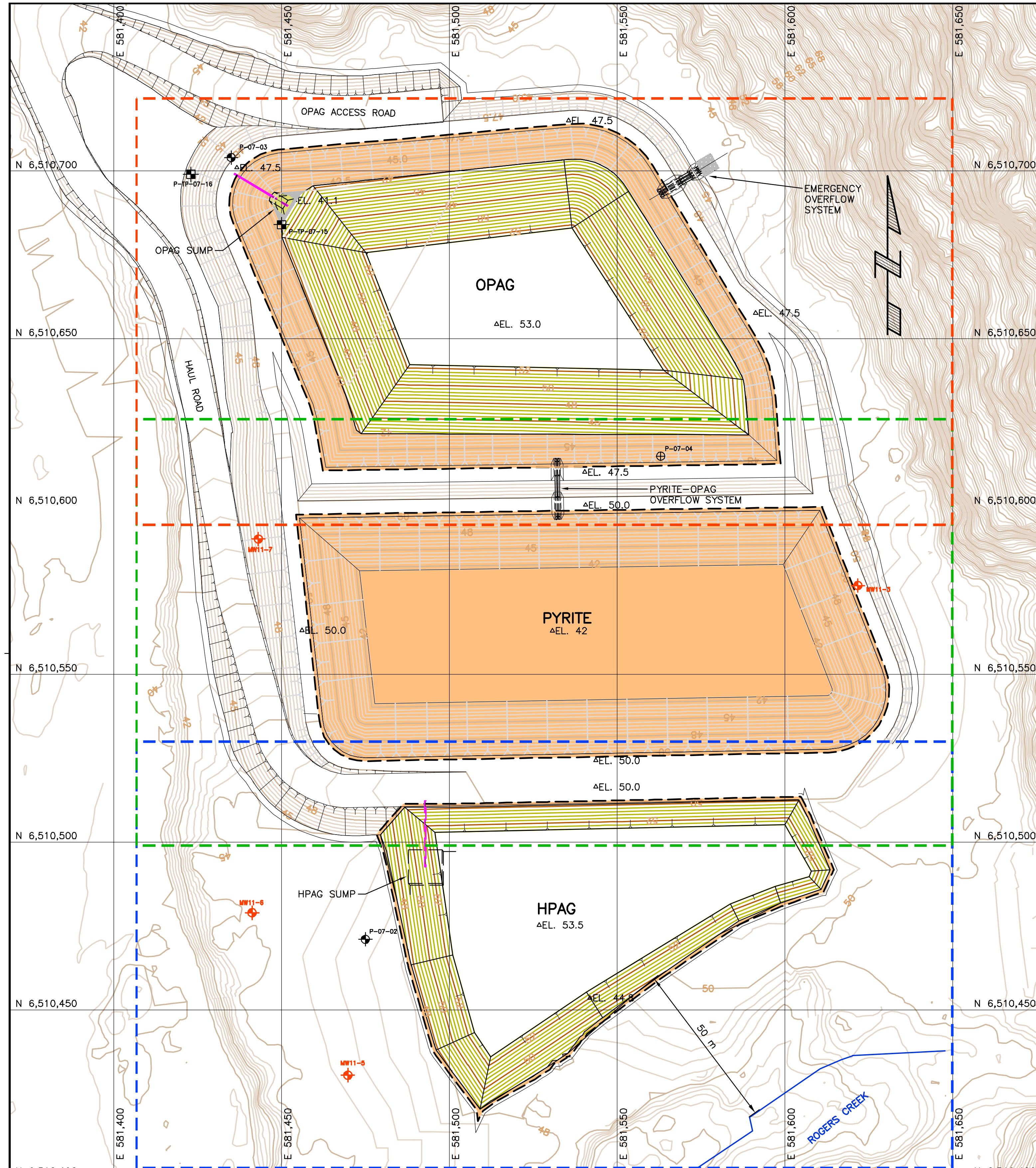
The main areas of focus of the closure program will be erosion control, embankment stability, storm-water management, and revegetation. Establishing a surface cover of vegetation will reduce the potential for adverse environmental impacts such as erosion, as well as improving wildlife habitat and visual 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 re-contoured to conform to the surrounding terrain and reduce the visual impact. Tailings deposits are not anticipated to require re-contouring 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. 73.8 m to approximately El. 73 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. 73 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.

After the closure works have been completed and the TMF no longer has a water storage pond, the Consequence classification of the facility will reduce from “High” to “Significant” as the potential for tailings run-out will be reduced.

18.26 PAG / Pyrite Facilities

The historical potentially acid-generating (HPAG), operating potentially acid-generating (OPAG) and pyrite tailings storage facilities are located approximately 1 km south of the main plant site. The general arrangement of the facilities is shown in Figure 18.15. 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 140,000 kt of HPAG, 120,000 kt of OPAG and 75,000 t 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).



PLAN

NOTES:

1. FOUNDATION DELETERIOUS AND SOFT MATERIAL SHALL BE REMOVED. FOUNDATION SHALL BE COMPACTED WITH 4 PASSES OF A SMOOTH DRUM VIBRATORY ROLLER TO OBTAIN FIRM AND NON PROTRUDING SURFACE.

2. NOTE 2:

LLDPE LINER			
FACILITY	ZONE	TEXTURE	THICKNESS (mm)
HPAG	Basin	Double	1.00
HPAG	Cover	Smooth	1.00
Pyrite	Basin	Smooth	1.00
OPAG	Basin	Double	1.50

3. LINER PROTECTION FILL WILL CONSIST OF SILTY SAND AND GRAVEL WITH A MAXIMUM PARTICLE SIZE OF 200mm AND A MINIMUM THICKNESS OF 500mm

4. COMPACTED FILL WILL BE PLACED IN 300mm LOOSE LIFTS, WETTED AND COMPACTED TO 95% MAXIMUM STANDARD PROCTOR DRY DENSITY. FOR LINER ANCHOR TRENCHES A MAXIMUM 50mm PARTICLE SIZE SHALL BE USED.

5. HDPE PIPE WILL BE 300mm IN DIAMETER WITH AN SDR OF 26. IT WILL HAVE (4) 10mm PERFORATIONS, SPACED EVERY 200 mm AND A BOTTOM END CAP WITH (9) 10mm PERFORATIONS.

6. CORRUGATED STEEL PIPE (CSP) SHALL BE CAN/CSA-G401 WISH INTERIOR AND EXTERIOR GALVANIZED COATING.

7. HPAG SUMP WILL HAVE AT TOP PLATFORM OF 11mx11m. OPAG SUMP WILL HAVE AT TOP PLATFORM OF 5mx3m.

8. " " DENOTES BOREHOLES FROM WHICH GROUND WATER LEVEL INFORMATION WAS OBTAINED.

9. " " DENOTES MONITORING WELLS WITH APPROXIMATE LOCATION. NO SURVEY DATA AVAOLABLE.

NOT FOR CONSTRUCTION | DRAFT

SCALE A 0 50 m

DRAWING NO.	REFERENCE DRAWING	NO.	DATE	ISSUE / REVISION	DRAWN	CHK'D	DESIGN	APP'D

CLIENT: CHIEFTAIN METALS INC

Klohn Crippen Berger

PROJECT: TULSEQUAH CHIEF PROJECT

TITLE: GENERAL LAYOUT

SCALE	PROJECT No.	DWG. No.	REV.
AS SHOWN	M09780A01	D-1001	0

CANCEL PRINTS BEARING PREVIOUS REVISION

KCB-C-MLD

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Drawing File: M:\VCR\M09780A01 - ChieftonPyritePond\400 Drawings\2012-design-report\FC draft\D-1001TO0-1005.dwg (persyem)

18.26.1 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 has been 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.

18.26.2 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.

18.26.3 OPAG

The OPAG facility will store up to 120,000 t 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.

If the quantity of OPAG exceeds the design capacity of 120,000 t, surplus will be stored in the pyrite pond until it can be re-handled underground as backfill.

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.

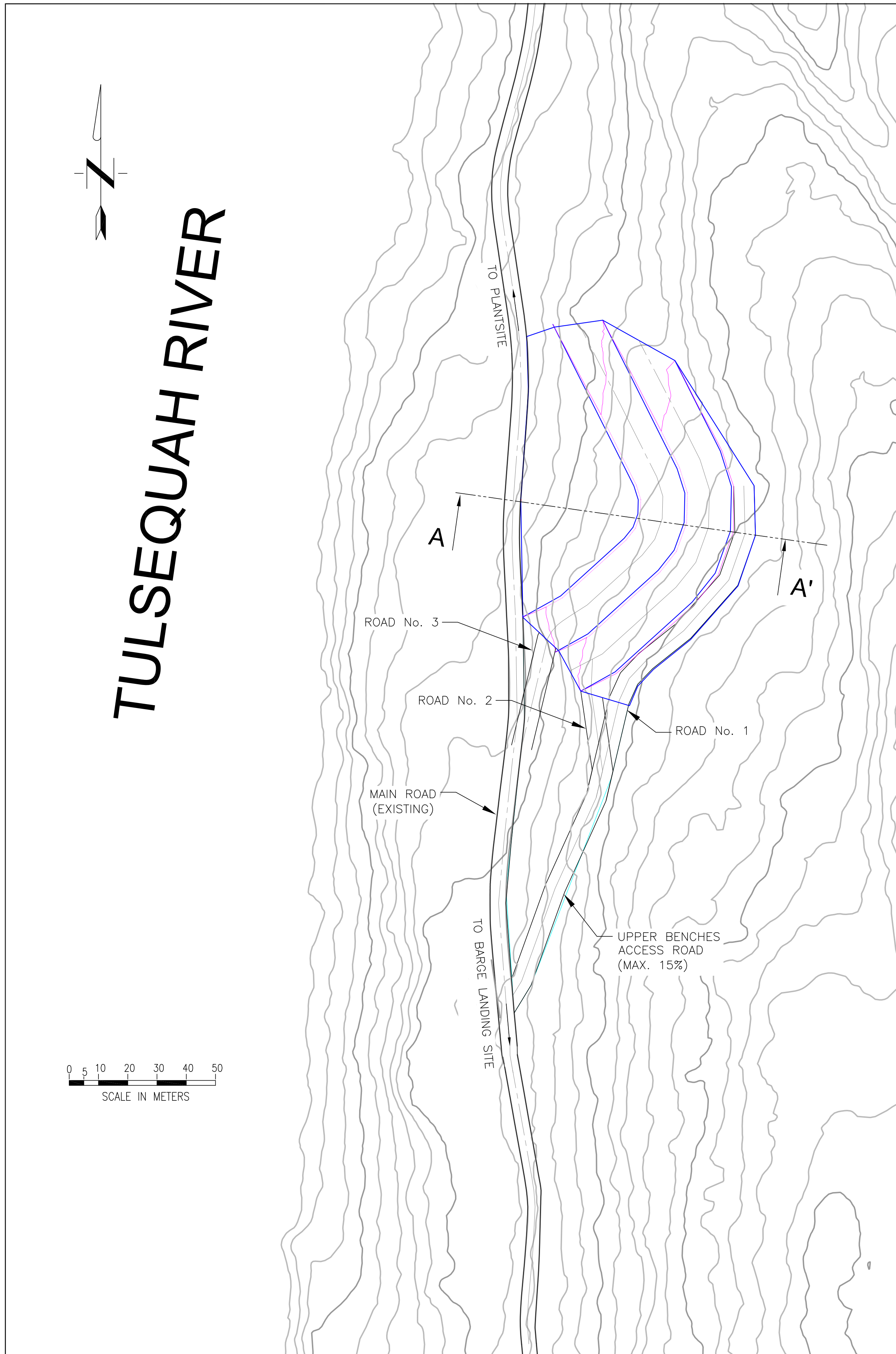
18.26.4 Closure

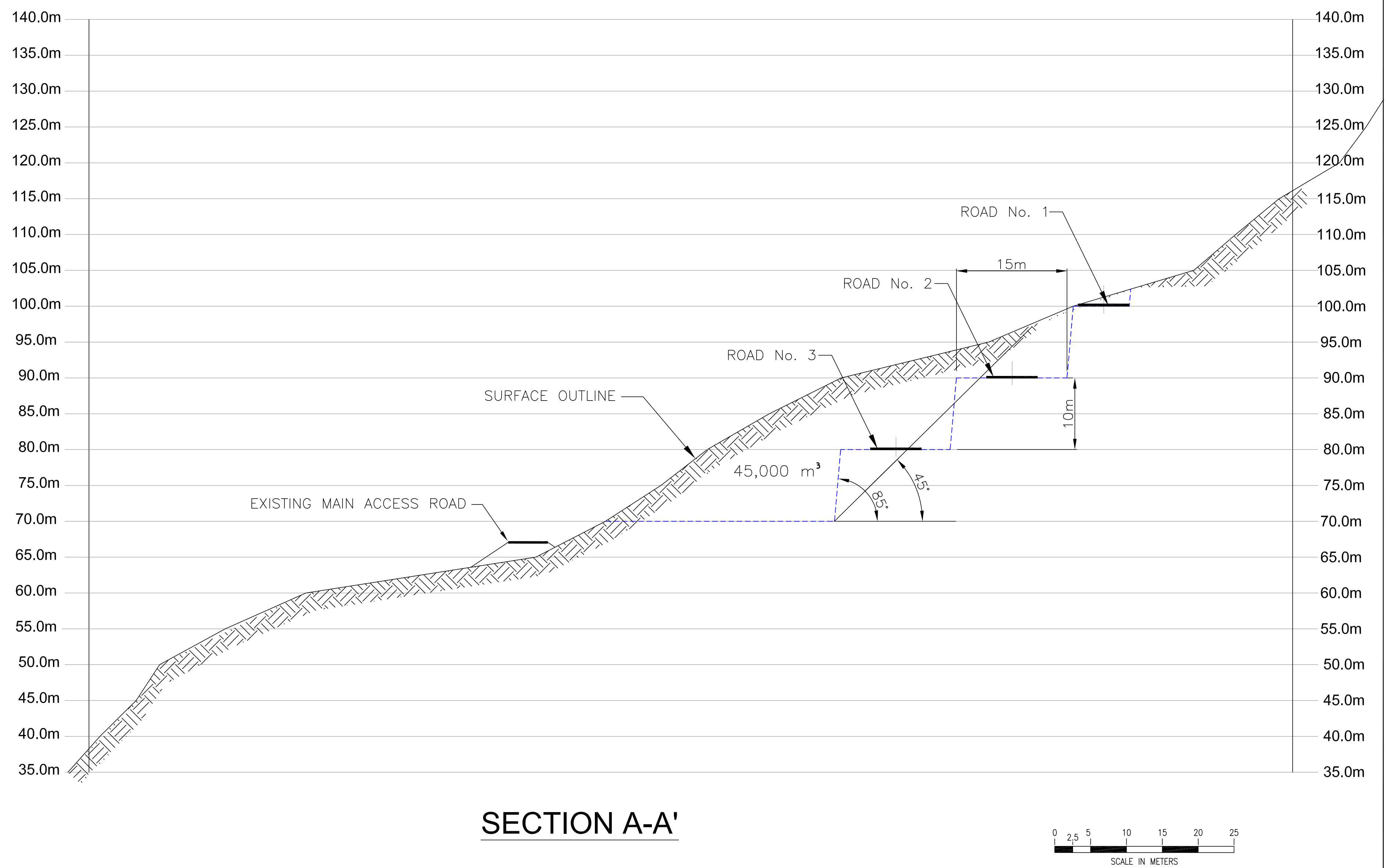
At the end of mine operations all remaining OPAG, HPAG, pyrite tailings and any other contaminated fill will be removed from the storage facilities and stored underground. The embankments will be decommissioned and the basins filled, and the geomembrane liners will be disposed of appropriately. The disturbed areas will be re-contoured, covered with topsoil and re-vegetated / re-seeded.

18.26.5 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, crushing and stockpiling of limestone from the quarry will be conducted by mine personnel on an as-needed basis. Stockpiled limestone will be transported via highway haul truck to the processing plant at a rate of up to 40 tpd.

Figure 18.16 shows the general arrangement of the limestone quarry.





SECTION A-A'

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	S.K.		25/09/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	S.K.		15/09/2014

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:		
DESIGNED:	J.D.S.	Sept 13 2014
DRAWN:	B. Wong	Sept 13 2014
CHECKED:		
APPROVED:		

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
LIMESTONE QUARRY
GENERAL ARRANGEMENT
PLAN AND SECTION

PROJECT No.	DRAWING No.	REV No.
	2010-10-001	F1

19. MARKET STUDIES AND CONTRACTS

19.1 Market Studies

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 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 Gordon Doerksen, P. Eng.

Smelter terms were identified for copper, zinc, lead, and silver and gold doré and are considered to be in line with the current market conditions and have been considered in the economic analysis.

A Letter of Intent has been signed with a marine transportation company for the barging of concentrate from the Tulsequah Chief project to the port of Seattle. No contractual arrangements for concentrate shipping, port usage, shipping, smelting or refining exists at this time. Table 19.1 through Table 19.4 outline the concentrate transportation cost and smelter terms used in the economic analysis.

Table 19.1: Gravity Concentrate Smelter Terms

NSR Assumptions	Unit	Gravity Concentrate
Recoveries		
Au	%	41
Ag	%	0.5
Smelter Payables		
Au Payable	%	99.9
Ag Payable	%	99
Refining Charge		
Au	US $/oz	0.65
Ag	US $/oz	0.65
Shipping Cost	US$/payable oz	1.15

Table 19.2: Copper Concentrate Smelter Terms

NSR Assumptions	Unit	Cu Concentrate
Recoveries		
Cu	%	89
Au	%	47
Ag	%	77.6
Concentrate Grade	%	21
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	96.5
Au Payable	%	95
Ag Payable	%	90
Minimum Deduction in Concentrate	%	1
Au Minimum Deduction	g/t	0
Ag Minimum Deduction	g/t	30
TC/RCs		
Treatment Charge	US$/dmt concentrate	150
Refining Charge		
Cu	US$/lb	0.15
Au	US$/oz	6
Ag	US$/oz	0.5
Deleterious Element Penalties		
As	US$/dmt concentrate	41.2
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

Table 19.3: Pb Concentrate Smelter Terms

NSR Parameters	Unit	Pb Concentrate
Recoveries		
Pb	%	65
Au	%	2.8
Ag	%	6.3
Concentrate Grade	%	60
Moisture Content	%	8
Smelter Payables		
Pb Payable	%	95
Au Payable	%	95
Ag Payable	%	95
Minimum Deduction in Conc	%	3
Au Minimum Deduction	g/t	1.5
Ag Minimum Deduction	g/t	50
TC/RCs		
Treatment Charge	$/dmt concentrate	100
Refining Charge		
Au	US $/oz	25
Ag	US $/oz	1.5
Escalator Costs		
Pb	$/dmt concentrate	1.70
Threshold	*$/tonne*	2000
Charge	*$/tonne*	0.04
Threshold	*$/tonne*	2500
Charge	*$/tonne*	0.06
Threshold	*$/tonne*	3000
Charge	*$/tonne*	0.08
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

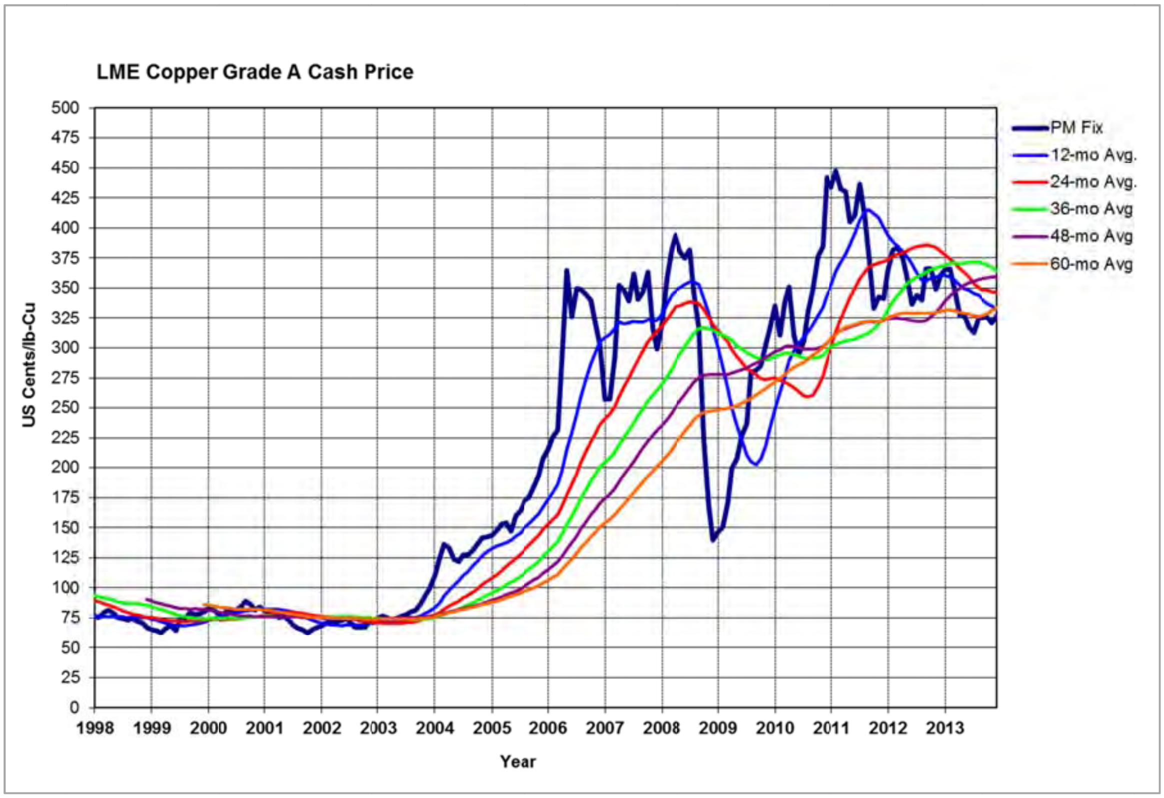
Table 19.4: Zn Concentrate Smelter Terms

NSR Parameters	Unit	Zn Concentrate
Recoveries		
Zn	%	90
Concentrate Grade	**%**	**60**
Moisture Content	%	8
Smelter Payables		
Zn Payable	%	85
Minimum Deduction in Conc	%	8
TC/RCs		
Treatment Charge	$/dmt concentrate	165
Escalator Costs		
Zn	$/dmt concentrate	13.20
Threshold	*$/tonne*	2000
Charge	*$/tonne*	0.04
Threshold	*$/tonne*	2500
Charge	*$/tonne*	0.06
Threshold	*$/tonne*	3000
Charge	*$/tonne*	0.08
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

19.2 Metal Prices

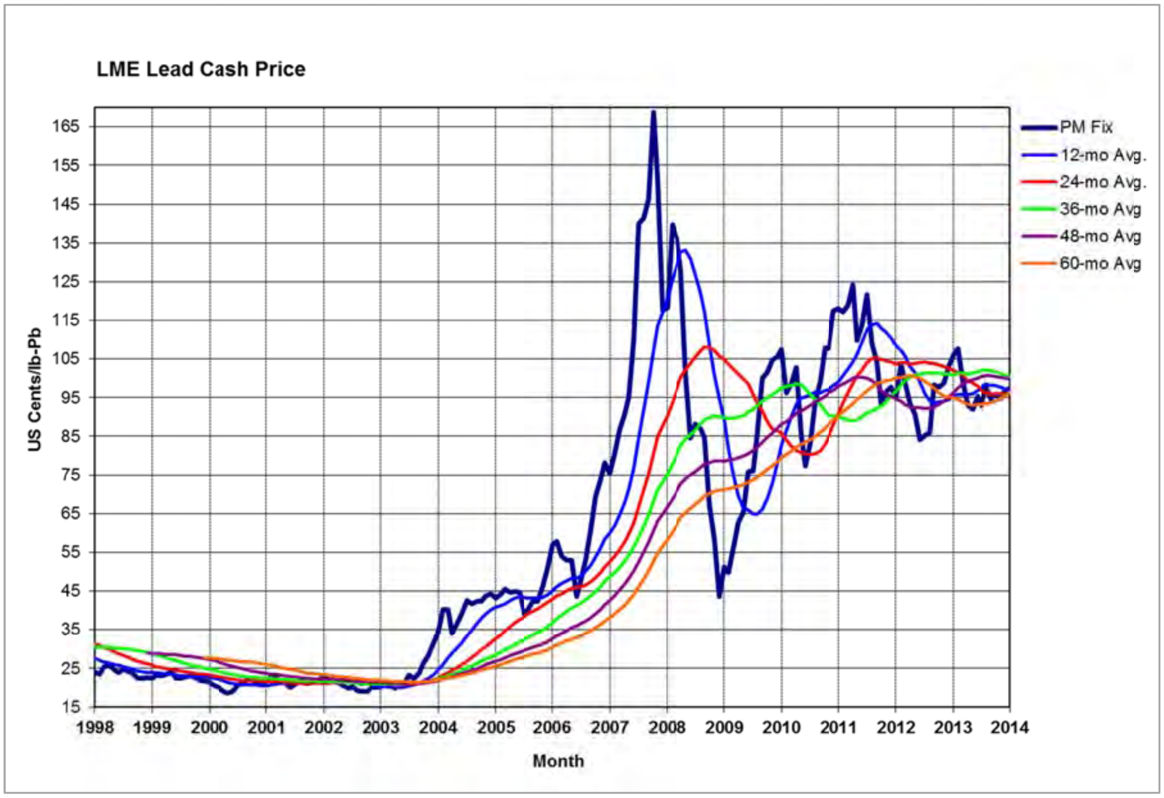
The base and precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo and Hong Kong). Historical metal prices are shown in Figure 19.1 through Figure 19.5.and demonstrate the change in metal prices from 1998 to 2014.

Figure 19.1: Historical Copper Price



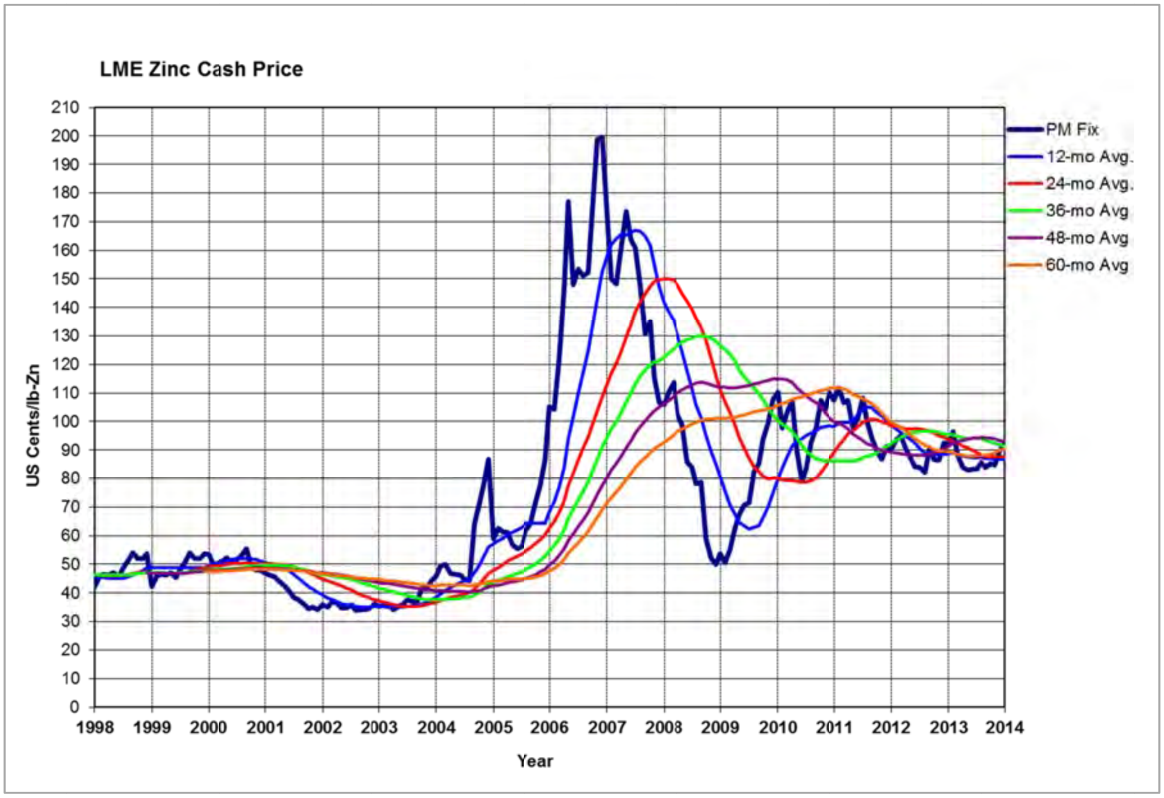
Source: JDS 2014

Figure 19.2: Historical Lead Price



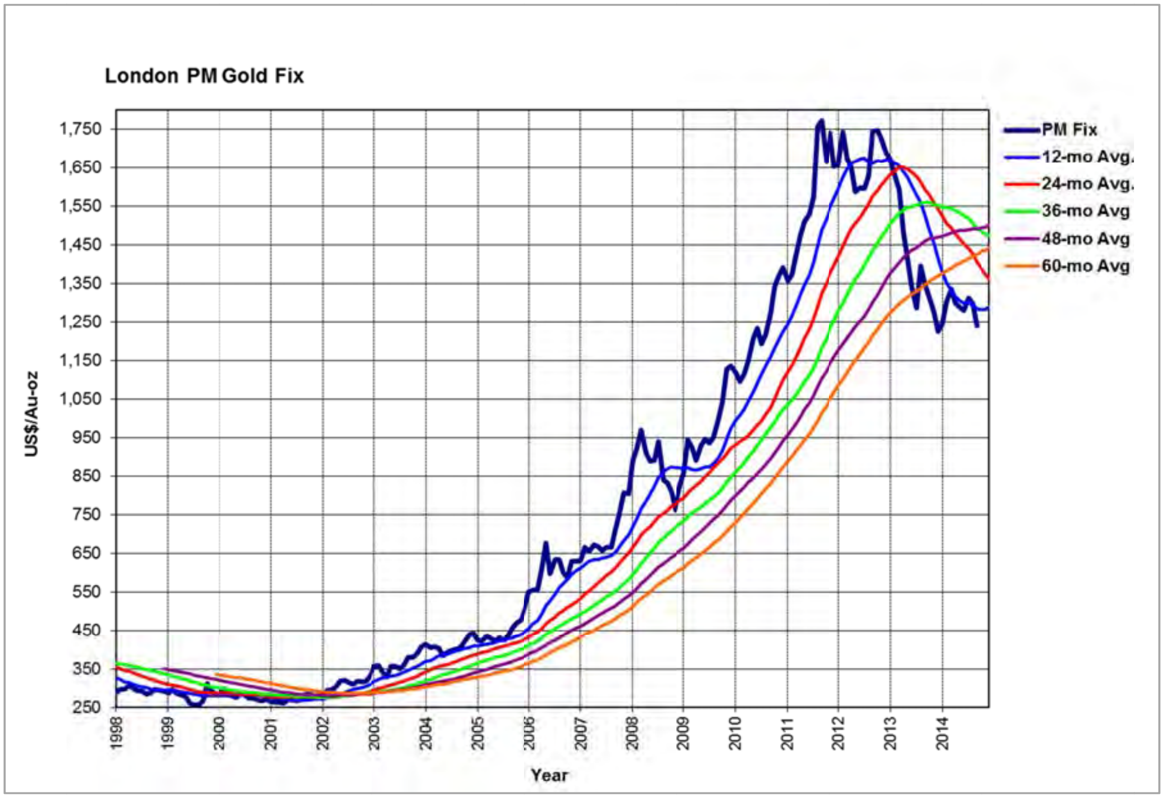
Source: JDS 2014

Figure 19.3: Historical Zinc Price



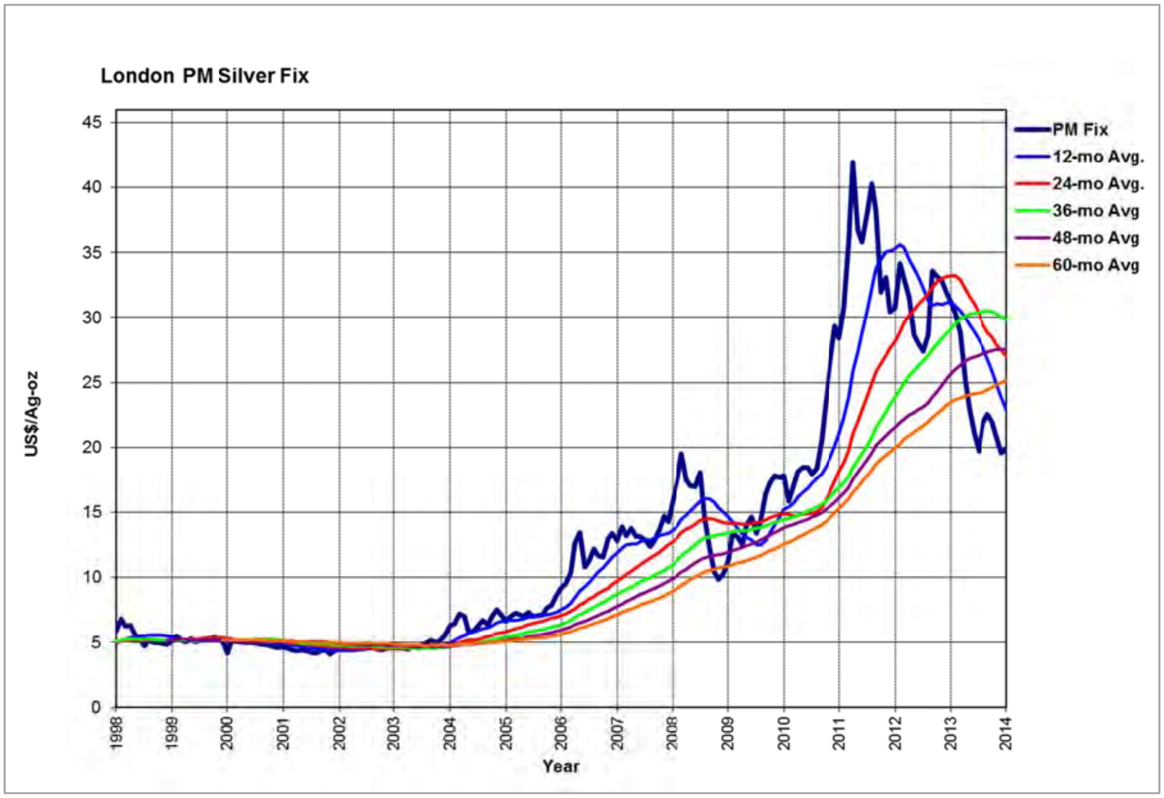
Source: JDS 2014

Figure 19.4: Historical Gold Price



Source: JDS 2014

Figure 19.5: Historical Silver Price



Source: JDS 2014

The metal prices used in the Base Case economic analysis are spot metal prices as at October 15, 2014. An additional scenario was evaluated, utilizing forward-looking metal prices published by Consensus Economics (October 2014), an independent macroeconomic survey firm that prepares monthly compilations of metal prices using more than 30 analysis covering over 25 commodities.

Table 19.5 summarizes the spot metal prices and exchange rate as of October 15, 2014, and Consensus Economics forward-looking prices and exchange rates.

Table 19.5: Metal Prices Used in the Economic Analysis

Commodity	Unit	Base Case Spot as at 15-Oct-14	Forward Pricing Consensus Economics Publication Oct-14
Copper Price	US$/lb	3.08	3.38
Lead Price	US$/lb	0.93	1.10
Zinc Price	US$/lb	1.06	1.18
Gold Price	US$/oz	1,238	1,373
Silver Price	US$/oz	17.00	23.07
Exchange Rate	US$:C$	0.89	0.90

Source: JDS 2014

19.3 Contracts

19.3.1 Streaming Contract with Royal Gold

In December 2011, Chieftain entered into a gold and silver purchase transaction with Royal Gold Inc. to sell a portion of the precious metals expected to be produced at the Tulsequah Chief mine. This agreement was then updated in July 2014 to reflect the reduction in production from 2,000 tpd to 1,100 tpd. Chieftain has received $10M in upfront payments upon the signing of the contract, and will receive an additional US$45M for the project build (to be received upon progression of construction completion for the project).

The advance and future proceeds will allow Royal Gold to purchase, upon production of the Tulsequah Chief mine:

- 17.50% of payable gold up to 65,000 oz, payable at 30% of the daily London price quotation, and 8.75% of the gold production thereafter
- 25.00% of payable silver up to 3,000,000 oz, payable at 25% of the daily London price quotation, and 12.50% of the silver production thereafter.

The contract has been included in the economic analysis of the project. Total gold and silver ounces expected to be sold to Royal Gold Inc. under this contract total 62.3koz and 2.7Moz, respectively.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Issues

The Tulsequah Chief Mine 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 upon the development of the Project for the historical waste rock is intended to be disposed of underground in workings that will become flooded upon 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 dealt with.

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 the winter of 2011-2012, 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 the AMD is to backfill the historic stopes early on during mine operations, which is expected 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 for disposal in the underground workings which will eventually be flooded at mine closure. Section 18.26 provides a detailed explanation of the PAG waste rock management system used for temporary storage of these materials during operations.

20.2.2 Tailings

All tailings from the Tulsequah Chief mine that are not required for backfill will be desulphidized then co-disposed with limestone in the tailings management facility to provide an adequate NPR that will mitigate any possible future generation of AMD. For a detailed discussion of tailings disposal refer to Section 18.25. The pyrite concentrate produced in the desulphidization process will be deposited in mine voids as backfill, below the ultimate groundwater table, preventing future AMD.

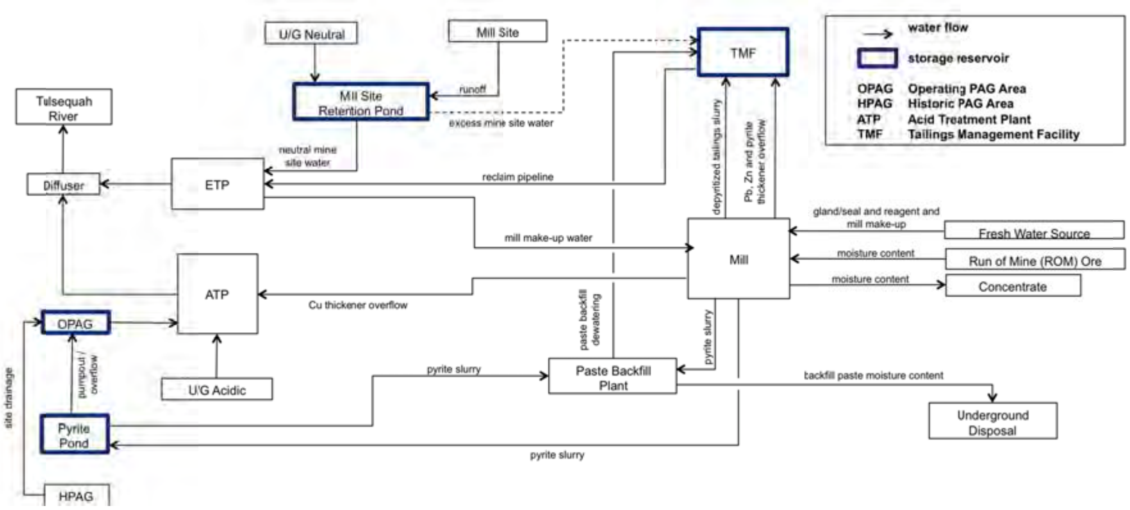
20.3 Water Management Plan / Water Balance

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 11-years of production, as well as a 2-year start-up and a 1-year closure period.

The WBM is illustrated schematically in Figure 20.1 and represents site-wide flow paths within the proposed project.

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 2014).

Figure 20.1: Water Balance Schematic, Years 3 - 11



20.3.1 Water Balance Modelling Platform

A robust Monte-Carlo Simulation Program, Goldsim, was used to set-up the WBM. It divided the balance up into six 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.

20.3.2 Weather Dataset

The dataset comprised of 65 years’ worth of precipitation, temperature and snowmelt data collected at the Juneau Airport weather station between the period of 1950 to 2008. The Juneau data was then adjusted to reflect the inland conditions experienced at the Tulsequah mine site. From this, 53 realizations were created, to test the WBM using real, historic data providing a realistic approximation of environmental conditions that can be expected at the site.

20.3.3 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. As new mine development proceeds, mine water flows are expected to increase, but the drainage from the new workings will be at neutral pH because of a the lack of sulphide oxidation and the presence of cemented paste backfill.

20.3.4 Tailings Management Facility (TMF)

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.

20.3.5 PAG Waste Storage Facility (HPAG/OPAG/Pyrite Pond)

Three lined 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 and as such it has not been incorporated into the proposed water management system.

20.3.6 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 produced into concentrate in the mill and the two main tailings streams will be the pyrite tailings (PAG) and the de-pyritized tailings (NAG). The retention pond collects mill site runoff and UG 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 paste fill and the lower workings with pyritic paste fill.

20.3.7 Acid Treatment Plant and 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, prior to discharging in the receiving environment through a diffuser buried below the scour depth in the Tulsequah River flood plain.

20.3.8 Contingency Measures

It can be expected that the mine water management system will undergo intermittent events where there is either a deficit of water supply or excess 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 of mine impacted water to the environment. Contingency measures are summarized in Table 20.1.

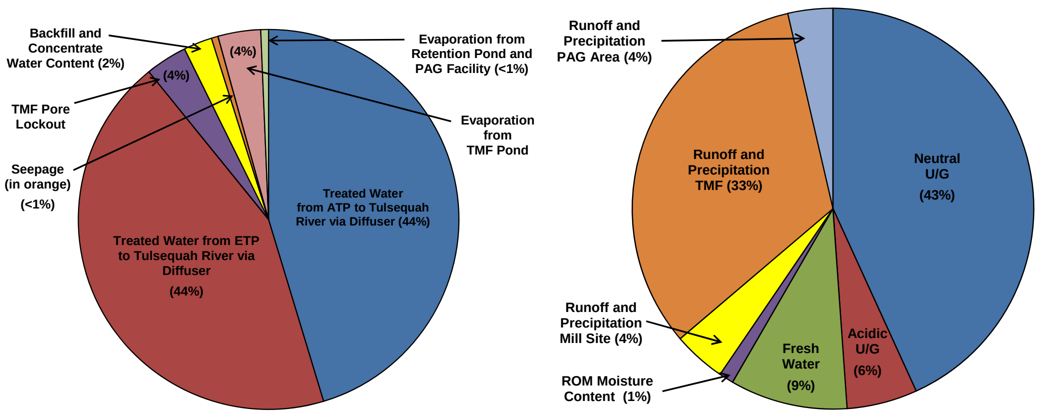
Table 20.1: Contingency Measures for Water Management during intermittent Events

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 or freshwater from the river 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.

20.3.9 Site-Wide Inflows and 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.

Figure 20.2: Water Balance Inflows and Outflows



20.3.10 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.

Environmental Conditions

The sensitivity of the model was tested using 30 unique weather scenarios. The development of the WBM was iterative and with each change the WBM was run with all 30 realizations. Parameters were constantly adjusted to ensure that no uncontrolled releases occurred during the design process.

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 30 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.

Underground Mine Inflow

The peak UG mine flow at 110m^3/h was adjusted by +100/-20% to test the WBM against variation in the UG flow. Since UG water normally reports to the ETP, it and the discharge piping will be sized to handle the potential additional increase in flow.

Processing Variation

The ore processing base case, 1,100 tpd, 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. The TMF dam crest height can readily be increased to accommodation additional tailings, should the higher rate persist throughout mine life (i.e., with increasing ore reserves).

20.3.11 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 or freshwater supplies.
- The mill make-up water will be taken from TMF reclaim. If water quality is acceptable, the treated effluent from the ETP will supply the mill with water for the gland, seal and reagent water as well. 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 53 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.

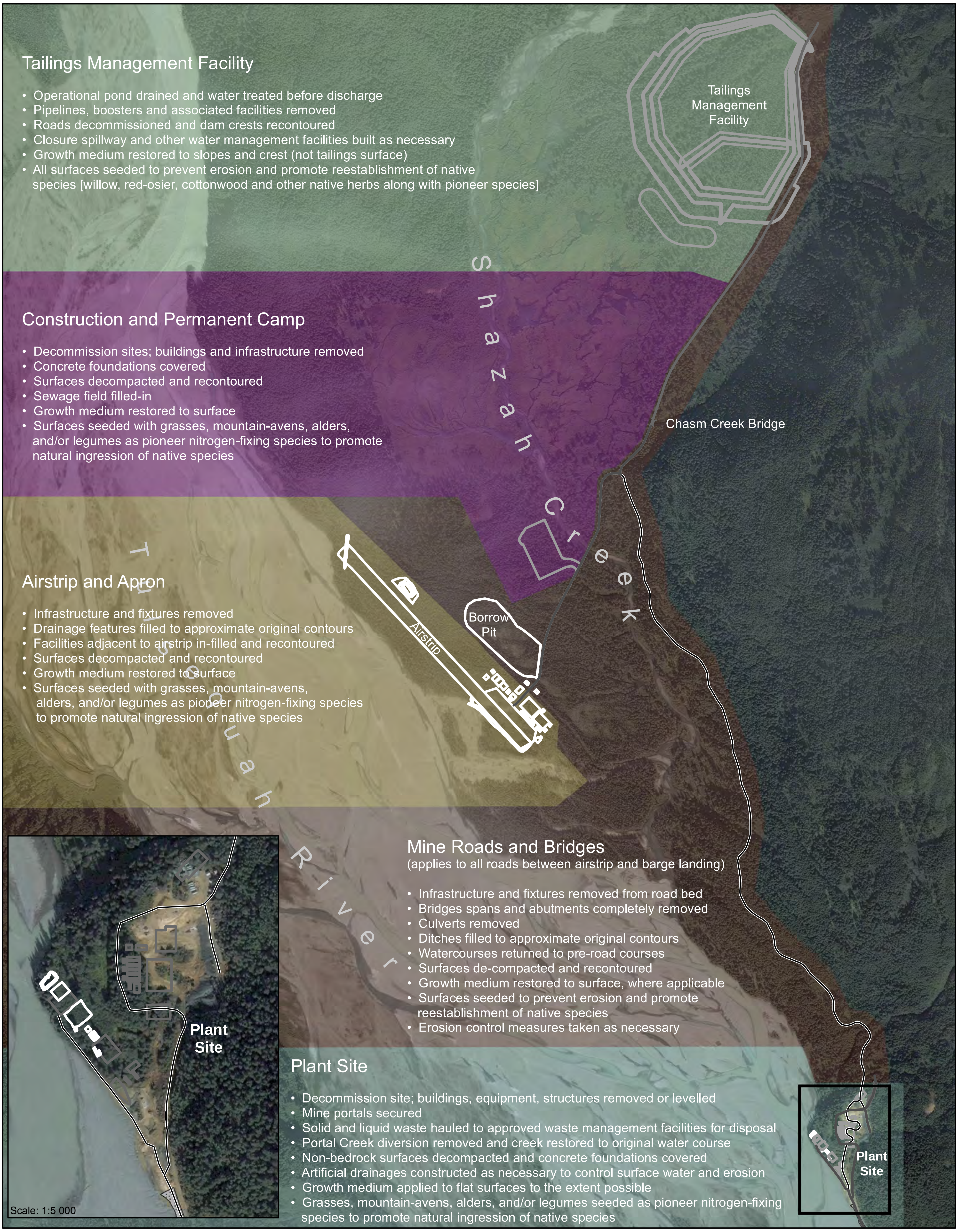
20.4 Mine Closure Plan

The Project is expected to result in a total disturbance at end of mine life of approximately 165 ha. The existing area of disturbance at the site is approximately 110 ha. Remaining on surface at mine closure will be a tailings management facility (TMF) containing non-acid generating tailings, a non-acid generating waste rock storage facility and a demolition debris landfill incorporated within 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. Generally speaking, 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. Re-contouring 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.

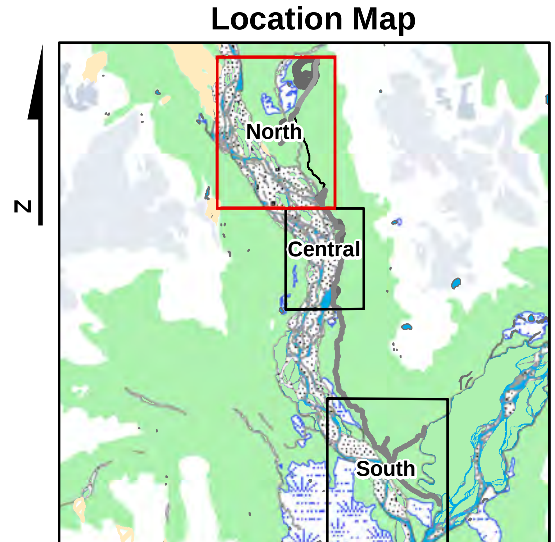
Figure 20.3, 20.4 and 20.5 summarize the closure concepts. Details are provided in the Tulsequah Chief Mine Closure Plan (MEA, 2012b). The costs associated with closure implementation, which forms the basis for the reclamation bonding discussed in Section 20.7, are provided in the Capital Cost estimate chapter of the Feasibility Study.



Projection: NAD83 UTM Zone 08N
Source: Chieftain Metals Inc., Government of BC, Quickbird Imagery (Aug. 26, 2003) Copyright 2007 Digital Globe Inc.
Scale: 1:15 000

0 250 500 1,000
Metres

Location Map



Tulsequah Chief Mine Project

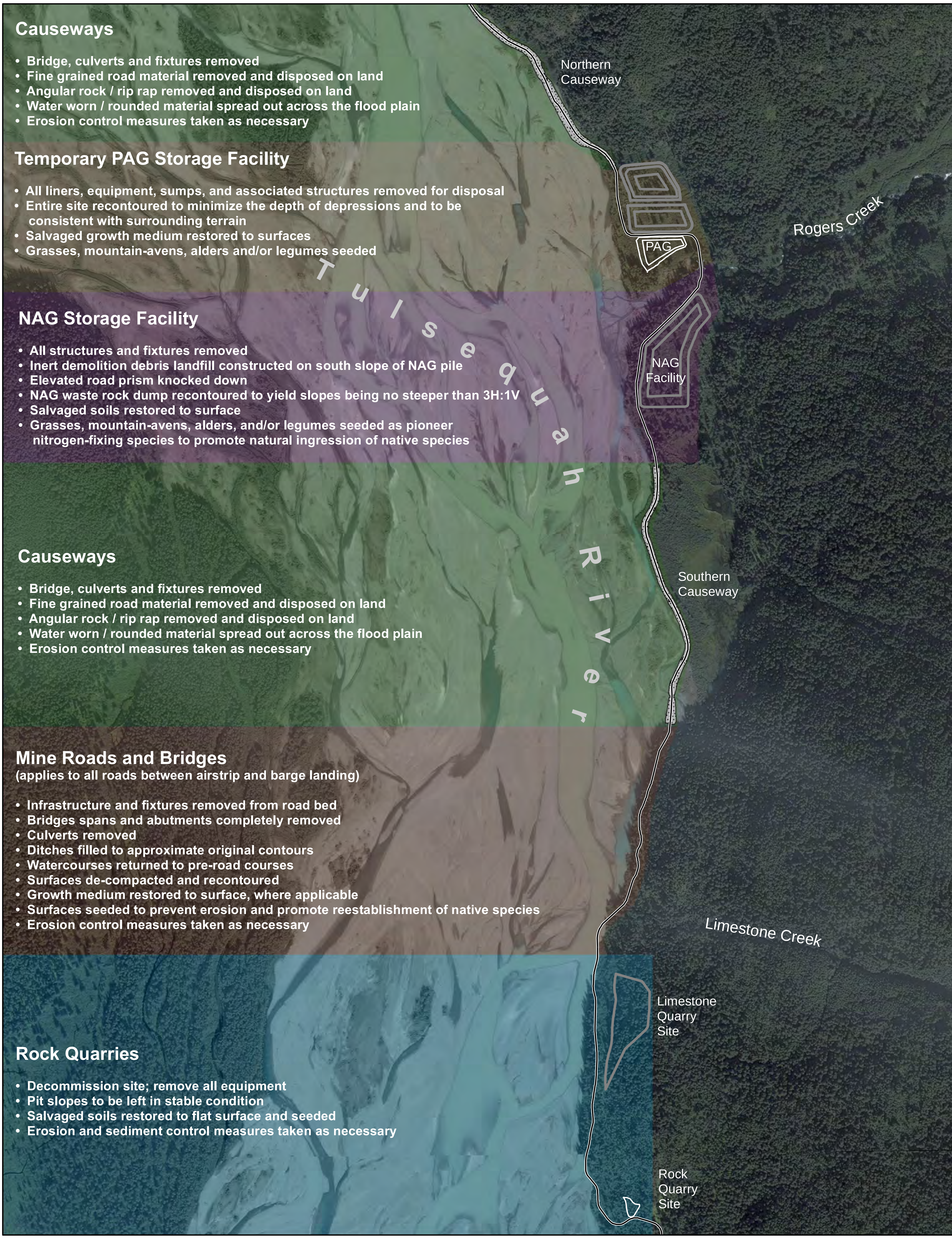
Reclamation Plan - North

CHIEFTAIN METALS CORP

Legend

- Existing Site Roads
- Causeway
- displayed in white — Existing Infrastructure
- Proposed Infrastructure

Date: November 20, 2014
Project: GIS12-0011



Projection: NAD83 UTM Zone 08N
Source: Chieftain Metals Inc., Government of BC, Quickbird
Imagery (Aug. 26, 2003) Copyright 2007 Digital Globe Inc.
Scale: 1:10 000

0 150 300 600
Metres

Tulsequah Chief Mine Project

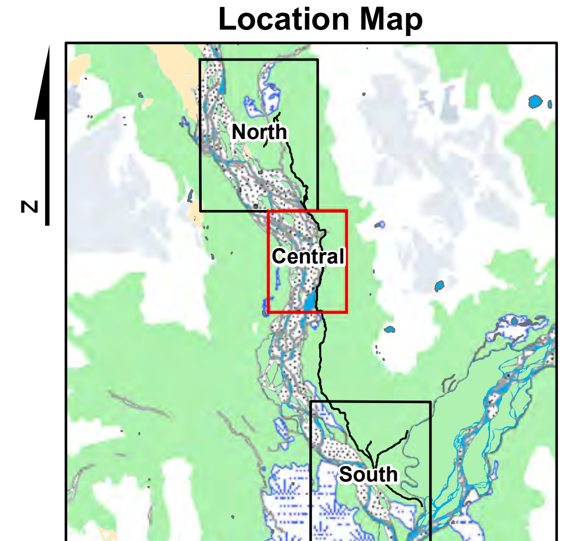
Reclamation Plan - Central

CHIEFTAIN METALS CORP

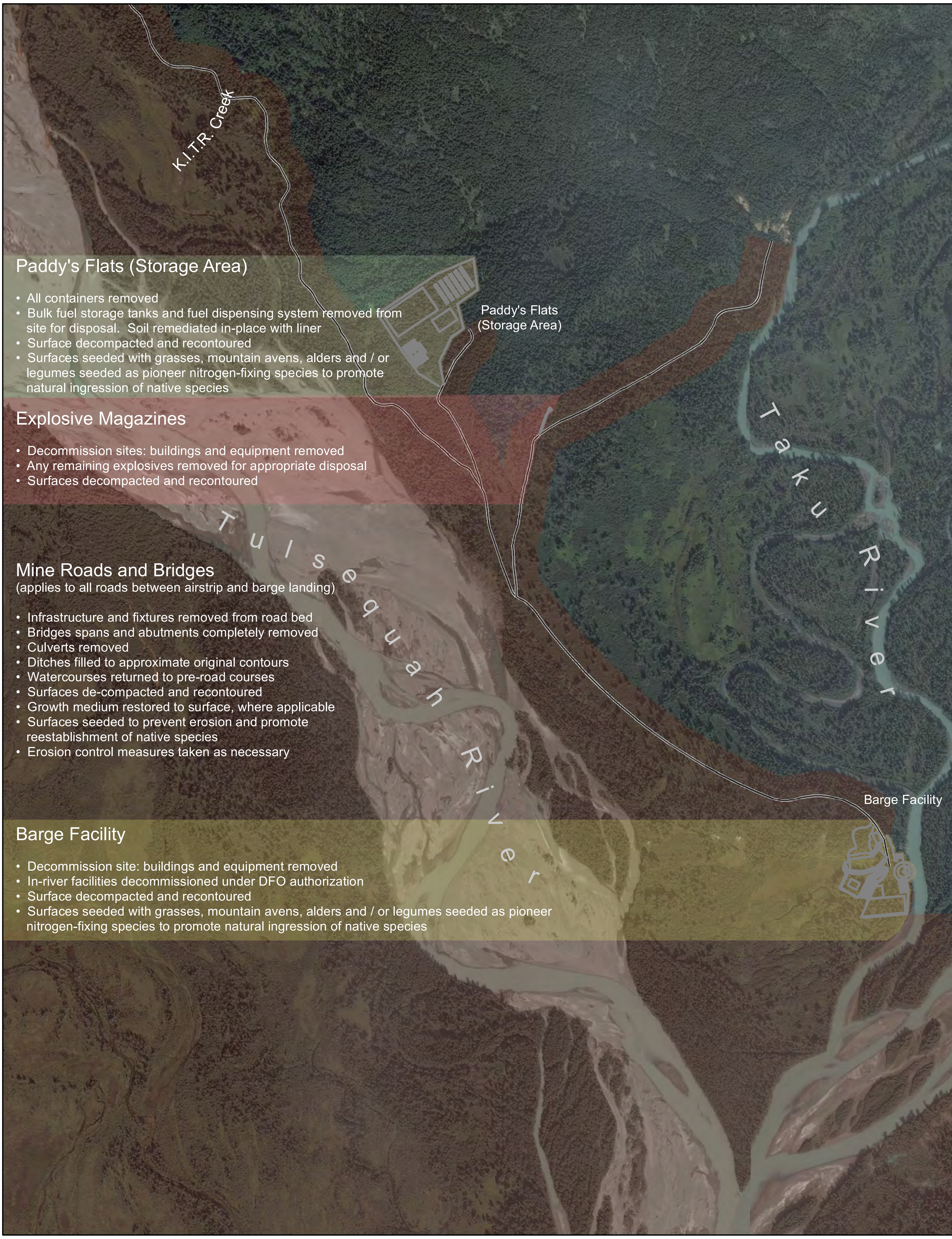
Legend

- Existing Site Roads
- Causeway
- displayed in white: Existing Infrastructure
- Proposed Infrastructure

Location Map



Date: November 20, 2014
Project: GIS12-0011



Projection: NAD83 UTM Zone 08N
Source: Chieftain Metals Inc., Government of BC, Quickbird Imagery (Aug. 26, 2003) Copyright 2007 Digital Globe Inc.
Scale: 1:15 000

0 250 500 1,000
Metres

Tulsequah Chief Mine Project

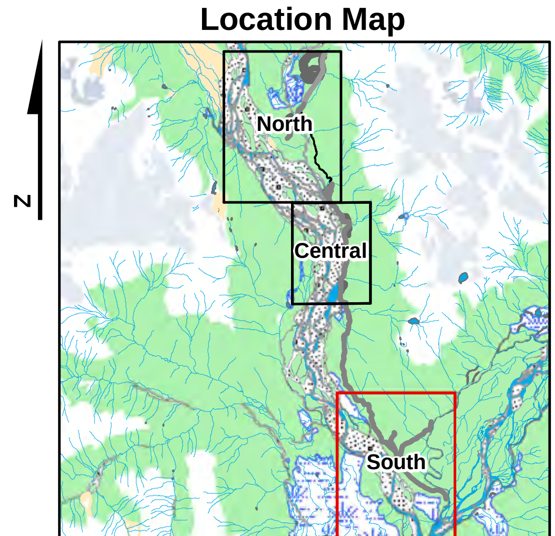
Reclamation Plan - South

CHIEFTAIN METALS CORP

Legend

	Existing Site Roads
	Causeway
displayed in white	Existing Infrastructure
	Proposed Infrastructure

Location Map



Date: November 20, 2014
Project: GIS12-0011

20.5 Permitting

The Tulsequah Chief Mine 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. This decision was set aside by the BC Supreme Court, on July 11, 2014, and the Minister must now render a new decision on the substantial start of the project. This decision is expected by the end of December 2014. After that, the Environmental Assessment certificate will remain in effect for the life of the project. Amendment #5 to Environmental Assessment Certificate M02-01 was received on October 19, 2012. A further amendment (#6) will be needed to incorporate the recent changes to the project design, such as the reduced mill throughput, the inclusion of the starter impoundment within the TMF and the use of conventional river barges for seasonal barging of concentrates. This process will commence once the substantially started decision has been rendered, and is expected to take 2-3 months to complete. This is a similar process to the one undertaken for Amendment #2 in 2007.

Chieftain has secured all necessary permits to commence construction at the mine site.

Several permits related to the construction of the Tulsequah Chief Mine Project were issued to the previous owner and have since been transferred to Chieftain. Table 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.

Table 20.2: Status of Project Permits, Licenses and 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 Metals Inc. on November 1, 2010 Project originally deemed substantially started on 30 May 2012; anticipated that new determination will be made in December 2014, certificate will then be 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 Metals Inc. 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 Metals Inc. on November 1, 2010
4	BCEAA Environmental Assessment Approval Amendment #3	Amendment #3 to EAC M02-01	BC Environmental Assessment Office	Amendment to allow ACB access for Project	26-Feb-09	Transferred to Chieftain Metals Inc. on November 1, 2010.
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	BCEAA Environmental Assessment Approval Amendment #6	Amendment #6 to EAC M02-01	BC Environmental Assessment Office	Amendment to allow changes to project design	Planned	Similar in concept to Amendment #2.
8	CEAA Screening Environmental Assessment Approval	36077	Canadian Environmental Assessment Agency	Overall Project Environmental Approval to proceed	05-Jul-05	Already in Place
9	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 Metals completed 17 February 2012; Revised permit issued January 25, 2013
10	Occupant License to Cut and amendments	L47498, Amendments #1-6	BC Ministry of Forests, Lands and Natural Resource Operations	Required for removal of timber from construction areas	December 6, 2007.	Revised permits issued January 25, 2013 for SUP and PUP
11	Parks Use Permit	N/A	BC Parks	Permits all-weather access road through Nakina-Inklin Reserve	January 25, 2013	
12	Roadworks Permit	N/A	BC Ministry of Transportation and Infrastructure	Permit to undertake roadworks on unmaintained BC MoTI right of way for mine access	January 30, 2014	
13	Intersection Permit	N/A	BC Ministry of Transportation and Infrastructure	Permit for mine access road to intersect with existing BC road network	January 30, 2014	
14	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
15	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
16	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
17	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
19	Mines Act Permit Initial (MA1)	M232	BC Ministry of Mining, Energy and Natural Gas	All surface roads and infrastructure development	08-Feb-08	Received February 2011

20	Mines Act Permit Amendment (MA2)	M232	BC Ministry of Mining, Energy and Natural Gas	Inclusion of Waste storage, plantsite surface development, underground preparatory work (slash adits)	14-Nov-08	Received February 2011
21	Mines Act Permit Amendment	M232	BC Ministry of Mining, Energy and Natural Gas	Approving Acid Water Treatment Plant	07-Jul-11	
22	Mines Act Permit Amendment	M232	BC Ministry of Mining, Energy and Natural Gas	Approving road bridge and camp construction activities	07-Jun-12	
23	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 underground development
24	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
25	Discharge Diffuser Authorization		Fisheries and Oceans Canada	Installation of buried diffuser pipe in Tulsequah River floodplain	Planned	Submission planned to ensure issuance prior to operations
26	Airstrip Extension Authorization		Fisheries and Oceans Canada	Extension of airstrip 150m		Re-submission planned upon completion of detailed engineering to ensure issuance prior to installation.
27	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
28	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.
29	Waste Discharge Authorization	#105719	BC Ministry of Environment	Authorization for discharges during construction period	01-Apr-12	Allows for up to 2,640 m^3/d

Permit Issued

Permit Pending

Permit to be applied for at a later date

Table 20.3: Status of Project Permits, Licenses and Authorizations Required During Operation

	Permit	Permit Number	Issuing Authority	Description	Status/Issue date	Comments
30	Waste Discharge Authorization amendment		BC Ministry of Environment	Amendment to increase discharge rate for Operations	Planned	To be applied for pending finalisation of site water balance and operating parameters
31	Air Emissions Authorization		BC Ministry of Environment	For incinerator and diesel generators	Planned	
32	Water License, Portal Creek diversion	C126606	BC Ministry of Environment	Diversion of intermittent creek	12-Jul-11	
33	Conditional Water License	C120434 and F014293	BC Ministry of Environment	Diversion of Camp Creek for power generation and potable water extraction	16-Feb-05	Transferred to Chieftain April 28, 2011
34	Dawn Creek Water License	C126660	BC Ministry of Environment	Water Usage licenses	November 6, 2012	
35	Tulsequah River Water License	C126460	BC Ministry of Environment	Water Usage license	August 30, 2012	
36	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.
37	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.
38	Section 9 approval diffuser		BC Ministry of Environment			Submission planned to ensure issuance prior to installation.
39	Section 9 approval airstrip extension		BC Ministry of Environment			Submission planned to ensure issuance prior to construction.
40	Section 9 approval barge landing		BC Ministry of Environment			Submission planned to ensure issuance prior to construction.

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20.6 Monitoring and 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 Pre-Construction 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).

20.7 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 $2,100,000 for initial 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 mill and TMF 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 Study. It is estimated that an additional $4,700,000 in security will need to be posted over a 5 year period.

20.8 Social and Community

20.8.1 Requirements and 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.

20.8.2 Status of Negotiations and 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.

A Land Use Plan ("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 also 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 which 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 Metals Inc. 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 AND OPERATING COST ESTIMATES

21.1 Capital Costs

21.1.1 Introduction & Summary Data

Preparation of the capital cost estimates is 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 -15%/+15%, which represents a JDS Feasibility Study Budget / Class 3 Estimate. A detailed Basis of Estimate document has been completed to supplement the Capital Direct & Indirect Costs for the Tulsequah FS project and should be referenced for more detail.

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 (1,100 tpd).
- Sustaining Capital Cost – These are costs incurred during operations for ongoing waste development, underground equipment acquisitions and underground infrastructure installations.

The following project costs are not discussed in this section:

- Sunk costs are not considered in this study; and
- Owners reserve is not considered in this study.

All cost estimates are based on the following key parameters:

- Owner-performed preproduction mining (contractors will be used only for Alimak raises); and
- The specific scope and execution plans described in this study. Deviations from these plans will affect the capital costs.

21.2 Initial Capital Cost Estimate

21.2.1 Summary of Costs & Distribution

Initial capital costs include all costs to develop the property to a nameplate production of 1,100 tpd. Initial capital costs total $198.0M over two pre-production years.

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	347,046	117,869,000	59.5
Pre-Production Opex	34,281	12,274,000	6.2
Indirect Costs	157,065	28,739,000	14.5
Owners Cost	121,177	21,280,000	10.7
Contingency (11.4%)	-	18,435,000	9.3
Total	659,569	198,596,000	100

* All cost data are presented in Q3/Q4 2014 dollars.

A Level 3 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	11,900	3,859,000
1010	Plant Site Area	4,430	1,901,000
1020	Ancillary Areas	3,176	682,000
1030	Site Roads	3,857	1,120,000
1040	Limestone Crushing	436	155,000
15	Underground Mining	140,763	28,988,000
1510	Underground Mining	115,679	18,449,000
1560	Underground Processing Facilities	25,084	10,539,000
25	Processing Plant	98,321	44,589,000
2510	Grinding	33,799	15,202,000
2520	Separation / Concentrating	33,910	17,051,000
2530	Concentrate Dewatering/Drying/Loadout	19,377	7,619,000
2540	Reagents	1,789	1,072,000
2550	Tailings	3,237	1,644,000
2560	Process Plant Utilities	6,209	2,001,000
30	Tailings & Waste Rock Management	28,517	6,562,000
3010	Tailings Area	21,482	4,773,000
3020	HPAG, PAG, and Pyrite Storage Area	7,035	1,789,000
35	On-Site Infrastructure	67,545	33,870,000
3510	Accommodation and Administration Facilities	25,877	11,777,000
3520	Ancillary Facilities	2,304	2,269,000
3530	Power Plant	21,502	5,848,000
3540	Bulk Diesel Storage And Distribution	11,606	3,559,000
3550	Fresh, Fire, and Potable Water Systems	2,350	1,176,000
3560	Effluent Water Treatment Plant Upgrades	1,575	718,000
3570	Garbage and Waste Management	984	695,000
3580	Plant Mobile Fleet	0	7,054,000
3590	Miscellaneous Infrastructure	1,347	774,000
40	Off-Site Infrastructure	-	-
	Direct Costs Subtotal	347,046	117,869,000
	Indirect Costs		
90	Project Indirects	72,404	15,265,000
9030	Construction Indirects - Others	57,377	4,087,000
9040	Freight	10,027	9,065,000
9060	Capital Spares and Initial Fills	-	1,447,000
9070	Commissioning and Start-up	5,000	666,000
95	Engineering & EPCM	84,661	13,473,000
9510	Detailed Engineering and Procurement Management	-	6,200,000
9520	Project and Construction Management	84,661	7,273,000
	Indirect Costs Subtotal	157,065	28,739,000
98	Owners Costs	121,177	21,280,000
9810	Owners Costs	121,177	21,280,000
	Owner Costs Subtotal	121,177	21,280,000
	Direct Costs	347,046	117,869,000
	Pre-Production Opex	34,281	12,274,000
	Indirect Costs	157,065	28,739,000
	Owners Costs	121,177	21,280,000
99	Contingency (11.4%)	0	18,435,000
	Grand Total	659,569	198,596,000

* All cost data are presented in Q3/Q4 2014 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.

Table 21.3: Initial Capital Cost Estimate Summary, by Commodity Group

Commodity Group	Labour $	Material $	Equip $	Equip Usage $	Other $	Total $
Mining	-	-	-	-	18,449,000	18,449,000
Architectural and Buildings	4,412,000	3,842,000	10,220,000	-	111,000	18,584,000
Civil Works	308,000	188,000	11,000	217,000	240,000	963,000
Concrete	2,498,000	2,115,000	-	-	50,000	4,613,000
Electrical	2,555,000	2,825,000	3,923,000	-	50,000	9,354,000
Earthworks	2,696,000	2,478,000	-	2,925,000	20,000	8,120,000
Instrumentation	943,000	1,248,000	791,000	-	303,000	3,285,000
Mechanical and Equipment	3,924,000	188,000	29,061,000	15,000	1,240,000	34,429,000
Piping	3,550,000	1,407,000	130,000	-	-	5,087,000
Plate-work	1,648,000	3,339,000	-	-	-	4,987,000
Structural Steel	809,000	2,134,000	-	-	-	2,944,000
Mobile Equipment	-	-	7,012,000	-	42,000	7,054,000
Total Direct Costs	23,343,000	19,766,000	51,148,000	3,158,000	20,455,000	117,869,000

* All cost data are presented in Q3/Q4 2014 dollars

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
Equipment	
Major Equipment	Multiple budget quotations using general engineering specifications and data sheets based on the design criteria and process flow diagrams. Also includes single source pricing from select designated suppliers. Tank costs are based on budget quotations based on brief specifications and/or process flow diagram information. Where quotations were not received costing used from previous similar projects was used.
Minor Equipment	Budget quotations based on brief specifications and/or process flow diagram information. Where quotations were not received costing used from previous similar projects was used.
Materials	
Bulk Earthworks	Bulk earthwork quantities were generated from rough grading designs prepared using Maptek's Vulcan™ software or provided by the party responsible for the area. The hours required to complete each job are calculated from the earthworks quantities, equipment fleet required and productivity. The equipment includes: dozers (D8, D6), excavators (20 and 65 ton), wheel loaders (IT28), articulated trucks (40 ton), graders, packers, water trucks, surveyors, quality control personnel & labourers equipped with hand tools. A combination of contract and owner's equipment hourly costs are used, as detailed in Appendix C. Productivity is dependent on the activity or cycle time and number of trucks for loading and hauling. Productivities and cycles times are provided in Appendix C and D respectively. Unit costs for earthworks materials (LLDPE liner, bridge decking, etc.) were obtained through budgetary quotations, recent estimates for similar projects or in-house estimates. Unit costs for materials are shown in the Basis of Estimate document.
Concrete	Preliminary concrete quantities are estimated based on the GA drawings and experience with similar projects. A 5% allowance has been added in the build-up for spillage and over pour. The concrete unit rates include aggregate, screening, rebar, forming, pouring and finishing. Unit rate costs for concrete supply and finishing hours are based on experience with similar projects.
Structural Steel	Structural steel quantities have been estimated based on the GA drawings and experience with similar projects. Unit rate costs for steel supply and erection hours are based on experience with similar projects.
Mechanical Bins, Pump Boxes, Tanks & Chutes	Mechanical bins, chutes and tank plate quantities have been estimated based on the GA drawings, simple material take offs and/or experience with similar projects. Rubber lining for pump-boxes have been included where identified on the Mechanical Equipment List. Unit rate costs for plate steel supply and erection hours are based on experience with similar projects.
Process Piping and Valves	Process piping quantities have been estimated based on the GA drawings, simple material take offs and/or experience with similar projects. Unit rate costs for piping and installation hours are based on experience with similar projects. Valves allowances are based on % of piping material costs.
Electrical	Based on an estimated number of motors and total connected horsepower derived from mechanical equipment list. HV/LV distribution based on Single Line Diagrams and estimated MTOs from plant layout with budget costs for materials based on experience with recent projects. Budget quotations were used for major equipment: transformers, switchgear, MCC's, and back up gensets.
Instrumentation	Instrumentation equipment, cable quantities and costs were based on budget quotations.
Control System	Control system hardware, cable quantities and costs were based on budget quotations.
Installation	
Installation Labour	Manhours provided by JDS based on similar project work under similar conditions or 1st principle estimates, and benchmarked against unit labour from recent JDS projects. Installation labour rates were built up using typical labour rates including overtime, small tools, supervision and standard labour burdens for a project located in Canada.
Underground Mining	
UG Development Labour and Consumables	Estimated from first principles utilizing the same methodologies described in the Operating Cost section.
UG Equipment Supply	Multiple budget quotations and firm prices were received based on project specific specifications and data sheets.
UG Infrastructure Supply	Budget quotations or firm prices were received based on brief specifications or standard off-the-shelf equipment requests.
Tailings Management Facility (TMF)	
TMF Earthworks	Tailings impoundment and waste storage areas earthworks quantities were based on the designs and take offs provided by Klohn Crippen Berger Ltd. (KCB). Unit rates were developed by JDS from first principles.
On-Site Infrastructure	
Camp	Construction and Permanent camp costs are based on budgetary quotations from vendors who have provided camps in the area. Installation costs for camp are also from the budgetary quotations.
Infrastructure Services & Buildings	Infrastructure buildings are based on budgetary quotations and/or experience with similar projects.

Commodity	Estimate Basis
On-site Electrical Distribution	Material take offs are based on site GA drawings, and pricing based on previous projects. Budgets quotations for on-site power distribution equipment.
Indirect Costs	
Construction Indirects	Construction indirect costs to allow for contractor administration infrastructure, medical services, site orientation and safety training, and contractor mobilizations are estimated based on experience with similar projects.
Consultants - EPCM	JDS estimated Project Management and Construction Management services have been based on project management organization and schedule durations. JDS project management and construction management based on an organizational chart and scheduling man-hours. Detailed engineering has been estimated using project scheduling and approximate weekly hours, based on JDS experience. 3rd Party consultants have been estimated using project scheduling and approximate weekly hours, based on JDS experience.
Site Operations Milling	Labour costs incurred prior to the start of commercial production are considered in this estimate. Labour costs are based on an organization build-up for project operations preparation for start-up and commissioning of plant utilizing fully burdened wage rates. The majority of operations management, technical services, security, and the process plant personnel prior to operational phase for one full year. Site services equipment purchases and operation is estimated for 18 months prior to commercial operations.
Mine Site G&A Operating	Includes camp catering & housekeeping, generator fuel, temporary communications, first aid & medical contractor, early camp care & maintenance, insurance, passenger travel & charted flights, licences and permits, IBA's, outside monitoring, safety equipment. Costs were calculated based on similar projects and input from CMC.
Freight	Freight consists of trucking of materials and equipment to Prince Rupert Port where low-draft barges pick-up the materials and equipment at the mouth of the Taku River and deliver to the site. A detailed trade-off has been completed to calculate the ground, air and barged freight to site.
Plant capital Spares & Initial Fills	Spare parts estimate is based on 3.5 % of fixed capital equipment costs to allow for commissioning and start up spare parts. Initial fills are based on requirements to start the mill process plus one months of supply.
Commissioning and Start Up technical services, trades and vendor representatives	Commissioning & start up and Vendor representatives has been calculated based on similar projects.
Contingency	Contingency is allocated at 11.4% of all costs with the exception of the underground mining costs and fleet.

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 P5 and P95 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 P85 contingency amounts for each cost grouping. Results concluded that the use of an 11.4% blended contingency was appropriate.

21.2.2 Sustaining Capital Cost Estimate

Summary of Costs & Distribution

The primary sustaining capital cost is capital mine development occurring during the operations phase. Capital underground mining 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, and mining equipment.

Other sustaining capital cost items include the ultimate tailings dam and effluent treatment plant.

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 (CA$)	%
Underground Mining	60,996,000	72.6
Tailings & Waste Rock	12,673,000	15.1
On-site Infrastructure	4,266,000	5.1
Closure & Salvage	3,758,000	4.5
Contingency	2,359,000	2.8
Total	84,052,000	100

* All cost data are presented in Q3/Q4 2014 dollars

For further information regarding the Annual Sustaining Capital by Activity, refer to the Tulsequah Financial Model.

Basis of Sustaining Capital Cost Estimate

Sustaining capital was estimated in the same manner as the initial capital costs.

21.2.3 Closure & Reclamation Cost Estimate

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;
- Re-establishing natural creek channels; and
- 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; and
- 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 21.6: 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
Total	13,826,000	100

* All cost data are presented in Q4 2012 dollars.

For further information regarding the Closure and Salvage Annual Costs by WBS, refer to the Tulsequah Financial Model.

Basis of Closure Cost Estimate

Table 21.9 presents additional details of the closure and reclamation cost estimate, arranged by WBS.

Table 21.7: Basis of Closure Estimate Summary, by WBS

Phase / Item	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.
Monitoring/Maintenance	Conservative cost allowances were used based on similar projects.
Re-vegetation	Costs were estimated using historic pricing of seed and seedlings with an assumed 50% re-seeding rate

Source: JDS 2014

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 t of waste will be disposed of at a landfill constructed on site and 8,119 m^3 or 3,466 t 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.

Contingency

A blended 15% contingency was applied to the closure and reclamation estimate utilizing professional judgement, based on the level of scope definition.

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 barging off site to Prince Rupert, BC.

Table 21.8: Salvage Value Estimate

Item	**Capital Cost**	**% Residual Value**	**Cash Value†**
UG Mining Equipment Fleet	$19,103,000	0%	
Jaw Crusher	$375,000	0%	
Grinding Mills	$5,699,000	10%	$570,000
Pressure Filters	$530,000	0%	
Paste Backfill Equipment	$3,590,000	10%	$359,000
Powerplant	$13,900,000	25%	$3,475,000
Other Generators	$630,000	0%	
Construction Camp Complex	$586,000	0%	
Main Camp Complex	$7,244,000	0%	
Administration/Dry Complex	$850,000	0%	
Ancillary Buildings	$652,000	0%	
Assay Lab	$1,190,000	0%	
Effluent Treatment Equipment	$1,722,000	0%	
Surface Equipment Fleet	$7,054,000	0%	
Bridges	$600,000	0%	
Total	$63,724,000	7%	$4,404,000

All cost data presented in Q4 2014 dollars. † "Cash Value" denotes net cash value of salvageable equipment - after consideration of disassembly and shipment costs to Juneau, AK.

Capital Cost Exclusions

The following items have been excluded from the pre-production direct and indirect capital costs estimate:

- Force majeure;
- Escalation in costs after the Q3/Q4 2014 base date;
- Currency fluctuations;
- Any and all scope changes;
- Any and all project financing costs, including interest during construction and all costs; associated with borrowed funds;
- Bonding costs;
- Project sunk costs including this study;
- Mine reclamation and closure costs;
- Cost for a completed Impact Benefit Agreement with First Nation

21.3 Operating Costs

21.3.1 Introduction & Summary

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined proven project execution strategies. The estimate was developed using first principles and applying 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 packages 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 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;
- Transportation; and
- General & Administrative.

Certain items within the operating costs begin during the construction phase (assumed to be 2015 through 2016) and continue through the life of the mine. Some of the costs incurred during the pre-production period relate to the costs to purchase items such as consumables required for the following year of production. The timing of these costs has been accounted for in the economic analysis.

Underground lateral and vertical waste development after the pre-production period has been capitalized and will not appear as an operating cost (refer to Section 21.1.1.5 – Sustaining Capital Cost). Capital waste development represents the mine’s permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, ore pass accesses and permanent explosive storage cut-outs, as well as main ventilation raises.

The total operating unit cost is $159.49 per tonne processed exclusive of ocean transportation. Average annual operating costs and total unit costs are summarized in Table 21.1.

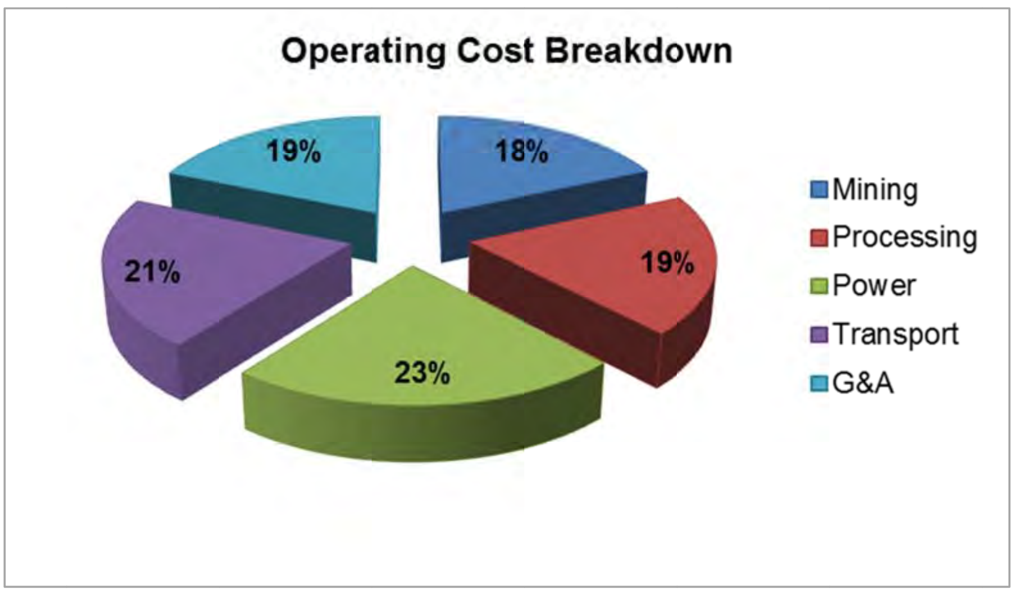
Figure 21.1 and Figure 21.2 illustrate the operating cost distribution. Annual operating costs by year are outlined in Table 21.2.

Table 21.9: Estimated Average Operating Costs by Area

Operating Costs	Average $M/yr	LOM $M	$/t processed
Mining	11.8	130.2	29.36
Processing	12.9	143.0	32.24
Power	14.5	160.4	36.16
Transport	13.3	147.4	33.23
G&A	11.4	126.4	28.50
Total Operating Costs	**63.9**	**707.5**	**159.49**

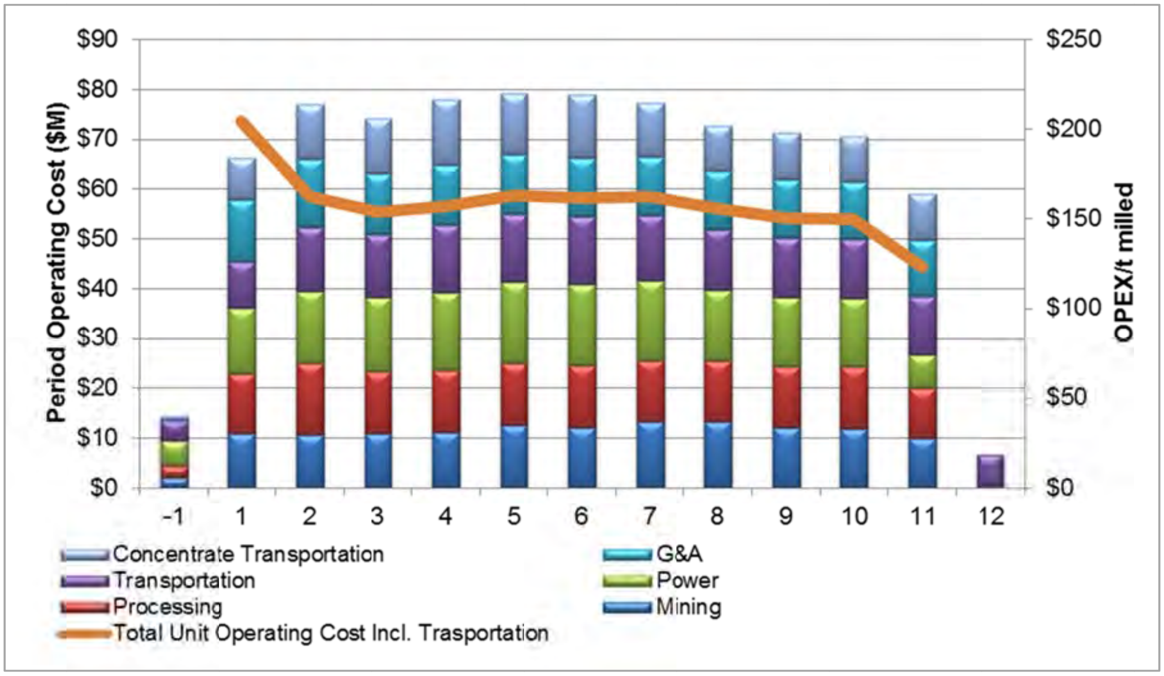
Source: JDS 2014

Figure 21.1: Total Operating Cost Distribution by Area



Source: JDS 2014

Figure 21.2: Annual Operating Cost by Area



Source: JDS 2014

Table 21.10: Annual Operating Costs by Area

Operating Cost	Units	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
Annual Operating Cost (C$M)															
Mining	M$	130.2	2.1	10.9	10.7	10.8	11.1	12.4	12.0	13.0	13.2	12.0	11.9	10.0	0.0
Processing	M$	143.0	2.3	12.6	14.7	12.9	13.0	12.9	12.9	12.9	12.8	12.8	12.8	10.3	0.0
Power	M$	160.4	4.9	13.2	14.6	14.9	15.5	16.2	16.2	16.0	14.1	14.0	13.8	6.9	0.0
Transportation	M$	147.4	4.2	9.3	12.8	12.8	13.6	13.7	13.7	13.1	12.1	11.9	11.8	11.8	6.6
G&A	M$	126.4	0.0	12.0	13.2	11.6	11.5	11.4	11.2	11.2	11.2	11.1	11.1	10.7	0.0
Concentrate Transportation*	M$	116.6	0.8	8.4	10.9	11.1	13.0	12.3	12.7	10.8	9.2	9.4	8.9	9.2	0.0
Total Operating Costs	M$	**824.0**	**14.4**	**66.4**	**76.9**	**74.0**	**77.7**	**79.0**	**78.8**	**77.0**	**72.6**	**71.3**	**70.4**	**58.9**	**6.6**
Unit Operating Cost by Year (C$/tonne processed)															
Mining	$/t milled	29.36		33.80	26.39	26.30	26.95	30.40	29.30	31.93	32.34	29.31	29.01	24.69	29.36
Processing	$/t milled	32.24		38.79	36.18	31.62	31.39	31.63	31.56	31.46	31.47	31.33	31.20	25.59	32.24
Power	$/t milled	36.16		40.74	36.01	36.39	37.51	39.59	39.67	39.27	34.53	34.20	33.65	16.98	36.16
Transportation	$/t milled	33.23		28.74	31.52	31.29	32.97	33.41	33.55	32.02	29.68	29.11	28.67	29.14	33.23
G&A	$/t milled	28.50		37.06	32.64	28.25	27.87	27.89	27.47	27.48	27.57	27.12	27.08	26.54	28.50
Concentrate Transportation*	$/t milled	26.28		25.79	26.96	27.01	31.45	30.05	30.95	26.37	22.45	22.97	21.66	22.88	26.28
Total Unit Operating Cost Incl. Transportation	$/t milled	**185.78**		**204.92**	**189.70**	**180.86**	**188.14**	**192.96**	**192.50**	**188.54**	**178.05**	**174.04**	**171.27**	**145.84**	**185.78**

(*) Concentrate Transportation costs were estimated as part of the economic model. They are shown here to demonstrate all-in operating costs.

Source: JDS 2014

21.3.2 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 21.3 summarizes the total planned workforce during project operations.

Table 21.11: Planned Operations Workforce

Department	Total Persons Employed (Peak)
Mining	82
Processing	66
G&A	48
Contractors*	32
Total	**228**

(*) 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, etc.

Source: JDS 2014

Labour base rates were determined through experience and benchmarked against the Costmine Canadian Mine Salaries, Wages, Benefits 2014 Survey Results and 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 7.7% for hourly employees;
- Travel pay of eight hours per rotation for hourly employees;
- Production bonus for underground production and development miners;
- CPP, EI, 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; and
- Health and welfare allowance of $2,500 per year for all employees.

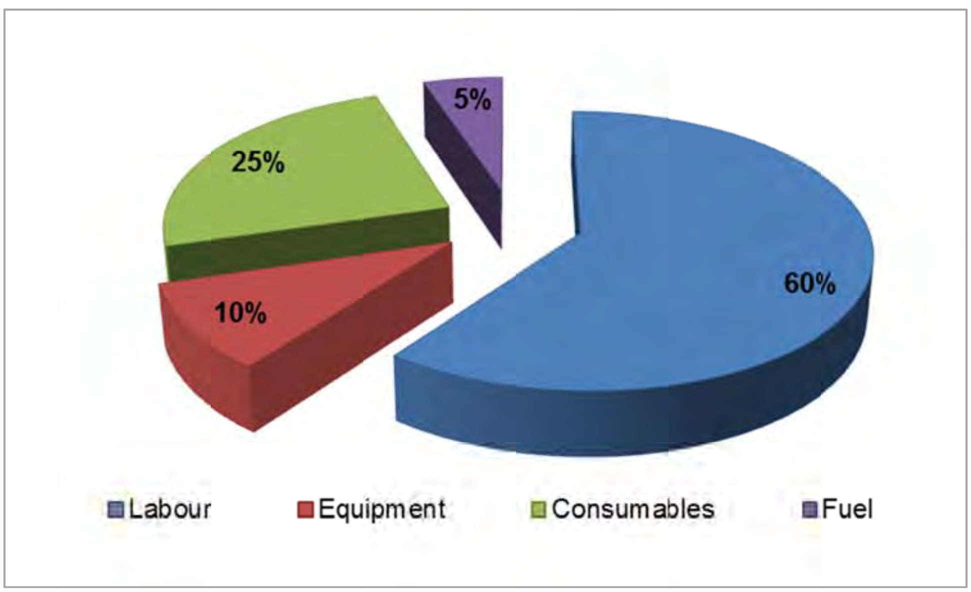
Mine Operating Costs

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 past 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; and
- Mine Maintenance – Costs include labour and shop consumables to maintain and repair the underground mining mobile equipment.

The estimated total mining operating cost is $29.36 per tonne processed. Average annual operating costs and total unit costs are summarized in Table 21.4. Figure 21.3 presents the mining operating cost distribution. The subsections below further describe the basis of estimate for major items within each grouping.

Figure 21.3: Mine Operating Cost Distribution



Source: JDS 2014

Table 21.12: Mine Operating Costs by Area

Total Operating Cost - By Area	**Average $M/yr**	**LOM $M**	**$/tonne milled**
Production	6.6	72.6	16.36
Backfill	2.0	22.4	5.04
Mine General	1.6	17.4	3.92
Mine Maintenance	1.6	17.9	4.04
Operating Cost - Total	11.8	130.2	29.36

Source: JDS 2014

Mining Labour

Mining labour was calculated using the personnel numbers summarized in Section 16.11 of this report. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations. Table 21.5 summarizes the mining workforce labour rates.

Table 21.5: Mine Labour Rates

Position	Salary/Hourly	Total Annual Salary Hourly Rate ($)
Mining Operations		
Mine Superintendent	Staff	220,996
Mine Captain	Staff	162,091
Mine Supervisor/Shift Boss	Staff	127,606
Production Drill Operator	Hourly	74.47
Jumbo Operator	Hourly	81.80
Ground Support/Bolter/Shotcrete	Hourly	81.80
Development Service	Hourly	81.80
Blaster	Hourly	74.47
LHD Operator	Hourly	65.99
Truck Driver	Hourly	63.87
Backfill	Hourly	58.84
Utility Vehicle Operator/Nipper	Hourly	47.21
Paste Backfill Plant		
Paste Backfill Plant Operators	Hourly	48.23
Mining Maintenance		
Mine Maintenance Superintendent	Staff	144,787
Mine Maintenance Supervisor/Shift Boss	Staff	113,494
Mechanical General Foreman	Staff	127,606
Maintenance Planner	Staff	113,494
HD Mechanic/Welder, Mobile	Hourly	65.46
Electrician	Hourly	65.73
Dry/Lapman/Bitman	Hourly	42.20
Mining Technical Services		
Chief Mining Engineer	Staff	162,091
Senior Mine Engineer & Planner	Staff	144,787
Mine Ventilation/Project Engineer	Staff	127,606
Geotechnical Engineer	Staff	127,606
Sr. Mine Technician	Staff	113,494
Surveyor/Mine Technician	Staff	103,921
Chief Geologist	Staff	162,091
Production Geologist	Staff	103,921
Geotechnical Technician/Sampler	Staff	84,992

Source: JDS 2014

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. 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);
- Hoses and fittings; and
- Electrical wiring, sensors.

Life expectancy for major underground mine equipment is summarized in Table 21.6.

Table 21.13: Major 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
40 Tonne Truck	45,000
Mechanized Bolter	45,000
ANFO Loader	60,000

Source: JDS 2014

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 21.7 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 21.8.

Table 21.14: Major Underground Equipment Tire Life Expectancy

Equipment Type	Expected Life (Hours)
7 m^3 LHD with Remote	1,750
40 Tonne Truck	3,500

Source: JDS 2014

Table 21.15: Major Underground Equipment Hourly Operating Cost

Equipment Type	Fuel $/hr	Oil/Lube $/hr	Parts $/hr	Tires $/hr	Total $/hr
Two Boom Jumbo	1.15	0.40	5.00	1.25	7.81
LH Drill	3.17	1.11	5.00	1.25	10.53
7 m^3 LHD with Remote	24.19	8.47	74.99	10.29	117.93
40 Tonne Truck	28.22	9.88	44.80	9.14	92.05
Mechanized Bolter	2.07	0.73	6.00	1.25	10.05
ANFO Loader	4.75	1.66	10.00	0.50	16.92

Source: JDS 2014

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 types are summarized in Table 21.9.

Table 21.16: Underground Mining Consumables Unit Costs

Drift/Stope Type	Ground Control ($/m)	Services ($/m)	Jumbo/Bolter Drilling ($/m)	LH Drilling ($/t)	Explosives ($/m)	Total ($/m)
Ramp (5 x 5)	89.69	323.45	132.29	-	353.21	898.63
Ore Drift (5 x 5)	86.84	214.79	98.65	-	229.89	630.16
Level/X-cut (4.6 x 4.6)	86.86	297.86	115.82	-	316.89	817.42
MCF Access (5x4)	85.12	223.59	101.48	-	264.89	675.08
LH Stoping	$0.03/t	-	-	0.47	$0.75/t	$1.25/t

Source: JDS 2014

Paste Backfill

Paste Backfill costs were based on an average cement content of 2.7% by weight. Other consumables used include pipe, barricades and an allowance for emergency. Underground drilling was estimated from experience at similar mining operations utilizing past backfill.

Processing Operating Costs

Operating costs for the 1,100 tpd concentrator plant were assembled using first principles and include costs for processing operations, maintenance, and technical service labour, as well as 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 $32.24 per tonne processed. Average operating costs are summarized in Table 21.10.

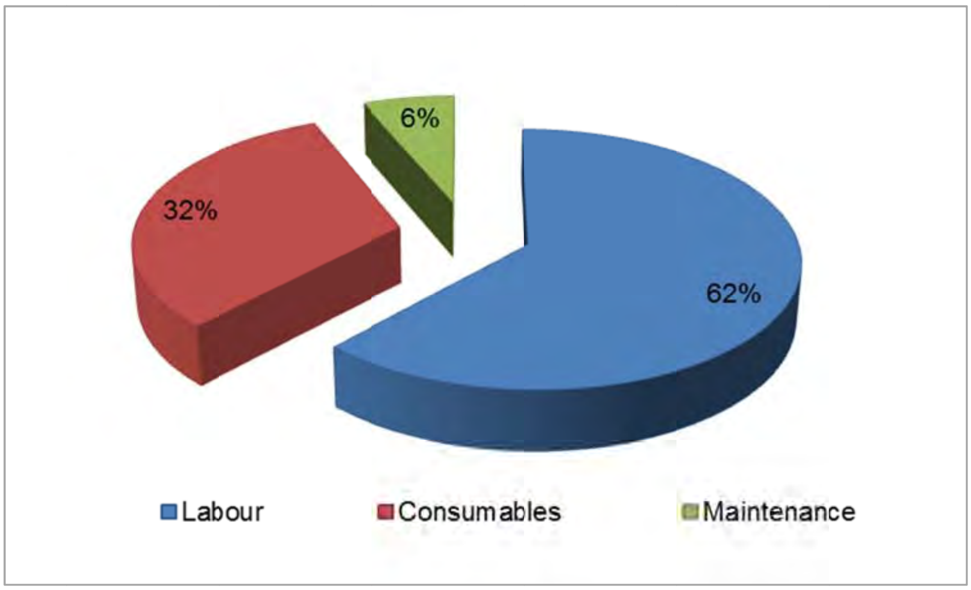
Figure 21.4 presents the processing operating cost distribution.

Table 21.17: Average Processing Operating Costs

Area	Average $M/yr	LOM $M	$/t milled
Labour	8.0	88.5	19.96
Consumables	4.1	45.5	10.25
Maintenance	0.8	9.0	2.03
Total	12.9	143.0	32.24

Source: JDS 2014

Figure 21.4: Process Operating Cost Distribution



Source: JDS 2014

Processing Labour

Processing Labour includes a peak of 68 employees: 26 operations employees, four effluent treatment plant operators, 26 maintenance employees, and 12 technical services employees. The majority of the processing workforce will operate on a 2 x 2 rotation, with supervisory personnel (foreman and above) working 8 x 6 rotations. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations. Table 21.11 summarizes the processing workforce labour rates.

Table 21.18: Process Labour Rates

Position	Salary/Hourly	Total Annual Salary/ Hourly Rate ($)
Processing Operations		
Mill Process Superintendent	Staff	182,340
Mill General Foreman	Staff	127,606
Mill Shift Foreman	Staff	113,494
Crushing Operator	Hourly	58.17
Crushing Helpers	Hourly	45.03
Grinding Mill Operators	Hourly	58.17
Gold Room/Gold Recovery Operators	Hourly	58.17
Control Room Operators	Hourly	58.17
Flotation Operators	Hourly	58.17
Filtration/Bagging	Hourly	58.17
Reagent Prep	Hourly	49.30
Tailing Delivery/General Labours (Shared)	Hourly	45.03
Limestone Prep Plant Operators	Hourly	45.03
Effluent Treatment Plant Operators		
Effluent Treatment Plant Operators	Hourly	45.03
Process Maintenance		
Mill Maintenance Shift Foreman	Staff	127,606
Mechanics/Millrights	Hourly	58.17
Mechanic Apprentice	Hourly	49.30
Maintenance Supervisor	Staff	127,606
Electricians	Hourly	58.41
Electrical Apprentices	Hourly	49.30
Welders	Hourly	55.36
Instrument Technicians	Hourly	55.36
Crane / Equipment Operators (Shared)	Hourly	51.73
General Labour for Maintenance	Hourly	42.79
Process Technical Services		
Chief Metallurgist	Staff	127,606
Plant Metallurgist	Staff	113,494
Metallurgical Technicians	Staff	103,921
Senior Metallurgist	Staff	113,494
Assay Technicians	Staff	84,992
Sample Preparation	Hourly	38.49

Source: JDS 2014

Operating & Maintenance Supplies

Reagent consumption rates for the concentrator and effluent treatment plant were determined through metallurgical test work 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 an as-needed basis by the mine personnel.

An allowance was made in each processing area for maintenance supplies, based on the capital cost and complexity of the equipment in each area. The total annual allowance for maintenance supplies between the concentrator and effluent plant is $0.8M.

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 generator sets. The average unit energy cost is estimated at $0.326/kWh.

The primary energy consumer on site is the concentrator plant, followed by mining operations and then the site infrastructure. Table 21.12 summarizes the site energy consumption by area.

Table 21.19: Average Annual Energy Consumption by Area

Area	**Average kWh/yr**	**%**
Underground Mine	138,783,000	28
Processing Facilities	324,224,000	64
Camp	36,178,000	7
Tailings Reclaim Pump	5,086,000	1
Total	504,271,000	100

Source: JDS 2014

The total estimate power plant operating unit cost is $36.16 per tonne processed. The subsections below further describe the basis of estimate for major items within each grouping.

Fuel

Fuel consumption rates were determined using manufacturer consumption curves and account for the average load factor per operating year (average 79% loading). An average 11.9 Ml of diesel is required per year. Fuel build-up details are outlined in Table 21.13.

Table 21.20: Fuel Rate Build-up

Component	Mobile Equipment ($)	Power Generation ($)
Diesel Rack Rate - Juneau, AK*	0.97	0.97
Transport to Site	0.04	0.04
FET	0.04	0.00
PET	0.03	0.03
Carbon Tax	0.08	0.08
Total Fuel Cost ($/L)	**1.15**	**1.11**

* August 24, 2014

Source: JDS 2014

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 internal 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 similar installations with similar infrastructure.

General & Administrative Operating Costs

G&A costs are grouped into the following categories:

- G&A Labour;
- G&A On-Site Items;
- Support Equipment Fleet;
- Satellite Office and Off-Site Warehousing;
- Freight; and
- Employee Travel.

The total G&A operating unit cost is estimated at $28.50 per tonne processed. Table 21.21 summarizes the annual G&A operating costs.

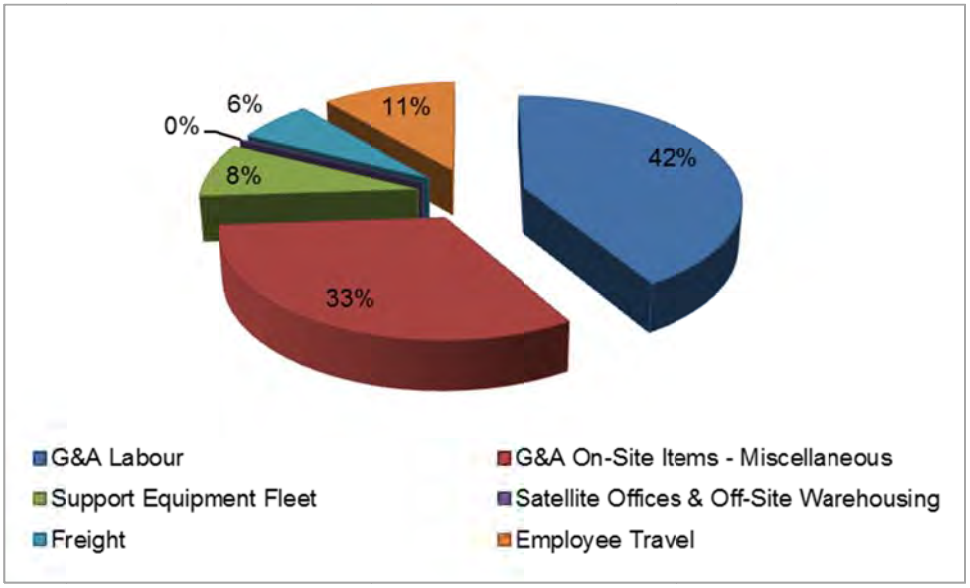
Figure 21.5 illustrates the G&A operating cost distribution.

Table 21.21: G&A Operating Cost Summary

Area	Average $M/yr	LOM ($)	$/tonne milled
G&A Labour	4.7	52.4	11.82
G&A On-Site Items - Miscellaneous	3.7	41.0	9.25
Support Equipment Fleet	0.9	10.5	2.37
Satellite Offices & Off-Site Warehousing	0.0	0.2	0.04
Freight	0.7	8.0	1.80
Employee Travel	1.3	14.2	3.21
Total	11.4	126.4	28.50

Source: JDS 2014

Figure 21.5: G&A Operating Cost Distribution



Source: JDS 2014

Table 21.22: G&A Detailed Costs

Area	Average $M/yr	LOM ($)	$/tonne milled
G&A Labour			
Surface Infrastructure & Maintenance Labour	2.4	26.6	5.99
First Aid	0.2	2.5	0.56
Environmental	0.4	4.1	0.91
Administration	0.8	9.0	2.02
HSE - Health & Safety	0.2	2.3	0.52
Human Resources	0.3	3.9	0.87
IT & Communications	0.2	2.3	0.52
Security	0.2	1.9	0.42
Total Labour	4.7	52.4	11.82
G&A On-Site Items			
Camp Catering & Cleaning	2.3	25.4	5.73
Health & Safety, Medical, First Aid	0.3	2.8	0.63
Environmental	0.2	2.5	0.57
Human Resources	0.1	1.2	0.27
Insurance & Legal	0.4	4.9	1.11
External Consulting	0.0	0.5	0.10
IT & Communications	0.2	1.7	0.38
Offices & Miscellaneous Costs	0.2	2.0	0.46
Total G&A On-Site Items	3.7	41.0	9.25
Support G&A			
Support Equipment	0.9	10.5	2.37
Satellite Office	0.0	0.2	0.04
Freight	0.7	8.0	1.80
Employee Travel	1.3	14.2	3.21
Total	11.4	126.4	28.50

Source: JDS 2014

G&A Labour

G&A Labour includes 48 employees (at peak). 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 21.16 summarized the G&A workforce labour rates.

Table 21.23: G&A Labour Rates

Position	Salary/Hourly	Total Annual Salary/ Hourly Rate ($)
Surface Infrastructure & Maintenance		
Maintenance Superintendent	Staff	182,340
Chief Electrician	Staff	162,091
Surface Foreman	Staff	113,494
Electrician - Surface Shops	Hourly	58.41
Mechanic - Surface Shops	Hourly	58.17
Carpenter - Surface Shops	Hourly	55.36
Labourer - Surface Shops	Hourly	42.79
Mobile Equipment Operator	Hourly	51.73
Surface Infrastructure & Maintenance - Total		
First Aid		
Nurse/First Aid/Security	Staff	113,494
First Aid - Total		
Environment		
Sustainability Manager	Staff	127,606
Environmental Officer	Staff	103,921
Environmental Technician	Hourly	38.49
Environment - Total		
Administration		
Mine/General Manager	Staff	220,996
Controller/Accountant	Staff	103,921
Payroll Supervisor	Staff	103,921
Transport dispatch Supervisor	Staff	113,494
Warehouse Clerk/Tech	Staff	68,276
Administration - Total		
Health & Safety		
Safety Training Coordinator	Staff	103,921
Health & Safety - Total		
Human Resources		
Human Resources Manager	Staff	162,091
Human Resources Clerk	Staff	84,992
Community Relations Coordinator	Staff	103,921
Human Resources - Total		
IT & Communications		
IT/Telecom. Technician	Staff	103,921
IT & Communications - Total		
Security		
Protective Services Supervisor	Staff	103,921
Protective Services Officials (Site)	Hourly	47.62
Security - Total		

Source: JDS 2014

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.

Miscellaneous On-Site Items

The miscellaneous G&A grouping includes 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; and
- Offsite Office Costs.

Support Equipment Fleet

Costs for fuel and maintenance for each piece of support equipment were based on an allowance of operating hours per year.

Satellite Offices

A small satellite office will be established in Atlin, primarily to support Human Resources, payroll, and accounting. Costs were estimated for the building lease.

General Freight

General freight includes barging costs for the following items:

- Fuel;
- Cement;
- Oil & Lube;
- Explosives;
- Drilling Consumables;
- Ground Support;
- Pipe and Cable;
- Ventilation;
- Maintenance Parts;
- Annual Operating Spares;
- Grinding Media;
- Liners;
- Reagents; and
- Other Supplies.

An estimated annual average of 7,200t/year of barging was estimated for these items. An allowance for airfreight was not included, it has been assumed that any supplies that will be transported by air (primarily fresh food) will be loaded in passenger planes, and thus the costs are considered incidental to the passenger transportation costs.

Outbound Concentrate Shipping

Operating costs for the transportation of concentrate from the mine site to the transhipment point at the mouth of the Taku River were based on indicative shipping costs provided by a barging contractor that has extensive river and ocean barging experience in the area.

The total operating costs for contract concentrate freight is estimated to be $165.06/wmt this includes a river barging and floating marine facility (transsshipment site) components.

River barge operating costs were estimated to be $144.01/wmt and include the following items:

- Fixed annual ownership fee for barges and tugs;
- Fixed annual maintenance fees (dry-docking) for barges and tugs;
- Variable maintenance costs (daily) for tugs;
- Personnel costs for operations, maintenance and supervision;
- River guides;
- Fuel;
- Insurance; and
- Contractor overhead and profit.

Contract floating marine facilities (transshipment facilities) operating costs were estimated to be $21.05/wmt and include the following:

- Fixed annual ownership fee for crane;
- Fixed annual maintenance fees (dry-docking) for barges and tugs;
- Rental costs for floating facility (barge, camp) and support equipment;
- Variable maintenance costs (daily) for tugs;
- Personnel costs for operations;
- Miscellaneous costs for housing, maintenance and fuel;
- Insurance; and
- Contractor overhead and profit.

Inbound Material Shipping

Operating costs associated with shipment of inbound materials are based on unit prices provided by a local contractor for ocean freight. The following table provides the unit price for various materials shipped to the transshipment site.

Table 21.24: Inbound Freight Unit Prices

	Unit Price
Fuel	$0.035/liter
Cement	$92.39/tonne
Reagents & Grinding Media	$108.70/tonne
Other Supplies	$157.61/tonne

Source: JDS 2014

Passenger Transportation

All workers will be transported to site via charter aircraft departing from Erik Neilson 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 1,520 commercial flights and 225 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;
- The majority of employees on a 8 & 6 rotation will reside in the Whitehorse area and will not require commercial flights;
- 25% of the remaining employees will reside in the Whitehorse area and will not require commercial flights; and
- Charter flight costs were based on contractor quotations for the utilization of a 15 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 and allowance for contractors). A 90% capacity factor was included to allow for cancellations and visitors.

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, grades, recoveries, 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 Metals Inc. for the Tulsequah project with respect to permitting.

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

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21.0 of this report and are presented in 2014 dollars. The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

Two 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. The metal price assumptions used in the economic analysis are outlined in Table 22.1.

All costs and economic results are reported in Canadian dollars ($C), unless otherwise noted, while metal pricing is reported in US dollars (US$). Both cases use identical LOM plan tonnage and grade estimates which are outlined in Table 22.2. On-site and off-site costs and production parameters were also held constant for each case.

Table 22.1: Metal Prices used in the Economic Analysis

Commodity	Unit	Spot as at 15-Oct-14	Consensus Economics Publication Oct-14
Copper Price	US$/lb	3.08	3.38
Lead Price	US$/lb	0.93	1.10
Zinc Price	US$/lb	1.06	1.18
Gold Price	US$/oz	1,238	1,373
Silver Price	US$/oz	17.00	23.07
Exchange Rate	US$:C$	0.89	0.90

Source: JDS 2014

Table 22.2: Life of Mine Plan Summary

Parameter	Unit	Value
Mine Life	Years	11.1
Total Ore	M tonnes	4.4
Throughput Rate	tpd	1,100
Average Head Grade		
Cu	%	1.46
Pb	%	1.29
Zn	%	6.95
Au	g/t	2.85
Ag	g/t	103.72
Metal Production		
Cu Concentrate Produced	dmt	274,256
	Average dmt/yr	24,760
Pb Concentrate Produced	dmt	61,868
	Average dmt/yr	5,586
Zn Concentrate Produced	dmt	462,089
	Average dmt/yr	41,718
Au Payable	k oz committed	62.3
	k oz uncommitted	293.7
	Total k oz	356.0
	Average k oz/yr	32.1
Ag Payable	k oz committed	1,860.4
	k oz uncommitted	9,095.3
	Total k oz	10,955.7
	Average k oz/yr	989.1

Source: JDS 2014

Other economic factors used include the following:

- Discount Rate of 8% (sensitivities using other discount rates have been calculated for each scenario);
- Reclamation costs of $8.2M;
- Salvage value of $4.4M;
- Nominal 2014 dollars;
- No Inflation;
- No PST;
- Numbers are presented on 100% ownership and do not include management fees or financing costs; and
- Exclusion of all pre-development and sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.). However, pre-development and sunk costs are utilized in tax calculations.

22.1.1 Timing of Revenues and Working Capital

Working capital has been considered in the economic analysis by the timing difference between cash outflows and cash inflows with respect to the operating costs. The following describes how the operating costs were scheduled to occur in the economic analysis:

Mining Operating Costs

- 50% of consumables and fuel required for mine operations is assumed to be purchased one year prior to the actual consumption. The remaining 50% of costs is assumed in the year the costs are assumed to occur. This models the incurrence of the costs for a portion of the consumables before their actual use due to the seasonal barge re-supply;
- Labour costs are assumed to be incurred as required; and
- A total of $2.1M of mine operating costs are assumed to occur in the pre-production period of cash flows and include labour, equipment, consumables and fuel.

Processing Operating Costs

- 50% of consumables, concentrate bags, and maintenance costs are assumed to be incurred one year prior to the actual occurrence/requirement based on the proposed mine plan. The remaining 50% of costs is assumed in the year the costs are assumed to occur;
- Labour costs are assumed to be incurred as required; and
- A total of $2.3M of processing operating costs is calculated to occur in the pre-production period of cash flows.

Power Operating Costs

- 50% of fuel costs are assumed to occur in the year prior to actual consumption. The remaining 50% are assumed to occur in the year the consumption is to take place;
- Lease payments for the power plant are assumed to occur in the period in which they occur; and
- A total of $4.9M of power operating costs is assumed to occur in the pre-production period.

Transportation Operating Costs

- A total of $4.3M of transportation operating costs is accounted for in the pre-production period but relate to the operating costs to occur in Year 1 of production.

22.2 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 leading industry participants who have provided Chieftain with indicative terms of the market conditions with respect to the concentrates to be produced. These details can be found in Section 19.0 of this report. Concentrate production and sale of concentrate is assumed to begin in 2017 for a period of 11 years, ending in 2028. Tables 22.3 to 22.7 indicate the NSR parameters that were used in the economic analysis. Figure 22.1 demonstrates the amount of concentrate produced during the mine life. Figure 22.2 shows the base case life of mine NSR by product.

Table 22.3: Gravity Concentrate Smelter Terms

NSR Assumptions	Unit	Gravity Concentrate
Recoveries		
Au	%	41
Ag	%	0.5
Smelter Payables		
Au Payable	%	99.9
Ag Payable	%	99
Refining Charge		
Au	US $/oz	0.65
Ag	US $/oz	0.65
Shipping Cost	US$/payable oz	1.15

Source: JDS 2014

Table 22.4: Copper Concentrate Smelter Terms

NSR Assumptions	Unit	Cu Concentrate
Recoveries		
Cu	%	89
Au	%	47
Ag	%	77.6
Concentrate Grade	**%**	**21**
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	96.5
Au Payable	%	95
Ag Payable	%	90
Minimum Deduction in Conc	%	1
Au Minimum Deduction	g/t	0
Ag Minimum Deduction	g/t	30
TC/RCs		
Treatment Charge	US$/dmt concentrate	150
Refining Charge		
Cu	US $/lb	0.15
Au	US $/oz	6.00
Ag	US $/oz	0.50
Deleterious Element Penalties		
As	US $/dmt concentrate	41.20
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

Source: JDS 2014

Table 22.5: Zn Concentrate Smelter Terms

NSR Parameters	Unit	Pb Concentrate
Recoveries		
Pb	%	65
Au	%	2.8
Ag	%	6.3
Concentrate Grade	**%**	**60**
Moisture Content	%	8
Smelter Payables		
Pb Payable	%	95
Au Payable	%	95
Ag Payable	%	95
Minimum Deduction in Conc	%	3
Au Minimum Deduction	g/t	1.5
Ag Minimum Deduction	g/t	50
TC/RCs		
Treatment Charge	$/dmt concentrate	100
Refining Charge		
Au	US $/oz	25.00
Ag	US $/oz	1.50
Escalator Costs		
Pb	$/dmt concentrate	1.70
Threshold	*$/tonne*	2000
Charge	*$/tonne*	0.04
Threshold	*$/tonne*	2500
Charge	*$/tonne*	0.06
Threshold	*$/tonne*	3000
Charge	*$/tonne*	0.08
Transport Costs		
Ocean Freight	US$wmt concentrate	119.10
	US$/dmt concentrate	129.46

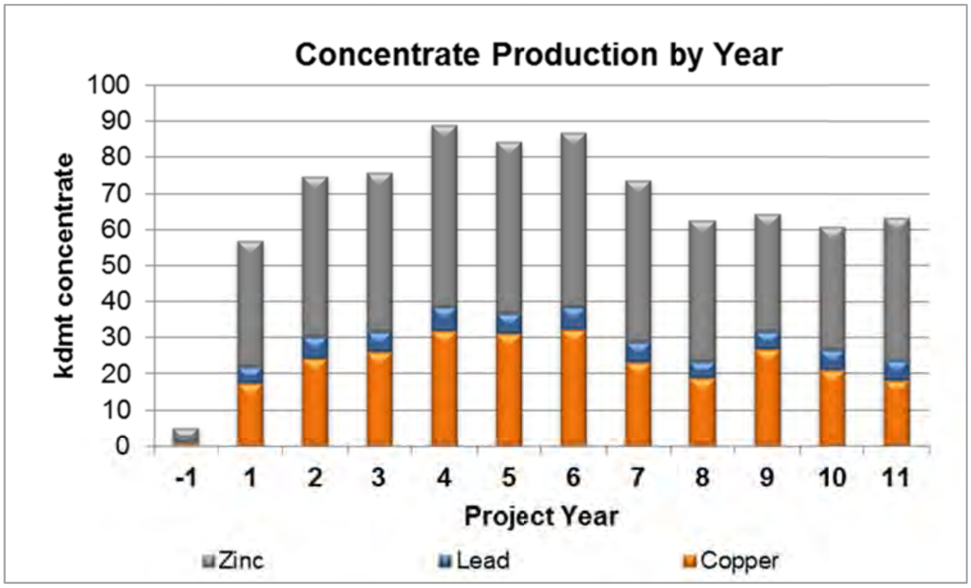
Source: JDS 2014

Table 22.6: Zn Concentrate Smelter Terms

NSR Parameters	Unit	Zn Concentrate
Recoveries		
Zn	%	90
Au	%	0
Ag	%	0
Concentrate Grade	**%**	**60**
Moisture Content	%	8
Smelter Payables		
Zn Payable	%	85
Minimum Deduction in Conc	%	8
Au Minimum Deduction	g/t	1
Ag Minimum Deduction	g/t	93
TC/RCs		
Treatment Charge	$/dmt concentrate	165
Escalator Costs		
Zn	$/dmt conc	13.20
Threshold	*$/tonne*	2000
Charge	*$/tonne*	0.04
Threshold	*$/tonne*	2500
Charge	*$/tonne*	0.06
Threshold	*$/tonne*	3000
Charge	*$/tonne*	0.08
Transport Costs		
Ocean Freight	US$/wmt concentrate	119.10
	US$/dmt concentrate	129.46

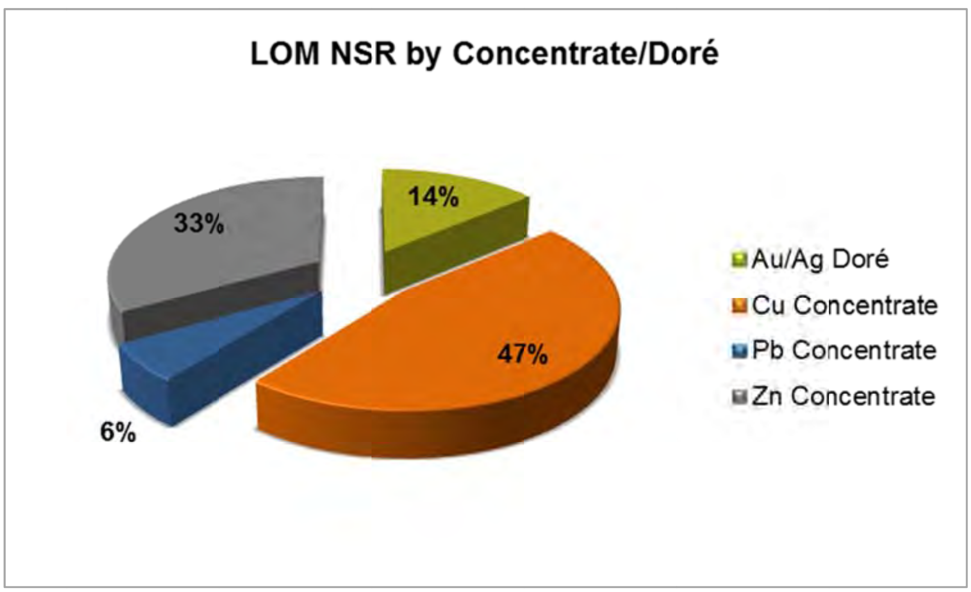
Source: JDS 2014

Figure 22.1: Life of Mine Concentrate Production by Year



Source: JDS 2014

Figure 22.2: Base Case Life of Mine NSR by Product



Source: JDS 2014

22.3 Summary of Capital Costs

The following capital costs were used for the economic analysis. Detailed information can be found in Section 21.0 of this report.

Table 22.7: Summary of Capital Costs

Pre-Production CAPEX	Pre-Production $M	Production $M	LOM $M
Underground Mining	18.4	61.0	79.4
Underground Infrastructure	10.5	0.0	10.5
Site Development	3.9	0.0	3.9
Processing Plant	44.6	0.0	44.6
Tailings & Waste Rock Management	6.6	12.7	19.2
On-Site Infrastructure	33.9	4.3	38.1
Off-Site Infrastructure	0.0	0.0	0.0
Project Indirects	15.3	0.0	15.3
Engineering & EPCM	13.5	0.0	13.5
Owner's Costs	21.3	0.0	21.3
Closure & Salvage	0.0	3.8	3.8
Pre-Production OPEX	12.3	0.0	12.3
Subtotal	180.2	81.7	261.9
Contingency (11.4%)	18.4	2.4	20.8
Total Capital Costs	198.6	84.1	282.6

Source: JDS 2014

22.4 Summary of Operating Costs

Total operating costs amount to $708M. This translates to an average cost of $159.49/tonne processed over the life of mine. These costs are shown in Table 22.9.

Table 22.8: Breakdown of Operating Costs

Operating Costs	Avg $M/yr	LOM $M	$/t processed
Mining	11.8	130.2	29.36
Processing	12.9	143.0	32.24
Power	14.5	160.4	36.16
Transport	13.3	147.4	33.23
G&A	11.4	126.4	28.50
Total Operating Costs	**63.9**	**707.5**	**159.49**

Note: Concentrate transport costs are calculated as part of the Net Revenue calculation in the economic analysis and are not shown in this table. All in operating costs amount to $185.78/tonne processed.

Source: JDS 2014

22.5 Taxes

The project has been evaluated on an after-tax basis in order 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 consultants for the purpose of the After-Tax valuation of the project. PST has been excluded from the economic analysis.

A detailed tax analysis was completed specifically for the purpose of evaluating the Tulsequah Chief project. Specific assumptions and methodology in the analysis includes the following:

BC Mineral Tax

- The BC Mineral tax is comprised of 2 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 pre-production exploration and development expenses. If the mine has an operating loss, no net current proceeds tax (Tier 1 Tax) is payable; and
- 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 & Provincial Corporate Income Tax

- Federal tax rate of 15.0% and a combined BC (10.0%) and Ontario (11.5%) rate were used to determine a blended 25.0% rate which 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 pre-production 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 in order to appropriately determine the taxable income for the project.

The tax analysis completed amount to a LOM taxes payable of $136.6M. The after-tax values are determined solely for project valuation purposes.

22.6 Streaming Contract with Royal Gold

In December 2011, Chieftain entered into a gold and silver purchase transaction with Royal Gold Inc. to sell a portion of the precious metals produced at the Tulsequah Chief mine. The details of this contract are outlined in Section 19.0 of this report. Under the agreement, Chieftain will receive a total of US$45M during the pre-production period of the mine. A summary of the streaming contract with Royal Gold is shown in Table 22.9 and 22.10.

Table 22.9: Summary of Streaming Contract with Royal Gold

Area	Unit	Silver	Gold
Precious metal sold via streaming contract	oz ('000)	2,739	62
Total revenue from streaming during production*	USD$ M	11.6	23.0
Total precious metal available (Pre-Streaming)	oz ('000)	10,956	356
Total precious metal uncommitted (Post-Streaming)	oz ('000)	8,217	294

*Utilizing Base Case Metal Pricing

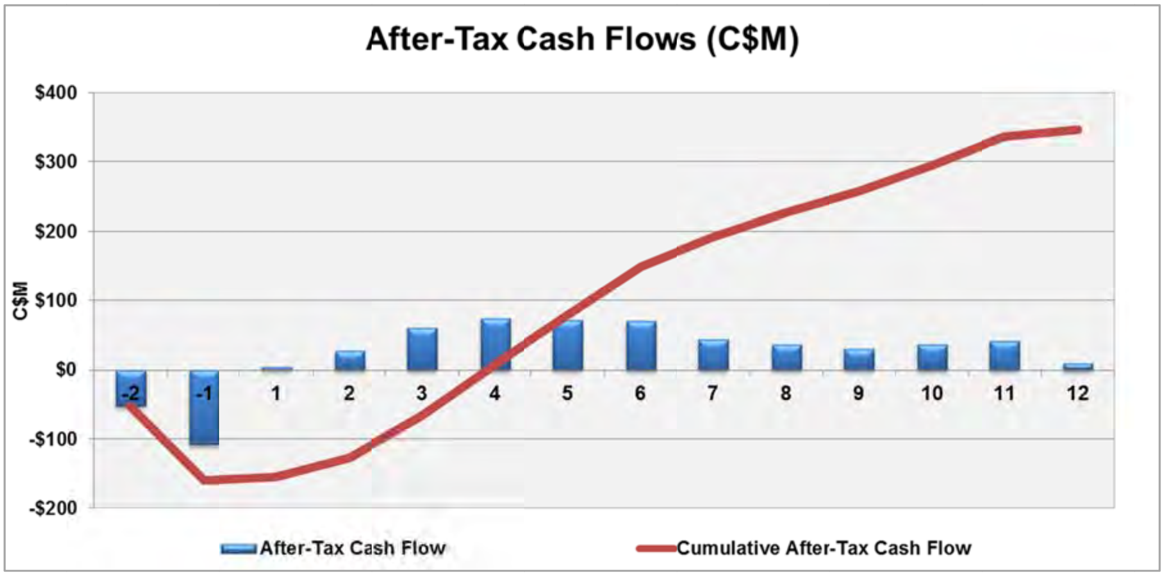
Source: JDS 2014

22.7 Economic Results

The project is economically viable with an after-tax internal rate of return (IRR) of 21.8% and net present value at 8% (NPV8%) of $145.6M for the Base Case which was calculated based on spot metal prices and exchange rate as at October 15, 2014. One additional case was measured based on projected long-term metal prices and exchange rate by Consensus Economic' October 2014 report (Forward Pricing Scenario).

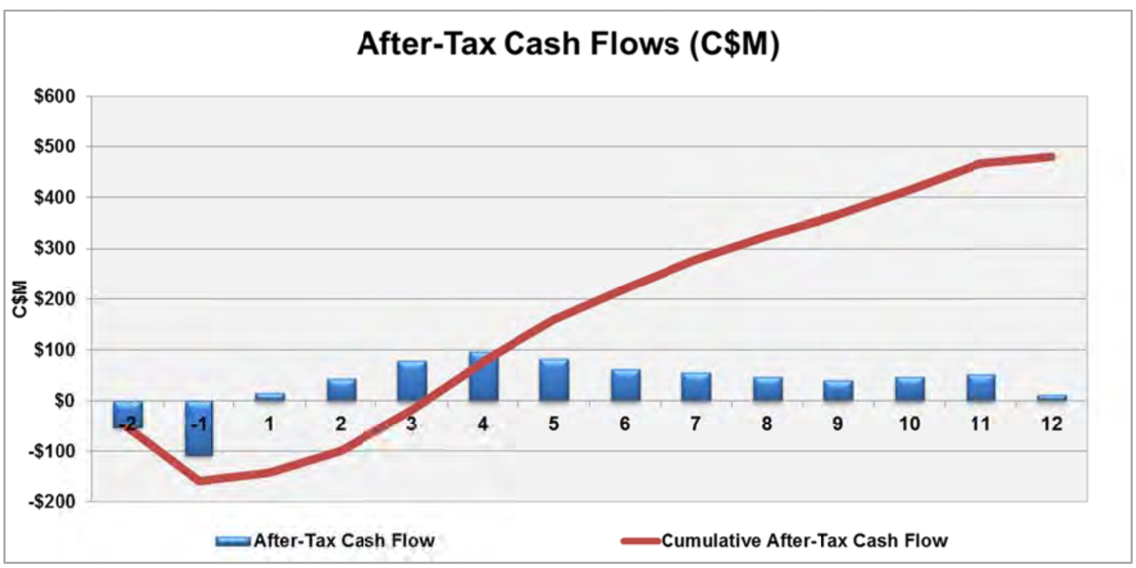
The Forward Pricing Scenario resulted in the highest performance and project value. Figure 22.3 and Figure 22.4 show the projected cash flows for the project used in the economic analysis. Table 22.10 shows the economic results of each of the cases evaluated.

Figure 22.3: Annual and Cumulative Life of Mine Cash Flows – Base Case Pricing



Source: JDS 2014

Figure 22.4: Annual and Cumulative Life of Mine Cash Flows – Forward Pricing



Source: JDS 2014

Table 22.10: Summary of Base Case Pricing Economic Results

Category	Unit	Base Case Metal Prices	Forward Metal Prices
Net Revenues	$M	1,421.1	1,629.2
Operating Costs	$M	707.5	707.5
Cash Flows from Operations	$M	713.7	921.7
Capital Costs*	$M	282.6	282.6
Up-Front Streaming Revenues	$M	50.7	50.0
Net Pre-Tax Cash Flow	$M	481.7	689.0
Pre-Tax $NPV_{8\%}$	$M	211.7	334.4
Pre-Tax IRR	%	25%	33%
Pre-Tax Payback	Years	3.8	3.2
Total Taxes	$M	136.5	210.7
Net After-Tax Cash Flow	$M	345.2	478.4
After-Tax $NPV_{8\%}$	$M	145.6	228.4
After-Tax IRR	%	22%	29%
After-Tax Payback	Years	3.9	3.2

(*) Includes pre-production, sustaining, closure and reclamation capital costs

Source: JDS 2014

22.8 Sensitivity

A sensitivity analysis was performed to test project value drivers on average annual operating cash flows and project Net Present Values (NPV) using an 8% discount rate. The results of this analysis are demonstrated in Table 22.11 through Table 22.14 and illustrated in Figure 22.5 and Figure 22.6. The project proved to be most sensitive to changes in metal prices and head grades, followed by operating costs. The project showed least sensitive to capital costs.

A sensitivity analysis of the pre-tax and after-tax results was performed using various discount rates. The results of this analysis are demonstrated in Table 22.15 and Table 22.16.

Table 22.11: After-Tax NPV8% Sensitivity Test Results – Base Case Pricing

Factor	-10%	100%	10%
Metal Price	76.6	145.6	213.2
Head Grade	87.7	145.6	203.1
OPEX	175.0	145.6	115.9
CAPEX	170.5	145.6	120.7

Source: JDS 2014

Table 22.12: Average Annual Operating Cash Flow Sensitivity – Base Case Pricing

Factor	-10%	100%	10%
Metal Price	49.7	64.8	79.9
Head Grade	52.1	64.8	77.6
OPEX	71.0	64.8	58.6

Source: JDS 2014

Table 22.13: After-Tax NPV8% Sensitivity Test Results – Forward Looking Pricing

Factor	-10%	100%	10%
Metal Price	152.8	228.4	303.1
Head Grade	163.3	228.4	293.2
OPEX	257.3	228.4	199.4
CAPEX	253.2	228.4	203.5

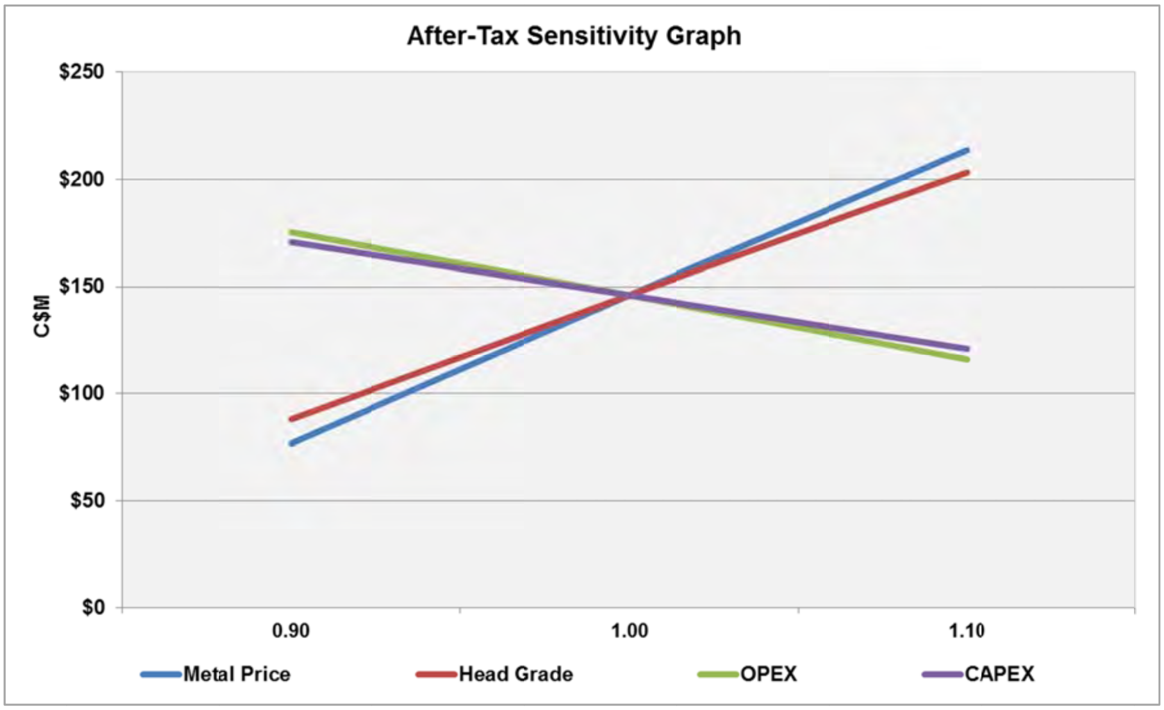
Source: JDS 2014

Table 22.14: Average Annual Operating Cash Flow Sensitivity – Forward Looking Pricing

Factor	-10%	100%	10%
Metal Price	66.5	83.4	100.3
Head Grade	68.8	83.4	98.1
OPEX	89.6	83.4	77.2

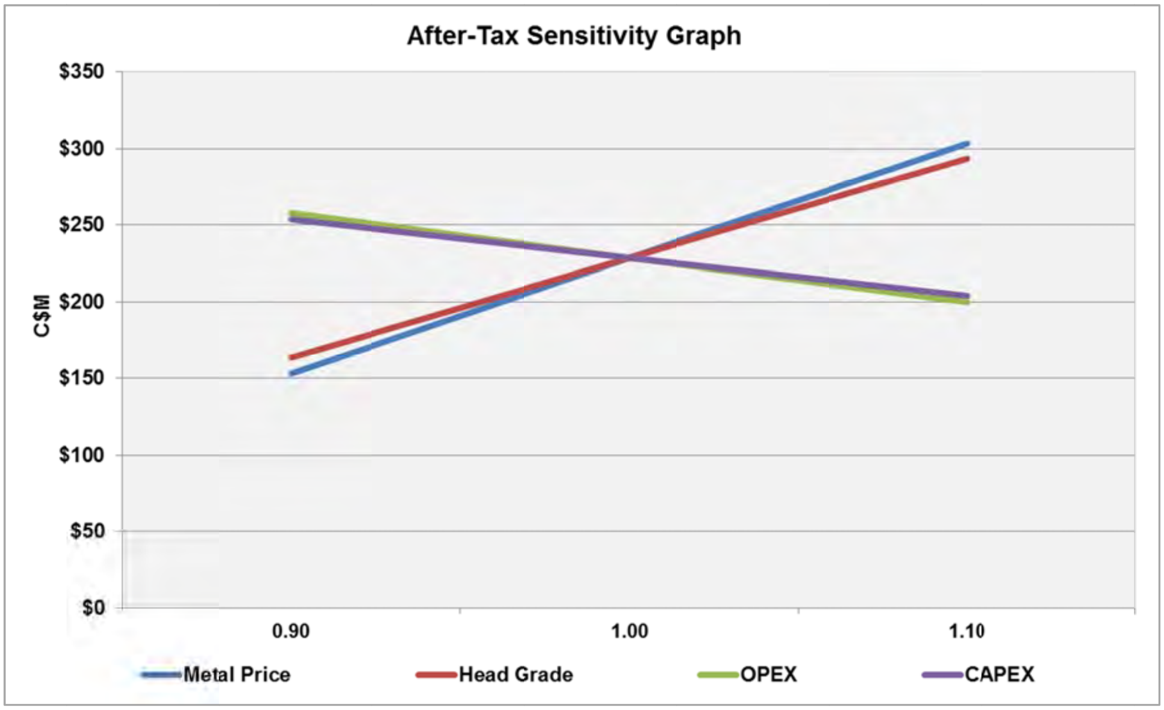
Source: JDS 2014

Figure 22.5: After-Tax NPV8% Sensitivity Graph – Base Case Pricing



Source: JDS 2014

Figure 22.6: After-Tax NPV8% Sensitivity Graph – Forward Pricing



Source: JDS 2014

Table 22.15: Discount Rate Sensitivity Test Results – Base Case Pricing

Discount Rate	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0%	481.7	345.2
5%	290.5	204.6
8%	211.7	145.6
10%	169.5	113.8
12%	134.0	86.7

Source: JDS 2014

Table 22.16: Discount Rate Sensitivity Test Results – Forward Pricing

Discount Rate	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0%	689.0	478.4
5%	438.3	302.4
8%	334.4	228.4
10%	278.7	188.3
12%	231.6	154.2

Source: JDS 2014

23. ADJACENT PROPERTIES

There are no adjacent properties that impact the results of this FS Technical Report in any way.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution

24.1.1 Introduction & Philosophy

The project execution plan for the Tulsequah Chief FS 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;
- 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; and
- Same camp accommodation status applied to all site personnel (no management quarters).
-

24.1.2 Project Execution Plan (PEP) Summary

The PEP utilizes seasonal barging as the primary delivery method for equipment and materials that are required for the construction of the project. Construction and utilization of an all-weather access road as the primary method for deliveries is no longer a feasible option.

Chieftain retained the services of Ausenco Engineering Canada Inc. (Ausenco) to assess, analyze and report on the navigability of the Taku River. A logistics study and execution plan for successful access to the mine site for construction and operations related to river transportation was developed. Based on estimates developed by JDS and the analysis conducted by Ausenco it was determined that the 9,806 t of construction freight can be feasibly transported to site by means of river barging.

The majority of construction freight will be mobilized utilizing conventional river barges with fixed wing aircraft delivering fuel in the first year of construction and select bulk freight, and passengers through the duration of construction.

24.1.3 Existing Site Development

There is currently a 1,050 m airstrip adjacent to the Tulsequah River near the mine site, which is in a reasonable condition to accommodate light aircraft such as Dornier and Caravans to provide passenger and freight service to the site.

A pioneer camp consisting of 50 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.

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.

24.1.4 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.

24.1.5 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; and
- Project risk procedures.

24.1.6 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 P6 as the primary scheduling software. All scheduling will be performed utilizing the critical path method (CPM). The schedules will be resource loaded as manpower constraints exist due to camp size. The master schedule owned by Project Controls will include all major procurement activities, milestones and construction activities.

The schedule is operated by Project Controls, but is built as a team including the Project Manager, Engineering Manager, Construction Managers, and Superintendents. The Project Manager will report any potentially serious issues to the Owner as soon as they are identified. The Project Manager is ultimately responsible and accountable for the Project performance in particular the budget and the schedule.

24.1.7 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.

Process equipment considered to be "long lead," 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. A detailed list of "long lead" items of the estimated committed funds required once financing has been achieved is outlined in Table 24.1.

Table 24.1: Long Lead Items

Package Description	Lead Time	Required Order Date	Estimated Arrival Date	Commitment Upon Funding(1)
Long Lead Items: Year -2 (2015)				
SAG Mill & Ball Mills	50 weeks	30-Apr-15	15-Apr-16	$575,000
Power plant	42 weeks	25-June-15	15-Apr-16	$4,000,000
Construction Camp	9 weeks	16-Jan-15	16-Mar-15	$50,000
Permanent Camp	13 weeks	31-Jan-15	30-Apr-15	$800,000
Process Plant Buildings (x3)	13 weeks	2-Mar-15	30-May-15	$450,000
Internal Steel – Flotation & Filtration Buildings	12 weeks	2-Mar-15	23-May-15	$500,000
Fuel Tanks (x2 – 5,000,000 Liter)	20 weeks	2-Mar-15	19-July-15	$650,000
Thickeners	26 weeks	15-Feb-15	13-Aug-15	$175,000
Flotation Cells	17 weeks	16-Jan-15	15-May-15	$515,000
Screens, Cyclone, Feeders, Gravity Concentrators	20 weeks	2-Mar-15	19-July-15	$150,000
Mining Equipment	17 weeks	2-Mar-15	29-June-15	$1,300,000
Primary Jaw Crusher, Dust Collectors PC, Rock Breaker, Pressure Filters, Bagging System, Pumps Belt Feeders	20 weeks	1-Apr-15	18-Aug-15	$275,000
MCC's	20 weeks	2-Mar-15	19-July-15	$275,000
Gold Room	30 weeks	1-Apr-15	27-Oct-15	$150,000
Total Deposits (for Year -2)				**$9,865,000**

Source: JDS 2014

24.1.8 Freight & Logistics

The most critical aspect of the schedule revolves around the Project Logistics Plan. The logistics plan completed by Ausenco removes the project's dependence on the completion of the site access road, and instead relies primarily on two river barging campaigns for delivery of equipment and materials. A fixed wing aircraft campaign will support construction activities with delivery of fuel, select bulk materials and passengers through the duration of construction.

The plan addresses the requirements for barge freight, airfreight and truck freight, personnel transport to support the project schedule, as well as the available source data and methods used to determine the feasible capacity of each delivery method.

Based on estimates developed by JDS and the analysis conducted by Ausenco it was determined that the 9,806 t of construction freight can be feasibly transported to site by means of river barging. For further information on Freight and Logistics, refer to Section 3.0 of the Tulsequah: Project Execution Plan.

24.1.9 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.

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.

24.1.10 Key Schedule Milestones

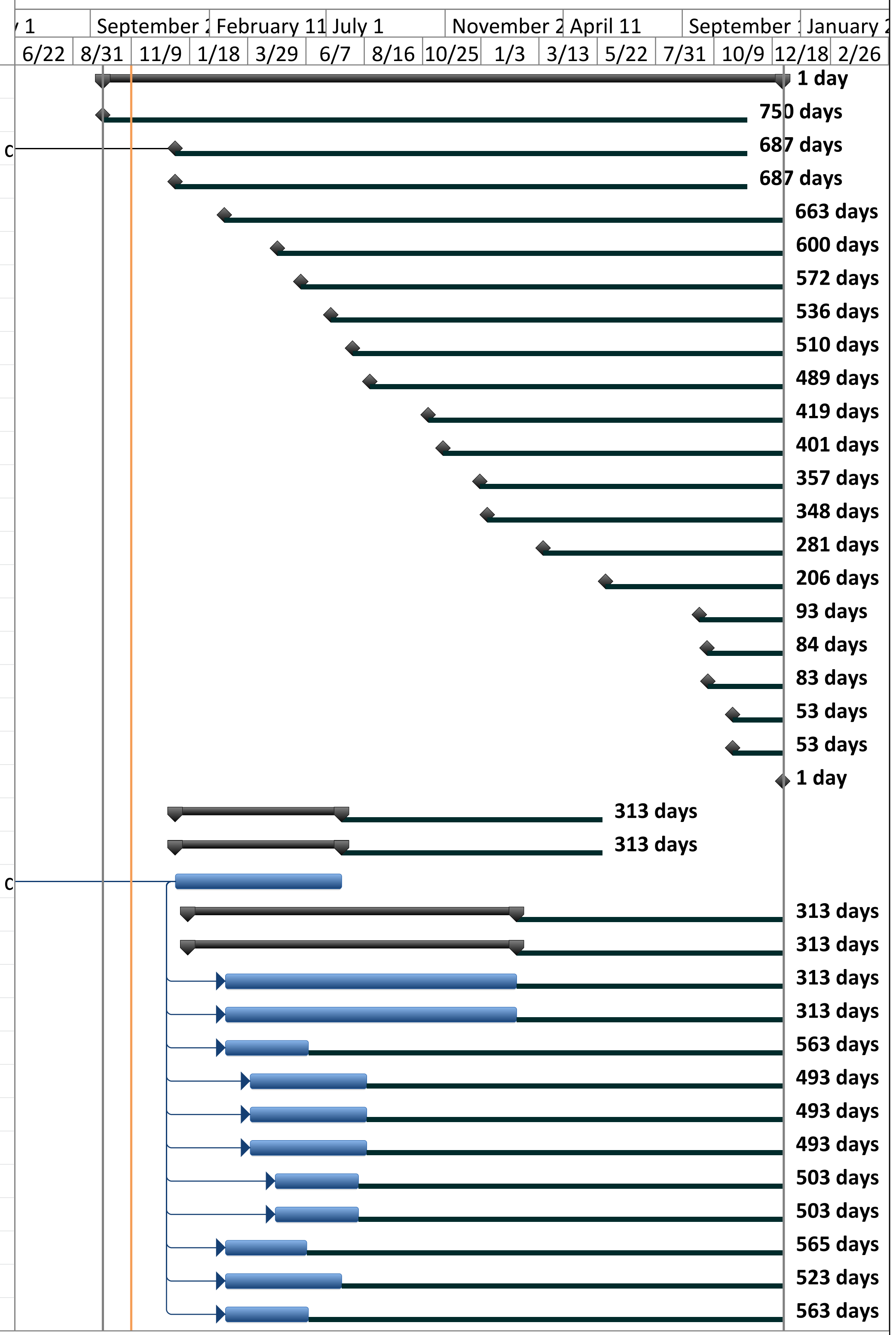
The construction execution schedule is driven by the spring/summer barging transportation windows in 2015 and 2016. A detailed, resource loaded schedule has been developed for the site construction activities, utilizing the feasibility cost estimate as the basis for the required manhours. This scheduling exercise indicates that mechanical completion and wet commissioning can be accomplished by the end of Q4 2016, providing that materials and equipment can be ordered and shipped to site during the Q2 2015 barging season. The key schedule milestones are presented in Table 24.2. The summary project execution Gantt chart schedule is shown on Figure 24.3.

Table 24.2: Key Schedule Milestones

Description	Milestone Date
Project Financing Approved	January 1, 2015
Detailed Engineering Awarded	January 1, 2015
Award Barging Contract	March 1, 2015
Early Works Construction Begins	May 4, 2015
Year 1 River Barge Campaign	June 1-August 22, 2015
Crusher/Batch Plant Established	July 7, 2015
Construction Camp Commissioned	August 2, 2015
Plant Site Earthworks Completed	August 23, 2015
Operations Camp Commissioned	November 1, 2015
Grinding Building Erected	November 19, 2015
Underground Mining Begins	January 2, 2016
Flotation Building Erected	January 11, 2016
Filtration Building Erected	March 18, 2016
Year 2 River Barge Campaign	June 1-August 1, 2016
Tailings Facilities Completed	September 22, 2016
Process Plant Mechanical Completion	October 1, 2016
Wet Commissioning Complete	October 31, 2016
Process Commissioning Complete	December 31, 2016

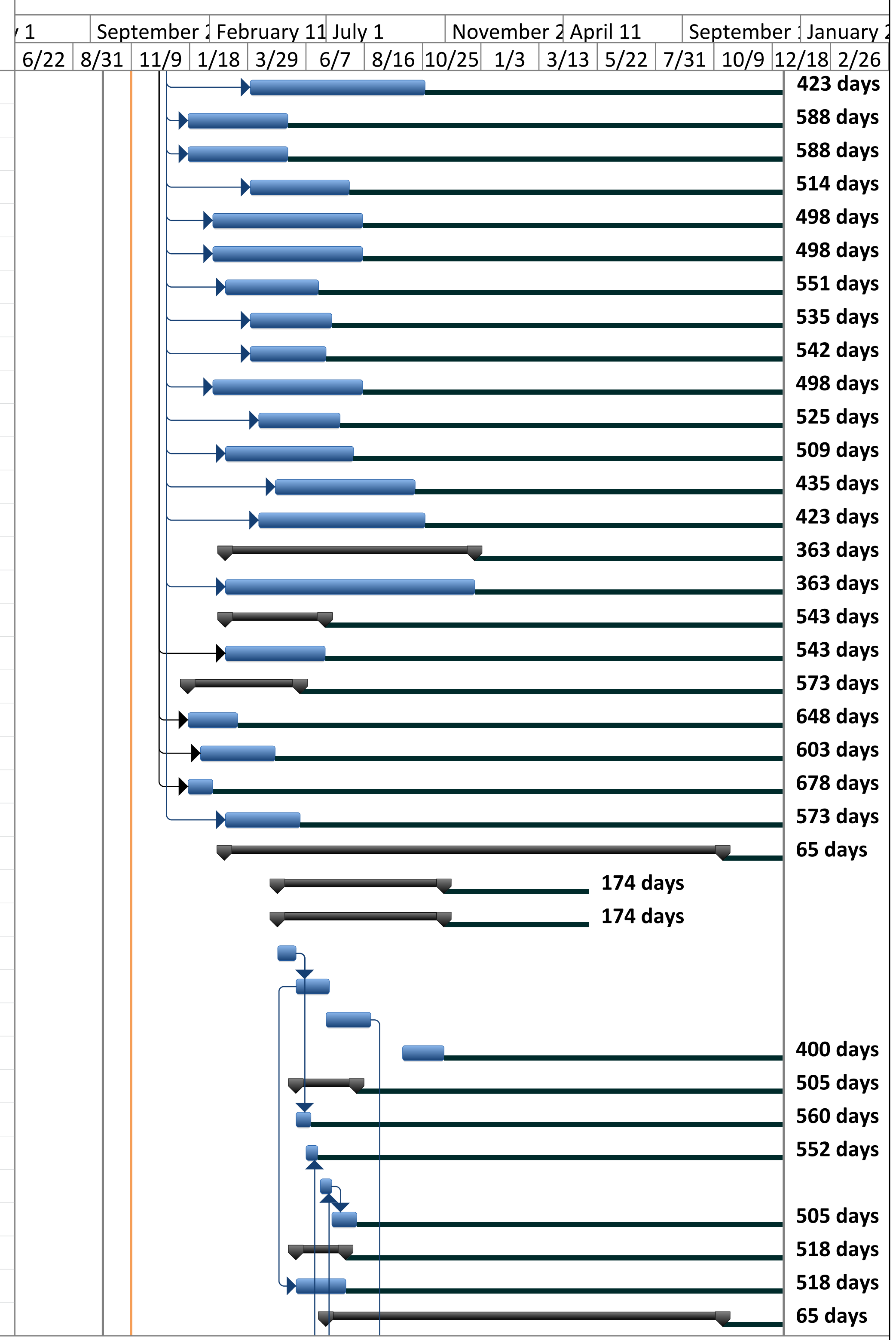
Source: JDS 2014

ID	Task Mode	Name	Leveling Delay	Duration	Start	Finish	Successors
1		**PROJECT MILESTONES**	**0 edays**	**785 days**	**Mon 10/6/14**	**Sat 12/31/16**	
2		Submit Feasibility Report	0 edays	0 days	Mon 10/6/14	Mon 10/6/14	
3		Funding Approved	0 edays	0 days	Thu 1/1/15	Thu 1/1/15	56SS+60 days,60SS+15 c
4		Engineering Commences	0 edays	0 days	Thu 1/1/15	Thu 1/1/15	
5		Award Barging Contract	0 edays	0 days	Sun 3/1/15	Sun 3/1/15	
6		Early Works Construction Commences	0 edays	0 days	Mon 5/4/15	Mon 5/4/15	
7		First Barge Arrives at Site - Year 1	0 edays	0 days	Mon 6/1/15	Mon 6/1/15	
8		Setup Crusher/Batch Plant	0 edays	0 days	Tue 7/7/15	Tue 7/7/15	
9		Construction Camp Established	0 edays	0 days	Sun 8/2/15	Sun 8/2/15	
10		Plant Site Earthworks Complete	0 edays	0 days	Sun 8/23/15	Sun 8/23/15	
11		Permanent Camp Established	0 edays	0 days	Sun 11/1/15	Sun 11/1/15	
12		Grinding Building Erected	0 edays	0 days	Thu 11/19/15	Thu 11/19/15	
13		Underground Mining Begins	0 edays	0 days	Sat 1/2/16	Sat 1/2/16	
14		Flotation Building Erected	0 edays	0 days	Mon 1/11/16	Mon 1/11/16	
15		Filtration Building Erected	0 edays	0 days	Fri 3/18/16	Fri 3/18/16	
16		First Barge Arrives at Site - Year 2	0 edays	0 days	Wed 6/1/16	Wed 6/1/16	
17		Tailings & Waste Facilities Complete	0 edays	0 days	Thu 9/22/16	Thu 9/22/16	
18		Process Plant Mechanical Completed	0 edays	0 days	Sat 10/1/16	Sat 10/1/16	
19		Wet Commissioning Begins	0 edays	0 days	Sat 10/1/16	Sat 10/1/16	
20		Wet Commissioning Complete	0 edays	0 days	Mon 10/31/16	Mon 10/31/16	
21		Initial Process Commissioning Begins	0 edays	0 days	Tue 11/1/16	Tue 11/1/16	
22		Commissioining Complete	0 edays	0 days	Sat 12/31/16	Sat 12/31/16	
23		**ENGINEERING**	**0 edays**	**200 days**	**Thu 1/1/15**	**Sun 7/19/15**	
24		**DETAILED ENGINEERING**	**0 edays**	**200 days**	**Thu 1/1/15**	**Sun 7/19/15**	
25		Detailed Engineering	0 edays	200 days	Thu 1/1/15	Sun 7/19/15	28SS+60 days,29SS+60 c
26		**PROCUREMENT (Delivered to Prince Rupert)**	**0 edays**	**395 days**	**Fri 1/16/15**	**Sun 2/14/16**	
27		**Major Equipment - Mechanical**	**0 edays**	**395 days**	**Fri 1/16/15**	**Sun 2/14/16**	
28		SAG Mill	0 edays	350 days	Mon 3/2/15	Sun 2/14/16	
29		Ball Mills (Primary & Secondary)	0 edays	350 days	Mon 3/2/15	Sun 2/14/16	
30		SAG Mill Feed Conveyor	0 edays	100 days	Mon 3/2/15	Tue 6/9/15	
31		Primary Jaw Crusher	0 edays	140 days	Wed 4/1/15	Tue 8/18/15	
32		Primary Crusher Dust Collector	0 edays	140 days	Wed 4/1/15	Tue 8/18/15	
33		Rock Breaker	0 edays	140 days	Wed 4/1/15	Tue 8/18/15	
34		Storage Bins Belt Feeder	0 edays	100 days	Fri 5/1/15	Sat 8/8/15	
35		Storage Bin Dust Collector	0 edays	100 days	Fri 5/1/15	Sat 8/8/15	
36		Screens	0 edays	98 days	Mon 3/2/15	Sun 6/7/15	
37		Cyclones (Primary & Secondary)	0 edays	140 days	Mon 3/2/15	Sun 7/19/15	
38		Leach Aid Feeder	0 edays	100 days	Mon 3/2/15	Tue 6/9/15	



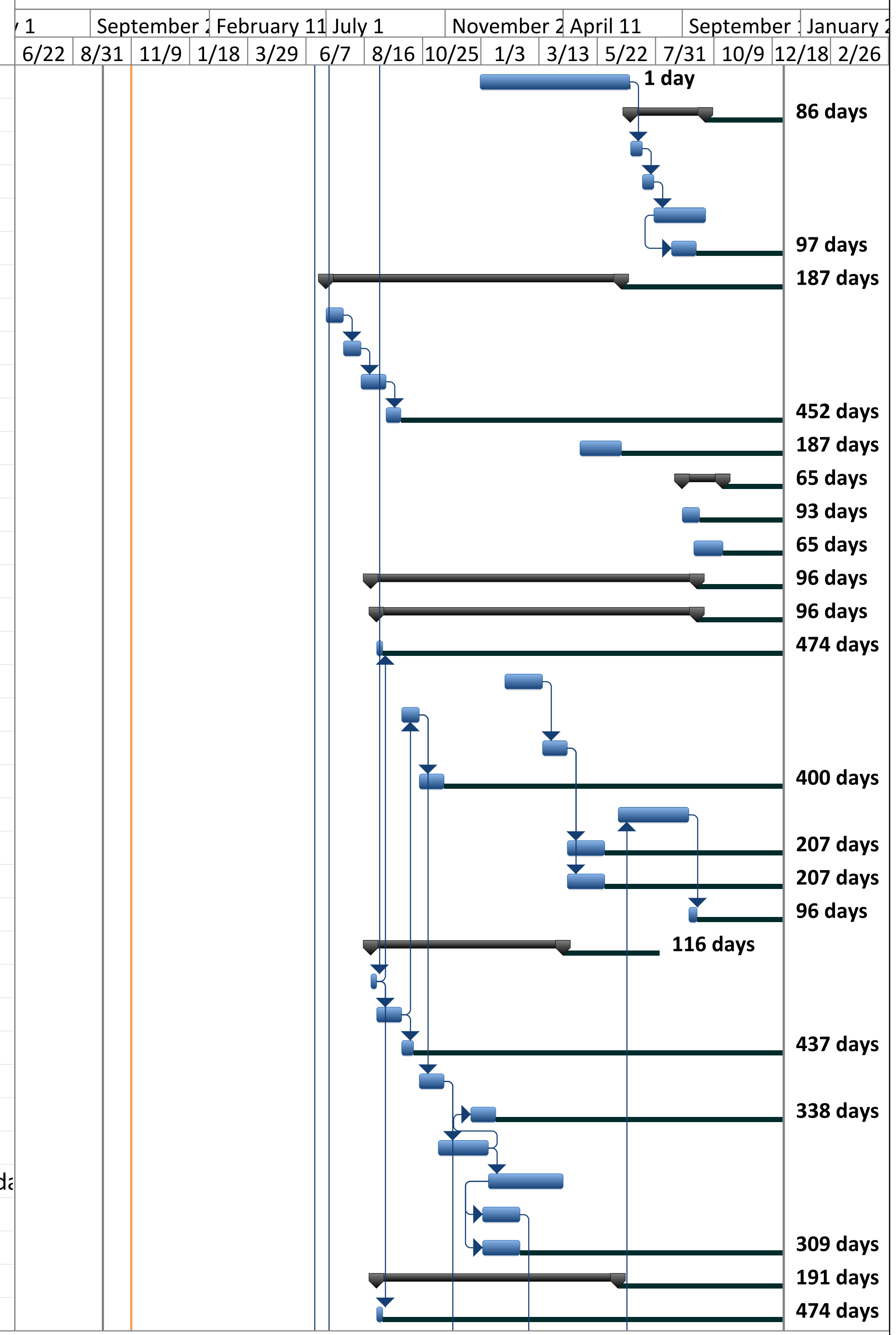
Critical
Critical Split
Task
Split
Milestone
Slack
Slippage
Summary
Project Summary
Rolled Up Critical
Rolled Up Critical Split
External Tasks
External Milestone
Inactive Task
Inactive Milestone
Inactive Summary
Manual Task
Duration-only
Manual Summary Rollup
Manual Summary
Start-only
Finish-only
Deadline
Progress

ID	Ta M	Name	Leveling Delay	Duration	Start	Finish	Successors
39		Gold Room - Pkg	0 edays	210 days	Wed 4/1/15	Tue 10/27/15	
40		Rougher Flotation Cells (Cu, Zn, Pb)	0 edays	120 days	Fri 1/16/15	Fri 5/15/15	
41		Cleaner Flotation Cells (Cu, Zn, Pb)	0 edays	120 days	Fri 1/16/15	Fri 5/15/15	
42		Pressure Filter (Cu, Zn, Pb)	0 edays	119 days	Wed 4/1/15	Tue 7/28/15	
43		Concentrate Thickeners (Cu x2)	0 edays	180 days	Sun 2/15/15	Thu 8/13/15	
44		Concentrate Thickeners (Pyrite, Zn, Pb)	0 edays	180 days	Sun 2/15/15	Thu 8/13/15	
45		Gravity Concentrators	0 edays	112 days	Mon 3/2/15	Sun 6/21/15	
46		Pumps (Feed, Tailings, Misc. Large HP Pumps)	0 edays	98 days	Wed 4/1/15	Tue 7/7/15	
47		Bagging System	0 edays	91 days	Wed 4/1/15	Tue 6/30/15	
48		Final Tails Thickener	0 edays	180 days	Sun 2/15/15	Thu 8/13/15	
49		Reagent - Pkgs	0 edays	98 days	Sat 4/11/15	Fri 7/17/15	
50		Assay Lab	0 edays	154 days	Mon 3/2/15	Sun 8/2/15	
51		Blowers (x3)	0 edays	168 days	Fri 5/1/15	Thu 10/15/15	
52		Paste Backfill Equipment	0 edays	200 days	Sat 4/11/15	Tue 10/27/15	
53		**Major Equipment - Electrical**	**0 edays**	**300 days**	**Mon 3/2/15**	**Sat 12/26/15**	
54		Power Plant	0 edays	300 days	Mon 3/2/15	Sat 12/26/15	
55		**Major Equipment - Mining**	**0 edays**	**120 days**	**Mon 3/2/15**	**Mon 6/29/15**	
56		Mining Equipment	0 edays	120 days	Mon 3/2/15	Mon 6/29/15	
57		**On-Site Infrastructure/Process Plant**	**0 edays**	**135 days**	**Fri 1/16/15**	**Sat 5/30/15**	
58		Construction Camp	0 edays	60 days	Fri 1/16/15	Mon 3/16/15	
59		Permanent Camp	0 edays	90 days	Sat 1/31/15	Thu 4/30/15	
60		Construction Power	0 edays	30 days	Fri 1/16/15	Sat 2/14/15	
61		Process Plant - Pre-Engineered (x3 Buildings)	0 edays	90 days	Mon 3/2/15	Sat 5/30/15	
62		**CONSTRUCTION**	**0 edays**	**599 days**	**Sun 3/1/15**	**Wed 10/19/16**	
63		**10 - SITE DEVELOPMENT**	**0 edays**	**200 days**	**Mon 5/4/15**	**Thu 11/19/15**	
64		**1010 - PLANT SITE AREA**	**0 edays**	**200 days**	**Mon 5/4/15**	**Thu 11/19/15**	
65		Install Liner HPAG	0 edays	22 days	Mon 5/4/15	Mon 5/25/15	66,70
66		Excavate HPAG Rock & Haul to HPAG Facility	0 edays	40 days	Tue 5/26/15	Sat 7/4/15	75SS
67		Plant Site Area - Site Preparation	0 edays	54 days	Wed 7/1/15	Sun 8/23/15	104
68		OPAG Storage	0 edays	50 days	Thu 10/1/15	Thu 11/19/15	
69		**1020 - ANCILLARY AREAS**	**0 edays**	**73 days**	**Tue 5/26/15**	**Thu 8/6/15**	
70		Start Camp Pad / Site Prep - Utilize Existing Equipment	0 edays	18 days	Tue 5/26/15	Fri 6/12/15	
71		Complete Camp Pad - Use Barged Equipment	0 edays	14 days	Sun 6/7/15	Sat 6/20/15	
72		Setup Crusher/Batch Plant	0 edays	14 days	Wed 6/24/15	Tue 7/7/15	73
73		Construct Laydown Area - Paddy's Flats	0 edays	30 days	Wed 7/8/15	Thu 8/6/15	
74		**1030 - SITE ROADS/ 1040 - LIMESTONE CRUSHING**	**0 edays**	**60 days**	**Tue 5/26/15**	**Fri 7/24/15**	
75		Complete Site Road Upgrades, Bridge Repairs, Creek Bridges	0 edays	60 days	Tue 5/26/15	Fri 7/24/15	
76		**15 - UNDERGROUND MINING**	**0 edays**	**477 days**	**Wed 7/1/15**	**Wed 10/19/16**	



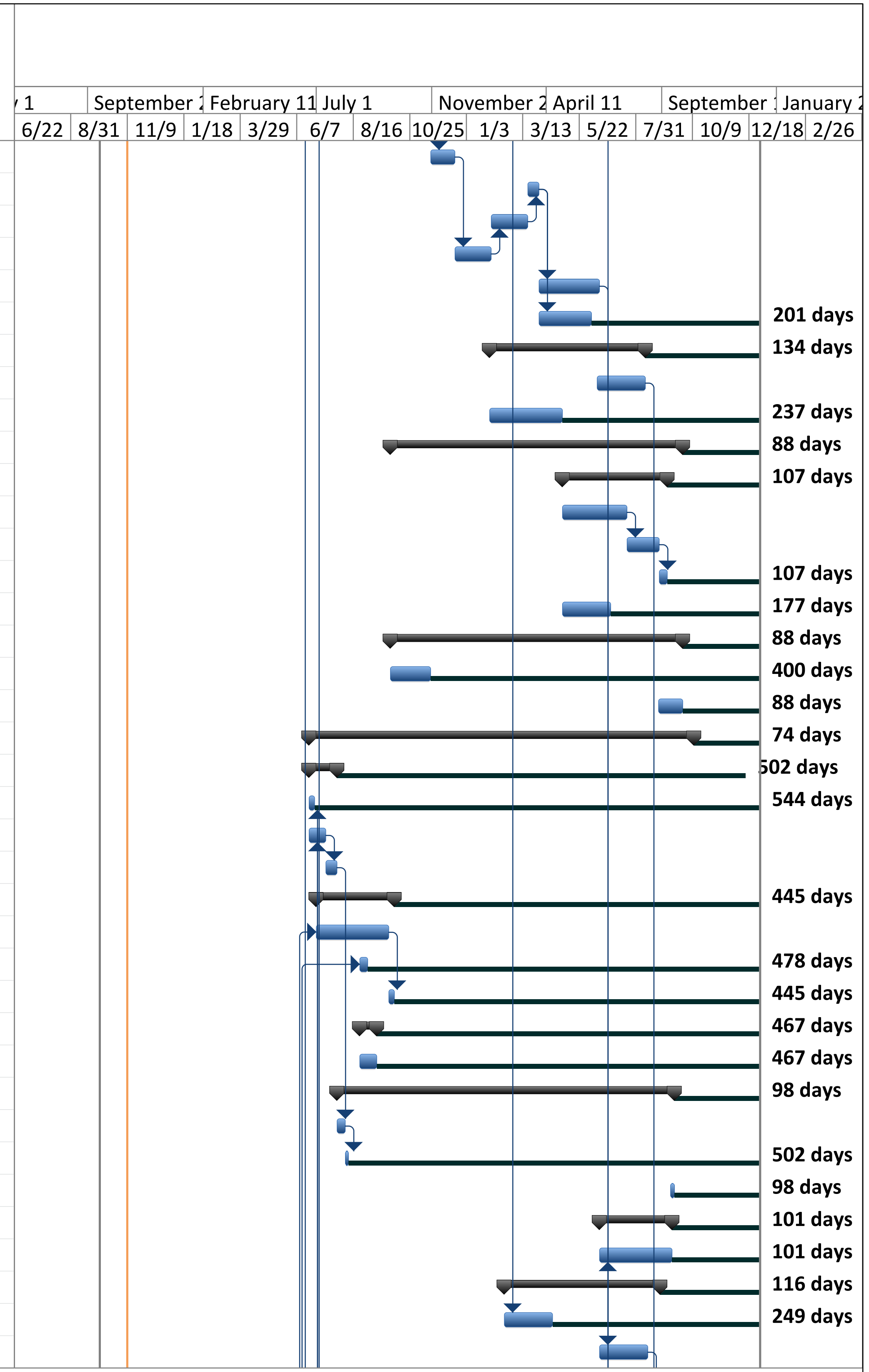
Critical		Slack		Rolled Up Critical Split		Inactive Summary		Start-only	
Critical Split		Slippage		External Tasks		Manual Task		Finish-only	
Task		Summary		External Milestone		Duration-only		Deadline	
Split		Project Summary		Inactive Task		Manual Summary Rollup		Progress	
Milestone		Rolled Up Critical		Inactive Milestone		Manual Summary			

ID	Task Mode	Name	Leveling Delay	Duration	Start	Finish	Successors
77		1505 - MINE DEVELOPMENT	0 edays	180 days	Sat 1/2/16	Wed 6/29/16	79
78		**1560 - UG CRUSHING EQUIPMENT & UTILITIES**	**0 edays**	**90 days**	**Fri 7/1/16**	**Wed 9/28/16**	
79		Install U/G Concrete	0 edays	14 days	Fri 7/1/16	Thu 7/14/16	80
80		Install Structural Steel	0 edays	14 days	Fri 7/15/16	Thu 7/28/16	81
81		Install U/G Crushing & Utilities - All Equipment	0 edays	62 days	Fri 7/29/16	Wed 9/28/16	82SS+21 days
82		Install Electrical & Instrumentation	0 edays	30 days	Fri 8/19/16	Sat 9/17/16	
83		**1560 - FINE ORE BIN & CONVEYORS**	**0 edays**	**355 days**	**Wed 7/1/15**	**Sun 6/19/16**	
84		Install Concrete	0 edays	21 days	Wed 7/1/15	Tue 7/21/15	85
85		Install Steel	0 edays	21 days	Wed 7/22/15	Tue 8/11/15	86
86		Install Mechanical Equipment	0 edays	30 days	Wed 8/12/15	Thu 9/10/15	87
87		Install Electrical & Instrumentation	0 edays	18 days	Fri 9/11/15	Mon 9/28/15	
88		Install Fine Ore bin	0 edays	50 days	Sun 5/1/16	Sun 6/19/16	
89		**1560 - UG PASTE PLANT**	**0 edays**	**49 days**	**Thu 9/1/16**	**Wed 10/19/16**	
90		Install U/G Concrete	0 edays	21 days	Thu 9/1/16	Wed 9/21/16	
91		Install U/G Paste Plant - Package	0 edays	35 days	Thu 9/15/16	Wed 10/19/16	
92		**25 - PROCESSING PLANT**	**0 edays**	**392 days**	**Mon 8/24/15**	**Sun 9/18/16**	
93		**2510 - GRINDING**	**0 edays**	**385 days**	**Mon 8/31/15**	**Sun 9/18/16**	
94		Detailed Excavation of Grinding	0 edays	7 days	Mon 8/31/15	Sun 9/6/15	
95		Install Equipment Bases	0 edays	45 days	Mon 2/1/16	Wed 3/16/16	97
96		Install Grade Beams, Piers & Walls	0 edays	21 days	Wed 9/30/15	Tue 10/20/15	98,107
97		Install Slab-on-Grade	0 edays	30 days	Thu 3/17/16	Fri 4/15/16	100,101
98		Install Structural Steel/Pre-Eng Building	0 edays	30 days	Wed 10/21/15	Thu 11/19/15	
99		Install Mechanical Equipment - Mills (Barging Constraint)	0 edays	85 days	Thu 6/16/16	Thu 9/8/16	102
100		Install Piping	0 edays	45 days	Sat 4/16/16	Mon 5/30/16	
101		Install Electrical & Instrumentation	0 edays	45 days	Sat 4/16/16	Mon 5/30/16	
102		Tie-In Electrical & Instrumentation to Mills	0 edays	10 days	Fri 9/9/16	Sun 9/18/16	
103		**2520 - FLOTATION**	**0 edays**	**231 days**	**Mon 8/24/15**	**Sun 4/10/16**	
104		Detailed Excavation of Flotation Building	0 edays	7 days	Mon 8/24/15	Sun 8/30/15	105,94,114
105		Install Flotation Building Foundations	0 edays	30 days	Mon 8/31/15	Tue 9/29/15	106,96
106		Rough-Set Mechanical Equipment	0 edays	14 days	Wed 9/30/15	Tue 10/13/15	
107		Install Grade Beams, Piers & Walls	0 edays	30 days	Wed 10/21/15	Thu 11/19/15	109FS-7 days,115
108		Install Slab-on-Grade	0 edays	30 days	Tue 12/22/15	Wed 1/20/16	
109		Install Structural Steel/Pre-Eng Building	0 edays	60 days	Fri 11/13/15	Mon 1/11/16	108FS-21 days,110
110		Install Mechanical Equipment (Includes Tailing Thickener)	0 edays	90 days	Tue 1/12/16	Sun 4/10/16	111SS-7 days,112SS-7 da
111		Install Piping	0 edays	45 days	Tue 1/5/16	Thu 2/18/16	151
112		Install Electrical & Instrumentation	0 edays	45 days	Tue 1/5/16	Thu 2/18/16	
113		**2530 - FILTRATION, REAGENTS & TAILINGS THICKENING**	**0 edays**	**290 days**	**Mon 8/31/15**	**Wed 6/15/16**	
114		Detailed Excavation of Filtration Building	0 edays	7 days	Mon 8/31/15	Sun 9/6/15	



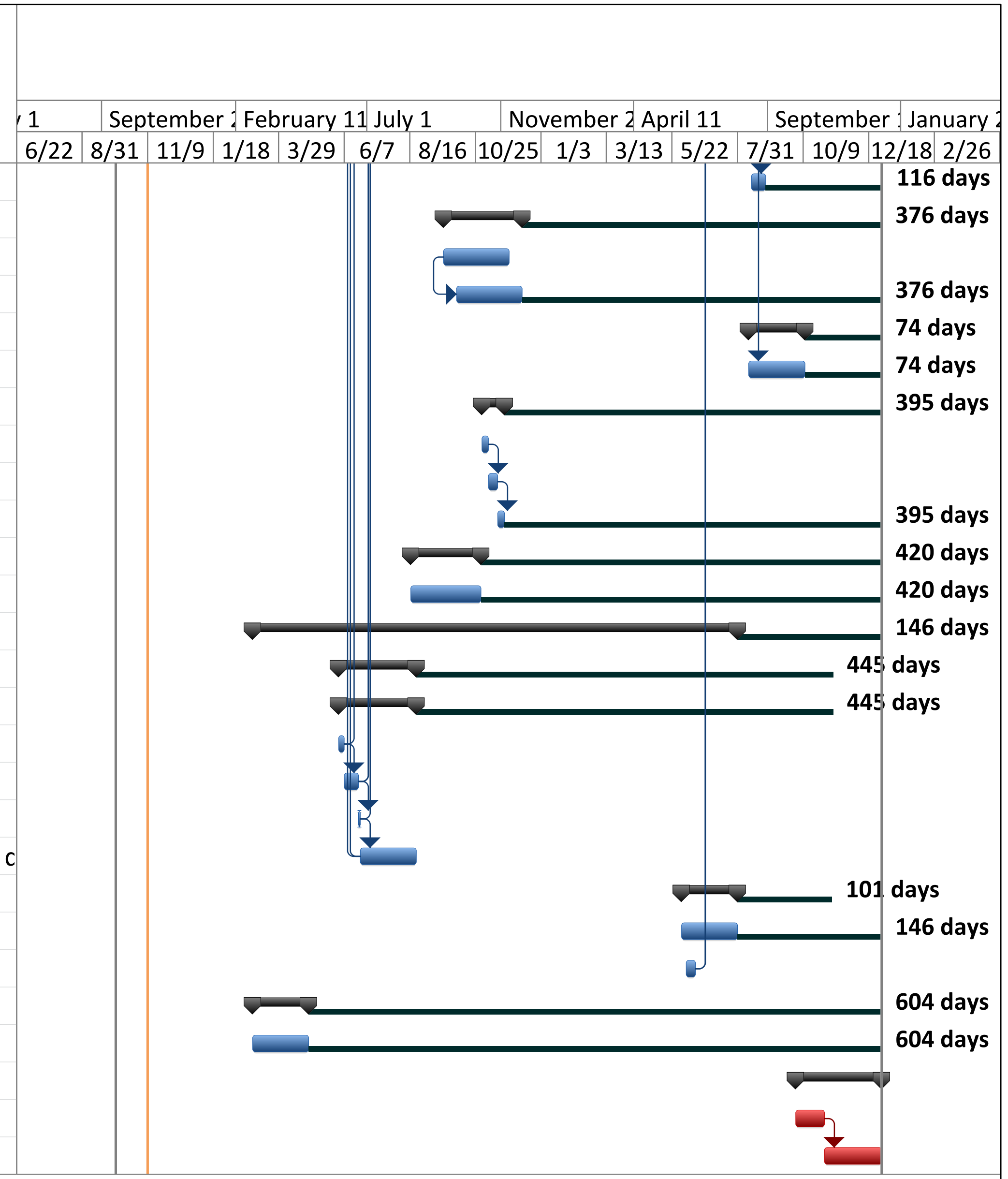
Critical		Slack		Rolled Up Critical Split		Inactive Summary		Start-only	
Critical Split		Slippage		External Tasks		Manual Task		Finish-only	
Task		Summary		External Milestone		Duration-only		Deadline	
Split		Project Summary		Inactive Task		Manual Summary Rollup		Progress	
Milestone		Rolled Up Critical		Inactive Milestone		Manual Summary			

ID	Task Mode	Name	Leveling Delay	Duration	Start	Finish	Successors
115		Install Grade Beams, Piers & Walls	0 edays	30 days	Fri 11/20/15	Sat 12/19/15	118
116		Install Slab-on-Grade	0 edays	14 days	Sat 3/19/16	Fri 4/1/16	119,120
117		Install Structural Steel/Pre-Eng Building	0 edays	45 days	Wed 2/3/16	Fri 3/18/16	116
118		Install Mechanical Equipment (Includes Tailing Thickener)	0 edays	45 days	Sun 12/20/15	Tue 2/2/16	117
119		Install Piping	0 edays	75 days	Sat 4/2/16	Wed 6/15/16	152
120		Install Electrical & Instrumentation	0 edays	65 days	Sat 4/2/16	Sun 6/5/16	
121		**2560 - PROCESS PLANT UTILITIES**	**0 edays**	**193 days**	**Mon 2/1/16**	**Thu 8/11/16**	
122		Process Water, Gland Water, Instrument Air	0 edays	60 days	Mon 6/13/16	Thu 8/11/16	158
123		Plant Control System	0 edays	90 days	Mon 2/1/16	Sat 4/30/16	
124		**30 - TAILINGS & WATER MANAGEMENT**	**0 edays**	**362 days**	**Thu 10/1/15**	**Mon 9/26/16**	
125		**3010 - TAILINGS BASIN, PIPELINE & BARGE**	**0 edays**	**130 days**	**Sun 5/1/16**	**Wed 9/7/16**	
126		Construct Tailings Basin - Stripping & Excavation	0 edays	80 days	Sun 5/1/16	Tue 7/19/16	127
127		Install Tailings Basin Liner	0 edays	40 days	Wed 7/20/16	Sun 8/28/16	128
128		Install Barge & Other Mech Equipment	0 edays	10 days	Mon 8/29/16	Wed 9/7/16	
129		Instal Pipeline	0 edays	60 days	Sun 5/1/16	Wed 6/29/16	
130		**3020 - PAG & PYRITE STORAGE FACILITY**	**0 edays**	**362 days**	**Thu 10/1/15**	**Mon 9/26/16**	
131		Bulk Earthworks for Pyrite Tailings Pond	0 edays	50 days	Thu 10/1/15	Thu 11/19/15	
132		Install Liners & Pipeline	0 edays	30 days	Sun 8/28/16	Mon 9/26/16	
133		**35 - ON-SITE INFRASTRUCTURE**	**0 edays**	**477 days**	**Mon 6/22/15**	**Mon 10/10/16**	
134		**3510 - CONSTRUCTION CAMP (ON SKIDS)**	**0 edays**	**35 days**	**Mon 6/22/15**	**Sun 7/26/15**	
135		Install Skid Camp incl. WTP & STP (Bunkhouse #1)	0 edays	7 days	Mon 6/22/15	Sun 6/28/15	
136		Install Kitchen/Diner (Permanent Camp Units used for Construction)	0 edays	21 days	Mon 6/22/15	Sun 7/12/15	137
137		Install Skid (Bunkhouse #2)	0 edays	14 days	Mon 7/13/15	Sun 7/26/15	145
138		**3510 - MAN PERMANENT CAMP**	**0 edays**	**97 days**	**Wed 7/1/15**	**Mon 10/5/15**	
139		Install Man Permanent Camp	0 edays	90 days	Wed 7/1/15	Mon 9/28/15	141
140		Install Fresh/Fire Water Systems at Camp	0 edays	10 days	Mon 8/24/15	Wed 9/2/15	
141		Tie-Ins to Kitchen/Diner	0 edays	7 days	Tue 9/29/15	Mon 10/5/15	
142		**3510 - ADMIN BUILDING**	**0 edays**	**21 days**	**Mon 8/24/15**	**Sun 9/13/15**	
143		Install Admin Building	0 edays	21 days	Mon 8/24/15	Sun 9/13/15	
144		**3520 - ANCILLARY FACILITIES**	**0 edays**	**418 days**	**Mon 7/27/15**	**Fri 9/16/16**	
145		Install Truck Shop (Sprung)	0 edays	10 days	Mon 7/27/15	Wed 8/5/15	146
146		Install Cold Storage (Seacans)	0 edays	4 days	Thu 8/6/15	Sun 8/9/15	
147		Install Assay Lab (Skids)	0 edays	5 days	Mon 9/12/16	Fri 9/16/16	
148		**3530 - POWER GENERATION & DISTRIBUTION**	**0 edays**	**90 days**	**Thu 6/16/16**	**Tue 9/13/16**	
149		Install Power Plant Generators (~12MW) & Distribution	0 edays	90 days	Thu 6/16/16	Tue 9/13/16	
150		**3530 - HEAT RECOVERY**	**0 edays**	**193 days**	**Fri 2/19/16**	**Mon 8/29/16**	
151		Install Interconnecting Glycol Piping - Flotation	0 edays	60 days	Fri 2/19/16	Mon 4/18/16	
152		Install Interconnecting Glycol Piping - Grinding & Filtration	0 edays	60 days	Thu 6/16/16	Sun 8/14/16	153



Critical		Slack		Rolled Up Critical Split		Inactive Summary		Start-only	
Critical Split		Slippage		External Tasks		Manual Task		Finish-only	
Task		Summary		External Milestone		Duration-only		Deadline	
Split		Project Summary		Inactive Task		Manual Summary Rollup		Progress	
Milestone		Rolled Up Critical		Inactive Milestone		Manual Summary			

ID	Task Mode	Name	Leveling Delay	Duration	Start	Finish	Successors
153		Install Boiler House, Unit Heaters, Etc	0 edays	15 days	Mon 8/15/16	Mon 8/29/16	
154		**3540 - BULK DIESEL STORAGE & DISTRIBUTION**	**0 edays**	**84 days**	**Mon 9/21/15**	**Sun 12/13/15**	
155		Install Fuel Tanks (5M L Tank #1)	0 edays	70 days	Mon 9/21/15	Sun 11/29/15	156SS+14 days
156		Install Fuel Tanks (5M L Tank #2)	0 edays	70 days	Mon 10/5/15	Sun 12/13/15	
157		**3550 - FRESH, FIRE, & POTABLE WATER SYSTEMS (PROCESS PLANT)**	**0 edays**	**60 days**	**Fri 8/12/16**	**Mon 10/10/16**	
158		Install Fresh/Fire Water Systems at Process Plant	0 edays	60 days	Fri 8/12/16	Mon 10/10/16	
159		**3570 - INCINERATOR FACILITY**	**0 edays**	**24 days**	**Sun 11/1/15**	**Tue 11/24/15**	
160		Install Foundations, Slabs, Curbs	0 edays	7 days	Sun 11/1/15	Sat 11/7/15	161
161		Install Incinerator Package	0 edays	10 days	Sun 11/8/15	Tue 11/17/15	162
162		Install Fencing	0 edays	7 days	Wed 11/18/15	Tue 11/24/15	
163		**3590 - MISC. INFRASTRUCTURE**	**0 edays**	**75 days**	**Mon 8/17/15**	**Fri 10/30/15**	
164		Barge Landing Upgrades	0 edays	75 days	Mon 8/17/15	Fri 10/30/15	
165		**90 - PROJECT INDIRECTS**	**0 edays**	**518 days**	**Sun 3/1/15**	**Sat 7/30/16**	
166		**9040 - FREIGHT & LOGISTICS - BARGING 2015 (125 Loads)**	**0 edays**	**83 days**	**Mon 6/1/15**	**Sat 8/22/15**	
167		**First Barge Arrives at Site**	**0 edays**	**83 days**	**Mon 6/1/15**	**Sat 8/22/15**	
168		Construction Equipment	0 edays	6 days	Mon 6/1/15	Sat 6/6/15	169,71
169		Camp Bunkhouse #1, WTP, STP, Kitchen/Diner, Equipment	0 edays	15 days	Sun 6/7/15	Sun 6/21/15	170,135,136
170		Crusher/Batch Plant	0 edays	2 days	Mon 6/22/15	Tue 6/23/15	72,171
171		Permanent Camp, Misc. Equipment, Materials	0 edays	60 days	Wed 6/24/15	Sat 8/22/15	139SS+7 days,140SS+6 c
172		**9040 - FREIGHT & LOGISTICS - BARGING 2016 (25 Loads)**	**0 edays**	**60 days**	**Wed 6/1/16**	**Sat 7/30/16**	
173		First Barge Arrives at Site - Equipment & Materials	0 edays	60 days	Wed 6/1/16	Sat 7/30/16	
174		Barge for Mills & Power Plant Arrive at Site	0 edays	10 days	Mon 6/6/16	Wed 6/15/16	99,149
175		**9040 - FREIGHT & LOGISTICS - GROUND 2015**	**0 edays**	**60 days**	**Sun 3/1/15**	**Wed 4/29/15**	
176		Ground Freight Commences	0 edays	60 days	Sun 3/1/15	Wed 4/29/15	
177		**COMMISSIONING & STARTUP**	**0 edays**	**84 days**	**Sat 10/1/16**	**Sat 12/31/16**	
178		Wet Commissioning	0 edays	31 days	Sat 10/1/16	Mon 10/31/16	179
179		Initial Process Commissioning	0 edays	53 days	Tue 11/1/16	Sat 12/31/16	



Critical		Slack		Rolled Up Critical Split		Inactive Summary		Start-only	
Critical Split		Slippage		External Tasks		Manual Task		Finish-only	
Task		Summary		External Milestone		Duration-only		Deadline	
Split		Project Summary		Inactive Task		Manual Summary Rollup		Progress	
Milestone		Rolled Up Critical		Inactive Milestone		Manual Summary			

24.1.11 Schedule Risks & Mitigation Plans

Risk – Seasonal river levels will not support the barging campaign

Low river discharge could cause interruptions to the barging campaign and prevent construction loads from being mobilized to support the construction schedule.

Mitigation – Monitor the snowpack prior to barging season to qualitatively predict the influence of run-off on the river discharge level for the upcoming barge season. If the barging season is shortened due to low river levels in the beginning of the season, secure more river barges for transport. Possible sources for additional river barges include Alaska Marine Lines from Juneau, AK or Northern Transportation Company Limited (NTCL) from Hay River, NWT. If river levels cannot support transporting all of the construction loads to site, review the cost/schedule benefit for utilizing alternate delivery methods, including helicopter and fixed wing. Prioritize barging loads to ensure that items that can only be barged are put on barges early in the season and those items that can be flown to site can be barged later in the season in case the barge season ends earlier than expected

Risk – Poor procurement performance will lead to schedule delays

Two phases of procurement activities are vital to maintaining the project schedule – the timely procurement of vendor data in year 1 to support the completion of structural designs in order to maintain the construction schedule on site, and the expediting of actual equipment orders in order to meet the detailed barging schedule. Failure to achieve procurement milestones will have a significant impact on the project schedule.

Mitigation – Establish fit for purpose procurement procedures early in the project; ensure adequate client resources to support review and approval of purchasing activities. If procurement items slip enough to miss barging windows, review the cost/benefit of alternate transportation methods to avoid schedule delays. If procurement dates cannot be met then explore the option of utilizing used equipment that could have significantly shorter lead times. Additional, if components do not meet the 2015 barge season then additional manpower could be added to the 2016 schedule to make up for the delay.

25. INTERPRETATIONS AND CONCLUSIONS

25.1 Conclusions

The financial analysis of this optimized 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.	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.
Barge Schedule	Shorter than planned barge season could impact fuel and consumable re-supply and concentrate shipping.	Monitor snow pack to predict water levels, arrange additional barge & tugs and ship concentrates in order of value. i.e. highest value first
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.
Lower Zone Mineralogy	Mineralogy of the lower zone indicates the minerals are finer grained. Additional primary grinding and rougher concentrate regrind may be required at around year 3 of operation to achieve sufficient liberation of the minerals.	Testwork is recommended to evaluate grind versus recovery.
Ore Variability	Limited information on ore variability may affect plant performance.	Testwork is recommended to better define the main ore zones.
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.	Well-developed and controlled construction and operating plans, along with experienced personnel will greatly mitigate potential cost overruns.
Development	The project development could be delayed for a	Well-developed and controlled

Schedule	number of reasons and a change in schedule would alter the project economics.	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
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.
Expansion of Reserves	The mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource and reserves.	A 6-year increase in mine life at the average FS reserve grade, would theoretically increase after-tax IRR and NPV to 23.8% and $230.8M, respectively. However, there is no basis, at the current time, for the addition of six years of mine life so the potential benefit is only to highlight the potential impact of finding more reserves, of which there is no guarantee whatsoever.
Increased Production	Production could potentially be increased by +10% with no additional capital investment and increased fixed costs.	Enhanced project economics and decreased overall unit costs. Current indications are an after-tax NPV of $155M and IRR of 23.5%.
Improved Copper Recovery	Additional metallurgical testing to improve copper splitting.	Improved copper metal recovery.
Used Equipment	Utilize used process and mining equipment.	Reduced capital costs and decreased lead times versus new equipment.
Improved Cement & Slag Prices	Cement and slag/cement pricing is virtually identical. Alternate slag sources may have decreased transportation costs.	Decreased paste backfill costs.
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 diesel power generation on site at a cost of about $ 0.326 /kWh.	If hydro power is used, especially from May to November, significant power cost savings could be realized and the project economics potentially improved. Current indications of $ 0.04/kWh for Hydro Power.

26. RECOMMENDATIONS

Based on the results of this study, the project is economic, and Chieftain should advance the project to the development stage. 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 Metallurgical Program

Test work has focused mainly on the upper zone ore. Additional test work is recommended in order to better understand the different ore zones, confirm plant design and production forecasts especially for years 1 to 3.

- Mineralogy on each zone;
- Grinding BWi on variability samples to reduce risk in unexpected throughput changes;
- Gravity concentration testing on selected variability samples with bench top machines;
- Include tests to leach gravity concentrate in an Acadia or similar circuit;
- Primary grind characterization - Grind Recovery Curves to identify opportunities for circuit optimization;
- Variability tests, bench scale, to confirm metallurgical forecast based on ore type and lenses as defined by mine schedule;
- Regrind studies on the copper rougher concentrates to improve cleaner recovery; and
- Produce a bulk concentrate to run thickening, filtration, and rheology tests to confirm equipment sizing.

The estimated cost for this test work is expected to be approximately $300K.

26.1.2 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; and
- 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.

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.

26.1.3 Environment & Permitting

On-going environmental monitoring and construction monitoring to fulfill permit requirements, including:

- Sampling of surface and groundwater;
- Sampling of HPAG;
- Sampling of PAG/NAG rock from underground development;
- Weather and stream-flow measurement, groundwater level monitoring; and
- Fish sampling in the Tulsequah river during low flow.

Environmental test work of mill design locked-cycle tailings supernatant and solids, for EMA Permit limit development will need to be undertaken.

Permit applications and EA Amendment will also have to be completed.

These costs are covered on the estimated owners costs in this report.

26.1.4 Paste Backfill

Additional UCS testing is recommended to optimize both the tailings management strategy and binder consumption (operating cost) as summarized in Kovit's backfill design report. The costs for this testing is covered in owners cost in this report.

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APPENDIX A

QP CERTIFICATES

PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

JDS Energy & Mining Inc.

CERTIFICATE OF AUTHOR

I, Gordon Doerksen do hereby certify that:

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

2. This certificate applies to the technical report titled Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date of October 20, 2014, prepared for Chieftain Metals Corp;

3. I am a Professional Mining Engineer (P.Eng. #32273) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Mining Engineer in Alaska, Wyoming and Yukon Territory. I am a Member of the Canadian Institute of Mining and Metallurgy and a Registered Member of the Society of Mining Engineers of the AIME.

   I am a graduate of Montana Tech with a B.Sc. in Mining Engineering (1990). I have been invlolved in Mining since 1985 and have practiced my profession continuously since 1990. I have held senior mine production and mine technical positions in mining operations in Canada, the US and in Africa. I have worked as a consultant for over eight years and have performed mine planning, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person, for a significant number of engineering studies and technical reports many of which were located in Latin America.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4. I visited the Tulsequah Project site on March 20-21, 2011;

5. I am responsible for Sections numbers 1,2,3,19, 21, 22, 24, 25, 26, 27;

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

7. I have had prior involvement with the property that is the subject of this Technical Report; I was QP for the "Technical Report for the Tulsequah Chief Project of Northern British Columbia, Canada", prepared for Chieftain Metals Inc. by JDS Energy & Mining Inc. with an effective date of December 12, 2012;

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

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;

Effective Date: October 20, 2014
Signing Date: November 27 2014

<u>"Original Signed and Sealed"</u>
Gordon Doerksen, P.Eng.

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date of October 20, 2014, prepared for Chieftain Metals Corp.;

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

3. I am a graduate of the University of Alberta with a BSc. In Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over seven years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining project worldwide;

5. I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have visited the Tulsequah Project site November 5-6 2012;

8. I am responsible for Sections15 and 16, excluding 16.3 and 16.9;

9. I have had prior involvement with the property that is the subject of this Technical Report; I was QP for the "Technical Report for the Tulsequah Chief Project of Northern British Columbia, Canada", prepared for Chieftain Metals Inc. by JDS Energy & Mining Inc. with an effective date of December 12, 2012;

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: October 20, 2014
Signing Date: November 27, 2014

Original Signed and Stamped
Michael E. Makarenko

CERTIFICATE OF AUTHOR

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

1. This certificate applies to the Technical Report entitled "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date of October 20, 2014, prepared for Chieftain Metals Corp.;

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

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

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

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

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have visited the Tulsequah Project site June 10 – 11, 2014;

8. I am responsible for Section 18 of the Technical Report, excluding 18.14.1(but including Potential Shortfall Effects), 18.25 and 18.26;

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

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date:October 20, 2014
Signing Date: November 27, 2014

"Signed and Sealed"

Scot Klingmann, P.Eng.

CERTIFICATE OF AUTHOR

I, Kelly Shea McLeod do hereby certify that:

This certificate applies to the Technical Report entitled Feasibility Study Technical Report with an effective date of October 20, 2014, prepared for Chieftain Metals Corp.;

1. I am currently employed as Senior Engineer, Metallugy, with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver, B.C.;
2. I am a graduate of McMaster University with a Bachelors of Engineering, Metallugy, 1984. I have practiced my profession intermittently since 1984;
3. I am a Registered Professional Mining Engineer in British Columbia;
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and engineering and mineral processing design, 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;
5. I have not visited the Project site;
6. I am responsible for Sections 1.5, 1.9, 13 (all), 17 (all), and 26.1.1;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
9. 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:October 20, 2014
Signing Date: November 27, 2014

Kelly McLeod

Kelly Shea McLeod, P.Eng.

SRK Consulting (Canada) Inc.
2100 – 25 Adelaide Street East
Toronto, Ontario, Canada
M5C 3A1

T: +1.416.601.1445
F: +1.416.601.9046

toronto@srk.com
www.srk.com

CERTIFICATE OF AUTHOR

I, Dr. Gilles Arseneau, P.Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada”, with an effective date of October 20, 2014, prepared for Chieftain Metals Corp.;
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.
4. 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;
5. I am a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia, License #25474;
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I 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;
8. I am responsible for Sections 4 to 12, 14 and 23 of the Technical report;
9. I have had prior involvement with the subject property. I am the author of three technical reports on the property dated December 12, 2013, November 2010 and September 2007;
10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-
12. 101 and Form 43-101F1.

Effective Date: October 20, 2014
Signing Date: November 27, 2014

Original Signed and Stamped
Gilles Arseneau

CERTIFICATE OF AUTHOR

I, Dave West, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date ofOctober20, 2014, prepared for Chieftain Metals Corp.;

2. I am currently employed as an Independent Consultant, with David West Consulting with an office at 106 David Street, Sudbury, Ontario P3E 1T2.

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 continuously since 1977.

4. I have worked in technical, operations and management positions at mines and projects throughout the world. I have been an independent consultant since 1987 and have performed mine design, mine planning, cost estimation, project management, technical due diligence reviews, economic assessments and report writing for mining projects;

5. I am a Registered Professional Engineer in Ontario, membership number 90523853;

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have visited the Tulsequah Chief Project site in March, 2012;

8. I am responsible for Section 16.3;

9. I have had prior involvement with the property that is the subject of this Technical Report; I was QP for the "Technical Report for the Tulsequah Chief Project of Northern British Columbia, Canada", prepared for Chieftain Metals Inc. by JDS Energy & Mining Inc. with an effective date of December 12, 2012;

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: October 20, 2014
Signing Date: November 27, 2014

Original Signed and Stamped
Dave West, P.Eng.

CERTIFICATE OF AUTHOR

I, Frank Palkovits, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report titled "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date of October 20, 2014, prepared for Chieftain Metals Corp.;

2. I am currently employed as President of Kovit Engineering Limited with an office at 311 Harrison Drive, Sudbury, Ontario, P3E 5E1;

3. I am a graduate of Laurentian University with a B.Eng. in Mining Engineering, 1988. I have practiced my profession continuously since 1988 (exclusively in paste backfill since 2000);

4. I have worked in technical, operations and management positions at mines in Canada. I have been a consultant for over fourteen years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Ontario (#90276379);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have not visited the Tulsequah Project site;

8. I am responsible for Section 16.9 of the said report;

9. I have had prior involvement with the property that is the subject of this Technical Report. I was a QP for the "Technical Report for the Tulsequah Chief Project of Northern British Columbia, Canada", prepared for Chieftain Metals Inc. by JDS Energy & Mining Inc. with an effective date of December 12, 2012;

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: October 20, 2014
Signing Date: November 27, 2014

Signed and Stamped

LICENSED PROFESSIONAL ENGINEER
F. S. PALKOVITS
PROVINCE OF ONTARIO

Frank Palkovits

Klohn Crippen Berger

CERTIFICATE OF AUTHOR

I, Harvey N. McLeod, do hereby certify that:

1. This certificate applies to the Technical Report entitled "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date of October 20, 2014, prepared for Chieftain Metals Corp..
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 18.25, and 18.26.
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: October 20, 2014
Signing Date: November 27, 2014

27 Nov. 2014

PROFESSIONAL PROVINCE OF H. N. McLEOD BRITISH COLUMBIA ENGINEER

Harvey N. McLeod, P.Eng., P.Geo.
KLOHN CRIPPEN BERGER LTD.; Principal

141127-CertificateOfAuthor-HMcLeod.docx
M09780A02.730

Ausenco

Certificates and Consents of Qualified Persons ||
Form

Nadia Krys, P.Eng.
Ausenco Engineering Canada Inc.
855 Homer Street
Vancouver, BC
V6B 2W2
Canada

Telephone: 1.604.684.9311
Facsimile: 1.604.688.5913
Email: nadia.krys@ausenco.com

I Nadia Krys, P.Eng. certify that:

I am Senior Marine Engineer at Ausenco Engineering Canada Inc., 855 Homer Street , Vancouver, BC , V6B 2W2, Canada.

This certificate applies to the Technical Report titled Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada, with an effective date of October 20, 2014.

My qualifications and relevant experiences are that:

1. I graduated with a B.Sc. from the University of Calgary in 2003.
2. I am a member of the APEGBC, license #34919
3. I have worked as a Marine Engineer for a total of 5 years. My work experience has included feasibility scale studies, project management, detailed design and construction supervision for terminal projects. Feasibility studies have included assessment for logistics, safe berthing and navigational considerations of proposed shipping and barging operations.
4. I have read the definition of Qualified Person set out in Nation Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have visited the Property for 2 days on July 1 and 2, 2014. I am responsible for the preparation of sections 18.2, 18.14.1 (except paragraph “Potential Shortfall Effects” under 18.14.1) of the above titled Technical Report.
6. I am independent of the issuer and related companies applying all of the tests per Section 1.5 of the NI 43-101.
7. I have not had prior involvement with the property(ies) that is(are) the subject of the Technical Report.
8. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
9. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated:	25 November 2014
Signature:	Original signed and sealed
Name:	Nadia Krys

Marsland
Environmental
Associates

CERTIFICATE OF AUTHOR

I, Robert Marsland, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", with an effective date of October 20, 2014, prepared for Chieftain Metals Corp.;

2. I am currently employed as a Senior Environmental Engineer with Marsland Environmental Associates Ltd. with an office at 203 West Beasley Street, Nelson, British Columbia, 1L 3K4;

3. I am a graduate of McGill University with a B.Eng. in Chemcial Engineering, 1986. I obtained a Master's degree in Environmental Engineering from the University of Alberta, in 1991. I have practiced my profession continuously since 1987;

4. I have worked as an environmental consultant in the mining industry since 1990. I have prepared Environmental Impact Assessments, Environmental Management Plans, Mine Closure Plans and provided technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Engineer in British Columbia (#25110);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have visited the Tulsequah Project site most recently July 24-30, 2014;

8. I am responsible for Section 20;

9. I have had prior involvement with the property that is the subject of this Technical Report; I was QP for the "Technical Report for the Tulsequah Chief Project of Northern British Columbia, Canada", prepared for Chieftain Metals Inc. by JDS Energy & Mining Inc. with an effective date of December 12, 2012;

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Effective Date: October 20, 2014
Signing Date: November 27, 2014

Original Signed and Stamped
Robert C. Marsland

CONSENT OF QUALIFIED PERSON

I, Michael E. Makarenko, P. Eng., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").

I also consent to any extracts from or a summary of the Technical Report in Chieftain's Press Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

Michael Makarenko, P. Eng.

Vancouver Office
T 604.687.7545 f 604.689.5041
#860 – 625 Howe Street Vancouver, BC V6C 2T6

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

CONSENT OF QUALIFIED PERSON

I, Gordon E. Doerksen, P.Eng., consent to the public filing by Chieftain Metals Corp. (“Chieftain”) of the technical report titled, “Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada”, dated effective October 20, 2014 and executed on November 27, 2014 (the “Technical Report”).

I also consent to any extracts from or a summary of the Technical Report in Chieftain’s Press Release (“PR”) dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

Gordon Doerksen, P.Eng.

Vancouver Office
T 604.687.7545 f 604.689.5041
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PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

JDS Energy & Mining Inc.

CONSENT OF QUALIFIED PERSON

I, Scot G. Klingmann, P.Eng., consent to the public filing by Chieftain Metals Corp. (“Chieftain”) of the technical report titled, “Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada”, dated effective October 20, 2014 and executed on November 27, 2014 (the “Technical Report”).

I also consent to any extracts from or a summary of the Technical Report in Chieftain’s Press Release (“PR”) dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

Scot Klingmann, P.Eng.

Vancouver Office
T 604.687.7545 f 604.689.5041
#860 – 625 Howe Street Vancouver, BC V6C 2T6

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

CONSENT OF QUALIFIED PERSON

I, Kelly S McLeod, P.Eng., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").

I also consent to any extracts from or a summary of the Technical Report in Chieftain's Press Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

Kelly McLeod

Kelly S McLeod, P.Eng.

PROFESSIONAL
PROVINCE OF
K. S. McLEOD
15868
BRITISH COLUMBIA
ENGINEER

SRK Consulting (Canada) Inc.
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toronto@srk.com
www.srk.com

Vancouver, B.C., November 27, 2014

To: Securities Regulatory Authorities
Toronto Stock Exchange (TSX)

CONSENT OF QUALIFIED PERSON

I, Dr Gilles Arseneau, P.Geo., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").

I also consent to any extracts from or a summary of the Technical Report in the Chieftain Metal Corp. Press Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

G. Arseneau

Gilles Arseneau, P.Geo.

Marsland
Environmental
Associates

CONSENT OF QUALIFIED PERSON

I, Robert Charles Marsland, P.Eng., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").
I also consent to any extracts from or a summary of the Technical Report in Chieftain's Press Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

Rob Marsland, P.Eng.

Klohn Crippen Berger

CONSENT OF QUALIFIED PERSON

I, Harvey N. McLeod, P.Eng., P.Geo., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").

I also consent to any extracts from or a summary of the Technical Report in Chieftain's Press Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

H. N. McLEOD

27 Nov. 2014

Harvey N. McLeod, P.Eng., P.Geo.
KLOHN CRIPPEN BERGER LTD.; Principal

141127-Consent Tulsequah FS-HMcLeod.docx
M09780A02.730

CONSENT OF QUALIFIED PERSON

I, Dave West, P.Eng., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").

I also consent to any extracts from or a summary of the Technical Report in Chieftain'sPress Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 27th day of November, 2014

Dave West, P.Eng.

KOVIT Engineering

CONSENT OF QUALIFIED PERSON

I, Frank Palkovits, P.Eng., consent to the public filing by Chieftain Metals Corp. ("Chieftain") of the technical report titled, "Feasibility Study Technical Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated effective October 20, 2014 and executed on November 27, 2014 (the "Technical Report").

I also consent to any extracts from or a summary of the Technical Report in the Chieftan's Press Release ("PR") dated October 20, 2014.

I confirm that I have read the PR and that they fairly and accurately represent the information in the Technical Report for which I am responsible.

Dated this 1st day of December, 2014

Frank Palkovits, P.Eng.

311 Harrison Drive, Sudbury, ON, Canada
Tel: 705-523-1040 Fax: 705-523-1041
www.koviteng.com

Ausenco

TO: British Columbia Securities Commission

RE: Chieftain Metals Corp.

Consent of: Nadia Krys, P.Eng. Senior Marine Engineer

I, Nadia Krys Senior Marine Engineer

a) do consent to the public filing, for regulatory purposes, of the Technical Report titled "*Feasibility Study Technical* Report, Tulsequah Chief Project, Northern British Columbia, Canada", dated with an effective date of October 20, 2014, (the "Technical Report") prepared for **Chieftain Metals Corp.** and to extracts from, or summaries of, the Technical Report in the press releases of **Chieftain Metals Corp.** with an effective date of October 20, 2014 (the "Press Releases")

b) confirm that I have read the Press Releases being filed and that they fairly and accurately represent the information contained in my contribution to the Technical Report that supports the disclosure.

Dated: 25 November 2014

Signature of Qualified Person

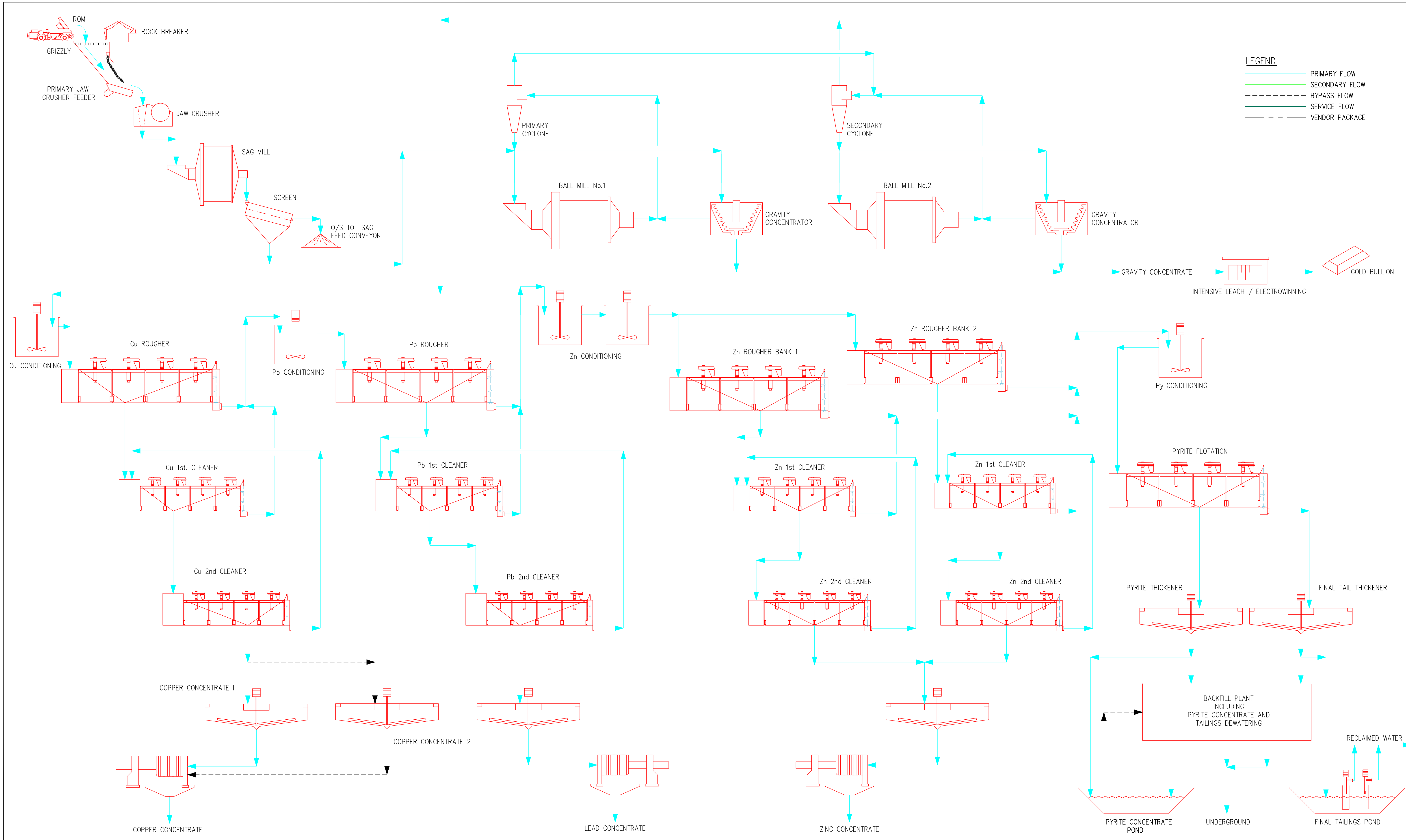
Name of Qualified Person: Nadia Krys

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA N. B. KRYS # 34919

11/25/14

APPENDIX B

METALLURGY



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

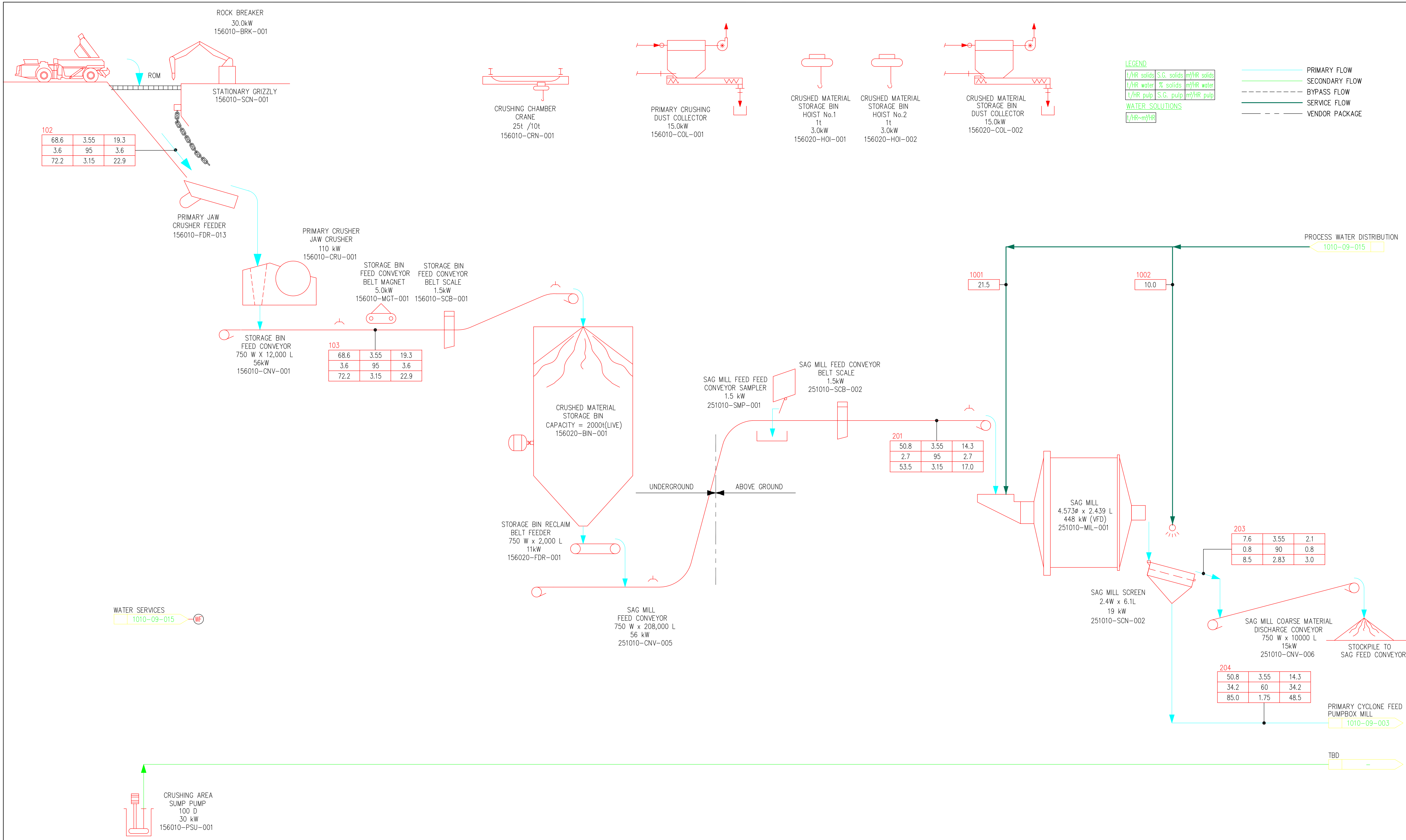
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
SIMPLIFIED FLOW DIAGRAM
PROCESS FLOW DIAGRAM

PROJECT No.	DRAWING No.	REV No.
	1010-09-001	E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

TULSEQUAH CHIEF PROJECT

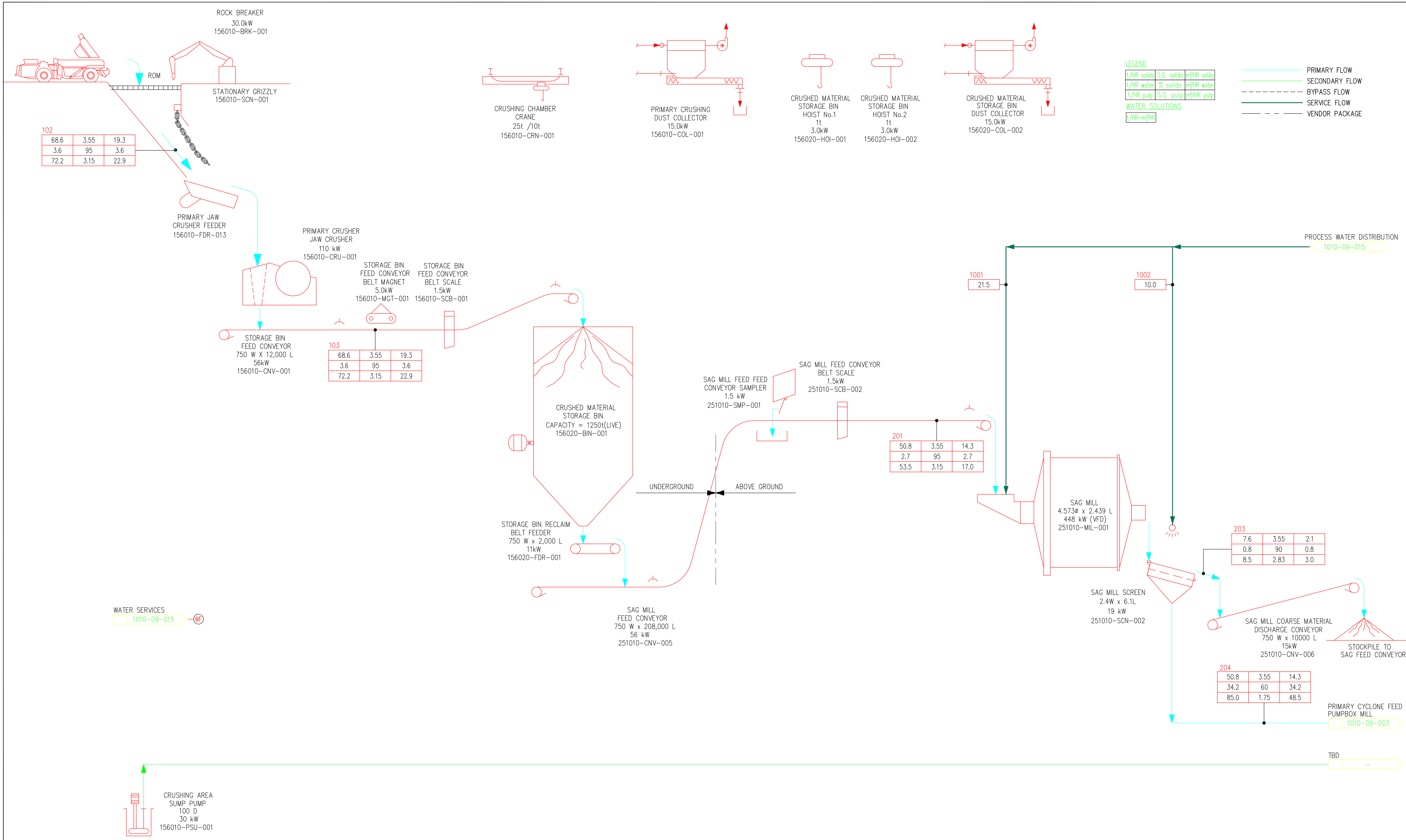
		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
PRIMARY CRUSHING AND SAG PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-002 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

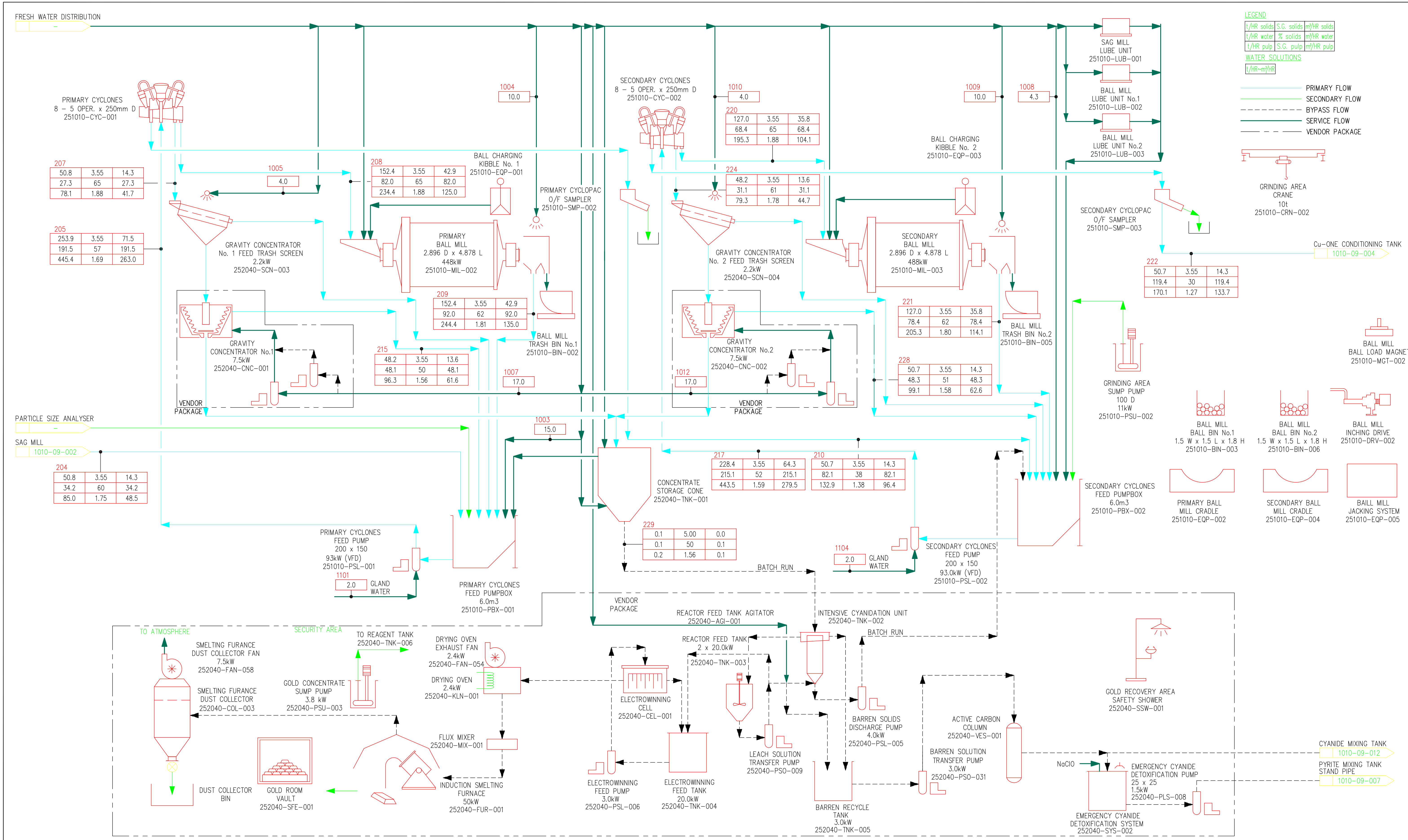
JDS Energy & Mining Inc.

TITLE
PRIMARY CRUSHING AND SAG
PROCESS FLOW DIAGRAM

PROJECT No.

DRAWING No. 1010-09-002

REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

SCALE:	None	DATE:
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

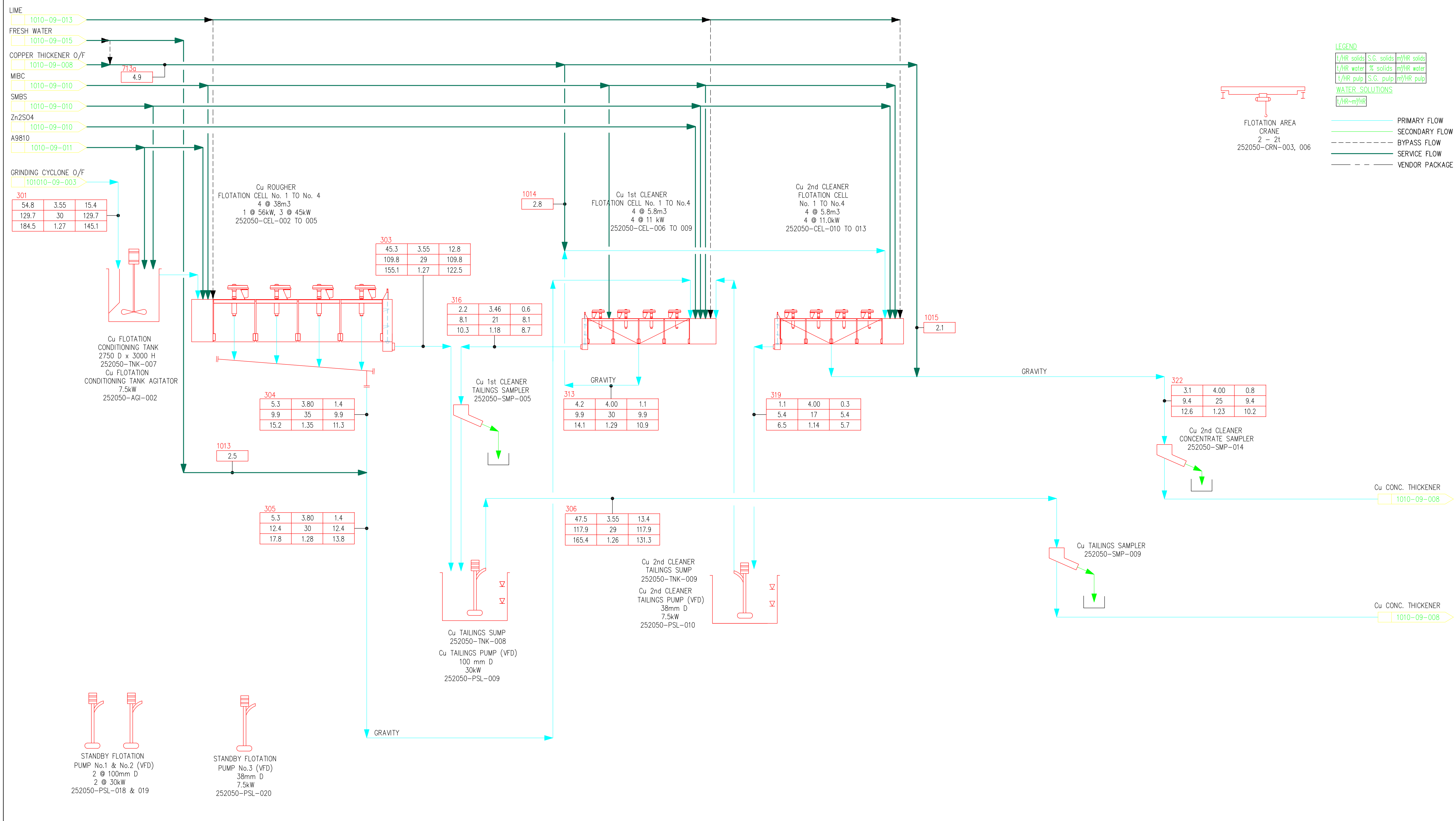
JDS Energy & Mining Inc.

TITLE
GRINDING CIRCUIT
PROCESS FLOW DIAGRAM

PROJECT No.

DRAWING No. 1010–09–003

REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B.Wong	June 22 2014
CHECKED:		
APPROVED:		

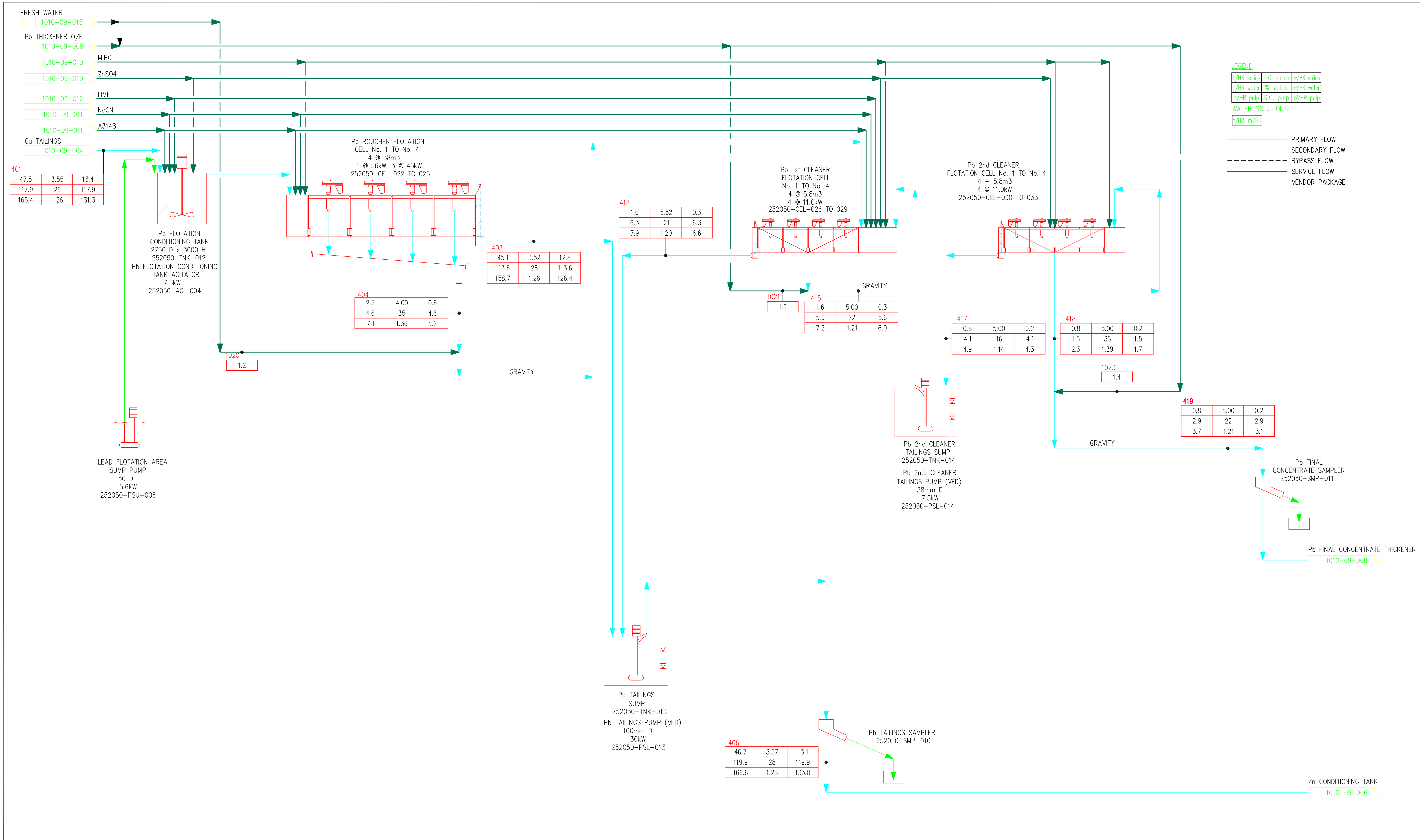
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
Cu - FLOTATION
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-004 | REV. No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

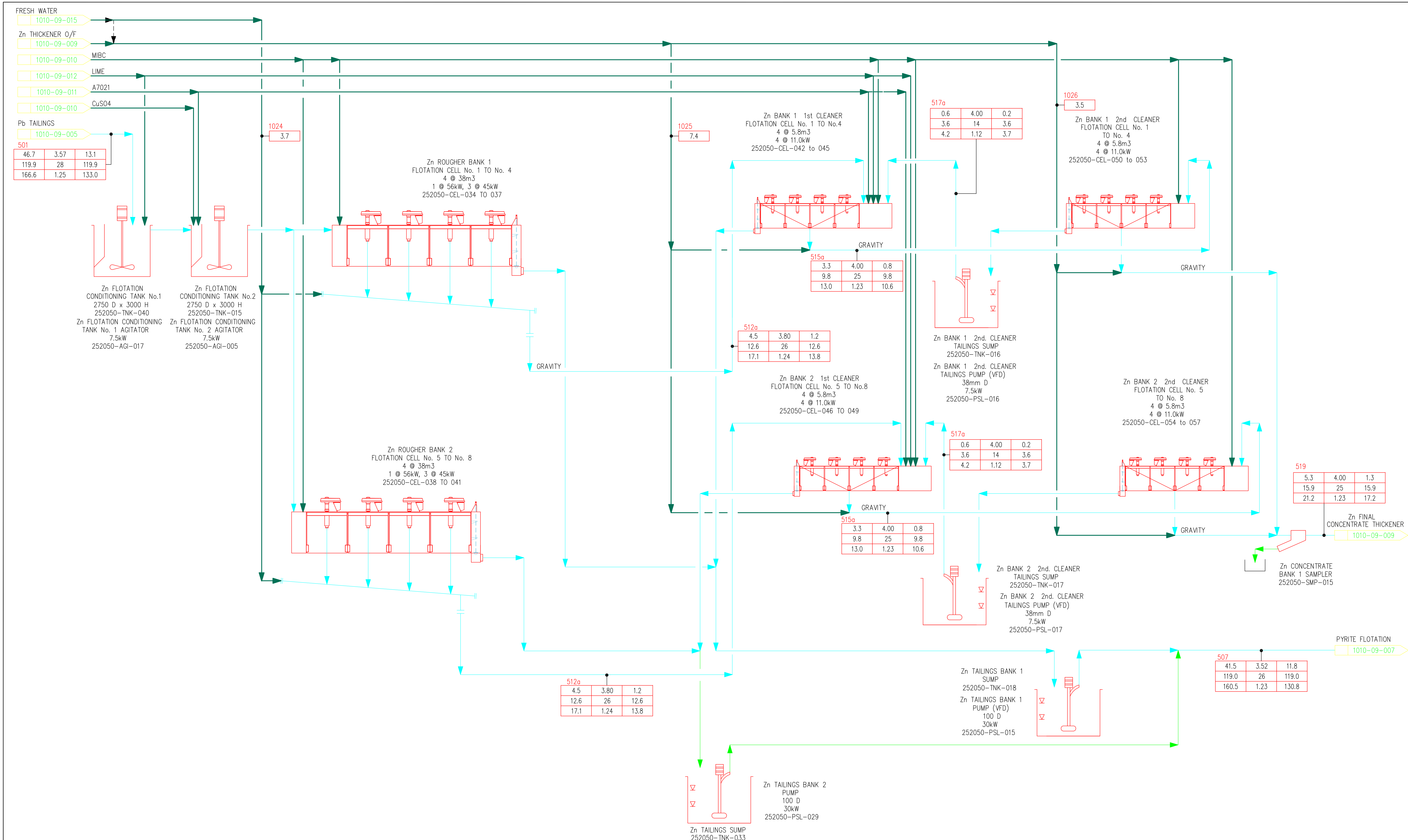
		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
Pb-FLOTATION
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-005 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

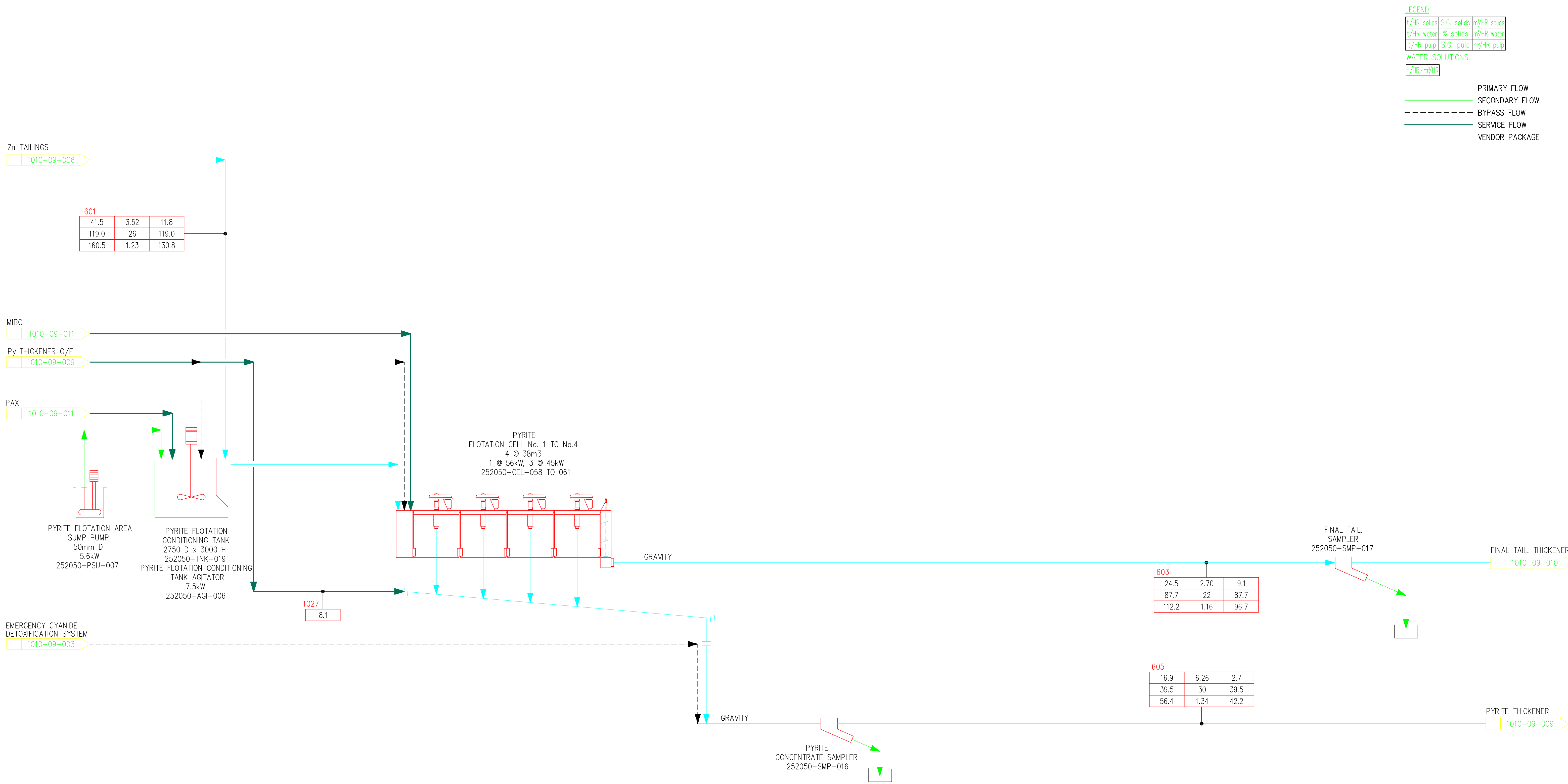
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
Zn – FLOTATION
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-006 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

SCALE: None		DATE:
DESIGNED: J.D.S.		June 22 2014
DRAWN: B. Wong		June 22 2014
CHECKED:		
APPROVED:		

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

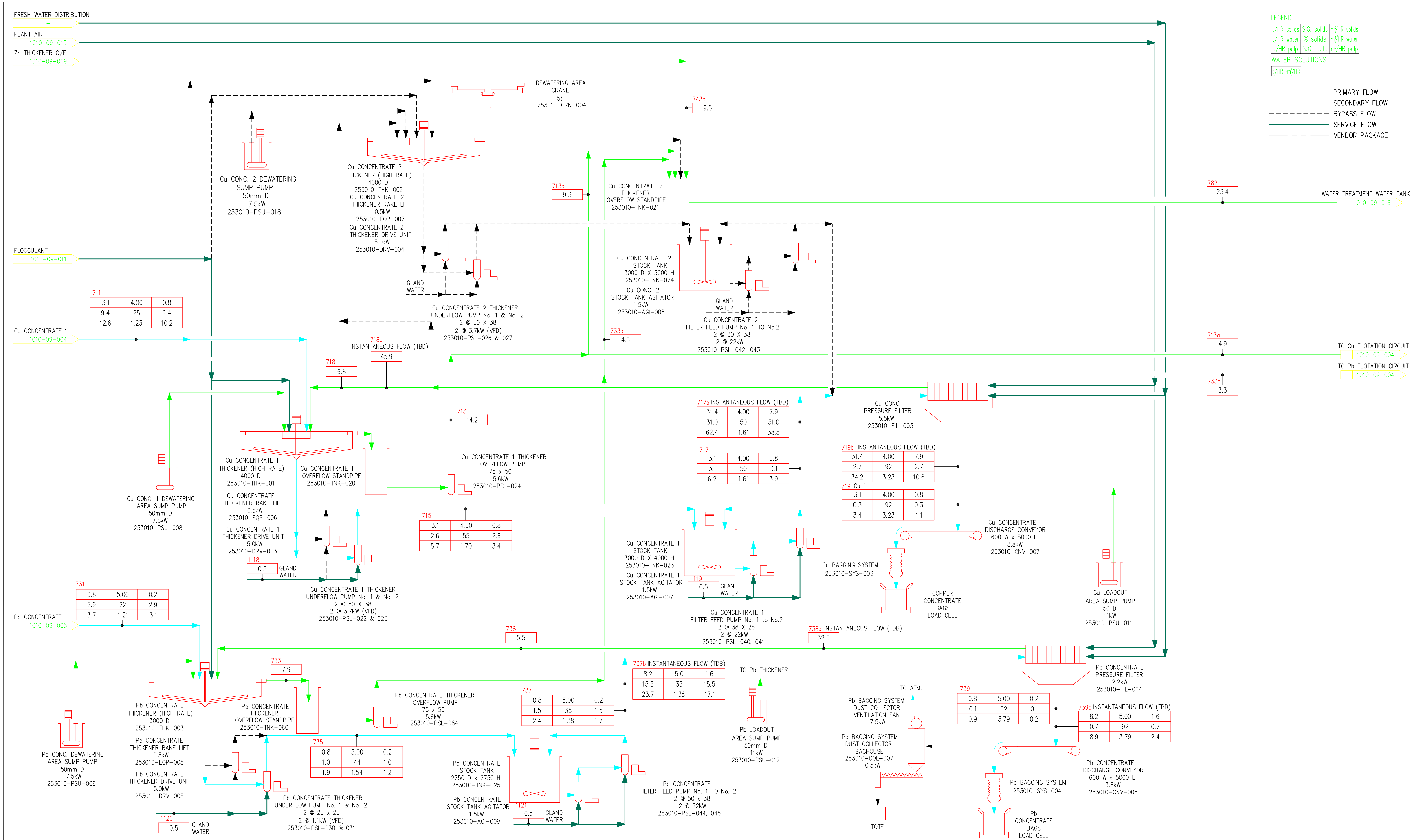
JDS Energy & Mining Inc.

TITLE
PYRITE FLOTATION
PROCESS FLOW DIAGRAM

PROJECT No.

DRAWING No. 1010-09-007

REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

TULSEQUAH CHIEF PROJECT

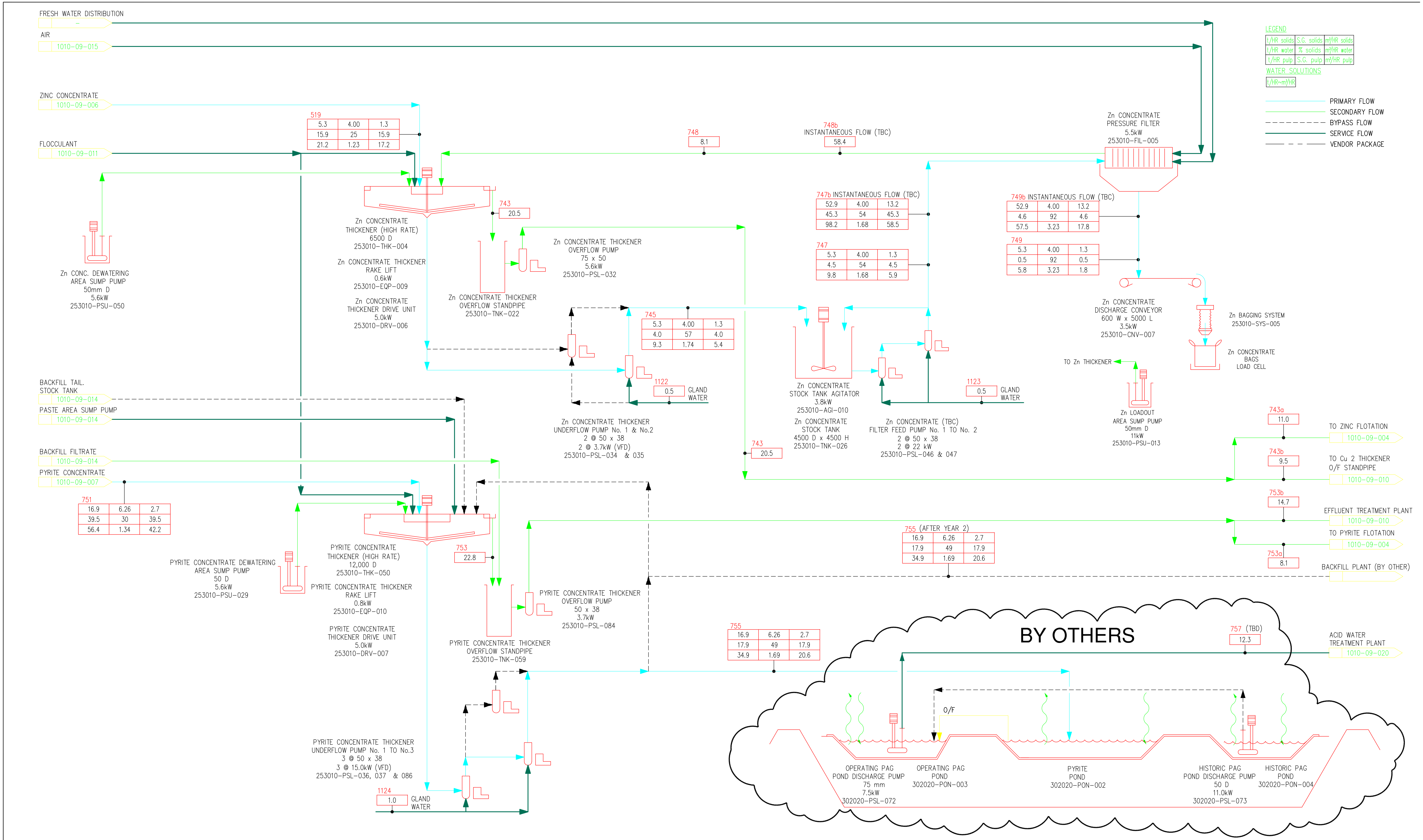
SCALE: None | DATE:
DESIGNED: J.D.S. | June 22 2014
DRAWN: B. Wong | June 22 2014
CHECKED:
APPROVED:

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
Cu CONC 1 & 2 – Pb CONCENTRATE DEWATERING PROCESS WATER DIAGRAM

PROJECT No. | DRAWING No. 1010-09-008 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: None | DATE:
DESIGNED: J.D.S. | June 22 2014
DRAWN: B. Wong | June 22 2014
CHECKED:
APPROVED:

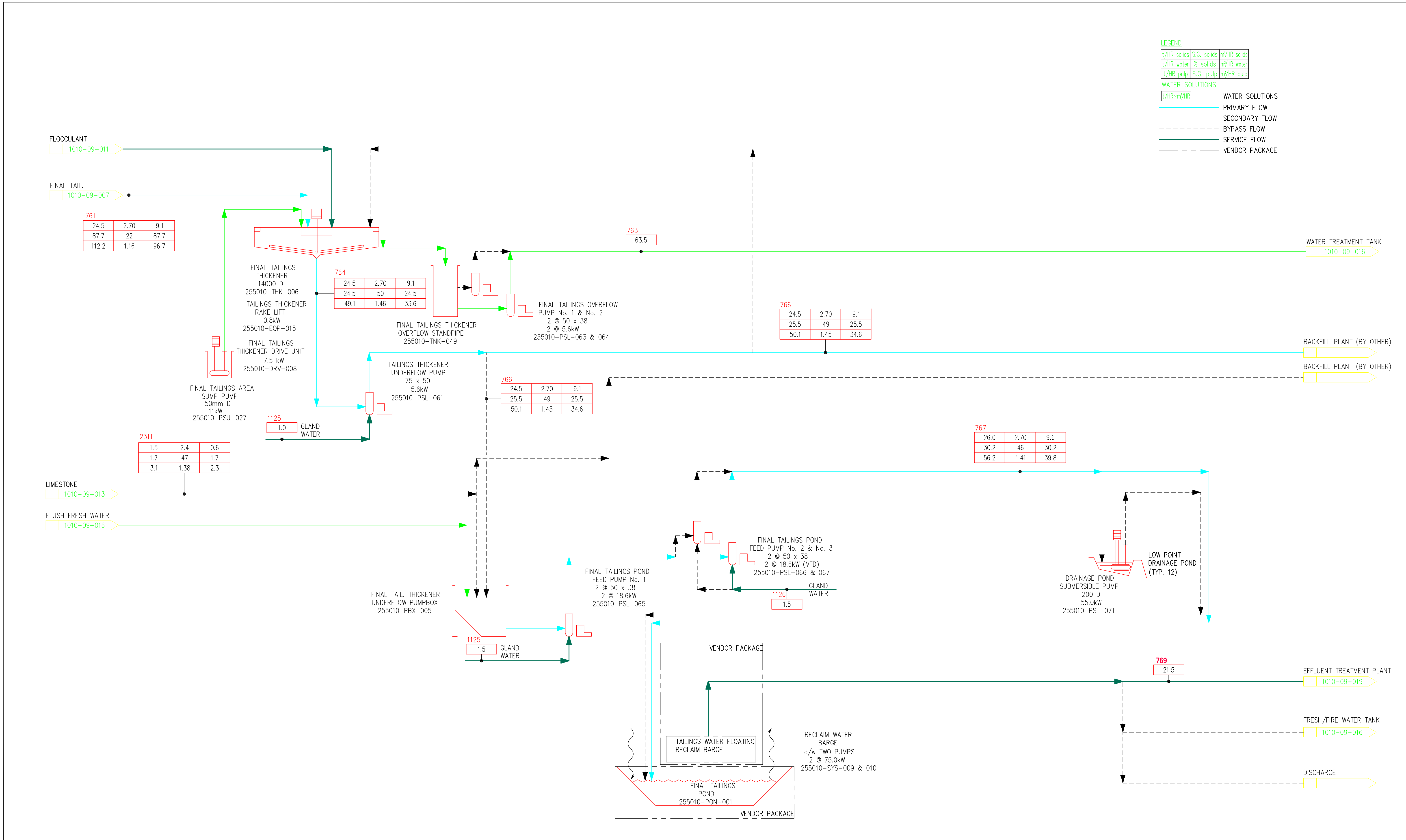
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
Zn – PYRITE CONCENTRATE DEWATERING PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-009 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: None		DATE:
DESIGNED: J.D.S.		June 22 2014
DRAWN: B. Wong		June 22 2014
CHECKED:		
APPROVED:		

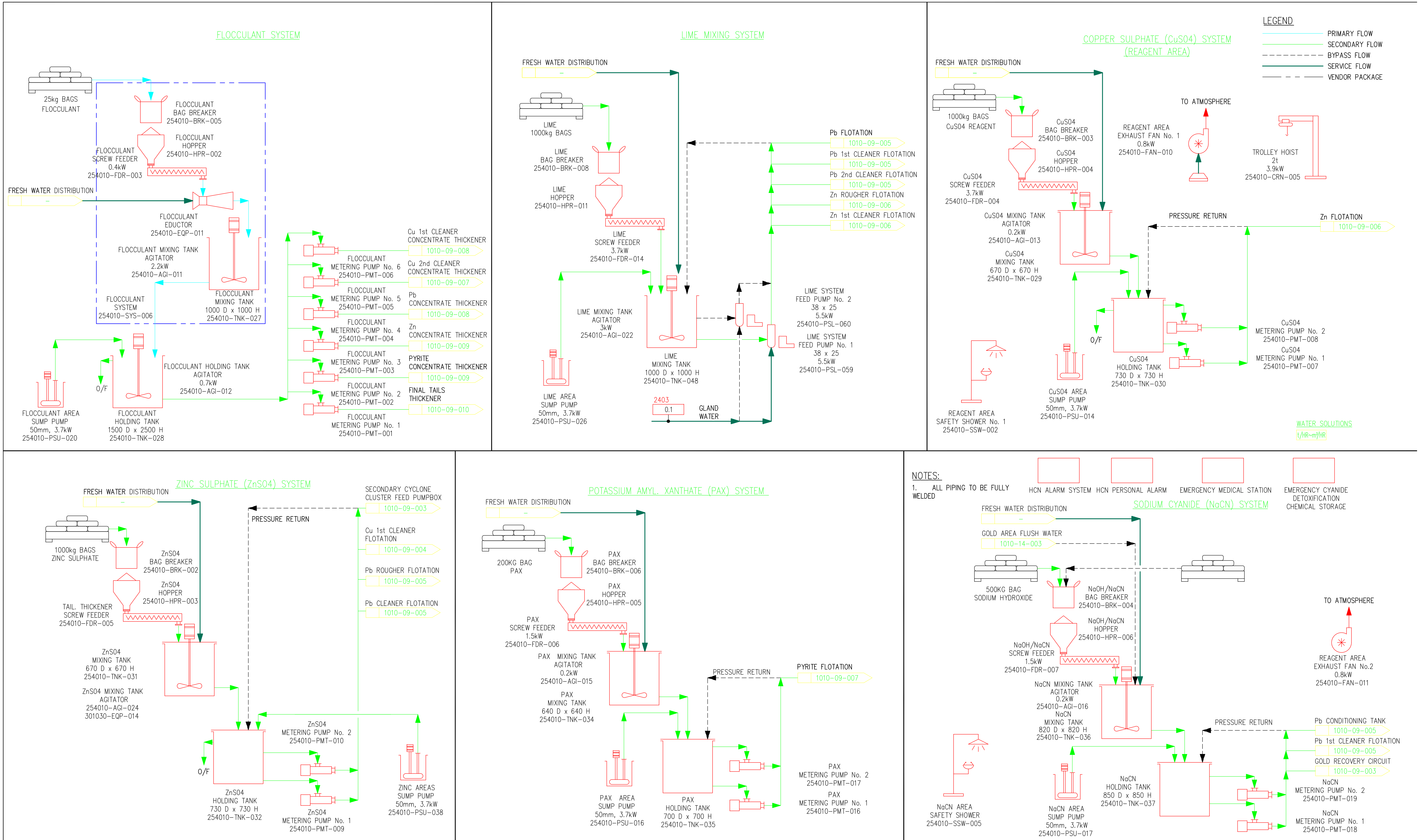
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
TAILINGS HANDLING PROCESS
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-010 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

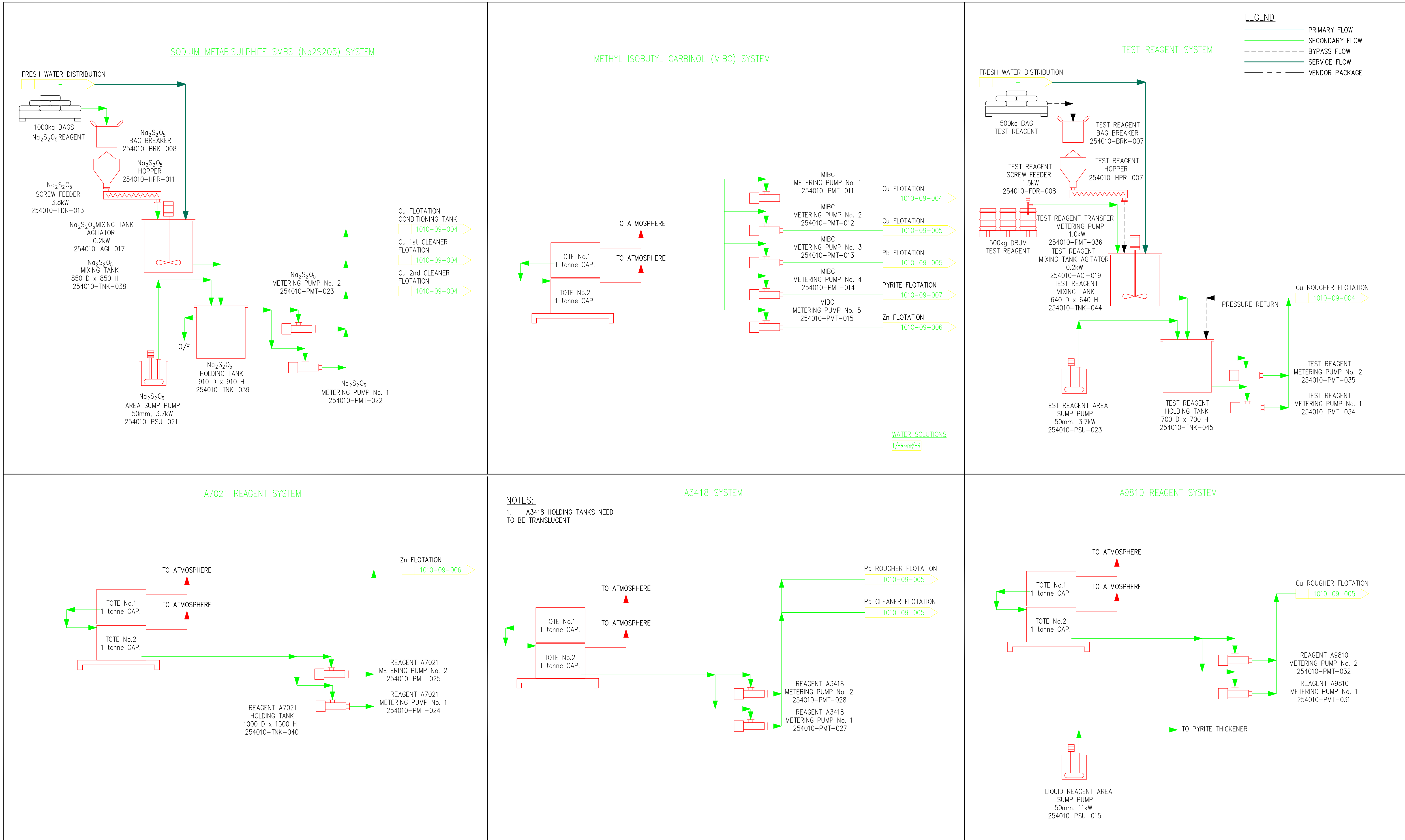
SCALE: None
DESIGNED: J.D.S. DATE: June 22 2014
DRAWN: B. Wong June 22 2014
CHECKED:
APPROVED:

Chieftain Metals Inc.
JDS Energy & Mining Inc.

TULSEQUAH CHIEF PROJECT

TITLE
REAGENTS PROCESS #1
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-011 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

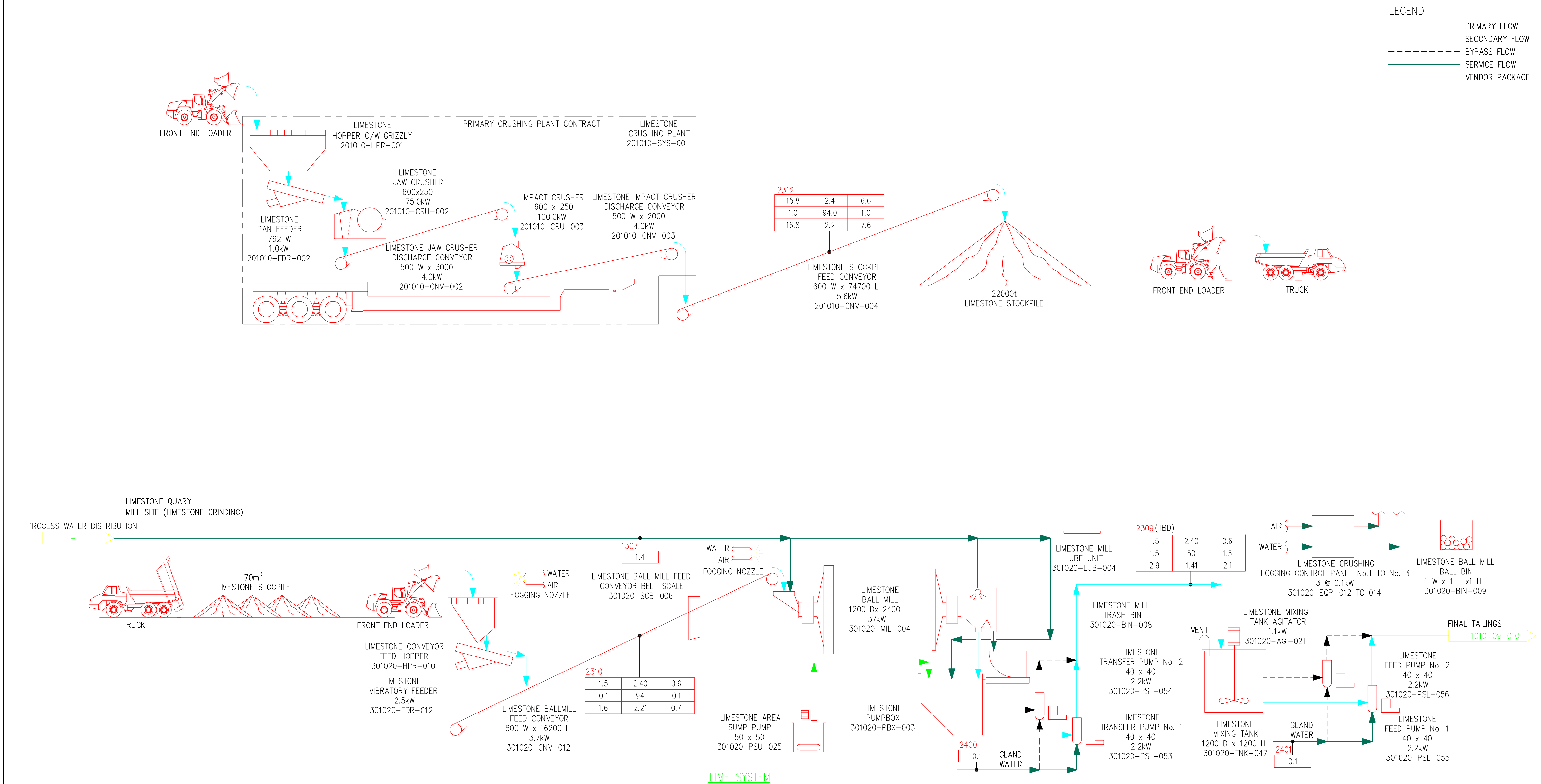
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
REAGENTS PROCESS #2
PROCESS FLOW DIAGRAM

PROJECT No.
DRAWING No. 1010-09-012
REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: None | DATE:
DESIGNED: J.D.S. | June 22 2014
DRAWN: B. Wong | June 22 2014
CHECKED:
APPROVED:

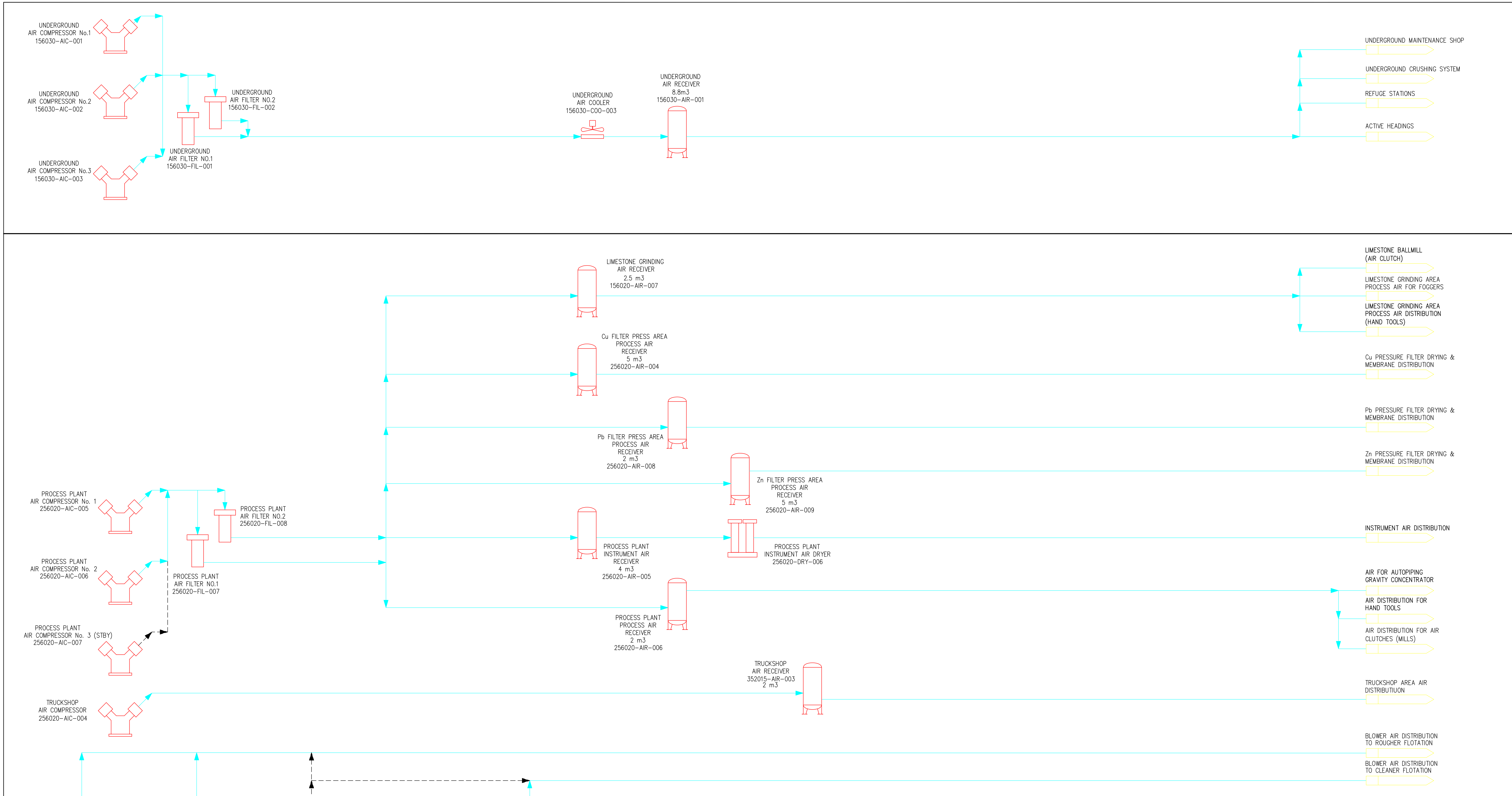
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
REAGENTS PROCESS #3
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010–09–013 | REV No. E2



NOTES:

1. UNDERGROUND AIR SYSTEM IS NOT SHOWN IN THIS DRAWING.
2. PRIMARY FILTERS SHOWN PROVIDE OIL AND PARTICLE REMOVAL.
3. SECONDARY FILTERS POLISH FILTRATION FOR INSTRUMENTS.
4. NOTE FLOWRATES SHOWN ARE PROCESS REQUIREMENTS AND DO NOT INDICATE SIZE OF COMPRESSOR FOR SPARES COMPATIBILITY.
5. ALL VOLUMES ARE IN SCFM.

LEGEND

DESIGN SCFM
NOMINAL SCFM

PRIMARY FLOW
SECONDARY FLOW
BYPASS FLOW
SERVICE FLOW
VENDOR PACKAGE

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: None | DATE:
DESIGNED: J.D.S. | June 22 2014
DRAWN: B. Wong | June 22 2014
CHECKED:
APPROVED:

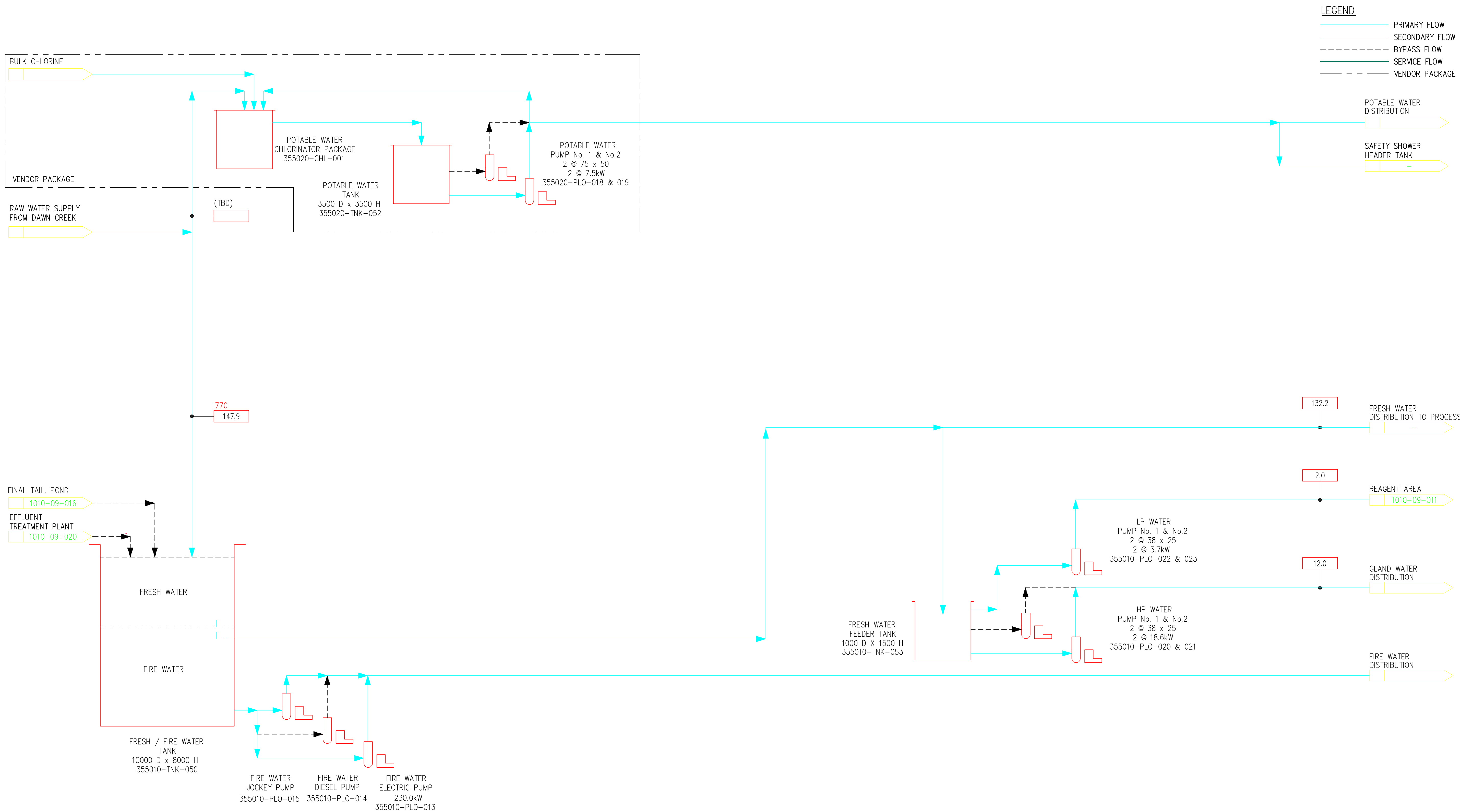
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
AIR SERVICES
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-015 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

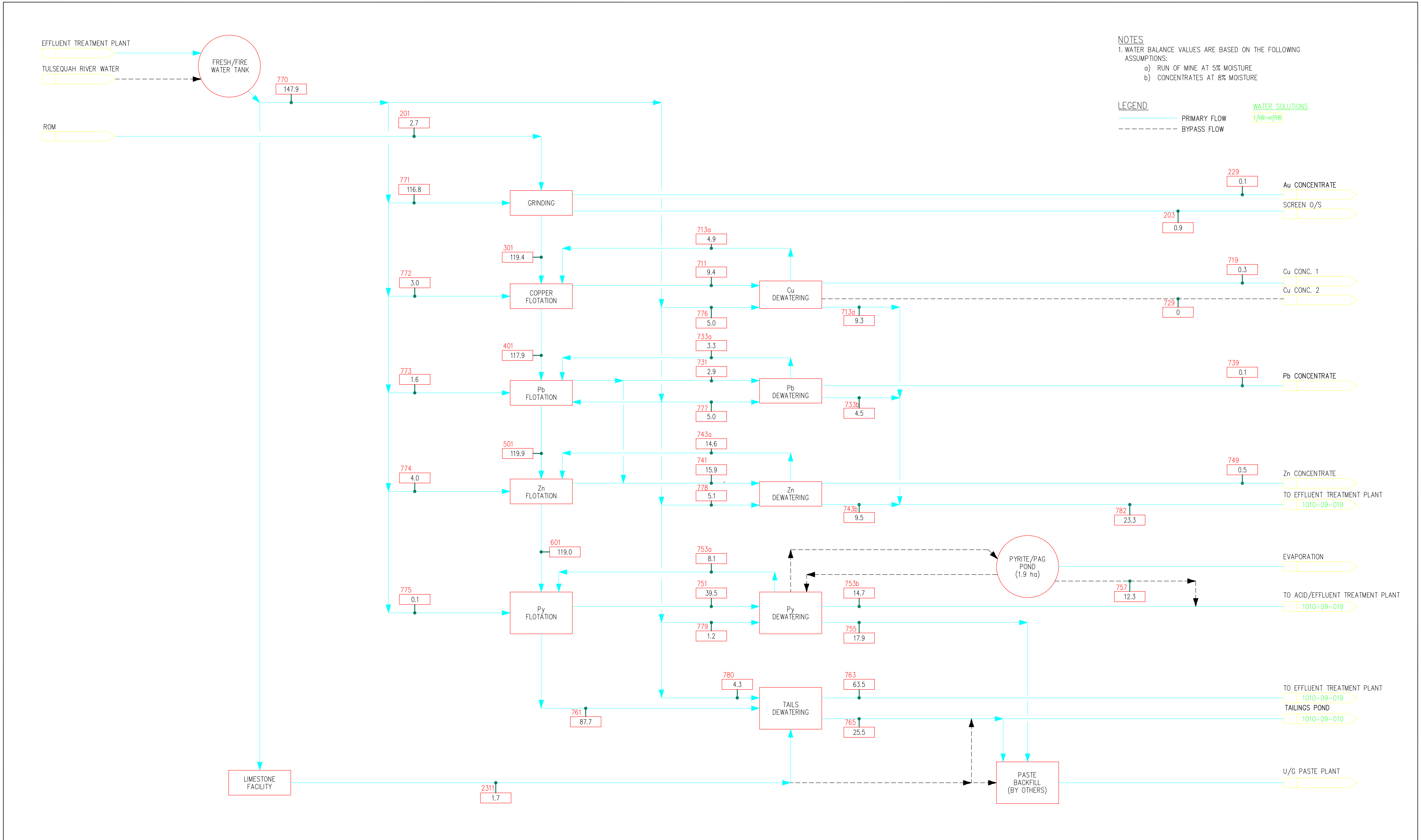
		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
WATER SERVICES
PROCESS FLOW DIAGRAM

PROJECT No. | DRAWING No. 1010-09-016 | REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

SCALE: None
DATE:
DESIGNED: J.D.S. June 22 2014
DRAWN: B. Wong June 22 2014
CHECKED:
APPROVED:

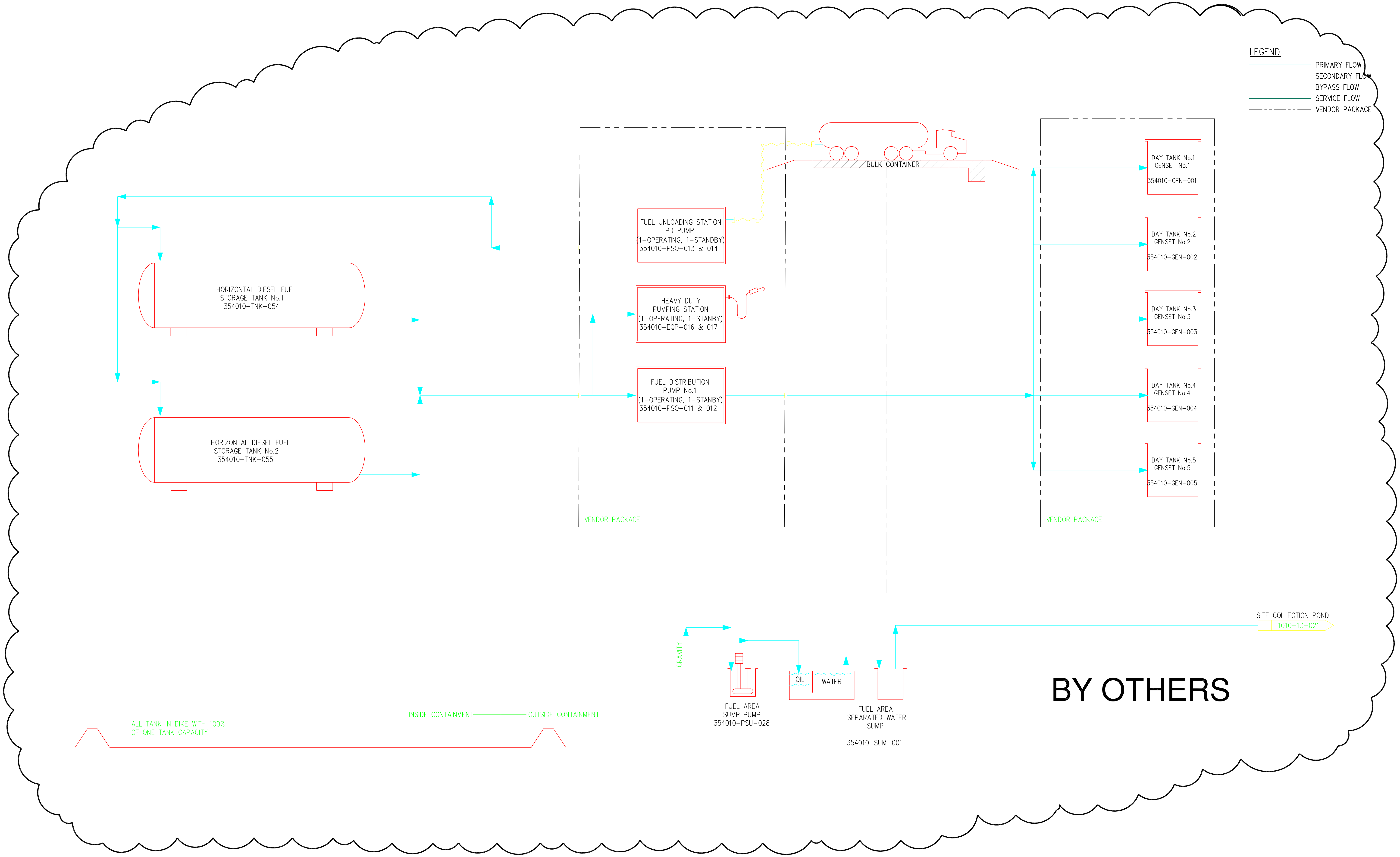
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
OVERALL SITE WATER BALANCE
PROCESS WATER DIAGRAM

PROJECT No.
DRAWING No. 1010-09-017
REV No. E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING	REF. DWG No.

		DATE:
SCALE:	None	
DESIGNED:	J.D.S.	June 22 2014
DRAWN:	B. Wong	June 22 2014
CHECKED:		
APPROVED:		

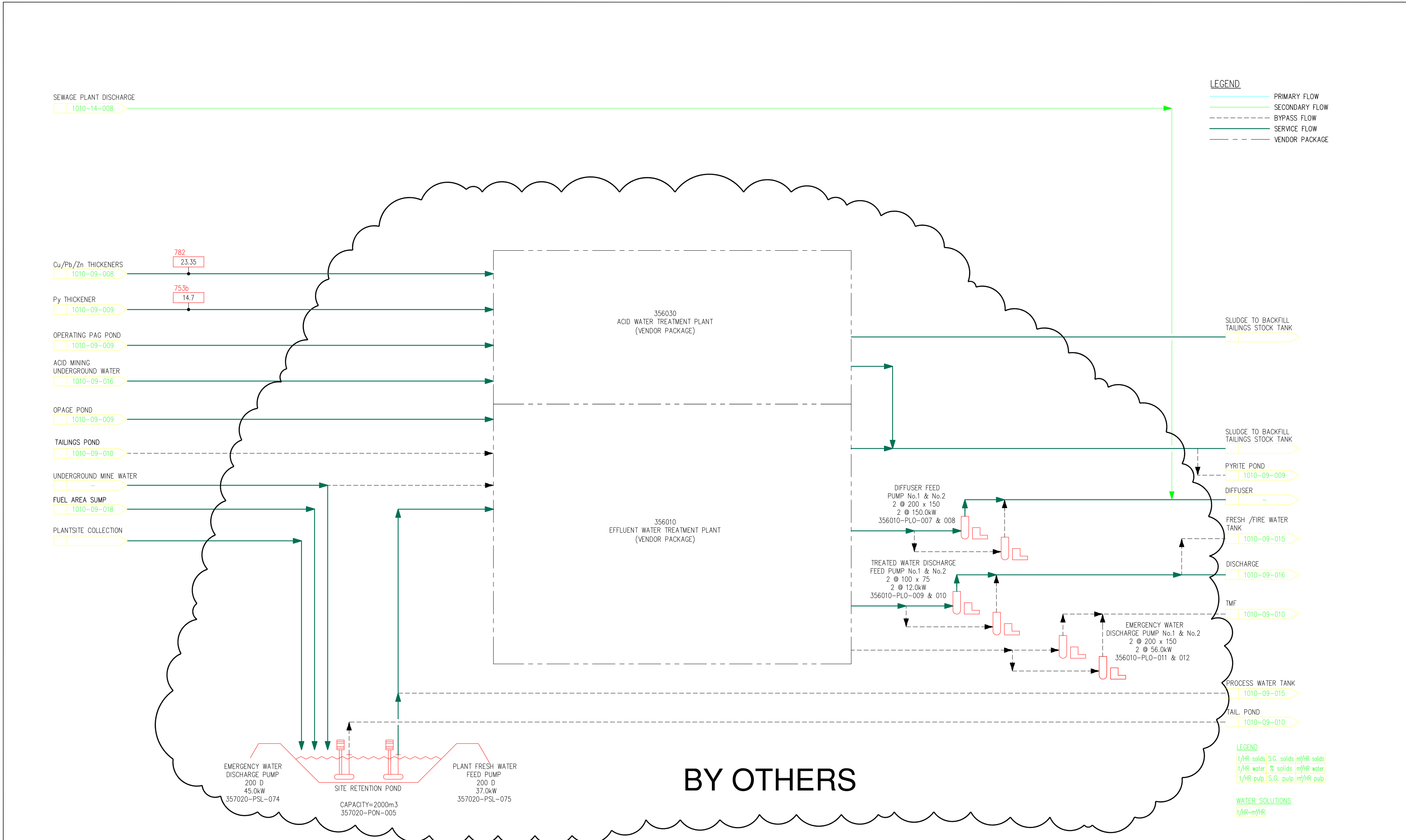
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
FUEL STORAGE SYSTEM
PROCESS WATER DIAGRAM

PROJECT No.	DRAWING No.	REV No.
	1010-09-018	E2



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
E2	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		03/11/14
E1	APPROVAL	ISSUED FOR FEASIBILITY STUDY	B.W.	K.M.		28/08/14
D1	REVIEW	DRAFT ISSUED FOR REVIEW	B.W.	K.M.		27/06/14

REFERENCE DRAWING | REF. DWG No.

SCALE: None | DATE:
DESIGNED: J.D.S. | June 22 2014
DRAWN: B. Wong | June 22 2014
CHECKED:
APPROVED:

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
WATER TREATMENT PLANT
PROCESS WATER DIAGRAM

PROJECT No. | DRAWING No. 1010-09-019 | REV No. E2

PROCESS DESIGN CRITERIA

CLIENT: Chieftain Metals Inc
PROJECT: Tulsequah 1,250 tpd Feasibility Study

DATE: October 8, 2014

PROJECT NUMBER:	CMITUL-05
DOCUMENT NUMBER:	CMITUL-05-09-001
Revision:	E2

Reference codes:

Assumed Indicative	A	Mass Balance	M
Client Data	C	Information by others	O
Design Basis	D	Published Data	P
Engineering Calculation	E	Test Work	T
Recommended	J	Vendor Originated Data	V
Industry Standard Practice	I		

BASE CASE PARAMETERS

Operating Data	Units	Nominal	Design	Ref. Code	Source	Comments
Daily Ore Throughput	t/d	1,097		J	Mine Plan Oct. 14, 2014	
Total Annual Ore Throughput	t/a	**400,405**		E		
Total Ore Throughput	t	**4,435,619**		J	Mine Plan Oct. 14, 2014	
LOM	y	11.1		E		
Crushing Throughput	t/h	69	82	E		
Grinding Throughput	t/h	51	61	E		
Design Factors						
Design factor	-	1.2		A		
Ore Characteristics						
General						
Ore Solids Density	SG	3.55		P	JDS Mine Plan Oct 14, 2014	
Ore Moisture	% w/w	5.00		P	JDS - Tech. Report 2,000 tpd FS	Doc. No. 1195650100-DBM-P0001-00
Crushed Ore Bulk Density	t/m^3	1.85		P	JDS - Tech. Report 2,000 tpd FS	PDC
Concentrate Bulk Density	t/m^3	2.30		I		
Rod Mill Work Index, Wi	kWh/t	8.8		T	Hazen Sept., 2014 Project 11936-01	
Bond Ball Mill Work Index, Wi	kWh/t	12.9		P	ALS A13775 : BRL Comminution August 2011	
Bond Abrasion Index, Ai	g	0.0743		P	A13902 - Burnie Comminution Testwork T12216	
Head Grade (Average LOM)	%Cu	1.46		P	JDS Mine Plan Oct 14, 2014	
Head Grade (Average LOM)	%Pb	1.29		P	JDS Mine Plan Oct 14, 2014	
Head Grade (Average LOM)	%Zn	6.95		P	JDS Mine Plan Oct 14, 2014	
Head Grade (Average LOM)	g/t Au	2.85		P	JDS Mine Plan Oct 14, 2014	
Head Grade (Average LOM)	g/t Ag	104		P	JDS Mine Plan Oct 14, 2014	
Head Grade (Average LOM)	% Pyrite	29		E,T	ALS Project T0897 Test No. T32	
Production Rates						
Overall Plant Availability	%	90		C	Ken Sangster June 16, 2014	
Operating Days per Year	days	365		P	JDS - Tech. Report 2,000 tpd FS	
Crushing Plant Operating Hours per Day	hours	16		C	Ken Sangster June 16, 2014	
Shifts per Day	shift/day	2		C	Ken Sangster June 16, 2014	
Process Plant Operating Hours per Day	hours	24		P	JDS - Tech. Report 2,000 tpd FS	
Shifts per Day	shift/day	2		P	JDS - Tech. Report 2,000 tpd FS	
Gold - Gravity						
Gold Concentrate Density	SG	5.00		M,P	JDS - Tech. Report 2,000 tpd FS	PFS
Gravity Gold Mass Pull	%	0.22		E		
Gravity Hourly Production	t/h	0.114		E		
Gold Grade	g/t	526		E		
Gold Recovery	g/t	1.17		E,V	FLSmidth Knelson Simulation July 2014	
	% Au	41		V,P	FLSmidth Knelson Simulation July 2014	
Silver Recovery	% Ag	0.5		P	JDS - Tech. Report 2,000 tpd FS	
Copper						
Copper Concentrate Density		4.60		C	Ken Sangster June 16, 2014	
Copper Concentrate Mass Pull	%	6.2		E		
Copper Concentrate Production, hourly	dry tph	3.14	3.77	E		
Copper Concentrate Production, daily	dry tpd	68	81.45	E		
Copper Concentrate Production, annually	dry tpa	24,776	29,731	E		
Copper Concentrate Grade	% Cu	21		P	JDS Projected Recoveries October 16, 2014	
Copper Recovery	% Cu	89		P	JDS Projected Recoveries October 16, 2014	
Gold Recovery	% Au	47		P	JDS Projected Recoveries October 16, 2014	
Silver Recovery	% Ag	78		P	JDS Projected Recoveries October 16, 2014	
Lead						
Lead Concentrate Density	SG	5.00		M,P	JDS - Tech. Report 2,000 tpd FS	PFS
Lead Concentrate Mass Pull	%	1.4		E		
Lead Concentrate Production, hourly	dry tph	0.71	0.85	E		
Lead Concentrate Production, daily	dry tpd	15	18	E		
Lead Concentrate Production, annually	dry tpa	5,596	6,715	E		
Lead Concentrate Grade	% Pb	60		P	JDS Projected Recoveries October 16, 2014	
Lead Recovery	% Pb	65		P	JDS Projected Recoveries October 16, 2014	
Gold Recovery	% Au	3		P	JDS Projected Recoveries October 16, 2014	
Silver Recovery	% Ag	6		P	JDS Projected Recoveries October 16, 2014	
Zinc						
Zinc Concentrate Density	SG	4.00		M,P	JDS - Tech. Report 2,000 tpd FS	PFS
Zinc Concentrate Mass Pull	%	10.4		E		
Zinc Concentrate Production, hourly	dry tph	5.29	6.35	E		
Zinc Concentrate Production, daily	dry tpd	114	137	E		
Zinc Concentrate Production, annually	dry tpa	41,742	50,091	E		
Zinc Concentrate Grade	% Zn	60		P	JDS Projected Recoveries October 16, 2014	
Zinc Recovery	% Zn	90		C	JDS Projected Recoveries October 16, 2014	
Total Concentrate Production						
Total Concentrate Production, hourly	dry tph	9.15	10.98	E		
Total Concentrate Production, daily	dry tpd	198	237	E		
Total Concentrate Production, annually	dry tpa	72,113	86,536	E		
Pyrite						
Pyrite Concentrate Density	SG	6.26		M		
Pyrite Concentrate Mass Pull	%	33		E		
Pyrite Concentrate Production, hourly	dry tph	16.93	20.31	E		
Pyrite Concentrate Production, daily	dry tpd	366	439	E		
Pyrite Concentrate Production, annually	dry tpa	133,448	160,138	E		
Pyrite Concentrate Grade	% Py	67		E,T	ALS Project T0897 Test No. T32	
Pyrite Recovery	% Py	77		E,T	ALS Project T0897 Test No. T32	
Tailings						
Tailings Density	SG	2.70		P	JDS Mine Plan Sept., 2014	
Tailings Production, hourly	dry tph	24.53	29.43	M		
Tailings Production, daily	dry tpd	530	636	M		
Tailings Production, annually	dry tpa	193,375	232,050	M		

PROCESS DESIGN CRITERIA

CLIENT: Chieftain Metals Inc PROJECT: Tulsequah 1,250 tpd Feasibility Study		PROJECT NUMBER:	CMITUL-05
		DOCUMENT NUMBER:	CMITUL-05-09-001
DATE: October 8, 2014		Revision:	E2

Crushing and Coarse Ore Stockpile

General						
Crusher Utilization	%	67		C	Ken Sangster June 16, 2014	
Crushing Throughput	t/h	69	82	E		
Angle of Repose	degrees	35-40		P	JDS - Tech. Report 2,000 tpd FS	PDC
Angle of Withdrawal	degrees	55-60		P	JDS - Tech. Report 2,000 tpd FS	PDC
Coarse Ore Bin/Feed System						
Maximum Feed Lump Size, Grizzly	mm	500		P	JDS - Tech. Report 2,000 tpd FS	PDC
Reclaim Rate	t/h	69	82	E		
Reclaim Method	-	Chute		C		
Dust Collection	wet/dry	Dry		A		
Primary Crusher						
Crusher type	-	Jaw		P	JDS - Tech. Report 2,000 tpd FS	PDC
Circuit Configuration	-	Open		P	JDS - Tech. Report 2,000 tpd FS	PDC
Size		30" x 40"		C		
Installed Power	kW	110		P	JDS - Tech. Report 2,000 tpd FS	PDC
Closed Side Setting	mm	70 - 75		P	JDS - Tech. Report 2,000 tpd FS	PDC
Estimated Feed F80	mm	500		V		Maximum crusher feed size 610 mm
Estimated Product P80	mm	100		V		
Fine Ore Surge Bin, Live	t	2,000		P	JDS - Tech. Report 2,000 tpd FS (1 day supply)	PDC

Grinding

Primary Grinding Circuit - SAG MILL						
Primary Grinding Feeder	-	Belt Conveyor, VSD		P	JDS - Tech. Report 2,000 tpd FS	PDC
Circuit Configuration	-	Open		P	JDS - Tech. Report 2,000 tpd FS	
Mill Type	-	SAG Mill		P	JDS - Tech. Report 2,000 tpd FS	
Number of Mills	#	1		P	JDS - Tech. Report 2,000 tpd FS	
Length, EGL	m	2.4		D,E,V	TBC by Chosen Vendor	
Diameter	m	4.6		D,E,V	TBC by Chosen Vendor	
Power Required	kW	401		E		
Installed Power	kW	448		D,E,V	TBC by Chosen Vendor	
% of Critical Speed	%	75		D,I		
Mill Discharge Density	% w/w	70		A		
Feed Size, F80	mm	100		A,D		
Product Size, P80	µm	425		E		
Mill Circulating Load	%	15		C,I		Stockpile and return by loader to SAG Feed Chute
Wear Materials						
Grinding Media Size	mm	125		O		
Consumption	g/t fresh feed	342		E		
Mill Liners	sets/years	0.5		C,J,V		based on low abrasion index
Primary Mill Discharge Screen						
Type	-	Vibrating		P	JDS - Tech. Report 2,000 tpd FS	
Make						
Number of Screens	#	1		P	JDS - Tech. Report 2,000 tpd FS	
Screen Width	m	2.4		V	TBC by Chosen Vendor	
Screen Length	m	6.1		V	TBC by Chosen Vendor	
Aperture Size	mm	10		D		
Power	kW	19		V	TBC by Chosen Vendor	
Feed Size, F100	mm	50		O		
Estimated Product Size, P80	mm	6.67		O		
Screen Feed	t/h	58	91	E		
Screen Oversize	t/h	8	9	E		
Screen Undersize	t/h	51	82	E		
Circulating Load	%	15		O		
Screen Deck Spray Water	m^3/hr	10		A		
Screen Oversize Density	% w/w	90		A		
Screen Undersize Density	% w/w	60		M		
Primary Gravity Separation						
Percent of New Feed to Gravity	%	100		V	FLS Knelson Model July 2014	
Gravity Feed	tph	50.8		E		
Type	-	Vibrating Single Deck Screen		V		
Size	m x m	0.915 x 2.439		V		
Aperture Size	mm	2		V		
Screen Oversize % Solids	%	90		P		
Screen Undersize % Solids	%	61		P		
Percent of screen U/S to concentrator	%	95		A		
Type	-	Centrifugal Concentrator,		V	FLS Knelson Model July 2014	
Number	#	1		V	FLS Knelson Model July 2014	
Feed	-	Cyclone Underflow		V	FLS Knelson Model July 2014	
Type		Semi Continuous, automated cycle		V	FLS Knelson Model July 2014	
Equipment Model		KC-XD20MS		V	FLS Knelson Model July 2014	
Operating Time	hour / day	24				
Fluidising Water Requirement	m^3/h	17		V	FLS Knelson Model July 2014 - G6 Cone	
Type of Water		Fresh		V	FLS Knelson Model July 2014	
Cycle Time per Flush/Discharge	min	10				
Concentrate Mass (20 min/batch)	kg / batch	19		E,V	FLS Knelson Model July 2014	
	dmt / day	1.4		E		
	kg/h	57		E,V	FLS Knelson Model July 2014	
Concentrate Solids Content	% wt.	93				
Concentrate Solids Content to Leach Circuit	% wt.	50				
Gold Recovery	g/t	0.58		E,V	FLS Knelson Model July 2014	
Recovery	%	20.5		V	FLS Knelson Model July 2014	
Secondary Grinding Circuit - Ball Mill No. 1						
Type	-	Ball Mill		C		
Number of mills	#	1		C		
Diameter	m	2.9		D,E,V	TBC by Chosen Vendor	
Length	m	4.9		D,E,V	TBC by Chosen Vendor	
Power Required	kW	371		E, V		
Installed Power	kW	448		D,E,V	TBC by Chosen Vendor	
Mill Discharge Density	%	65		P	JDS - Tech. Report 2,000 tpd FS	PFS
Feed Size, F80	µm	425		E		
Product Size, P80	µm	95		P	JDS - Tech. Report 2,000 tpd FS	
Mill Circulating Load	%	300		A		

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Wear Materials					
Grinding Media Size	mm	75	J		
Consumption	g/t fresh feed	226	E		
Mill Liners	sets per 2 years	1.0	C,J,V		based on low abrasion index
Mill Discharge Classification					
Type	-	Cyclopac	I		
Number of Cyclones	#	8 - 5 operating	E		
Size, (Diameter)	mm	254	E		
	in	10	E		
Cyclone Feed Density	% w/w	57	D, V		
Underflow Solids Density	% w/w	65	D		
Overflow Solids Density	% w/w	38	E		
Secondary Gravity Separation					
Percent of New Feed to Gravity	%	100	V	FLS Knelson Model July 2014	
Gravity Feed		50.79			
Type		Vibrating Single Deck Screen	V		
Size	m x m	0.915 x 2.439	V		
Aperture Size	mm	2	V		
Screen Oversize % Solids	%	90	P		
Screen Undersize % Solids	%	61	P		
Percent of screen U/S to concentrator	%	95	A		
Type	-	Centrifugal Concentrator,	V	FLS Knelson Model July 2014	
Number	#	1	V	FLS Knelson Model July 2014	
Feed	-	Cyclone Feed Pumpbox	V	FLS Knelson Model July 2014	
Type		Semi Continuous, automated cycle	V	FLS Knelson Model July 2014	
Equipment Model		KC-XD20MS	V	FLS Knelson Model July 2014	
Operating Time	hour / day	24			
Fluidising Water Requirement	m^3/h	17	V	FLS Knelson Model July 2014 - G6 Cone	
Type of Water		Fresh	V	FLS Knelson Model July 2014	10 - 15 minutes
Cycle Time per Flush/Discharge	min	10			
Concentrate Mass	kg / batch	19	E,V	FLS Knelson Model July 2014	
	dmt / day	1.37	E		
	kg/h	57	E,V	FLS Knelson Model July 2014	
Concentrate Grade	Au g/t				
Concentrate Solids Content	% wt.	93			
Concentrate Solids Content to Leach Circuit	% wt.	50			
Gold Recovery	g/t	0.584	E,V	FLS Knelson Model July 2014	
Recovery	%	20.5	V	FLS Knelson Model July 2014	
Tertiary Grinding Circuit - Ball Mill No. 2					
Mill Type	-	Ball Mill	C		
Number of mills	#	1	C		
Diameter	m	2.9	D,E,V	TBC by Chosen Vendor	
Length	m	4.9	D,E,V	TBC by Chosen Vendor	
Power Required	kW	385	E, V		
Total Power Installed	kW	448	D,E,V	TBC by Chosen Vendor	
Mill Discharge Density	% w/w	65	P	JDS - Tech. Report 2,000 tpd FS	
Feed Size, F80	µm	95	E		
Product Size, P80	µm	45	P	JDS - Tech. Report 2,000 tpd FS	
Mill Circulating Load	%	250	A		
Wear Materials					
Grinding Media Size	mm	50	J		
Consumption	g/t fresh feed	235	E		
Mill Liners	sets/years	1.0	C,J,V		based on low abrasion index
Mill Discharge Classification					
Type	-	Cyclopac	I		
Number of Cyclones	#	8 - 5 operating	E		
Size, (Diameter)	mm	254	E		
	in	10	E		
Cyclone Feed Density	% w/w	52	D, V		
Underflow Solids Density	% w/w	65	D		
Overflow Solids Density	% w/w	30	P	JDS - Tech. Report 2,000 tpd FS	
Intensive Leaching and Refining					
Feed Material	-	Gravity Concentrate	T,V	FLS Knelson Model July 2014	
Cyanide Solution Strength	%w/w	25-30	V	FLS Knelson Model July 2014	
Intensive Leaching Batch	batch/day	1	V	FLS Knelson Model July 2014	
Intensive Leaching Rate	t/batch	2.7	V	FLS Knelson Model July 2014	
Intensive Leaching Batch Time	hour/batch	16	V	FLS Knelson Model July 2014	
Water Requirements	m^3/h	15.0	V	FLS Knelson Model July 2014	30 m^3/30 minutes
Gold Recovery From Pregant Solution, PLS	-	Electrowinning	V	FLS Knelson Model July 2014	
Smelting Process Batch	batch/week	2	V	FLS Knelson Model July 2014	
Smelting Process Rate	kg/batch	4.5	V	FLS Knelson Model July 2014	
Flotation					
Design Factors					
Design Factor	-	1.2	J		
Flotation Retention Time Scale-up Factor					
Roughers		2.5	P	JDS - Tech. Report 2,000 tpd FS	PDC
Cleaners		4.0	J		
Froth Factor					
Roughers		1.0	J		
Cleaners		2.5	J		
Lip Length/ Solids Tonnage					
Roughers (Based on 200 kg/m/hr Lip Loading)	m - t/h	80/16	C,E	Ken Sangster June 16, 2014	
Cleaners (Based on 100 kg/m/hr Lip Loading)	m - t/h	40/4	C,E	Ken Sangster June 16, 2014	
Copper Flotation					
Copper Conditioning					
Conditioning Time	min	6.0	T	ALS Project T0662 Test No. T26-28	
Number of Tanks	#	1	A		
Slurry Feed Flowrate	m^3/hr	134 161	M		
Required Tank Volume	m^3	13 16	E		
Tank Size	m dia x m L	2.75 x 3.00	E		
Slurry pH	pH	7.2	T	ALS Project T0662 Test No. T26-28	

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Copper Rougher Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	38	38	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	133.9	160.7	M		
Feed Slurry Density	% w/w	30		M		
Conc. Slurry Density at the Lip	% w/w	35		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	30		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	9.5		T	ALS Project T0662 Test No. T26-28	
Retention Time (Design)	min	23.8		C,D		
Installed Retention Time	min	68.1	56.7	E		
Concentrate Mass Pull	%	10.5		T	ALS Project T0662 Test No. T26-28	
Slurry pH	pH	7.2		T	ALS Project T0662 Test No. T26-28	

Copper Cleaner Flotation

Copper 1st Cleaner Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	5.8	5.8	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	19.7	23.6	M		
Feed Slurry Density	% w/w	26		M		
Conc. Slurry Density at the Lip	% w/w	30		J		
Conc. Slurry Density	% w/w	25		J		
Retention Time (Laboratory)	min	6.0		T	ALS Project T0662 Test No. T26-28	
Retention Time (Design)	min	24.0		C,D		
Installed Retention Time	min	70.7	58.9	E		
Concentrate Mass Pull	%	8.3		E		
Slurry pH	pH	6.3		T	ALS Project T0662 Test No. T26-28	

Copper 2nd Cleaner Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	5.8	5.8	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	13.8	16.6	M		
Feed Slurry Density	% w/w	25		M		
Conc. Slurry Density at the Lip	% w/w	30		J		
Conc. Slurry Density	% w/w	25		J		
Retention Time (Laboratory)	min	6.5		T	ALS Project T0662 Test No. T26-28	
Retention Time (Design 4x)	min	26.0		C,D		
Installed Retention Time	min	100.7	83.9	E		
Concentrate Mass Pull	%	6.2		E	Grade/Recovery	
Slurry pH	pH	6.6		T	ALS Project T0662 Test No. T26-28	

Lead Flotation

Lead Conditioning Tank

Conditioning Time	min	7.0		T	ALS Project T0662 Test No. T26-28	
Number of Tanks	#	1		A		
Slurry Feed Flowrate	m^3/hr	131	158	M		
Required Tank Volume	m^3	15	18	E		
Tank Size	m dia x m L	2.75 x 3.00		E		
Slurry pH	pH	9.6		T	ALS Project T0662 Test No. T26-28	

Lead Rougher Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	38	38	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	131.6	157.9	M		
Feed Slurry Density	% w/w	29		M		
Conc. Slurry Density at the Lip	% w/w	35		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	30		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	9.5		T	ALS Project T0662 Test No. T26-28	
Retention Time (Design)	min	23.8		D,O		
Installed Retention Time	min	69.3	57.7	E		
Concentrate Mass Pull	%	4.9		P	JDS - Tech. Report 2,000 tpd FS	
Slurry pH	pH	9.5		T	ALS Project T0662 Test No. T26-28	

Lead Cleaner Flotation

Lead 1st Cleaner Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	5.8	5.8	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	10.6	12.8	M		
Feed Slurry Density	% w/w	24		M		
Conc. Slurry Density at the Lip	% w/w	30		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	22		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	6.5		T	ALS Project T0662 Test No. T26-28	PDC
Retention Time (Design)	min	26.0		D,O		
Installed Retention Time	min	131.0	109.2	E		
Concentrate Mass Pull	%	3.1		E		
Slurry pH	pH	10.3		T	ALS Project T0662 Test No. T26-28	

Lead 2nd Cleaner Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	5.8	5.8	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	6.0	7.2	M		
Feed Slurry Density	% w/w	22		M		
Conc. Slurry Density at the Lip	% w/w	35		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	22		P	JDS - Tech. Report 2,000 tpd FS	

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Retention Time (Laboratory)	min	11.5		T	ALS Project T0662 Test No. T26-28	PDC
Retention Time (Design)	min	46.0		D,O		
Installed Retention Time	min	233.3	194.4	E		
Concentrate Mass Pull	%	1.4		E	Grade/Recovery	
Slurry pH	pH	10.3		T	ALS Project T0662 Test No. T26-28	

Zinc Flotation

Zinc Conditioning Tank

Conditioning Time, per Tank	min	5.0		P	JDS - Tech. Report 2,000 tpd FS	PDC
Number of Tanks	#	2		A, I, T		pH control tank 1, $CuS0_4$ tank 2
Slurry Feed Flowrate	m^3/hr	133	160	M		
Required Tank Volume, per tank	m^3	11	13	E		
Tank Size	m dia x m L	2.75 x 3.00		E		
Slurry pH	pH	10.0		T	ALS Project T0662 Test No. T26-28	

Zinc Rougher Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	2		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	38	38	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	133.3	160.0	M		
Feed Slurry Density	% w/w	28		M		
Conc. Slurry Density at the Lip	% w/w	35		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	30		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	16.5		T	ALS Project T0662 Test No. T26-28	
Retention Time (Design)	min	41.3		D,O		
Installed Retention Time	min	136.8	114.0	E		
Rougher Concentrate Mass Pull	%	15.2		T	ALS Project T0662 Test No. T26-28	
Slurry pH	pH	10.0		T	ALS Project T0662 Test No. T26-28	

Zinc Cleaner Flotation

Zinc 1st Cleaner Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	2		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	5.8	5.8	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	27.5	33.1	M		
Feed Slurry Density	% w/w	26		M		
Conc. Slurry Density at the Lip	% w/w	35		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	25		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	7.0		T	ALS Project T0662 Test No. T26-28	
Retention Time (Design)	min	28.0		D,O		
Installed Retention Time	min	101.1	84.2	E		
Concentrate Mass Pull	%	12.8		E		
Slurry pH	pH	10.5		T	ALS Project T0662 Test No. T26-28	

Zinc 2nd Cleaner Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	2		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	5.8	5.8	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	21.1	25.4	M		
Feed Slurry Density	% w/w	25		M		
Conc. Slurry Density at the Lip	% w/w	30		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	25		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	12.0		T	ALS Project T0662 Test No. T26-28	PDC
Retention Time (Design)	min	48.0		D,O		
Installed Retention Time	min	131.6	109.7	E		
Concentrate Mass Pull	%	10.4		E	Grade/Recovery	
Slurry pH	pH	10.5		T	ALS Project T0662 Test No. T26-28	

Pyrite Flotation

Pyrite Conditioning Tank

Conditioning Time, per Tank	min	5.0		P	JDS - Tech. Report 2,000 tpd FS	PDC
Number of Tanks	#	1		A		
Slurry Feed Flowrate	m^3/hr	131	157	M		
Required Tank Volume	m^3	11	13	E		
Tank Size	m dia x m L	2.75 x 3.0		E		
Slurry pH	pH	8.9		E,T	ALS Project T0897 Test No. T32	

Pyrite Rougher Flotation

Cell Type	-	Conventional		C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Number of Banks	#	1		C		
Number of Cells per Bank	#	4	4	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Cell Volume	m^3	38	38	C	Email and Mill Layout Sketches Ken Sangster June 9, 2014	
Feed Flowrate	m^3/hr	130.9	157.1	M		
Feed Slurry Density	% w/w	26		M		
Conc. Slurry Density at the Lip	% w/w	35		P	JDS - Tech. Report 2,000 tpd FS	
Conc. Slurry Density	% w/w	29		P	JDS - Tech. Report 2,000 tpd FS	
Retention Time (Laboratory)	min	14.0		T	ALS Project T0897 Test No. T32	
Retention Time (Design)	min	35.0		D,O		
Installed Retention Time	min	69.7	58.1	E		
Rougher Concentrate Mass Pull	%	33		E	Grade/Recovery	
Slurry pH	pH	8.9		T	ALS Project T0897 Test No. T32	

Thickeners

Copper Concentrate

Thickener Type	-	High Rate				
Thickener Feed Rate	tph	3.1	3.8	M		
Thickener Underflow Density	%	60		V		
Thickener Loading	$t/h/m^2$	0.3		P,V	Previous Projects - TBC	
Thickener Settling Area Required	m^2	10	13	E		
Thickener Diameter, Calculated	m	4	4	E		
Thickener Diameter	m	4		J	Two - 4 m dia. thickeners installed low Cp/high Tn %As.	

Lead Thickener

Thickener Type	-	High Rate				

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Thickener Feed Rate	tph	0.8	1.0	M	
Thickener Underflow Density	%	60		V	
Thickener Loading	t/h/m^2	0.3		P, V	Previous Projects - TBC
Thickener Settling Area Required	m^2	3	3	E	
Thickener Diameter, Calculated	m	2	2	E	
Thickener Diameter	m	3		J	
Zinc Thickener					
Thickener Type	-	High Rate			
Thickener Feed Rate	tph	5.3	6.4	M	
Thickener Underflow Density	%	60		V	
Thickener Loading	t/h/m^2	0.3		P, V	Previous Projects - TBC
Thickener Settling Area Required	m^2	18	21	E	
Thickener Diameter, Calculated	m	5	5	E	
Thickener Diameter	m	6.5		J	
Pyrite Thickener (TBC)					
Thickener Type	-	High Rate			
Thickener Feed Rate	tph	16.9	20.3	M	
Thickener Underflow Density	%	50		T,V	
Thickener Loading	t/h/m^2	0.3		P, V	Previous Projects - TBC
Thickener Settling Area Required	m^2	56	68	E	
Thickener Diameter, Calculated	m	8	9	E	
Thickener Diameter	m	12		J	
Tailings Thickener (TBC)					
Thickener Type	-	High Rate			
Thickener Feed Rate	tph	24.5	29.4	M	
Thickener Underflow Density	%	50		V	
Thickener Loading	t/h/m^2	0.3		P,V	Previous Projects - TBC
Thickener Settling Area Required	m^2	82	98	E	
Thickener Diameter, Calculated	m	10	11	E	
Thickener Diameter	m	14		J	
Concentrate Storage					
Copper Stock Tank					
Retention Time	hr	8.0		I	
Number of Tanks	-	2			
Feed Flowrate	m^3/hr	3.4	4.1	M	
Tank Volume, Calculated	m^3	27	32	E	
Tank Size	m dia x m L	2.75 x 3.0		E	
Lead Stock Tank					
Retention Time	hr	8.0		I	
Feed Flowrate	m^3/hr	1.2	1.5	M	
Tank Volume, Calculated	m^3	10	12	E	
Tank Size	m dia x m L	2.75 x 2.75		E	
Zinc Stock Tank					
Retention Time	hr	8.0		I	
Feed Flowrate	m^3/hr	5.4	6.4	M	
Tank Volume, Calculated	m^3	43	51	E	
Tank Size	m dia x m L	4.50 x 5.00		E	
Concentrate Filtration (TBC)					
Copper Concentrate					
Filter Type	-	Pressure		I	
Number of Filters	#	1		A	
Filter Rate	Kg/m^2/h	582		P	Previous Projects - TBC
Filter Solids Feed Rate	tph	3.8	4.5	M	
Filter Availability	%	75			
Concentrate Density	%	92		C,D,V	TBC in next stage of project with testwork
Concentrate Moisture Content	%	8		C,D,V	TBC in next stage of project with testwork
Lead					
Filter Type	-	Pressure		I	
Number of Filters	#	1		A	
Filter Rate	Kg/m^2/h	395		P	Previous Projects - TBC
Filter Solids Feed Rate	tph	1.0	1.2	M	
Filter Availability	%	75			
Concentrate Density	%	92		C,D,V	TBC in next stage of project with testwork
Concentrate Moisture Content	%	8		C,D,V	TBC in next stage of project with testwork
Zinc					
Filter Type	-	Pressure		I	
Number of Filters	#	1		A	
Filter Rate	Kg/m^2/h	270		P	Previous Projects - TBC
Filter Solids Feed Rate	tph	6.4	7.6	M	
Filter Availability	%	75			
Concentrate Density	%	92		C,D,V	TBC in next stage of project with testwork
Concentrate Moisture Content	%	8		C,D,V	TBC in next stage of project with testwork
Reagents					
General					Testwork is based on ALS Project T0897, T0662 Test No. 26-28, LC01 to LC06 unless noted
Potassium Amyl Xanthate (PAX)					
Reagent Make-up Strength	%	10		V	
Reagent Usage per Tonne New Feed	g/t	154		T	ALS Project T0897 Test No. T32
	kg/d	169		E	
	kg/y	61,662		E	
Dosage Rate					
Py Rougher	g/t	154		T	ALS Project T0897 Test No. T32, Project T0662 LC04 to 06
MIBC					
Reagent Make-up Strength	%	100		V	
Reagent Usage	g/t	135		T	
	kg/d	148		E	
	kg/y	54,055		E	
Dosage Rate					
Cu Conditioning	g/t	30		T	
Cu 1st Cleaner	g/t	5		T	
Cu 2nd Cleaner	g/t	15		T	
Pb Conditioning	g/t	10		T	
Pb 1st Cleaner	g/t	10		T	
Pb 2nd Cleaner	g/t	5		T	
Zn Conditioning Tank	g/t	25		T	
Zn 1st Cleaner	g/t	10		T	

PROCESS DESIGN CRITERIA

CLIENT: Chieftain Metals Inc
PROJECT: Tulsequah 1,250 tpd Feasibility Study

DATE: October 8, 2014

PROJECT NUMBER:	CMITUL-05
DOCUMENT NUMBER:	CMITUL-05-09-001
Revision:	E2

Zn 2nd Cleaner	g/t	10		T		
Py Rougher	g/t	15		T	ALS Project T0897 Test No. T32	
3418a						
Reagent Make-up Strength	%		100	V		
Reagent Usage per Tonne New Feed	g/t		8	T		
	kg/d		8	E		
	kg/y		3,003	E		
Pb Conditioning	g/t	7		T		
Pb 1st Cleaner	g/t	1		T		
9810						
Reagent Make-up Strength	%		100	V		
Reagent Usage	g/t		7	T		
	kg/d		8	E		
	kg/y		2,803	E		
Dosage Rate						
Cu Conditioning	g/t	7		T		
SMBS						
Reagent Make-up Strength	%		20	V		
Reagent Usage	g/t		1605	T		
	kg/d		1,761	E		
	kg/y		642,650	E		
Dosage Rate						
Cu Conditioning	g/t	1,019		T		
Cu 1st Cleaner	g/t	382		T		
Cu 2nd Cleaner	g/t	204		T		
Sodium Cyanide						
Reagent Make-up Strength	%		10	V		
Reagent Usage	g/t		343	T		
	kg/d		526	E		
	kg/y		192,089	E		
Dosage Rate						
Pb Conditioning	g/t	302		T		
Pb 1st Cleaner	g/t	41		T		
Gold Recovery Circuit	kg/day	150		V	FlSmidth Knelson August 2014	
Aerofloat 7021						
Reagent Make-up Strength	%		100	V		
Reagent Usage	g/t		16	T		
	kg/d		18	E		
	kg/y		6,406	E		
Dosage Rate						
Zn Conditioning Tank	g/t	15		T		
Zn 2nd Cleaner	g/t	1.0		T		
Copper Sulphate						
Reagent Make-up Strength	%		10	V		
Reagent Usage	g/t		509	T		
	kg/d		558	E		
	kg/y		203,806	E		
Dosage Rate						
Zn Conditioning	g/t	509		T		
Lime						
Reagent Make-up Strength	%		20	A		
Reagent Usage	g/t		789	T		
	kg/d		866	E		
	kg/y		315,920	E		
Dosage Rate						
Pb Conditioning Tank	g/t	331		T		
Pb 1st Cleaner	g/t	20		T		
Pb 2nd Cleaner	g/t	5		T		
Zn Conditioning Tank	g/t	372		T		
Zn 1st Cleaner	g/t	61		T		
Zinc Sulphate						
Reagent Make-up Strength	%		10	V		
Reagent Usage	g/t		280	T		
	kg/d		307	E		
	kg/y		112,113	E		
Dosage Rate						
Cu 1st Cleaner	g/t	51		T		
Pb Conditioning	g/t	204		T		
Pb 1st Cleaner	g/t	25		T		
Flocculant						
Reagent Make-up Strength	%		0.50	V		
Reagent In- Line Mix Strength	%		0.25	V		
Reagent Usage	g/t concentrate		175	T		
	kg/d		43	T		
	kg/y		15,549	E		
Dosage Rate (TBC)						
Cu Concentrate Thickener	g/t conc.	35		P	JDS - Tech. Report 2,000 tpd FS - TBC	
Pb Concentrate Thickener	g/t conc.	35		P	JDS - Tech. Report 2,000 tpd FS - TBC	
Zn Concentrate Thickener	g/t conc.	35		P	JDS - Tech. Report 2,000 tpd FS - TBC	
Py Thickener	g/t conc.	35		P	JDS - Tech. Report 2,000 tpd FS - TBC	
Tailings Thickener	g/t conc.	35		P	JDS - Tech. Report 2,000 tpd FS - TBC	
Consumables						
Primary Crusher						
Jaw Crusher - Liner	kg/kWh		0.012	E		
	kg/d		105	E		
	kg/y		38,378	E		
SAG Mill						
SAG Mill Grinding Media	Type		Chrome Steel	I		
	kg/kWh		0.043	E		
	kg/d		375	E		
	kg/y		137,045	E		
SAG Mill-Liners	kg/kWh		0.005	E		
	kg/d		44	E		
	kg/y		15,980	E		
Ball Mill - Secondary						
Ball Mill Grinding Media	Type		Chrome Steel	I		

PROCESS DESIGN CRITERIA

CLIENT: Chieftain Metals Inc PROJECT: Tulsequah 1,250 tpd Feasibility Study DATE: October 8, 2014		**PROJECT NUMBER:**	**CMITUL-05**
		DOCUMENT NUMBER:	CMITUL-05-09-001
		Revision:	E2

	kg/kWh	0.031	E		
	kg/d	248	E		
	kg/y	90,566	E		
Ball Mill-Liners	kg/kWh	0.005	E		
	kg/d	41	E		
	kg/y	14,784	E		
Ball Mill - Tertiary					
Ball Mill Balls	Type	Chrome Steel	I		
	kg/kWh	0.031	E		
	kg/d	257	E		
	kg/y	93,984	E		
Ball Mill Liners	kg/kWh	0.005	E		
	kg/d	42	E		
	kg/y	15,342	E		

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS

CLIENT: Chieftain Metals Inc.

Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

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Black Bold - From another sheet
Blue - Input

STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
	ROM Ore	50.8	**95**	2.7	3.55	53.5	14.3	2.7	17.0	3.15
100	**Crushing Circuit**									
101	Feed to Grizzly	**68.6**	95	3.6	3.55	72.2	19.3	3.6	22.9	3.15
102	Feed to Jaw Crusher	**68.6**	95	3.6	3.55	72.2	19.3	3.6	22.9	3.15
103	Fine Ore Storage Bin Feed	**68.6**	95	3.6	3.55	72.2	19.3	3.6	22.9	3.15
200	**Grinding Circuit**									
201	SAG Mill New Feed	50.8	95	2.7	3.55	53.5	14.3	2.7	17.0	3.15
1001	**FW - SAG Mill Feed**			**21.5**				**21.5**		
202	SAG Mill Discharge	58.4	**70**	25.0	3.55	83.4	16.5	25.0	41.5	2.01
1002	**FW - Screen Wash Water**			**10.0**				**10.0**		
203	Primary Screen O/S to Waste (Returned to SAG feed conveyor)	**7.6**	**90**	0.8	3.55	8.5	2.1	0.8	3.0	2.83
204	Primary Screen U/S	50.8	60	34.2	3.55	85.0	14.3	34.2	48.5	1.75
205	BM1 Cyclone Feed	253.9	**57**	191.5	3.55	445.4	71.5	191.5	263.0	1.69
1003	**FW - BM1 Cyclone Feed**			**15.0**				**15.0**		
1101	**GW - BM1 Cyclone Feed Pump**			**2.0**				**2.0**		
206	Cyclone U/F	203.1	**65**	109.4	3.55	312.5	57.2	109.4	166.6	1.88
207	Cyclone U/F to Gravity	50.8	**65**	27.3	3.55	78.1	14.3	27.3	41.7	1.88
208	Cyclone U/F to BM1	**152.4**	**65**	82.0	3.55	234.4	42.9	82.0	125.0	1.88
209	BM1 Discharge	152.4	62	92.0	3.55	244.4	42.9	92.0	135.0	1.81
1004	**FW - BM1 Discharge Spray Water**			**10.0**				**10.0**		
210	BM1 Cyclone O/F	50.7	38	82.1	3.55	132.9	14.3	82.1	96.4	1.38
211	BM1 Trash Screen Overflow to Gravity Tailings Pumpbox	**2.5**	**90**	0.3	3.55	2.8	0.7	0.3	1.0	2.83
212	Gravity Concentrator Feed	48.2	61	31.1	3.55	79.3	13.6	31.1	44.7	1.78
1005	**FW - Gravity Trash Screen Spray Water**			**4.0**				**4.0**		
213	Gravity Concentrator Concentrate	0.1	**93**	0.0	5.00	0.1	0.0	0.0	0.0	3.01
1006	**Concentrate Dilution Water**			**0.1**				**0.1**		
214	Gravity Concentrator Concentrate to Gold Leaching Feed Tank	0.1	**50**	0.1	5.00	0.1	0.0	0.1	0.1	1.67
1007	**FW - Gravity Bowl Dilution Water**			**17.0**				**17.0**		
215	Gravity Concentrator Tailings	48.2	50	48.1	3.55	96.3	13.6	48.1	61.6	1.56
216	Gravity Concentrator Tailings + Trash Screen Overflow	50.7	51	48.3	3.55	99.1	14.3	48.3	62.6	1.58
217	BM2 Cyclone Feed	228.4	**52**	215.1	3.55	443.5	64.3	215.1	279.5	1.59
1008	**FW - BM2 Cyclone Feed**			**4.3**				**4.3**		
1104	**GW - BM2 Cyclone Feed Pump**			**2.0**				**2.0**		
218	Cyclone U/F	177.8	**65**	95.7	3.55	273.5	50.1	95.7	145.8	1.88
219	BM2 Cyclone U/F to Gravity	50.8	**65**	27.3	3.55	78.1	14.3	27.3	41.7	1.88
220	BM2 Cyclone U/F to BM2	127.0	**65**	68.4	3.55	195.3	35.8	68.4	104.1	1.88
221	BM2 Discharge	127.0	62	78.4	3.55	205.3	35.8	78.4	114.1	1.80
1009	**FW - BM2 Discharge Spray Water**			**10.0**				**10.0**		

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS
CLIENT: Chieftain Metals Inc.
Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

Legend
Black Bold - From another sheet
Blue - Input

STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
222	BM2 Cyclone O/F to Copper Circuit (Gravity)	50.7	30	119.4	3.55	170.1	14.3	119.4	133.7	1.27
223	BM1 Trash Screen Overflow to Gravity Tailings Pumpbox	2.5	90	0.3	3.55	2.8	0.7	0.3	1.0	2.83
224	Gravity Concentrator Feed	48.2	61	31.1	3.55	79.3	13.6	31.1	44.7	1.78
1010	**FW - Gravity Trash Screen Spray Water**			**4.0**				**4.0**		
225	Gravity Concentrator Concentrate	0.1	**93**	0.0	5.00	0.1	0.0	0.0	0.0	3.01
1011	**Concentrate Dilution Water**			**0.1**				**0.1**		
226	Gravity Concentrator Concentrate to Gold Leaching Feed Tank	0.1	**50**	0.1	5.00	0.1	0.0	0.1	0.1	1.67
1105	**GW - Gravity Concentrator Concentrate Pump**			**0.0**				**0.0**		
1012	**FW - Gravity Bowl Dilution Water**			**17.0**				**17.0**		
227	Gravity Concentrator Tailings	48.2	50	48.1	3.55	96.3	13.6	48.1	61.6	1.56
228	Gravity Concentrator Tailings + Trash Screen Overflow	50.7	51	48.3	3.55	99.1	14.3	48.3	62.6	1.58
229	Total Gravity Gold to Gold Leaching Feed Tank	0.1	50	0.1	5.00	0.2	0.0	0.1	0.1	1.56
	Grinding Area Water Balance									
300	**Copper Rougher Flotation Circuit**									
301	Copper Conditioning Feed	50.7	30	119.4	3.55	170.1	14.3	119.4	133.7	1.27
810	9810			**0.0**				0.0		
820	SMBS			**0.3**				0.3		
830	MIBC			**0.0**				0.0		
302	Copper Rougher Feed	50.7	30	119.7	3.55	170.3	14.3	119.7	133.9	1.27
303	Copper Rougher Tailings to Lead Circuit	45.3	29	109.8	3.55	155.1	12.8	109.8	122.5	1.27
304	Copper Rougher Concentrate at Lip	**5.3**	**35**	9.9	3.80	15.2	1.4	9.9	11.3	1.35
1013	**FW - Copper Rougher Conc. Launder Water**			**2.5**				**2.5**		
305	Copper Rougher Concentrate to Copper Cleaner (Gravity)	5.3	**30**	12.4	3.80	17.8	1.4	12.4	13.8	1.28
306	Copper Tailings to Lead Conditioning Tank	47.5	29	117.9	3.55	165.4	13.4	117.9	131.3	1.26
	Copper Rougher Circuit Water Balance									
310	**Copper Cleaner Flotation Circuit**									
310	Copper 1st Cleaner Circuit Feed	5.3	30	12.4	3.80	17.8	1.4	12.4	13.8	1.28
821	SMBS			**0.1**				0.1		
831	MIBC			**0.0**				0.0		
845	ZnSO4			**0.0**				0.0		
311	Copper 1st Cleaner Flotation Feed	6.4	26	18.0	3.80	24.4	1.7	18.0	19.7	1.24
312	Copper 1st Cleaner Tailings	2.2	21	8.1	3.46	10.3	0.6	8.1	8.7	1.18
313	Copper 1st Cleaner Conc. at Lip	**4.2**	**30**	9.9	4.00	14.1	1.1	9.9	10.9	1.29
1014	**FW - Copper 1st Cleaner Launder Water**			**2.8**				**2.8**		
315	Copper 1st Cleaner Conc. to Copper 2nd Cleaner (Gravity)	4.2	**25**	12.7	4.00	17.0	1.1	12.7	13.8	1.23
316	Copper 1st Cleaner Tailings to Lead Circuit	2.2	21	8.1	3.46	10.3	0.6	8.1	8.7	1.18

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS
CLIENT: Chieftain Metals Inc.
Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

Legend
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STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
317	Copper 2nd Cleaner Circuit Feed	4.2	25	12.7	4.00	17.0	1.1	12.7	13.8	1.23
832	MIBC			0.0				0.0		
822	SMBS			**0.1**				0.1		
318	Copper 2nd Cleaner Flotation Feed	4.2	25	12.8	4.00	17.0	1.1	12.8	13.8	1.23
319	Copper 2nd Cleaner Tailings to 1st Cleaner	1.1	17	5.4	4.00	6.5	0.3	5.4	5.7	1.14
320	Copper 2nd Cleaner Conc. at Lip	**3.1**	30	7.3	4.00	10.5	0.8	7.3	8.1	1.29
1015	**FW - Copper 2nd Conc. 1 Launder Water**			**2.1**				**2.1**		
322	Copper 2nd Cleaner Conc.	3.1	25	9.4	4.00	12.6	0.8	9.4	10.2	1.23
	Copper Cleaner Circuit Water Balance									
400	**Lead Rougher Flotation Circuit**									
401	Lead Conditioning Tank Feed	47.5	29	117.9	3.55	165.4	13.4	117.9	131.3	1.26
832	MIBC			0.0				0.0		
224	ZnSO4			**0.1**				0.1		
850	3418a			**0.0**				0.0		
860	Lime	0.0	**20**	**0.1**	2.60	0.1	0.0	0.1	0.1	0.00
870	Cyanide			**0.2**				0.2		
402	Lead Rougher Feed	47.5	29	118.2	3.55	165.7	13.4	118.2	131.6	1.26
403	Lead Rougher Tailings	45.1	28	113.6	3.52	158.7	12.8	113.6	126.4	1.26
404	Lead Rougher Concentrate at Lip	**2.5**	**35**	4.6	4.00	7.1	0.6	4.6	5.2	1.36
1020	**FW - Lead Rougher Conc. Launder Water**			**1.2**				**1.2**		
405	Lead Rougher Concentrate to Cleaner Circuit (Gravity)	2.5	**30**	5.8	4.00	8.2	0.6	5.8	6.4	1.29
406	Lead Tailings to Zinc Conditioning Tank	46.7	28	119.9	3.57	166.6	13.1	119.9	133.0	1.25
	Lead Rougher Circuit Water Balance									
410	**Lead Cleaner Flotation Circuit**									
411	Lead 1st Cleaner Circuit Feed	2.5	30	5.8	4.00	8.2	0.6	5.8	6.4	1.29
833	MIBC			0.0				0.0		
225	ZnSO4			**0.0**				0.0		
851	3418a			**0.0**				0.0		
861	Lime	0.0	**20**	0.0	2.60	0.0	0.0	0.0	0.0	1.14
871	Cyanide			**0.1**				0.1		
412	Lead 1st Cleaner Feed	3.2	24	10.0	4.00	13.3	0.6	10.0	10.6	1.25
413	Lead 1st Cleaner Tailings to Zinc Conditioning	1.6	21	6.3	5.52	7.9	0.3	6.3	6.6	1.20
414	Lead 1st Cleaner Concentrate at Lip	**1.6**	**30**	3.7	5.00	5.3	0.3	3.7	4.0	1.32
1021	**FW - Lead 1st Cleaner Conc. Launder Water**			**1.9**				**1.9**		
415	Lead 1st Cleaner Concentrate to 2nd Cleaner (Gravity)	1.6	**22**	5.6	5.00	7.2	0.3	5.6	6.0	1.21
1022	**FW - Lead 2nd Cleaner Feed**									
834	MIBC			0.0				0.0		
862	Lime	0.0	**20**	0.0	2.60	0.0	0.0	0.0	0.0	1.14
416	Lead 2nd Cleaner Feed	1.6	22	5.6	5.00	7.2	0.3	5.6	6.0	1.21
417	Lead 2nd Cleaner Tailings to 1st Cleaner	0.8	16	4.1	5.00	4.9	0.2	4.1	4.3	1.14

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS

CLIENT: Chieftain Metals Inc.

Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

Legend
Black Bold - From another sheet
Blue - Input

STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
418	Lead 2nd Cleaner Concentrate at Lip	**0.8**	**35**	1.5	5.00	2.3	0.2	1.5	1.7	1.39
1023	**FW - Lead 2nd Cleaner Conc. Launder Water**			**1.4**				**1.4**		
419	Lead 2nd Cleaner Concentrate to Lead Thickener (Gravity)	0.8	**22**	2.9	5.00	3.7	0.2	2.9	3.1	1.21
	Lead Cleaner Circuit Water Balance									
500	**Zinc Rougher Flotation Circuit**									
501	Zinc Conditioning Tank Feed	46.7	28	119.9	3.57	166.6	13.1	119.9	133.0	1.25
875	CuSO4			0.3				0.3		
863	Lime	0.0	**20**	0.1	2.60	0.1	0.0	0.1	0.1	1.14
890	7021			0.0				0.0		
835	MIBC			**0.0**				0.0		
502	Zinc Conditioning Tank Feed	46.7	28	120.2	3.57	167.0	13.1	120.2	133.3	1.25
503	Zinc Rougher Feed	46.7	28	120.2	3.57	167.0	13.1	120.2	133.3	1.25
504	Zinc Rougher Tailings to Pyrite Flotation	39.0	27	105.9	3.53	144.9	11.1	105.9	117.0	1.24
505	Zinc Rougher Concentrate at Lip	**7.7**	**35**	14.3	3.80	22.1	2.0	14.3	16.4	1.35
1024	**FW - Zinc Rougher Conc. Launder Water**			**3.7**				**3.7**		
506	Zinc Rougher Concentrate to 1st Cleaner Flotation (Gravity)	7.7	**30**	18.0	3.80	25.7	2.0	18.0	20.0	1.28
507	Zinc Tailings to Pyrite Conditioning Tank	41.5	26	119.0	3.52	160.5	11.8	119.0	130.8	1.23
	Zinc Rougher Circuit Water Balance									
510	**Zinc Cleaner Flotation Circuit**									
511	Zinc 1st Cleaner Feed	7.7	30	18.0	3.80	25.7	2.0	18.0	20.0	1.28
891	7021			**0.0**				0.0		
836	MIBC			**0.0**				0.0		
864	Lime	0.0	**20**	**0.0**	2.60	0.0	0.0	0.0	0.0	0.00
512	Zinc 1st Cleaner Feed	8.9	26	25.2	3.80	34.1	2.4	25.2	27.5	1.24
512a	Zinc 1st Cleaner Feed (per bank)	4.5	26	12.6	3.80	17.1	1.2	12.6	13.8	1.24
513	Zinc 1st Cleaner Tailings to Pyrite Flotation	2.4	16	13.1	3.35	15.5	0.7	13.1	13.8	1.12
514	Zinc 1st Cleaner Concentrate at Lip	**6.5**	**35**	12.1	4.00	18.6	1.6	12.1	13.7	1.36
1025	**FW - Zinc 1st Cleaner Conc. Launder Water**			**7.4**				**7.4**		
515	Zinc 1st Cleaner Concentrate to 2nd Cleaner (Gravity)	6.5	**25**	19.5	4.00	26.0	1.6	19.5	21.1	1.23
515a	Zinc 1st Cleaner Concentrate to 2nd Cleaner (per bank)	3.3	**25**	9.8	4.00	13.0	0.8	9.8	10.6	1.23
837	MIBC			**0.0**				0.0		
516	Zinc 2nd Cleaner Feed	6.5	25	19.5	4.00	26.0	1.6	19.5	21.1	1.23
517	Zinc 2nd Cleaner Tailings to 1st Cleaner	1.2	14	7.2	4.00	8.4	0.3	7.2	7.5	1.12
517a	Zinc 2nd Cleaner Tailings to 1st Cleaner (per bank)	0.6	14	3.6	4.00	4.2	0.2	3.6	3.7	1.12
518	Zinc 2nd Cleaner Concentrate at Lip	**5.3**	**30**	12.4	4.00	17.6	1.3	12.4	13.7	1.29
1026	**FW - Zinc 2nd Cleaner Conc. Launder Water**			**3.5**				**3.5**		

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS
CLIENT: Chieftain Metals Inc.
Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

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Black Bold - From another sheet
Blue - Input

STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
519	Zinc 2nd Cleaner Concentrate to Zinc Thickener (Gravity)	5.3	**25**	15.9	4.00	21.2	1.3	15.9	17.2	1.23
	Zinc Cleaner Circuit Water Balance									
600	**Pyrite Rougher Flotation Circuit**									
601	Pyrite Conditioning Tank Feed	41.5	26	119.0	3.52	160.5	11.8	119.0	130.8	1.23
838	MIBC			**0.0**				0.0		
885	PAX			0.1				0.1		
602	Pyrtie Rougher Feed	41.5	26	119.1	3.52	160.5	11.8	119.1	130.9	1.23
603	Pyrite Rougher Tailings to Final Tailings Thickener (Gravity)	24.5	22	87.7	2.70	112.2	9.1	87.7	96.7	1.16
604	Pyrite Rougher Concentrate at Lip	**16.9**	**35**	31.4	6.26	48.4	2.7	31.4	34.1	1.42
1027	**FW - Pyrite Rougher Conc. Launder Water**			**8.1**				**8.1**		
605	Pyrite Rougher Concentrate to Pyrite Thickener (Gravity)	16.9	**30**	39.5	6.26	56.4	2.7	39.5	42.2	1.34
	Pyrite Circuit Water Balance									
700	**Thickening/Dewatering**									
710	**Copper Concentrate Thickening/Dewatering**									
711	Copper Thickener Feed	3.1	25	9.4	4.00	12.6	0.8	9.4	10.2	1.23
840	Flocculant Addition			0.0				0.0		
712	Copper Thickener Feed	3.1	25	9.5	4.00	12.6	0.8	9.5	10.3	1.23
713	Copper Thickener O/F			14.2				14.2		
713a	Thickener O/F to Copper Flotation			4.9				4.9		
713b	Thickener O/F to Water Treatment			9.3				9.3		
714	Copper Thickener U/F	3.1	**60**	2.1	4.00	5.2	0.8	2.1	2.9	1.82
1118	**GW - Copper Thickener U/F Pump**			**0.5**				**0.5**		
715	Copper Thickener U/F To Filter	3.1	55	2.6	4.00	5.7	0.8	2.6	3.4	1.70
716	Copper Filter Feed	3.1	55	2.6	4.00	5.7	0.8	2.6	3.4	1.82
1119	**GW - Filter Feed Pump**			**0.5**				**0.5**		
717	Copper Filter Feed	3.1	50	3.1	4.00	6.2	0.8	3.1	3.9	1.61
718	Copper Filter Filtrate			6.8				6.8		
1215	**FW - Filter Spray**			**4.0**				**4.0**		
719	Copper Filter Concentrate	3.1	**92**	0.3	4.00	3.4	0.8	0.3	1.1	3.23
	Filtration at 75% availablility									
716a	Copper Filter Feed	3.8	55	3.1	4.00	6.9	0.9	3.1	4.1	0.00
1119a	**GW - Filter Feed Pump**			**0.5**				**0.5**		
717a	Copper Filter Feed	3.8	51	3.6	4.00	7.4	0.9	3.6	4.6	1.62
718a	Copper Filter Filtrate			7.3				7.3		
1215a	**FW - Filter Spray**			**4.0**				**4.0**		
719a	Copper Filter Concentrate (75% availability)	3.8	**92**	0.3	4.00	4.1	0.9	0.3	1.3	3.23
	Instantaneous Filter Flowrate (TBC)									

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS
CLIENT: Chieftain Metals Inc.
Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

Legend
Black Bold - From another sheet
Blue - Input

STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
717b	Pressure Filter Feed	31.4	50	31.0	4.00	62.4	7.9	31.0	38.8	1.61
1215b	Spray Water to Filter			**17.7**				**17.7**		
719b	Copper Cake	31.4	92	2.7	4.00	34.2	7.9	2.7	10.6	3.23
718b	Filtrate			**45.9**				**45.9**		
	Thickener/Filter Area Water Balance									
730	**Lead Thickening/Dewatering**									
731	Lead Thickener Feed	0.8	22	2.9	5.00	3.7	0.2	2.9	3.1	1.21
841	Flocculant Addition			0.0				0.0		
732	Lead Thickener Feed	0.8	22	2.9	5.00	3.7	0.2	2.9	3.1	1.21
733	Lead Thickener O/F			7.9				7.9		
733a	Lead Thickener O/F to Pb Flotation			3.3				3.3		
733b	Lead Thickener O/F to Water Treatment			4.5				4.5		
734	Lead Thickener U/F	0.8	**60**	0.5	5.00	1.4	0.2	0.5	0.7	1.92
1120	**GW - Lead Thickener U/F Pump**			**0.5**				**0.5**		
735	Lead Thickener U/F To Filter	0.8	44	1.0	5.00	1.9	0.2	1.0	1.2	1.54
736	Lead Filter Feed	0.8	44	1.0	5.00	1.9	0.2	1.0	1.2	1.92
1121	**GW - Filter Feed Pump**			**0.5**				**0.5**		
737	Lead Filter Feed	0.8	35	1.5	5.00	2.4	0.2	1.5	1.7	1.38
738	Lead Filter Filtrate			5.5				5.5		
1216	**FW - Filter Spray**			**4.0**				**4.0**		
739	Lead Filter Concentrate	0.8	**92**	0.1	5.00	0.9	0.2	0.1	0.2	3.79
	Filtration at 75% availablility									
736a	Lead Filter Feed	1.0	44	1.3	5.00	2.2	0.2	1.3	1.5	0.00
1120a	**GW - Filter Feed Pump**			**0.5**				**0.5**		
737a	Lead Filter Feed	1.0	36	1.8	5.00	2.7	0.2	1.8	2.0	1.40
738a	Lead Filter Filtrate			5.7				5.7		
1216a	**FW - Filter Spray**			**4.0**				**4.0**		
739a	Lead Filter Concentrate (75% availability)	1.0	**92**	0.1	5.00	1.1	0.2	0.1	0.3	3.79
	Instantaneous Filter Flowrate (TBC)									
737b	Pressure Filter Feed	8.2	35	15.5	5.00	23.7	1.6	15.5	17.1	1.38
1217b	Spray Water to Filter			**17.7**				**17.7**		
739b	Lead Cake	8.2	92	0.7	5.00	8.9	1.6	0.7	2.4	3.79
738b	Filtrate			**32.5**				**32.5**		
	Thickener/Filter Area Water Balance									
740	**Zinc Thickening/Dewatering**									
741	Zinc Thickener Feed	5.3	25	15.9	4.00	21.2	1.3	15.9	17.2	1.23

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS
CLIENT: Chieftain Metals Inc.
Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

DESIGN THROUGHPUT	1,097	TONNES PER DAY	
PLANT AVAILABILITY	90		
ORE S.G.	3.55		
TAILINGS S.G.	2.70		
	Rougher	1st Clnr	2nd Clnr
COPPER CONCENTRATE S.G.	3.80	4.00	4.00
LEAD CONCENTRATE S.G.	4.00	5.00	5.00
ZINC CONCENTRATE S.G.	3.80	4.00	4.00
PYRITE CONCENTRATE S.G.	6.26		

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Blue - Input

STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
842	Flocculant Addition			0.1				0.1		
742	Zinc Thickener Feed	5.3	25	16.0	4.00	21.3	1.3	16.0	17.3	1.23
743	Zinc Thickener O/F			20.5				20.5		
743a	Zinc Thickener O/F to Zinc Flotation			11.0				11.0		
743b	Zinc Thickener O/F to Water Treatment			9.5				9.5		
744	Zinc Thickener U/F	5.3	**60**	3.5	4.00	8.8	1.3	3.5	4.9	1.82
1122	**GW - Zinc Thickener U/F Pump**			**0.5**				**0.5**		
745	Lead Thickener U/F To Filter	5.3	57	4.0	4.00	9.3	1.3	4.0	5.4	1.74
746	Zinc Filter Feed	5.3	57	4.0	4.00	9.3	1.3	4.0	5.4	1.82
1123	**GW - Filter Feed Pump**			**0.5**				**0.5**		
747	Zinc Filter Feed	5.3	54	4.5	4.00	9.8	1.3	4.5	5.9	1.68
748	Zinc Filter Filtrate			8.1				8.1		
1217	**FW - Filter Spray**			**4.0**				**4.0**		
749	Zinc Filter Concentrate	5.3	**92**	0.5	4.00	5.8	1.3	0.5	1.8	3.23
	Filtration at 75% availablility									
746a	Zinc Filter Feed	6.4	25	19.1	4.00	25.4	1.6	19.1	20.6	1.23
1123a	**GW - Filter Feed Pump**			**0.5**				**0.5**		
747a	Zinc Filter Feed	6.4	25	19.6	4.00	25.9	1.6	19.6	21.1	1.23
748a	Zinc Filter Filtrate			23.0				23.0		
1218a	**FW - Filter Spray**			**4.0**				**4.0**		
749a	Zinc Filter Concentrate	6.4	**92**	0.6	4.00	6.9	1.6	0.6	2.1	3.23
	Instantaneous Filter Flowrate (TBC)									
747b	Pressure Filter Feed	52.9	54	45.3	4.00	98.2	13.2	45.3	58.5	1.68
1218b	Spray Water to Filter			**17.7**				**17.7**		
749b	Zinc Cake	52.9	92	4.6	4.00	57.5	13.2	4.6	17.8	3.23
748b	Filtrate			**58.4**				**58.4**		
	Thickener/Filter Area Water Balance									
750	**Pyrite Thickening/Dewatering**									
751	Pyrite Thickener Feed	16.9	30	39.5	6.26	56.4	2.7	39.5	42.2	1.34
843	Flocculant Addition			0.2				0.2		
752	Pyrite Thickener Feed	16.9	30	39.7	6.26	56.7	2.7	39.7	42.4	1.34
753	Pyrite Thickener O/F			22.8				22.8		
753a	Pyrite Thickener O/F to Flotation			8.1				8.1		
753b	Pyrite Thickener O/F to Water Treatment Plant			14.7				14.7		
754	Pyrite Thickener U/F	16.9	**50**	16.9	6.26	33.9	2.7	16.9	19.6	1.72
1124	**GW - Pyrite Thickener U/F Pump**			**1.0**				**1.0**		
755	Pyrite Thickener U/F To Backfill Plant or Pyrite Pond	16.9	49	17.9	6.26	34.9	2.7	17.9	20.6	1.69

PROCESS MASS BALANCE

PROJECT: Telsequah Chief
PROJ. No: CMITUL-05 Tulsequah 1250 tpd FS
CLIENT: Chieftain Metals Inc.
Document No. CMITUL-05-09-001
DATE: 10/8/2014
REVISION: E2

	Rougher	1st Clnr	2nd Clnr	
DESIGN THROUGHPUT	1,097	TONNES PER DAY		
PLANT AVAILABILITY	90			
ORE S.G.	3.55			
TAILINGS S.G.	2.70			
COPPER CONCENTRATE S.G.	3.80	4.00	4.00	
LEAD CONCENTRATE S.G.	4.00	5.00	5.00	
ZINC CONCENTRATE S.G.	3.80	4.00	4.00	
PYRITE CONCENTRATE S.G.	6.26			

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STREAM NO	DESCRIPTION	SOLIDS tph	% SOLIDS	SOL'N tph	SOLIDS SG	SLURRY tph	SOLIDS m3/hr	SOL'N m3/hr	SLURRY m3/hr	SLURRY SG
756	Settled Pyrite	16.9	**75**	5.6	6.26	22.6	2.7	5.6	8.3	2.70
757	Available Pond Water to Water Treatment			12.3				12.3		
758	Pyrite O/F to Water Treatment Plant			14.7				14.7		
	Thickener/Filter Area Water Balance									
760	**Final TailingsThickening/Dewatering**									
761	Tailings Thickener Feed	24.5	22	87.7	2.70	112.2	9.1	87.7	96.7	1.16
748	Flocculant Addition			0.3				0.3		
762	Tailings Thickener Feed	24.5	22	88.0	2.70	112.5	9.1	88.0	97.1	1.16
763	Tailings Thickener O/F to Water Treatment			63.5				63.5		
764	Tailings Thickener U/F	24.5	**50**	24.5	2.70	49.1	9.1	24.5	33.6	1.46
1125	**GW - Tailings U/F Pump**			**1.0**				**1.0**		
765	Tailings Thickener U/F To Backfill Plant	24.5	49	25.5	2.70	50.1	9.1	25.5	34.6	1.45
	Bypass to Tailings Pond									
766	Tailings Thickener U/F To Tailings Pond	24.5	49	25.5	2.70	50.1	9.1	25.5	34.6	1.45
1126	**GW - Final Tailings Pond Feed Pump No. 1**			**1.5**				**1.5**		
1127	**GW - Final Tailings Pond Feed Pump No. 2**			**1.5**				**1.5**		
1128	**GW - Final Tailings Pond Feed Pump No. 3**			**0.0**				**0.0**		
767	Tailings Thickener U/F To Tailings Pond	26.0	46	30.2	2.70	56.2	9.6	30.2	39.8	1.41
768	Settled Tailings	26.0	**75**	8.7	2.70	34.7	9.6	8.7	18.3	1.89
769	Pond Water to Water Treatment			21.5				21.5		
	Thickener/Filter Area Water Balance									
2310	Limestone BM Feed	1.5	94	0.1	2.40	1.6	0.6	0.1	0.7	2.21
1307	**FW - BM Mill Feed**			**1.4**				**1.4**		
2309	BM Mill Discharge	1.5	50	1.5	2.40	2.9	0.6	1.5	2.1	1.41
2400	**GW**			**0.1**				**0.1**		
2310	Limestone Pumpbox to Mixing Tank	1.5	48	1.6	2.40	3.0	0.6	1.6	2.2	1.39
2401	**GW**			**0.1**				**0.1**		
2311	Limestone Mixing Tank to Final Tailings Thickener	1.5	47	1.7	2.40	3.1	0.6	1.7	2.3	1.38
	Limestone Area Water Balance									

PROCESS EQUIPMENT LIST

CLIENT: Chieftain Metals Inc
PROJECT: Tulsequah 1,250 tpd Feasibility Study

DATE: October 31, 2014

Equipment Number			Description	Equipment Size	Power (kW)	Power (HP)
Area	Tag	Number				
156010						
156010	BRK	001	Rock Breaker	MB432	30	40
156010	CHU	001	Primary Crushing Grizzly Feed Chute			
156010	CHU	002	Jaw Crusher Feed Chute			
156010	CHU	003	Jaw Crusher Discharge Chute			
156010	CNV	001	Storage Bin Feed Conveyor	**750 W X 12,000 L**	56	75
156010	COL	001	Primary Crushing Dust Collector		15	20
156010	CRN	001	Crushing Chamber Crane	25t/10t	34	45
156010	CRU	001	Primary Crusher Jaw Crusher	C100 or equivalent	110	147
156010	FDR	013	Belt Feeder			
156010	MGT	001	Storage Bin Feed Conveyor Belt Magnet		5	7
156010	SCB	001	Storage Bin Conveyor Belt Scale		1.5	2
156010	SCN	001	Stationary Grizzly	6x10, 500 mm opening		
156020						
156020	ACA	001	Air Cannon No.1			
156020	ACA	002	Air Cannon No.2			
156020	ACA	003	Air Cannon No.3			
156020	ACA	004	Air Cannon No.4			
156020	BIN	001	Crushed Material Storage Bin	2,000 t		
156020	CHU	004	Storage Bin Feed Conveyor Head Chute			
156020	CHU	005	Storage Bin Belt Feeder No.1 Feed Chute			
156020	CHU	006	Storage Bin Belt Feeder No.1 Head Chute			
156020	COL	002	Crushed Material Storage Bin Dust Collector		15	20
156020	FDR	001	Storage Bin Reclaim Belt Feeder	750 W X 2000 L	11	15
156020	HOI	001	Crushed Material Storage Bin Hoist 1	1 tonne	3	4
156020	HOI	002	Crushed Material Storage Bin Hoist 2	1 tonne	3	4
201010						
201010	CHU	007	Limestone Jaw Crusher Inlet Chute			
201010	CHU	008	Limestone Stockpile Feed Conveyor Inlet Chute			
201010	CHU	009	Limestone Pan Feeder Chute			
201010	CNV	002	Limestone Jaw Crusher Discharge Conveyor	500W X 3000 L	4	5
201010	CNV	003	Limestone Impact Crusher Discharge Conveyor	500W X 2000 L	4	5
201010	CNV	004	Limestone Stockpile Feed Conveyor	500 W X 74700 L	5.6	8
201010	CRU	002	Limestone Jaw Crusher	C63 or equiv.	45	60
201010	CRU	003	Impact Crusher	HP100 or equiv.	90	121
201010	FDR	002	Limestone Pan Feeder	762 W	1	1
201010	HPR	001	Limestone Hopper C/W Grizzly			
201010	SYS	001	Limestone Crushing Plant			
251010						
251010	BIN	002	Ball Mill Trash Bin No.1			
251010	BIN	003	Ball Bin No.1	1.5W X 1.5 L X 1.8 H		
251010	BIN	005	Ball Mill Trash Bin No.2			
251010	BIN	006	Ball Bin No.2	1.5W X 1.5 L X 1.8 H		
251010	CHU	010	SAG Feed Chute			
251010	CHU	011	Primary Ball Mill Pumpbox Discharge Chute			
251010	CHU	012	Primary Ball Mill Trash Discharge Chute			
251010	CHU	013	Primary Cyclone Collecting Chute			
251010	CHU	014	Primary Ball Charging Feed Chute			
251010	CHU	015	Secondary Ball Mill Feed Chute			
251010	CHU	016	SAG Discharge Chute			
251010	CHU	017	Primary Ball Mill Feed Chute			
251010	CHU	018	Secondary Ball Mill Pumpbox Discharge Chute			
251010	CHU	019	Secondary Ball Mill Trash Discharge Chute			
251010	CHU	020	Secondary Cyclone Collecting Chute			
251010	CHU	021	Secondary Ball Charging Feed Chute			
251010	CNV	005	Sag Mill Feed Conveyor	750 W x 208000 L	56	75
251010	CNV	006	SAG Mill Coarse Material Discharge Conveyor	750W X 10000L	15	20
251010	CRN	002	Grinding Area Crane (need to be able to load the balls quickly)	10 t	7.5	10
251010	CYC	001	Primary Cyclones	8 - 5 oper, 3 stby 254 mm		
251010	CYC	002	Secondary Cyclones	8 - 5 oper, 3 stby 254 mm		
251010	DRV	001	SAG Mill inching Drive		2.2	3
251010	DRV	002	Ball Mill Inching Drive		2.2	3
251010	EQP	001	Ball Charging Kibble No. 1	0.5m3		
251010	EQP	002	Primary Ball Mill Cradle			
251010	EQP	003	Ball Charging Kibble No. 2	0.5m3		
251010	EQP	004	Secondary Ball Mill Cradle			
251010	EQP	005	Ball Mill Jacking System		1.1	1
251010	EQP	016	SAG Mill Jacking System		1.1	1
251010	LUB	001	SAG Mill Lube Unit		5.6	8
251010	LUB	002	Ball Mill Lube Unit No.1		5.6	8

251010	LUB	003	Ball Mill Lube Unit No.2		5.6	8
251010	MGT	002	Ball Mill Ball Load Magnet			
251010	MIL	001	SAG Mill	4573D X 2439L	448	600
251010	MIL	002	Primary Ball Mill	2896D X 4878L	448	600
251010	MIL	003	Secondary Ball Mill	2896D X 4878L	448	600
251010	PBX	001	Primary Cyclones Feed Pumpbox	6.0m3, 1.3 tonnes		
251010	PBX	002	Secondary Cyclones Feed Pumpbox	6.0m3, 1.3 tonnes		
251010	PSL	001	Primary Cyclones Feed Pump	200 X 150	93.0	125
251010	PSL	002	Secondary Cyclones Feed Pump	200 X 150	93.0	125
251010	PSL	003	Standby Cyclones Feed Pump	200 X 150	93.0	125
251010	PSU	002	Grinding Area Sump Pump	100	11	15
251010	SCB	002	SAG Mill Feed Conveyor Belt Scale		1.5	2
251010	SCN	002	Sag Mill Discharge Screen	2.4m x 6.1m L	19	25
251010	SMP	001	SAG Mill Feed Conveyor Sampler		1.5	2
251010	SMP	002	Primary Cyclopac O/F Sampler	SamStat-30	1	1.3
251010	SMP	003	Secondary Cyclopac O/F Sampler	SamStat-30	1	1.3
252040						
252040	AGI	001	Reactor Feed Tank Agitator		0.2	0.3
252040	CEL	001	Electrowinning Cell	EW3000,50 cu.ft., 18 SS Cath, 20 Anodes	4.0	5
252040	CHU	022	No.1 Feed Trash Screen Chute			
252040	CHU	023	No.1 Feed Trash Discharge Chute			
252040	CHU	024	No.1 Feed Trash Screen Underpan			
252040	CHU	025	No.2 Feed Trash Screen Chute			
252040	CHU	026	No.2 Feed Trash Discharge Chute			
252040	CHU	027	No.2 Feed Trash Screen Underpan			
252040	CNC	001	Gravity Concentrator No.1	KC-XD20MS	8	10
252040	CNC	002	Gravity Concentrator No.2	KC-XD20MS	8	10
252040	COL	003	Smelting Furnace Dust Collector			
252040	FAN	054	Drying Oven Exhaust Fan		2.4	3
252040	FAN	058	Smelting Furnace Dust Collector Fan		7.5	10
252040	FUR	001	Induction Smelting Furnace	Induction c/w hood, ducting, curtains	50	67
252040	KLN	001	Drying Oven	7 cu. Ft.	2.4	3
252040	MIX	001	Flux Mixer	3 cu. Ft.		
252040	PSL	005	Barren Solids Discharge Pump	75 x 75 Krebs Slurry Pump	4.0	5.4
252040	PSL	006	Electrowinning Feed Pump		3.0	4
252040	PSL	008	Emergency Cyanide Detoxification Pump	25 x 25	1.5	2
252040	PSO	009	Leach Solution Transfer Pump		4.0	
252040	PSO	031	Barren Solution Pump		3.0	4
252040	PSU	003	Gold Concentrate Sump Pump	38 mm	3.8	5
252040	SCN	003	Gravity Concentrator No.1 Feed Trash Screen	3'x8' SDH	2.2	3
252040	SCN	004	Gravity Concentrator No.2 Feed Trash Screen	3'x8' SDH	2.2	3
252040	SFE	001	Gold Room Vault			
252040	SSW	001	Gold Recovery Safety Shower			
252040	SYS	002	Emergency Cyanide Detoxification System	c/w 12 m3 tank, mixer	0.8	1.0
252040	SYS	011	Consep Acacia	CS3000 c/w leach aid feeder	0.8	1.0
252040	SYS	012	Gold Room and Refinery	c/w rectifier, molds, slag pot, crushers, scales	21.0	28
252040	TNK	001	GRG Concentrate Storage Cone	3 m3 Storage Cone		
252040	TNK	002	Intensive Cyanidation Unit			
252040	TNK	003	Reactor Feed Tank	8 m3 c/w 2 - 20 kW immersion heaters and agitator	40.0	54
252040	TNK	004	Electrowinning Feed Tank	8 m3 c/w1 - immersion heater, level detector, flow meter, control valves	20.0	27
252040	TNK	005	Barren Recycle Tank	8 m3 c/w level detector, flow meter, control valves		
252040	VES	001	Carbon Column	4.5 m3		
252050						
252050	AGI	002	Cu Flotation Conditioning Tank Agitator		7.5	10
252050	AGI	003	Zn Scav./Cu Conc. 2 Conditioning Tank Agitator		1.1	1.5
252050	AGI	004	Pb Flotation Conditioning Tank Agitator		7.5	10
252050	AGI	017	Zn Flotation Conditioning Tank No. 1 Agitator		7.5	10
252050	AGI	005	Zn Flotation Conditioning Tank No. 2 Agitator		7.5	10
252050	AGI	006	Pyrite Flotation Conditioning Tank Agitator		7.5	10
252050	CEL	002	Cu Rougher Flotation Cell No.1	38 m3	56.0	75
252050	CEL	003	Cu Rougher Flotation Cell No.2	38 m3	45	60
252050	CEL	004	Cu Rougher Flotation Cell No.3	38 m3	45	60
252050	CEL	005	Cu Rougher Flotation Cell No.4	38 m3	45	60
252050	CEL	006	Cu 1st Cleaner Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	007	Cu 1st Cleaner Flotation Cell No.2	5.8m3	11.0	15
252050	CEL	008	Cu 1st Cleaner Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	009	Cu 1st Cleaner Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	010	Cu 2nd Cleaner Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	011	Cu 2nd Cleaner Flotation Cell No.2	5.8m3	11.0	15
252050	CEL	012	Cu 2nd Cleaner Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	013	Cu 2nd Cleaner Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	014	Cu Conc. 1 Cleaner Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	015	Cu Conc. 1 Cleaner Flotation Cell No.2	5.8m3	11.0	15

252050	CEL	016	Cu Conc. 1 Cleaner Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	017	Cu Conc. 1 Cleaner Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	018	Zn Scav./Cu Conc. 2 Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	019	Zn Scav./Cu Conc. 2 Flotation Cell No.2	5.8m3	11.0	15
252050	CEL	020	Zn Scav./Cu Conc. 2 Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	021	Zn Scav./Cu Conc. 2 Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	022	Pb Rougher Flotation Cell No.1	38 m3	56.0	75
252050	CEL	023	Pb Rougher Flotation Cell No.2	38 m3	56.0	60
252050	CEL	024	Pb Rougher Flotation Cell No.3	38 m3	56.0	60
252050	CEL	025	Pb Rougher Flotation Cell No.4	38 m3	56.0	60
252050	CEL	026	Pb 1st Cleaner Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	027	Pb 1st Cleaner Flotation Cell No.2	5.8m3	11.0	15
252050	CEL	028	Pb 1st Cleaner Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	029	Pb 1st Cleaner Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	030	Pb 2nd Cleaner Flotation Cell No. 1	5.8m3	11.0	15
252050	CEL	031	Pb 2nd Cleaner Flotation Cell No. 2	5.8m3	11.0	15
252050	CEL	032	Pb 2nd Cleaner Flotation Cell No. 3	5.8m3	11.0	15
252050	CEL	033	Pb 2nd Cleaner Flotation Cell No. 4	5.8m3	11.0	15
252050	CEL	034	Zn Rougher Flotation Cell No.1	38 m3	56.0	75
252050	CEL	035	Zn Rougher Flotation Cell No.2	38 m3	56.0	60
252050	CEL	036	Zn Rougher Flotation Cell No.3	38 m3	56.0	60
252050	CEL	037	Zn Rougher Flotation Cell No.4	38 m3	56.0	60
252050	CEL	038	Zn Rougher Flotation Cell No.5	38 m3	56.0	75
252050	CEL	039	Zn Rougher Flotation Cell No.6	38 m3	56.0	60
252050	CEL	040	Zn Rougher Flotation Cell No.7	38 m3	56.0	60
252050	CEL	041	Zn Rougher Flotation Cell No.8	38 m3	56.0	60
252050	CEL	042	Zn 1st Cleaner Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	043	Zn 1st Cleaner Flotation Cell No.2	5.8m3	11.0	15
252050	CEL	044	Zn 1st Cleaner Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	045	Zn 1st Cleaner Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	046	Zn 1st Cleaner Flotation Cell No.5	5.8m3	11.0	15
252050	CEL	047	Zn 1st Cleaner Flotation Cell No.6	5.8m3	11.0	15
252050	CEL	048	Zn 1st Cleaner Flotation Cell No.7	5.8m3	11.0	15
252050	CEL	049	Zn 1st Cleaner Flotation Cell No.8	5.8m3	11.0	15
252050	CEL	050	Zn 2nd Cleaner Flotation Cell No.1	5.8m3	11.0	15
252050	CEL	051	Zn 2nd Cleaner Flotation Cell No.2	5.8m3	11.0	15
252050	CEL	052	Zn 2nd Cleaner Flotation Cell No.3	5.8m3	11.0	15
252050	CEL	053	Zn 2nd Cleaner Flotation Cell No.4	5.8m3	11.0	15
252050	CEL	054	Zn 2nd Cleaner Flotation Cell No.5	5.8m3	11.0	15
252050	CEL	055	Zn 2nd Cleaner Flotation Cell No.6	5.8m3	11.0	15
252050	CEL	056	Zn 2nd Cleaner Flotation Cell No.7	5.8m3	11.0	15
252050	CEL	057	Zn 2nd Cleaner Flotation Cell No.8	5.8m3	11.0	15
252050	CEL	058	Pyrite Flotation Cell No.1	38 m3	56.0	75
252050	CEL	059	Pyrite Flotation Cell No.2	38 m3	56.0	60
252050	CEL	060	Pyrite Flotation Cell No.3	38 m3	56.0	60
252050	CEL	061	Pyrite Flotation Cell No.4	38 m3	56.0	60
252050	CHU	028	Cu Rougher Flotation Overflow Chute			
252050	CHU	029	Cu 1st Cleaner Flotation Overflow Chute			
252050	CHU	030	Cu 2nd Cleaner Flotation Overflow Chute			
252050	CHU	031	Cu Conc. 1 Cleaner Flotation Overflow Chute			
252050	CHU	032	Zn Scav./Cu Conc. 2 Cleaner Flotation Overflow Chute			
252050	CHU	033	Pb Rougher Flotation Overflow Chute			
252050	CHU	034	Pb 1st Cleaner Flotation Overflow Chute			
252050	CHU	035	Pb 2nd Cleaner Flotation Overflow Chute			
252050	CHU	036	Zn Rougher Flotation Overflow Chute			
252050	CHU	037	Zn 1st Cleaner Flotation Overflow Chute			
252050	CHU	038	Zn 2nd Cleaner Flotation Overflow Chute			
252050	CHU	039	Py Rougher Flotation Overflow Chute			
252050	CRN	003	Flotation Area Crane - Rougher	2t	0.0	0
252050	CRN	006	Flotation Area Crane - Cleaner	2t	0.0	0
252050	PSL	009	Copper Tailings Pump	100 mm	30	40
252050	PSL	010	Copper 2nd Cleaner Tailings Pump	38 mm	7.5	10
252050	PSL	011	Copper 2nd Cleaner Concentrate Pump	38 mm	7.5	10
252050	PSL	012	Zn Scavenger Concentrate Pump	38 mm	7.5	10
252050	PSL	013	Pb Tailings Pump	100 mm	30	40
252050	PSL	014	Pb 2nd Cleaner Tailings Pump	38 mm	7.5	10
252050	PSL	015	Zn Tailings Bank 1 Pump	100 mm	30	40
252050	PSL	029	Zn Tailings Bank 2 Pump	100 mm	30	40
252050	PSL	016	Zn Bank 1 1st Cleaner Tailings Pump	38 mm	7.5	10
252050	PSL	017	Zn Bank 2 1st Cleaner Tailings Pump	38 mm	7.5	10
252050	PSL	018	Standby Flotation Pump No.1	100 mm	30	40
252050	PSL	019	Standby Flotation Pump No.2	100 mm	30	40
252050	PSL	020	Standby Flotation Pump No.3	38 mm	7.5	10
252050	PSU	006	Pb Flotation Area Sump Pump	50 mm	5.6	7.5
252050	PSU	007	Pyrite Flotation Area Sump Pump	50 mm	5.6	7.5
252050	SMP	004	Cu Cp Tailings Sampler	AnStat-230	1.0	1
252050	SMP	005	Cu Cl 1 Tailings Sampler	AnStat-230	1.0	1

252050	SMP	006	Cu Conc. 1 Sampler	AnStat-230F	1.0	1
252050	SMP	007	Cu Conc. 2 Sampler	SamStat-30F	1.0	1
252050	SMP	008	Zn Scavenger Concentrate Sampler	SamStat-30F	1.0	1
252050	SMP	009	Cu Tailings Sampler to Lead Conditioning Tank	SamStat-30	1.0	1
252050	SMP	010	Pb Tailings Sampler to Zinc Conditioning Tank	SamStat-30	1.0	1
252050	SMP	011	Pb Final Concentrate Sampler	SamStat-30F	1.0	1
252050	SMP	014	Cu Cl2 Concentrate Sampler	AnStat-230	1.0	1
252050	SMP	015	Zn Concentrate Sampler	SamStat-30F	1.0	1
252050	SMP	016	Pyrite Concentrate Sampler	SamStat-30F	1.0	1
252050	SMP	017	Final Tailings Sampler	SamStat-30	1.0	1
252050	TNK	007	Cu Conditioning Tank	2750 D X 3000 H, 2.7 tonne		
252050	TNK	008	Cu Tailings Sump	part of concrete (sump)		
252050	TNK	009	Cu 2nd Cleaner Tailings Sump	part of concrete (sump)		
252050	TNK	010	Zn Scav./ Cu Conc. 2 Conditioning Tank	1000 D X 1500 H, 0.6 tonne		
252050	TNK	011	Zn Scav. Conc. Sump	part of concrete (sump)		
252050	TNK	012	Pb Conditioning Tank	2750 D X 3000 H, 2.7 tonne		
252050	TNK	013	Pb Tailings Sump	part of concrete (sump)		
252050	TNK	014	Pb 2nd Cleaner Tailings Sump	part of concrete (sump)		
252050	TNK	015	Zn Conditioning Tank No. 1	2750 D X 3000 H, 2.7 tonne		
252050	TNK	040	Zn Conditioning Tank No. 2	2750 D X 3000 H, 2.7 tonne		
252050	TNK	016	Zn Bank 1 2nd Cleaner Tailings Sump	part of concrete (sump)		
252050	TNK	017	Zn Bank 2 2nd Cleaner Tailings Sump	part of concrete (sump)		
252050	TNK	018	Zn Tailings Bank 1 Sump	part of concrete (sump)		
252050	TNK	033	Zn Tailings Bank 2 Sump	part of concrete (sump)		
252050	TNK	019	Pyrite Conditioning Tank	2750 D X 3000 H, 2.7 tonne		
252050	TNK	058	Cu 2nd Cleaner Conc. Sump	750 D X 1000 H, 0.26 tonne		
253010						
253010	AGI	007	Cu Conc. 1 Stock Tank Agitator		1.5	2
253010	AGI	008	Cu Conc. 2 Stock Tank Agitator		1.5	2
253010	AGI	009	Pb Concentrate Stock Tank Agitator		1.5	2
253010	AGI	010	Zn Concentrate Stock Tank Agitator		3.8	5
253010	CHU	040	Cu Concentrate Pressure Filter Discharge Chute			
253010	CHU	042	Pb Concentrate Pressure Filter Discharge Chute			
253010	CHU	043	Zn Concentrate Pressure Filter Discharge Chute			
253010	CNV	007	Cu Concentrate Discharge Conveyor	600W X 5000L	3.8	5
253010	CNV	008	Pb Concentrate Discharge Conveyor	600W X 5000L	3.8	5
253010	CNV	009	Zn Concentrate Discharge Conveyor	600W X 5000L	3.8	5
253010	COL	007	Lead Bagging Area Dust Collector	2500 cfm c/w fan	6	8
253010	CRN	004	Dewatering Area Crane	15t/3t	0	0
253010	DRV	003	Cu Conc. 1 Thickener Drive Unit		5.0	7
253010	DRV	004	Cu Conc. 2 Thickener Drive Unit		5.0	7
253010	DRV	005	Pb Thickener Drive Unit		5.0	7
253010	DRV	006	Zn Concentrate Thickener Drive Unit		5.0	7
253010	DRV	007	Pyrite Concentrate Thickener Drive Unit		5.0	7
253010	EQP	006	Cu Conc. 1 Thickener Rake Lift		0.5	1
253010	EQP	007	Cu Conc. 2 Thickener Rake Lift		0.5	1
253010	EQP	008	Pb Concentrate Thickener Rake Lift		0.5	1
253010	EQP	009	Zn Concentrate Thickener Rake Lift		0.6	1
253010	EQP	010	Pyrite Concentrate Thickener Rake Lift		0.8	1
253010	FIL	003	Cu Concentrate Pressure Filter c/w motor		5.5	7
253010	FIL	004	Pb Concentrate Pressure Filter c/w motor		2.2	3
253010	FIL	005	Zn Concentrate Pressure Filter c/w motor		5.5	7
253010	PSL	022	Cu Conc. 1 Thickener Underflow No. 1	50 X 38	3.7	5
253010	PSL	023	Cu Conc. 1 Thickener Underflow No. 2	50 X 38	3.7	5.0
253010	PSL	024	Cu Conc. 1 Thickener Overflow	75 X 50	5.6	8
253010	PSL	25	Thickener Overflow Pump (Spare)	75 X 50	5.6	8
253010	PSL	026	Cu Conc. 2 Thickener Underflow Pump No. 1	50 X 38	3.7	5
253010	PSL	027	Cu Conc. 2 Thickener Underflow Pump No.2	50 X 38	3.7	5.0
253010	PSL	030	Pb Concentrate Thickener Underflow Pump No.1	50 X 38	3.7	5
253010	PSL	031	Pb Concentrate Thickener Underflow Pump No.2	50 X 38	3.7	5.0
253010	PSL	032	Zn Concentrate Thickener Overflow Pump	75 X 50	5.6	8
253010	PSL	034	Zn Concentrate Thickener Underflow Pump No.1	50 X 38	3.7	5
253010	PSL	035	Zn Concentrate Thickener Underflow Pump No.2	50 X 38	3.7	5
253010	PSL	036	Pyrite Concentrate Thickener Underflow Pump No.1	50 X 38	15	20
253010	PSL	037	Pyrite Concentrate Thickener Underflow Pump No.2	50 X 38	15	20
253010	PSL	038	Pyrite Concentrate Thickener Overflow	50 X 38	3.7	5
253010	PSL	040	Cu Conc. 1 Filter Feed Pump No.1	50 X 38	22.4	30
253010	PSL	041	Cu Conc. 1 Filter Feed Pump No.2	50 X 38	22.4	30
253010	PSL	042	Cu Conc. 2 Filter Feed Pump No.1	50 X 38	22.4	30
253010	PSL	043	Cu Conc. 2 Filter Feed Pump No.2	50 X 38	22.4	30
253010	PSL	044	Pb Concentrate Filter Feed Pump No.1	50 X 38	22.4	30
253010	PSL	045	Pb Concentrate Filter Feed Pump No.2	50 X 38	22.4	30
253010	PSL	046	Zn Concentrate Filter Feed Pump No.1	50 X 38	22.4	30
253010	PSL	047	Zn Concentrate Filter Feed Pump No.2	50 X 38	22.4	30
253010	PSL	76	Filter Feed Pump (Spare) No. 1	50 X 38	22.4	30
253010	PSL	77	Filter Feed Pump (Spare) No. 2	50 X 38	22.4	30
253010	PSL	084	Pb Concentrate Thickener Overflow Pump	75 X 50	5.6	8

253010	PSL	086	Pyrite Concentrate Thickener Underflow Pump No.3	50 X 38	15	20
253010	PSU	018	Cu Concentrate 2 Dewatering Area Sump Pump	50 mm	5.6	7.5
253010	PSU	008	Cu Concentrate 1 Dewatering Area Sump Pump	50 mm	5.6	7.5
253010	PSU	009	Pb Concentrate Dewatering Area Sump Pump	50 mm	5.6	7.5
253010	PSU	010	Zn Concentrate Dewatering Area Sump Pump	50 mm	5.6	7.5
253010	PSU	011	Cu Concentrate Loadout Area Sump Pump	50 mm	11	15
253010	PSU	012	Pb Concentrate Loadout Area Sump Pump	50 mm	11	15
253010	PSU	013	Zn Concentrate Loadout Area Sump Pump	50 mm	11	15
253010	PSU	029	Py Concentrate Dewatering Area Sump Pump	50 mm	5.6	7.5
253010	SYS	003	Cu Bagging System	c/w hopper, load cell weigh	3.0	4
253010	SYS	004	Pb Bagging System	module, feeder, bag loadout system,	3.0	4
253010	SYS	005	Zn Bagging System	control center	3.0	4
253010	THK	001	Cu Conc. 1 Thickener (High Rate)	4,000 D		
253010	THK	002	Cu Conc. 2 Thickener (High Rate)	4,000 D		
253010	THK	003	Pb Concentrate Thickener (High Rate)	3,000 D		
253010	THK	004	Zn Concentrate Thickener (High Rate)	6,500 D		
253010	THK	005	Pyrite Concentrate Thickener (High Rate)	12,000 D		
253010	TNK	020	Cu Conc. 1 Thickener O/F Standpipe	500 D x 1250 H, 0.2 tonnes		
253010	TNK	021	Cu Conc. 2 Thickener O/F Standpipe	750 D x 1000 H, 0.26 tonnes		
253010	TNK	022	Zn Conc. Thickener O/F Standpipe	750 D x 1250 H, 0.3 tonnes		
253010	TNK	023	Cu Conc. 1 Stock Tank	3000D X 4000H, 4.09 tonne		
253010	TNK	024	Cu Conc. 2 Stock Tank	3000D X 3000H, 2.85 tonne		
253010	TNK	025	Pb Concentrate Stock Tank	2750D X 2750H, 2.29 tonne		
253010	TNK	026	Zn Concentrate Stock Tank	4500D X 4500H, 6.06 tonne		
253010	TNK	059	Py Conc. Thickener O/F Standpipe	750 D x 1250 H, 0.3 tonnes		
253010	TNK	060	Pb Conc.Thickener O/F Standpipe	500 D x 1250 H, 0.2 tonnes		
254010						
254010	SYS	020	Reagent Mixing Systems	c/w tanks, pumps, agitators, controls		
254010	AGI	011	Flocculant Mixing Tank Agitator		2.2	2.9
254010	AGI	012	Flocculant Holding Tank Agitator		0.7	0.9
254010	AGI	013	CuSO4 Mixing Tank Agitator		0.2	0.3
254010	AGI	014	ZnSO4 Mixing Tank Agitator		0.2	0.3
254010	AGI	015	PAX Mixing Tank Agitator		0.2	0.3
254010	AGI	016	NaCN Mixing Tank Agitator		0.2	0.3
254010	AGI	019	Test Reagent Mixing Tank Agitator		0.2	0.3
254010	BRK	002	ZnSO4 Bag Breaker		1.0	1.3
254010	BRK	003	CuSO4 Bag Breaker		1.0	1.3
254010	BRK	004	NaOH/NaCN Bag Breaker		1.0	1.3
254010	BRK	005	Flocculant Bag Breaker		1.0	1.3
254010	BRK	006	PAX Bag Breaker		1.0	1.3
254010	BRK	007	Test Reagent Bag Breaker		1.0	1.3
254010	BRK	008	Lime Bag Breaker		1.0	1.3
254010	CRN	005	Reagent Area Trolley	2t	3.5	4.7
254010	EQP	011	Flocculant Water Eductor		3.5	4.7
254010	FAN	010	Reagent Area Exhaust Fan No. 1		0.8	1.1
254010	FAN	011	Reagent Area Exhaust Fan No. 2		0.8	1.1
254010	FDR	003	Flocculant Screw Feeder		0.4	0.5
254010	FDR	004	CuSO4 Screw Feeder		3.7	5.0
254010	FDR	005	ZnSO4 Screw Feeder		3.7	5.0
254010	FDR	006	PAX Screw Feeder		1.5	2.0
254010	FDR	007	NaOH/NaCN Screw Feeder (enclosed)		1.5	2.0
254010	FDR	008	Test Reagent Screw Feeder		1.5	2.0
254010	FDR	014	Lime Screw Feeder		1.5	2.0
254010	HPR	002	Flocculant Hopper			
254010	HPR	003	ZnSO4 Hopper			
254010	HPR	004	CuSO4 Hopper			
254010	HPR	005	PAX Hopper			
254010	HPR	006	NaOH/NaCN Hopper			
254010	HPR	007	Test Reagent Hopper			
254010	HPR	011	Lime Hopper			
254010	PMT	001	Flocculant Metering Pump No.1		1.0	1.3
254010	PMT	002	Flocculant Metering Pump No.2		1.0	1.3
254010	PMT	003	Flocculant Metering Pump No.3		1.0	1.3
254010	PMT	004	Flocculant Metering Pump No.4		1.0	1.3
254010	PMT	005	Flocculant Metering Pump No.5		1.0	1.3
254010	PMT	006	Flocculant Metering Pump No.6		1.0	1.3
254010	PMT	007	CuSO4 Metering Pump No.1		1.0	1.3
254010	PMT	008	CuSO4 Metering Pump No.2		1.0	1.3
254010	PMT	009	ZnSO4 Metering Pump No.1		1.0	1.3
254010	PMT	010	ZnSO4 Metering Pump No.2		1.0	1.3
254010	PMT	011	MIBC Metering Pump No.1		1.0	1.3
254010	PMT	012	MIBC Metering Pump No.2		1.0	1.3
254010	PMT	013	MIBC Metering Pump No.3		1.0	1.3
254010	PMT	014	MIBC Metering Pump No.4		1.0	1.3
254010	PMT	015	MIBC Metering Pump No.5		1.0	1.3
254010	PMT	016	PAX Metering Pump No.1		1.0	1.3
254010	PMT	017	PAX Metering Pump No.2		1.0	1.3

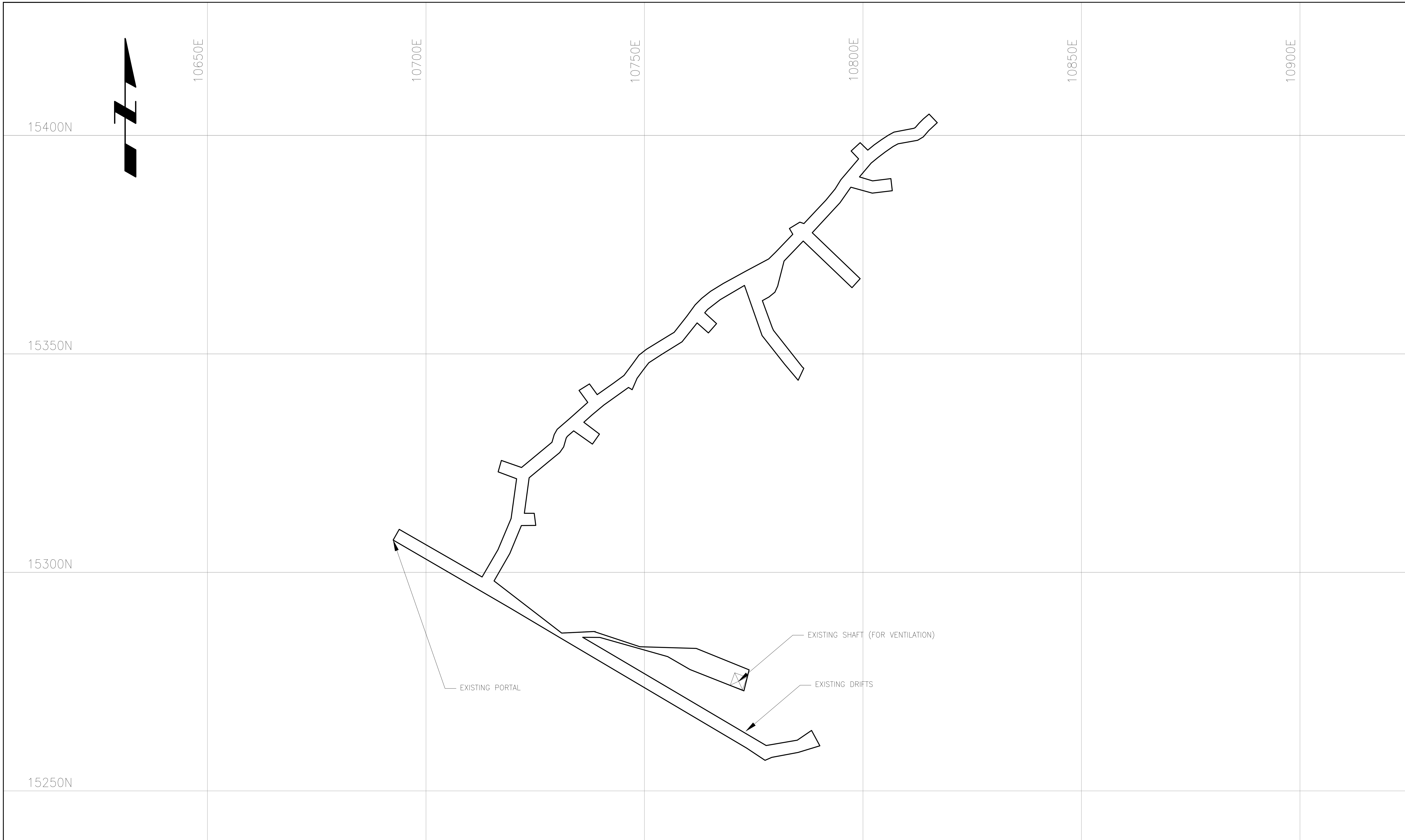
254010	PMT	018	NaCN Metering Pump No.1		1.0	1.3
254010	PMT	019	NaCN Metering Pump No.2		1.0	1.3
254010	PMT	022	Na2S2O5 Metering Pump No.1		1.0	1.3
254010	PMT	023	Na2S2O5 Metering Pump No.2		1.0	1.3
254010	PMT	024	Reagent A7021 Metering Pump No.1		1.0	1.3
254010	PMT	025	Reagent A7021 Metering Pump No.2		1.0	1.3
254010	PMT	027	Reagent A3418 Metering Pump No.1		1.0	1.3
254010	PMT	028	Reagent A3418 Metering Pump No.2		1.0	1.3
254010	PMT	031	Reagent A9810 Metering Pump No.1		1.0	1.3
254010	PMT	032	Reagent A9810 Metering Pump No.2		1.0	1.3
254010	PMT	034	Test Reagent Metering Pump No.1		1.0	1.3
254010	PMT	035	Test Reagent Metering Pump No.2		1.0	1.3
254010	PMT	036	Test Reagent Transfer Metering Pump		1.0	1.3
254010	PSU	014	CuSO4 Area Sump Pump	50 mm	3.8	5.0
254010	PSU	015	Liquid Reagents Areas Sump Pump - to Pyrite Thickener	50 mm	11	15
254010	PSU	016	PAX Area Sump Pump	50 mm	3.7	5.0
254010	PSU	017	NaCN Area Sump Pump	50 mm	3.7	5.0
254010	PSU	020	Flocculant Area Sump Pump	50 mm	3.7	5.0
254010	PSU	021	Na2S2O5 Area Sump Pump	50 mm	3.7	5.0
254010	PSU	023	Test Reagent Area Sump Pump	50 mm	3.7	5.0
254010	PSU	038	ZnS04 Area Sump Pump	50 mm	3.7	5.0
254010	SSW	002	Reagent Area Safety Shower No.1			
254010	SSW	005	NaCN Area Safety Shower			
254010	SYS	006	Flocculant System			
254010	TNK	027	Flocculant Mixing Tank	1000D X 1000H		
254010	TNK	028	Flocculant Holding Tank	1500D X 2500H		
254010	TNK	029	CuSO4 Mixing Tank	670D X 670H X670L		
254010	TNK	030	CuSO4 Holding Tank	730D X 730H X730L		
254010	TNK	031	ZnSO4 Mixing Tank	670D X 670H X670L		
254010	TNK	032	ZnSO4 Holding Tank	730D X 730H X730L		
254010	TNK	034	PAX Mixing Tank	640D X 640H X640L		
254010	TNK	035	PAX Holding Tank	700D X 700H X700L		
254010	TNK	036	NaCN Mixing Tank	820D X820H X820L		
254010	TNK	037	NaCN Holding Tank	850D X 850H X850L		
254010	TNK	038	Na2S2O5 Mixing Tank	850D X 850H X850L		
254010	TNK	039	Na2S2O5 Holding Tank	910D X 910H X 910L		
254010	TNK	044	Test Reagent Mixing Tank	640D X 640H X640L		
254010	TNK	045	Test Reagent Holding Tank	700D X 700H X700L		
254010	PSU	026	Lime Area Sump Pump	50 mm	3.7	5.0
254010	TNK	048	Lime Mixing Tank	1000D X 1000H		
254010	AGI	022	Lime Mixing Tank Agitator		3.0	4.0
254010	PSL	059	Lime System Feed Pump No.1	38 X 25	5.5	7.4
254010	PSL	060	Lime System Feed Pump No.2	38 X 25	5.5	7.4
255010						
255010	DRV	008	Final Tailings Thickener Drive Unit		7.5	10.1
255010	EQP	015	Tailings Thickener Rake Lift		0.8	1.1
255010	PBX	005	Final Tailings Thickener Underflow Pumpbox	1 m3, .42 tonne		
255010	PON	001	Final Tailings Pond			
255010	PSL	061	Tailings Thickener Underflow Pump No.1	75 X 50	5.6	7.5
255010	PSL	062	Tailings Thickener Underflow Pump No.2	75 X 50	5.6	7.5
255010	PSL	063	Tailings Thickener Overflow Pump No.1	50 X 38	5.6	7.5
255010	PSL	064	Tailings Thickener Overflow Pump No.2	50 X 38	5.6	7.5
255010	PSL	065	Final Tailings Pond Feed Pump No.1	50 X 38	18.6	24.9
255010	PSL	066	Final Tailings Pond Feed Pump No.2	50 X 38	18.6	24.9
255010	PSL	067	Final Tailings Pond Feed Pump No.3 (stby)	50 X 38	18.6	24.9
255010	PSL	071	Drainage Pond Submersible Pump	200 mm	55.0	73.7
255010	PSU	027	Final Tailings Area Sump Pump	50 mm	11	15
255010	THK	006	Final Tailings Thickener	14000D		
255010	TNK	049	Final Tailings Thickener Overflow Standpipe	1000 D x 2000 H, 0.6 tonnes		
255020						
255020	MIL	004	Limestone Ball Mill	1500D X 2400L	45	60
255020	PSU	025	Limestone Area Sump Pump	50 mm	3.7	5.0
255020	BIN	009	Limestone Ball Mill Ball Bin	1W X 1L X 1H		
255020	CHU	048	Limestone Ball Mill Discharge Chute			
255020	CNV	012	Limestone Ball Mill Feed Conveyor	600W X 16200L	3.0	4.0
255020	SCB	006	Limestone Ball Mill Feed Conveyor Belt Scale		0.56	0.75
255020	CHU	047	Limestone Ball Mill Feed Conveyor Head Chute			
255020	CHU	050	Limestone Belt Feeder Feed Chute	38 X 25		
255020	CHU	051	Limestone Belt Feeder Head Chute	3000D X 3700H		
255020	HPR	010	Limestone Conveyor Feed Hopper		0.56	0.75
255020	PSL	055	Limestone Feed Pump No.1	40 X 40	2.2	3.0
255020	PSL	056	Limestone Feed Pump No.2	40 X 40	2.2	3.0
255020	LUB	004	Limestone Mill Lube Unit		0.1	0.1
255020	BIN	008	Limestone Mill Trash Bin			
255020	TNK	047	Limestone Mixing Tank	2000D X 2000H		
255020	AGI	021	Limestone Mixing Tank Agitator		0.75	1.0
255020	PBX	003	Limestone Pumpbox	0.50m3		

255020	PSL	053	Limestone Transfer Pump No.1	40 X 40	2.2	3.0
255020	PSL	054	Limestone Transfer Pump No.2	40 X 40	2.2	3.0
255020	FDR	012	Limestone Vibratory Feeder		2.2	3.0
255020	EQP	012	Limestone Crushing Fogging Control Panel No.1		0.1	0.1
255020	EQP	013	Limestone Crushing Fogging Control Panel No.2		0.1	0.1
255020	EQP	014	Limestone Crushing Fogging Control Panel No.3		0.1	0.1
256020						
256020	AIC	005	Process Plant Air Compressor No.1	172 cfm@200 psig	38.0	50.9
256020	AIC	006	Process Plant Air Compressor No.2	172 cfm@200 psig	38.0	50.9
256020	AIC	008	Process Plant Air Compressor No.3 (stby)	172 cfm@200 psig	38.0	50.9
256020	AIR	004	Cu Filter Press Area Process Air Receiver	5m3		
256020	AIR	005	Process Plant Instrument Air Receiver	4m3		
256020	AIR	006	Process Plant Process Air Receiver	2m3		
256020	AIR	007	Limestone Grinding Air Receiver	2.5m3		
256020	AIR	008	Pb Filter Press Area Process Air Receiver	2m3		
256020	AIR	009	Zn Filter Press Area Process Air Receiver	5m3		
256020	DRY	006	Process Plant Instrument Air Dryer Dryer			
256020	FIL	007	Process Plant Air Filter No.1			
256020	FIL	008	Process Plant Air Filter No.2			
256030						
256030	BLO	003	Flotation Aeration Blower No.1 (BHP = 163/122 kW)	180 m3/min, 36Kpag	150	200
256030	BLO	004	Flotation Aeration Blower No.2	180 m3/min, 36Kpag	150	200
256030	BLO	005	Flotation Aeration Blower No.3	180 m3/min, 36Kpag	150	200
256030	BLO	006	Flotation Aeration Blower No.4	144 m3/min, 22Kpag	112	150
301010						
301010	AGI	020	Backfill Tailings Stock Tank Agitator		22.0	29.5
301010	AIR	002	Backfill Filter Receiver			
301010	BIN	007	Pneumatic Cement Container			
301010	BLO	002	Binder Blower Blower			
301010	CHU	045	Paste Mixer Discharge Chute			
301010	CHU	046	Disc Filter Discharge Chute			
301010	CNV	010	Backfill Filter Conveyor	1050W X 14200L	3.7	5.0
301010	CNV	011	Binder Weigh Belt Conveyor			
301010	COL	004	Cement Dust Collector		5.0	6.7
301010	FDR	009	Binder Screw Feeder		5.0	6.7
301010	FDR	010	Binder Rotary Feeder			
301010	FIL	006	Backfill Disc Filter			
301010	HEX	001	Vacuum Pump Seal Water Heat Exchanger		0.1	0.1
301010	HPR	008	Paste Hopper			
301010	HPR	009	Binder Hopper			
301010	MIX	002	Paste Mixer		5.0	6.7
301010	PLO	006	Emergency Flush Water Pump		525.0	703.8
301010	PSL	048	Backfill Filtrate Pump No.1		5.5	7.4
301010	PSL	049	Backfill Filtrate Pump No.2		5.5	7.4
301010	PSL	050	Backfill Tailings Disc Filter/Mixer Feed Pump No.1		5.5	7.4
301010	PSL	051	Backfill Tailings Disc Filter/Mixer Feed Pump No.2		5.5	7.4
301010	PSL	052	Backfill Disc Filter Discharge Pump		5.5	7.4
301010	PSU	024	Paste Area Sump Pump		5.5	7.4
301010	PVU	001	Backfill Filter Vacuum Pump No. 1		130.0	174.3
301010	PVU	002	Backfill Filter Vacuum Pump No. 2		130.0	174.3
301010	SCB	005	Backfill Filter Conveyor Belt Scale		0.5	0.7
301010	SCU	**001**	Binder Wet Scrubber			
301010	SEP	001	Vacuum Pump Seal Water Discharge Separator			
301010	SYS	007	Paste Backfill Pump System			
301010	TNK	046	Backfill Tailings Stock Tank	6500D X 7000H		
301010	WAS	001	Pressure Washer			
301020						
301020	SYS	009	Reclaim Water Barge (Pump No.1)	c/w TWO PUMPS	75.0	100.5
301020	SYS	010	Reclaim Water Barge (Pump No.2)		75.0	100.5
302030						
302030	PLS	072	OPAG Discharge Pump		7.6	10.2
302030	PLS	073	Historic PAG Discharge Pump		7.6	10.2
352030						
352030	ACU	011	Assay Lab Air Conditioning Unit No.1		1.5	2.0
352030	ACU	012	Assay Lab Air Conditioning Unit No.2		1.5	2.0
352030	ACU	013	Assay Lab Condensing Unit For Offices No.1		12.5	16.8
352030	ACU	014	Assay Lab Condensing Unit For Offices No.2		12.5	16.8
352030	COL	006	Assay Lab Dust Collector		0.1	0.1
352030	FAN	060	Assay Lab Exhaust Fan No.1		0.7	0.9
352030	FAN	061	Assay Lab Exhaust Fan No.2		0.7	0.9
352030	FAN	062	Assay Lab Baghouse Exhaust Fan		12.5	16.8
352030	FAN	063	Assay Lab Scrubber Exhaust Fan		15.0	20.1
352030	HEA	028	Assay Lab Electric Baseboard Heater No.1			
352030	HEA	029	Assay Lab Electric Baseboard Heater No.2			
352030	HEA	030	Assay Lab Electric Baseboard Heater No.3			
352030	HEA	031	Assay Lab Electric Baseboard Heater No.1			
352030	HEA	032	Assay Lab Electric Baseboard Heater No.1			

352030	HEA	033	Assay Lab Electric Baseboard Heater No.1			
352030	HEA	034	Assay Lab Electric Baseboard Heater No.1			
352030	HEA	035	Assay Lab Electric Baseboard Heater No.1			
352030	HEA	036	Assay Lab Electric Baseboard Heater No.1			
352030	HEA	037	Assay Lab Electric Baseboard Heater No.1			
352030	HTR	066	Assay Lab Water Heater #2		18.0	24.1
352030	MUA	007	Assay Lab Make Up Air Unit - Glycol		7.5	10.1
352030	MUA	008	Assay Lab Make Up Air Unit - Glycol		7.5	10.1
352030	PSO	027	Assay Lab Glycol Secondary Pump No.1		3.7	5.0
352030	PSO	028	Assay Lab Glycol Secondary Pump No.2		3.7	5.0
352030	PSO	029	Assay Lab Glycol Circulator		0.4	0.5
352030	PSO	030	Assay Lab Glycol Circulator		0.4	0.5
352030	SCU	002	Assay Lab Scrubber			
355010						
355010	PLO	013	Fire Water Electric Pump		230.0	308.3
355010	PLO	014	Fire Water Diesel Pump			
355010	PLO	015	Fire Water Jockey Pump			
355010	PLO	020	HP Water Pump No.1	38 X 25	18.6	24.9
355010	PLO	021	HP Water Pump No.2	38 X 25	18.6	24.9
355010	PLO	022	LP Water Pump No. 1	38 X 25	3.7	5.0
355010	TNK	050	Fresh/Fire Water Tank	10000D X 8000H		
355010	TNK	053	Fresh Water Feeder Tank	1000 D x 1500 H		

APPENDIX C

LEVEL PLANS



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:1200
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

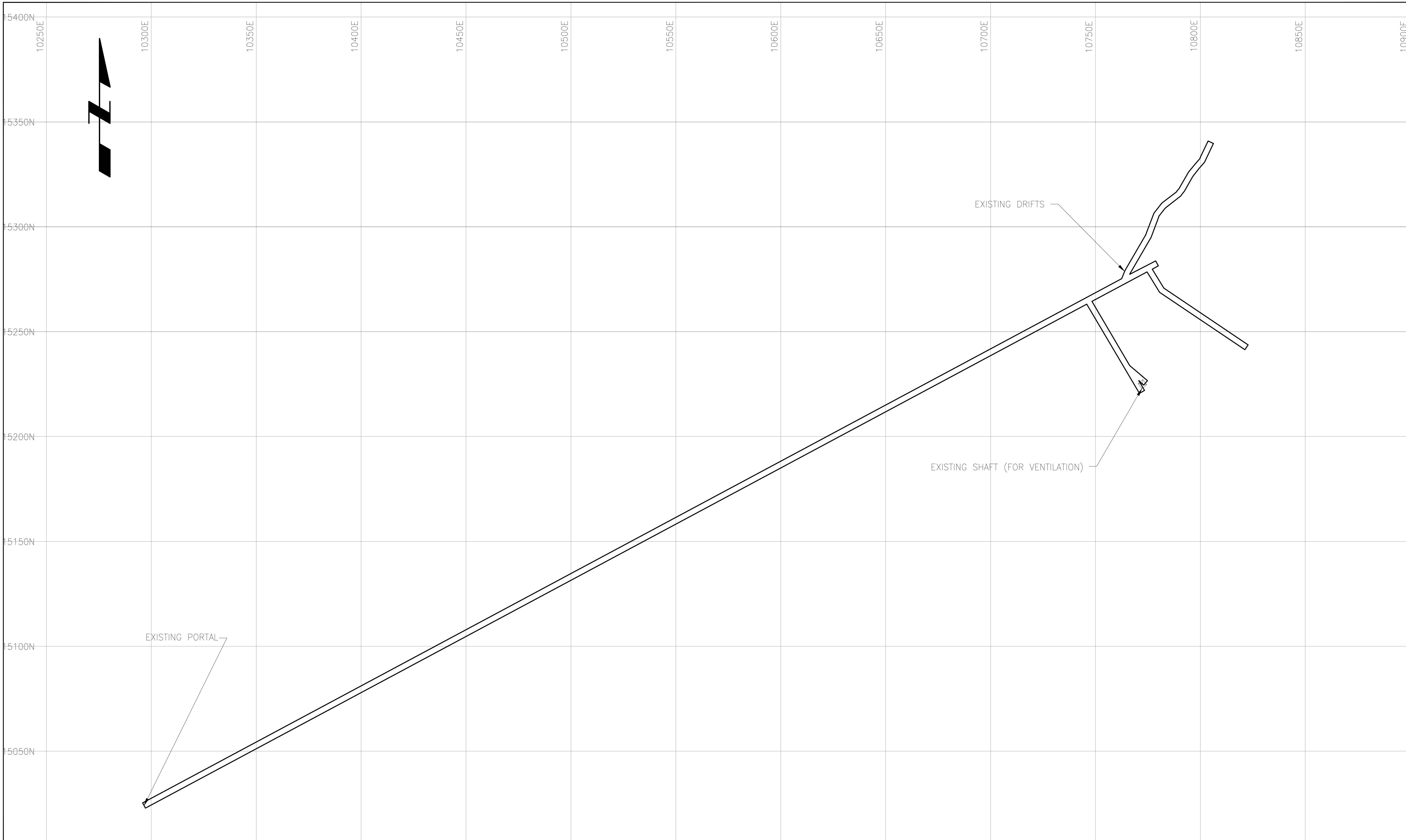
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
410m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-010
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:2000
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

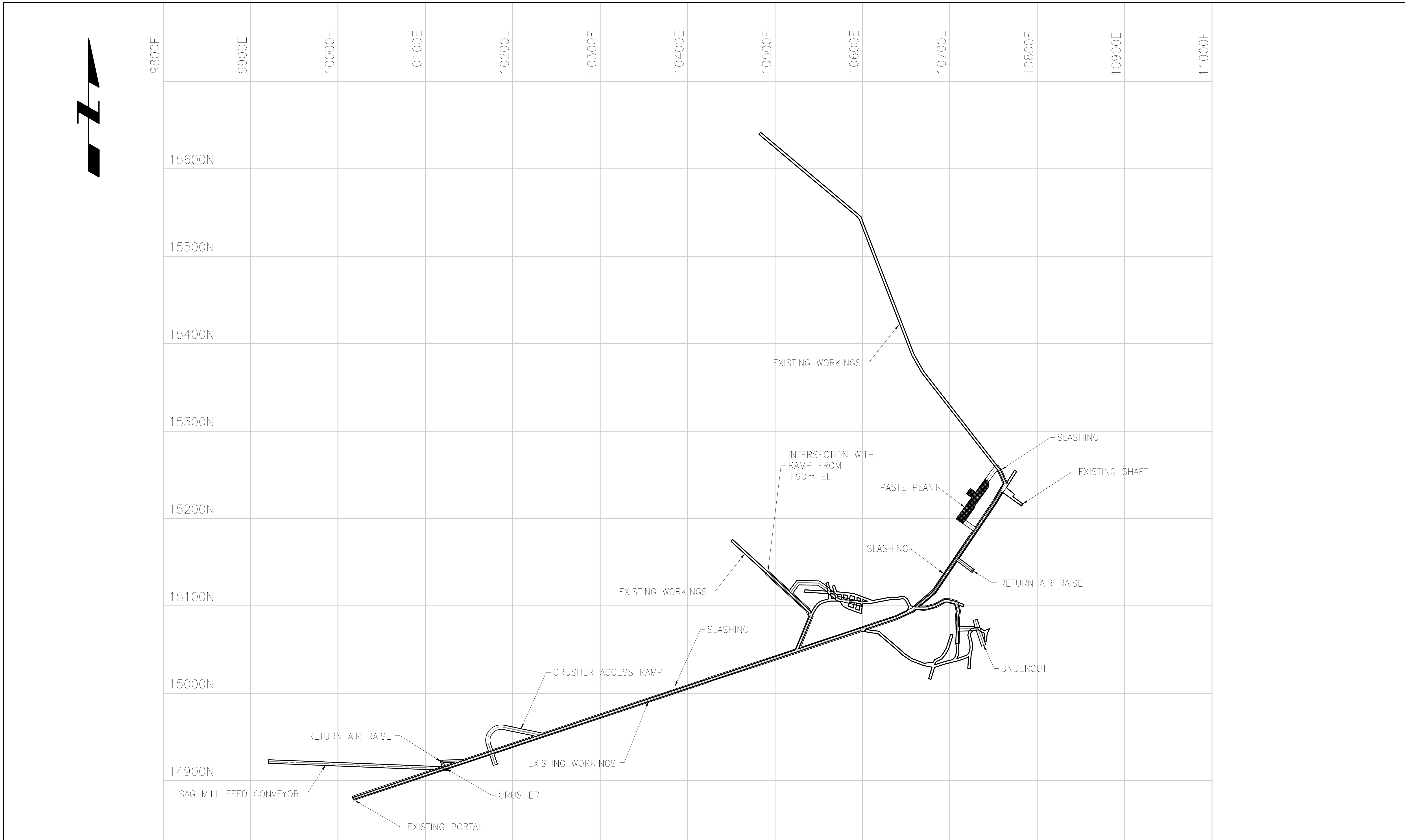
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
270m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-011
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500 DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

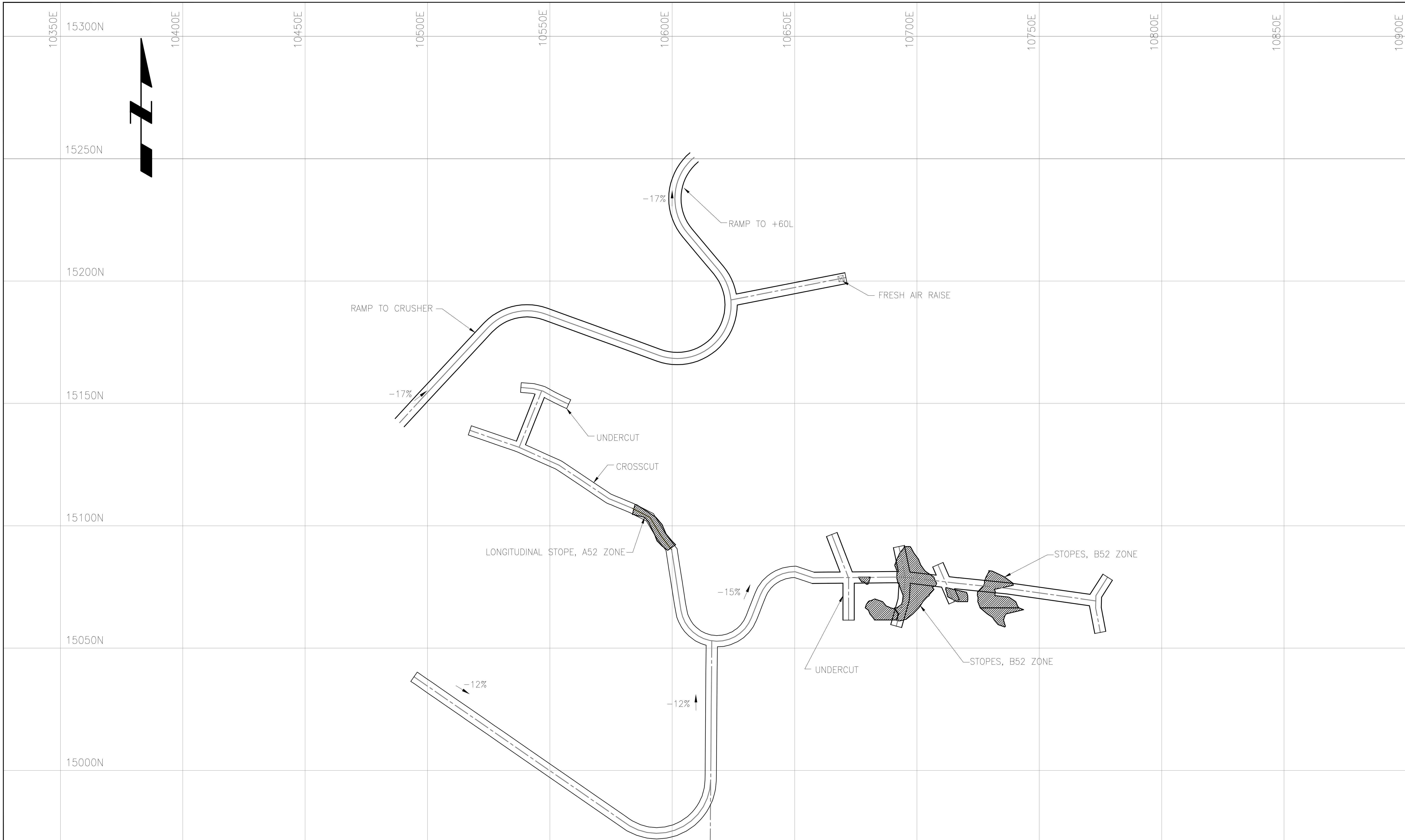
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
110m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No. | DRAWING No. 1500-10-012 | REV No. F1



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REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REFERENCE DRAWING / REF. DWG No.

SCALE: 1:4000 DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

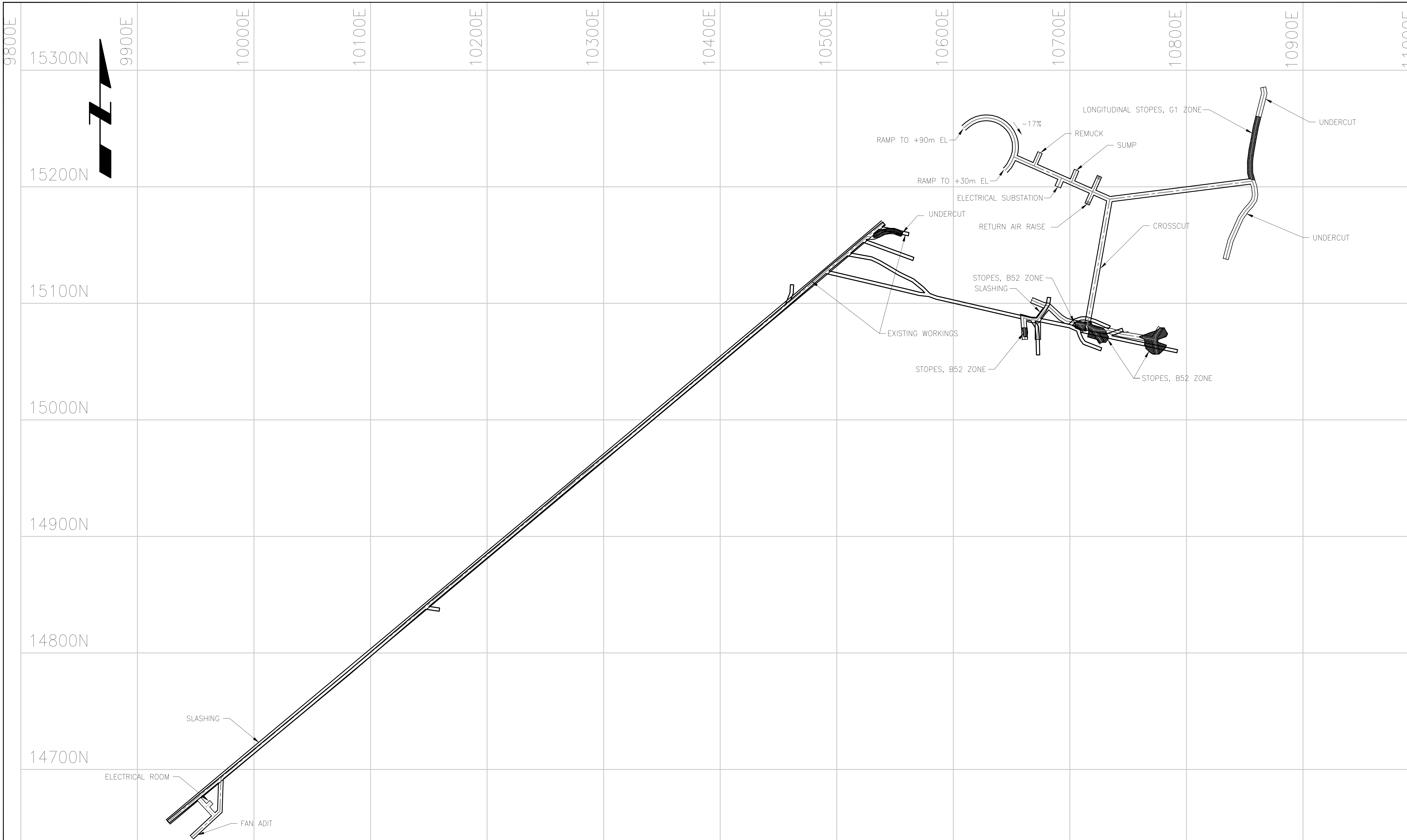
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
90m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No. | DRAWING No. 1500-10-013 | REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:1500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

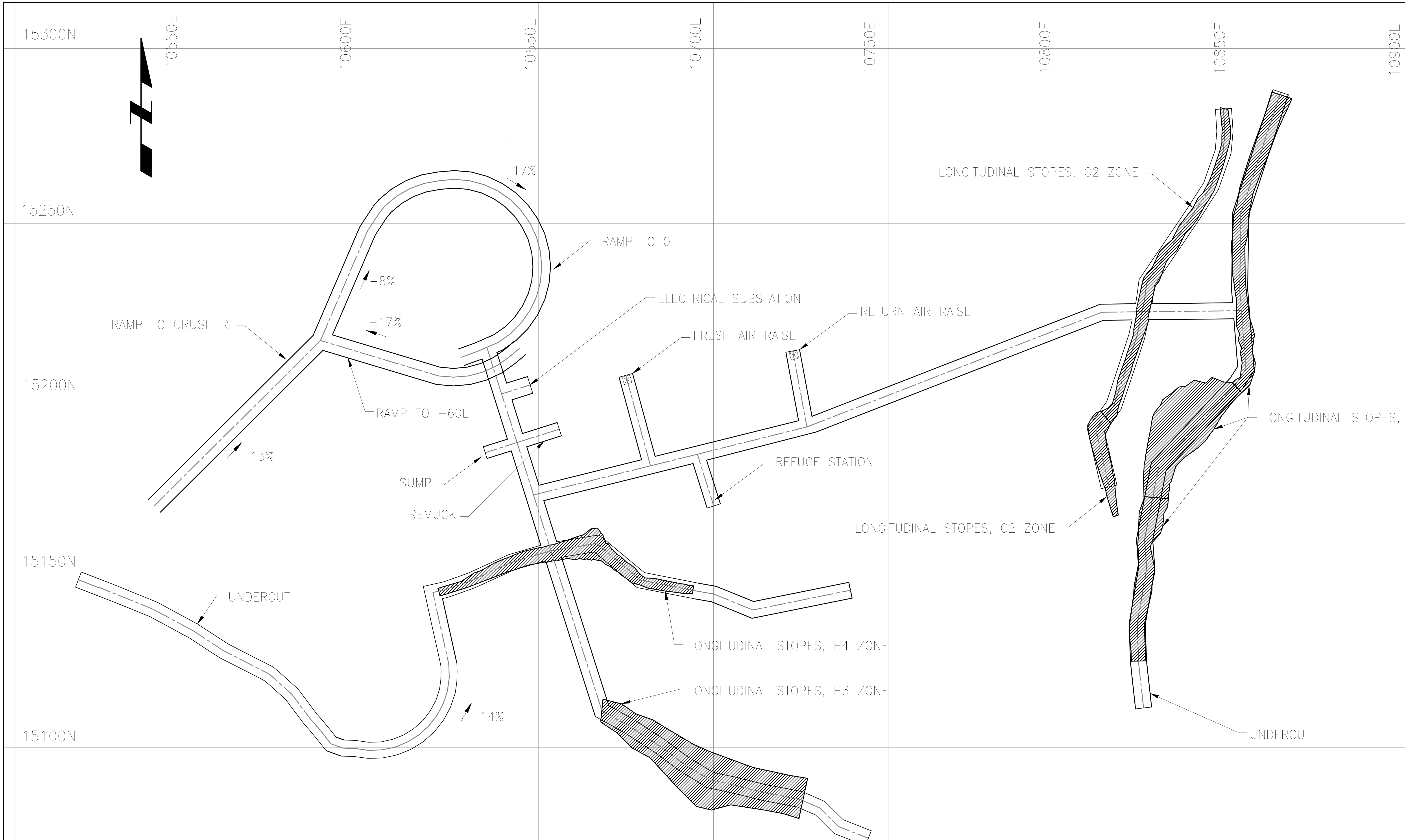
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
60m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-014
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

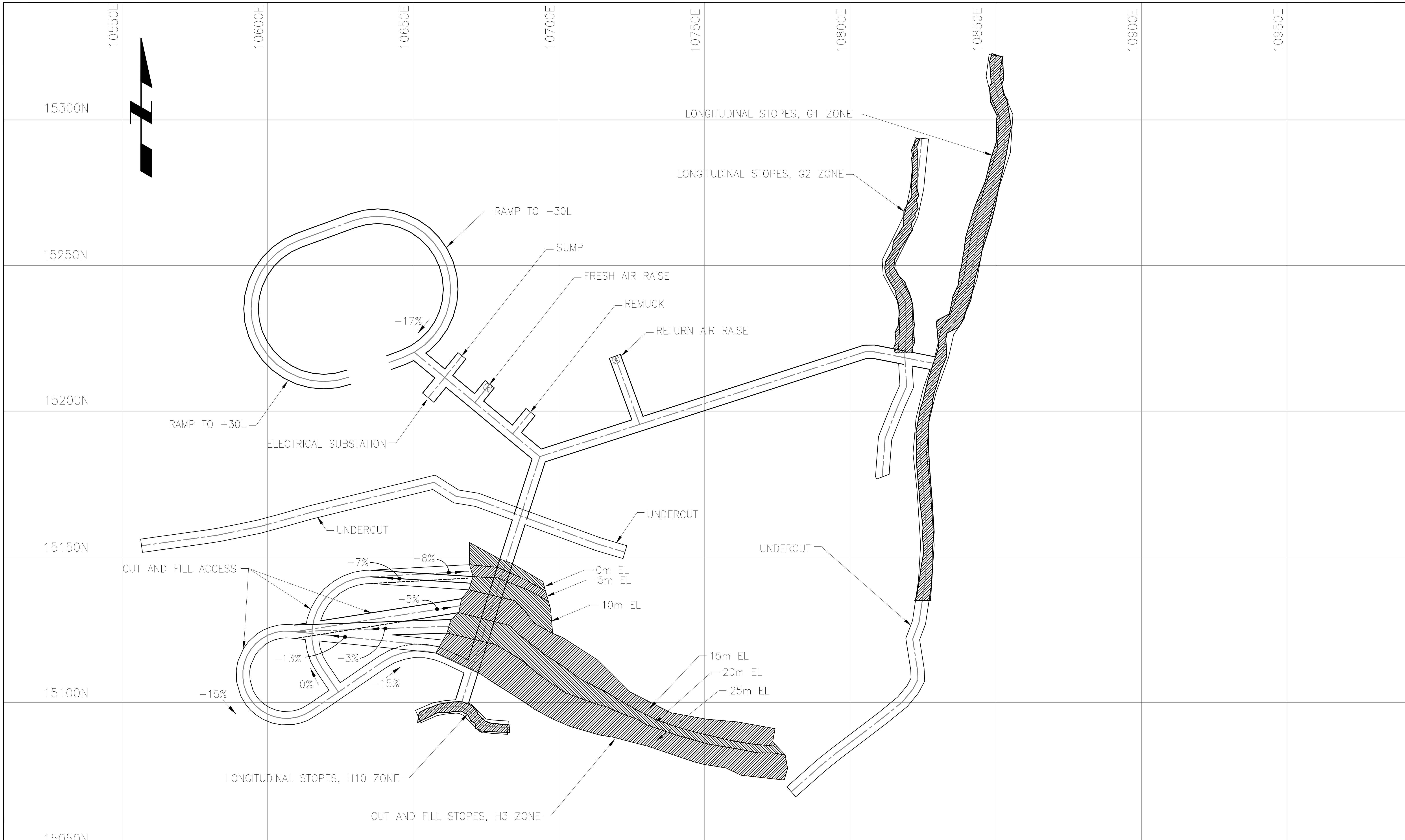
SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
30m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-015
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

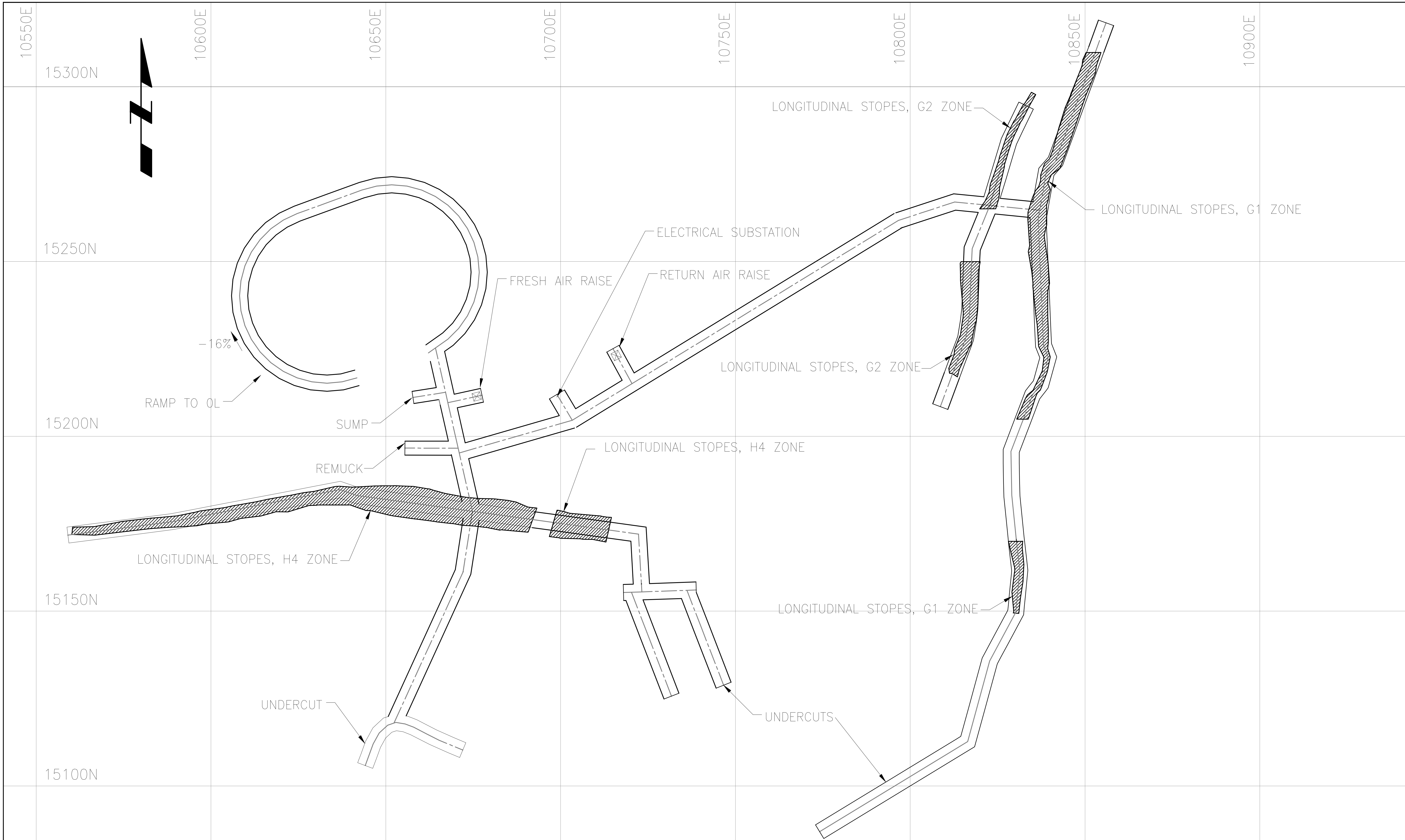
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
0m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-016
REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

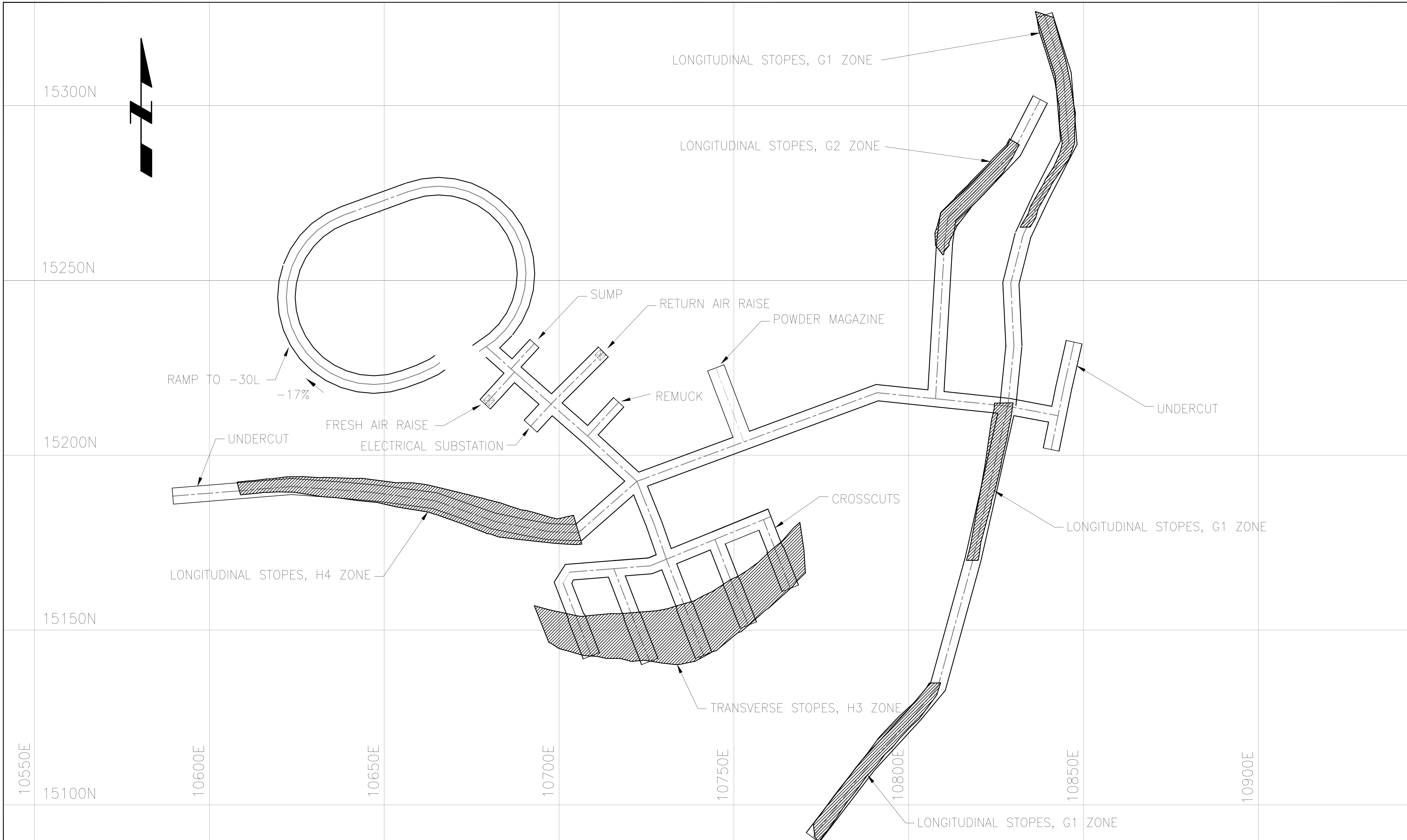
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-30m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.	DRAWING No.	REV No.
	1500-10-017	F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

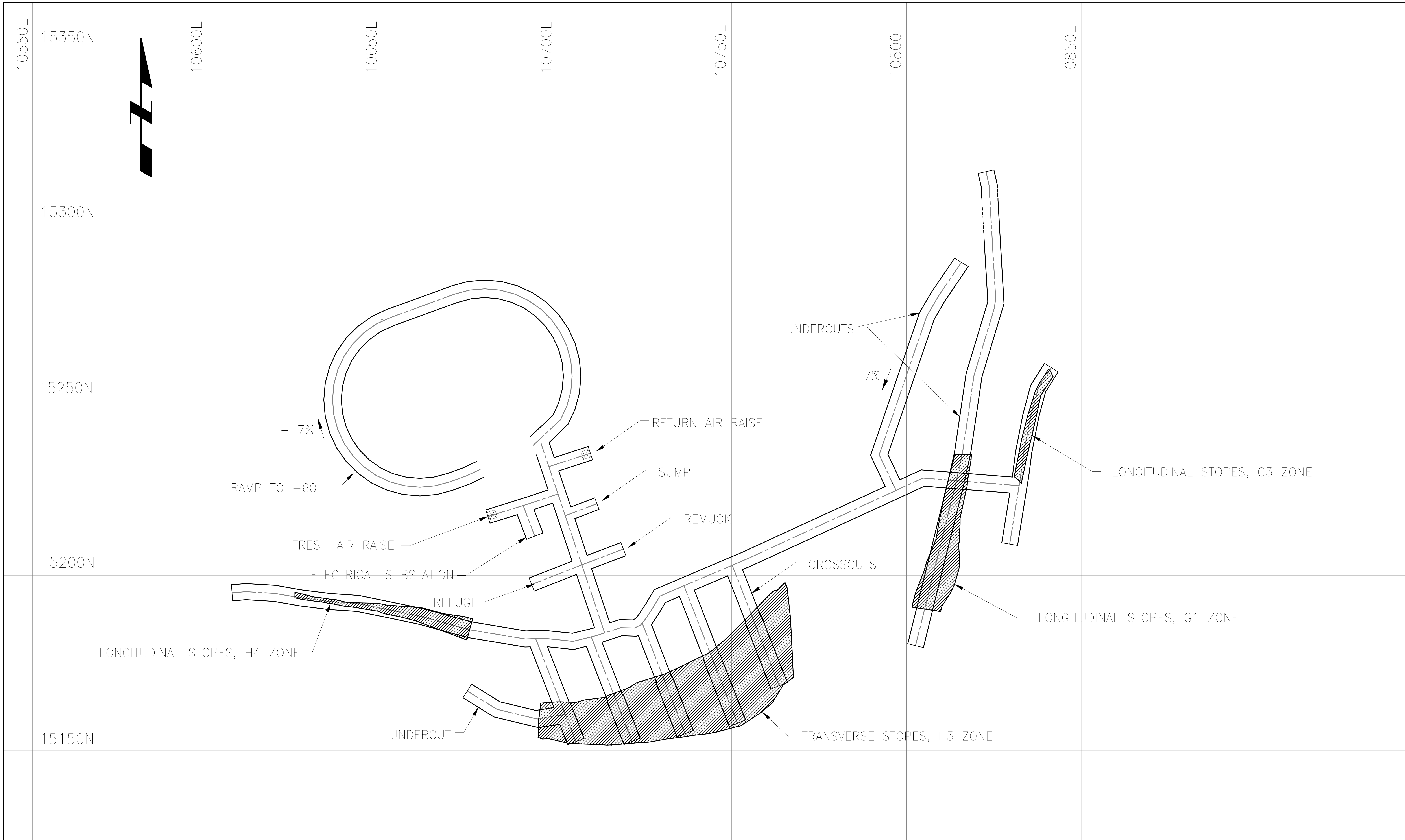
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-60m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-018
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING | REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

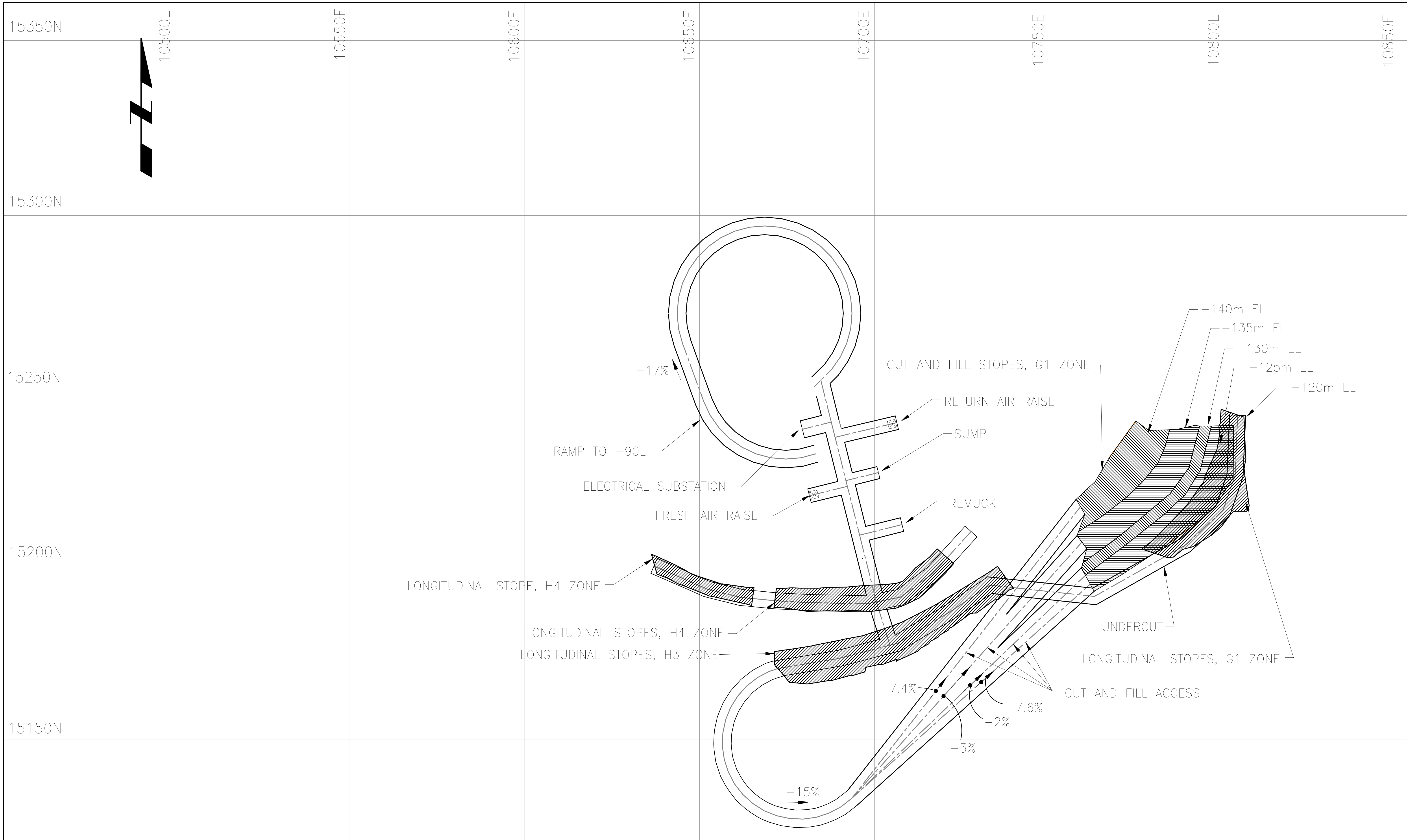
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-90m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-019
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

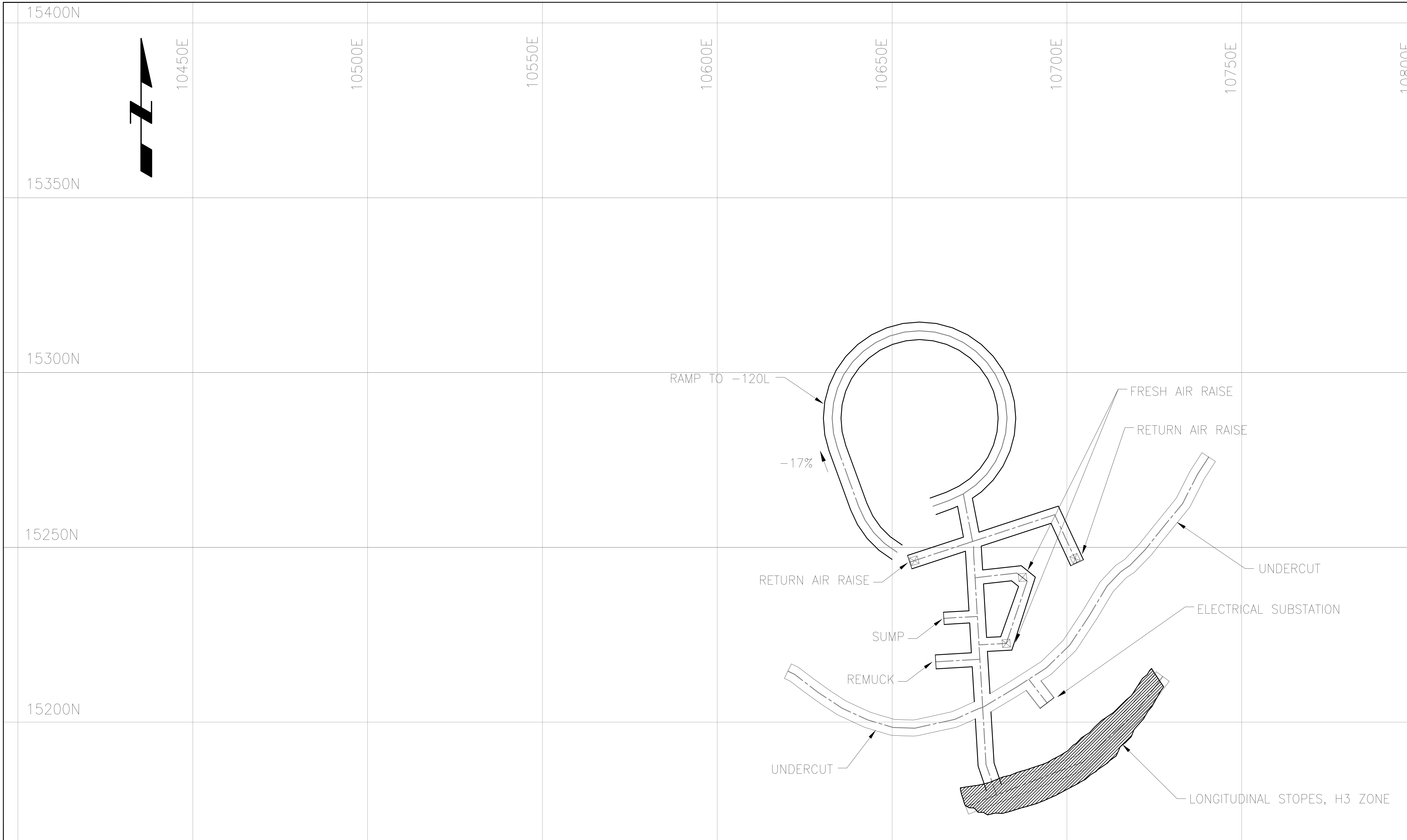
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−120m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500−10−020
REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

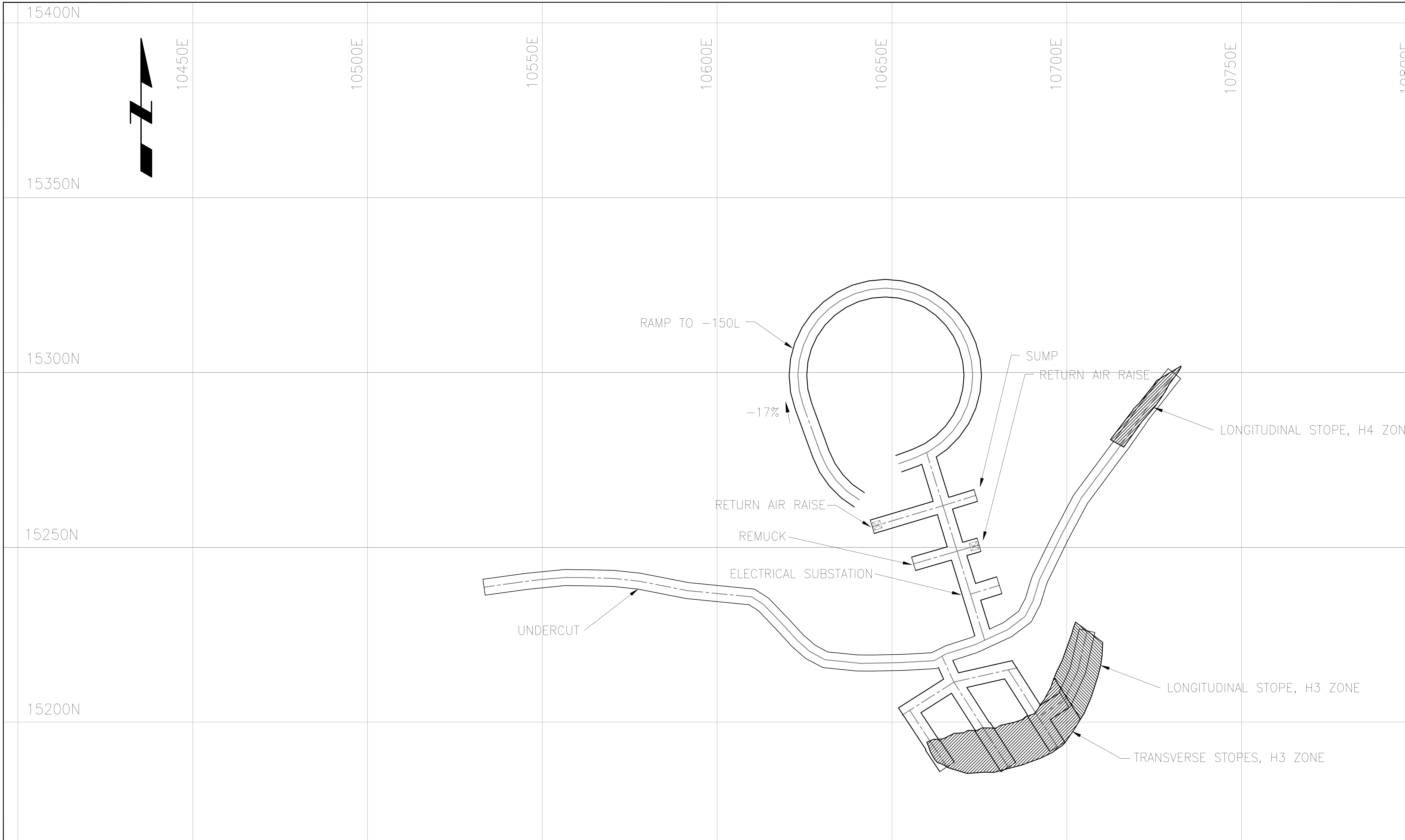
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-150m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-021
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500 DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

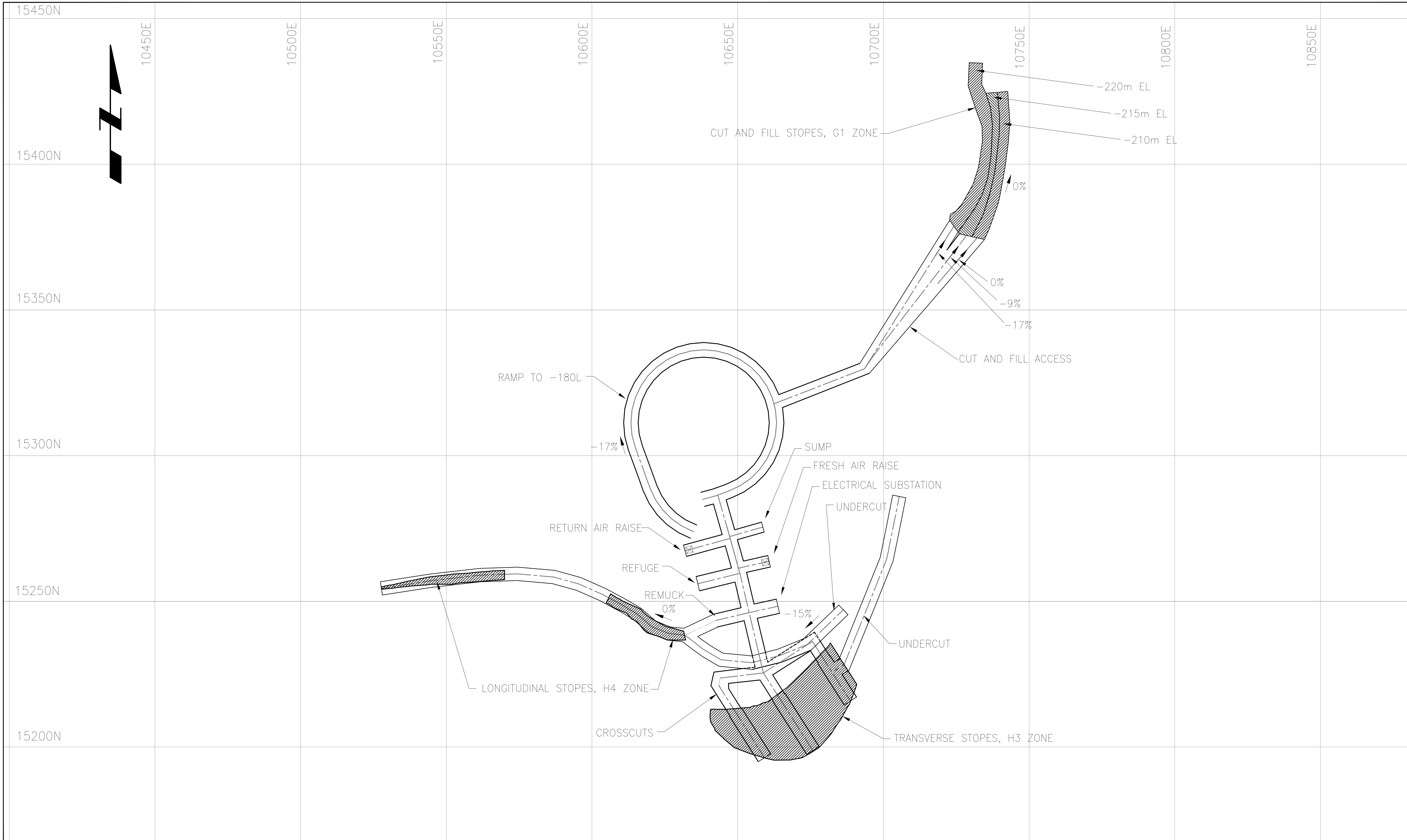
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−180m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500−10−022
REV No. F1



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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-210m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-023
REV No. F1

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE
F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

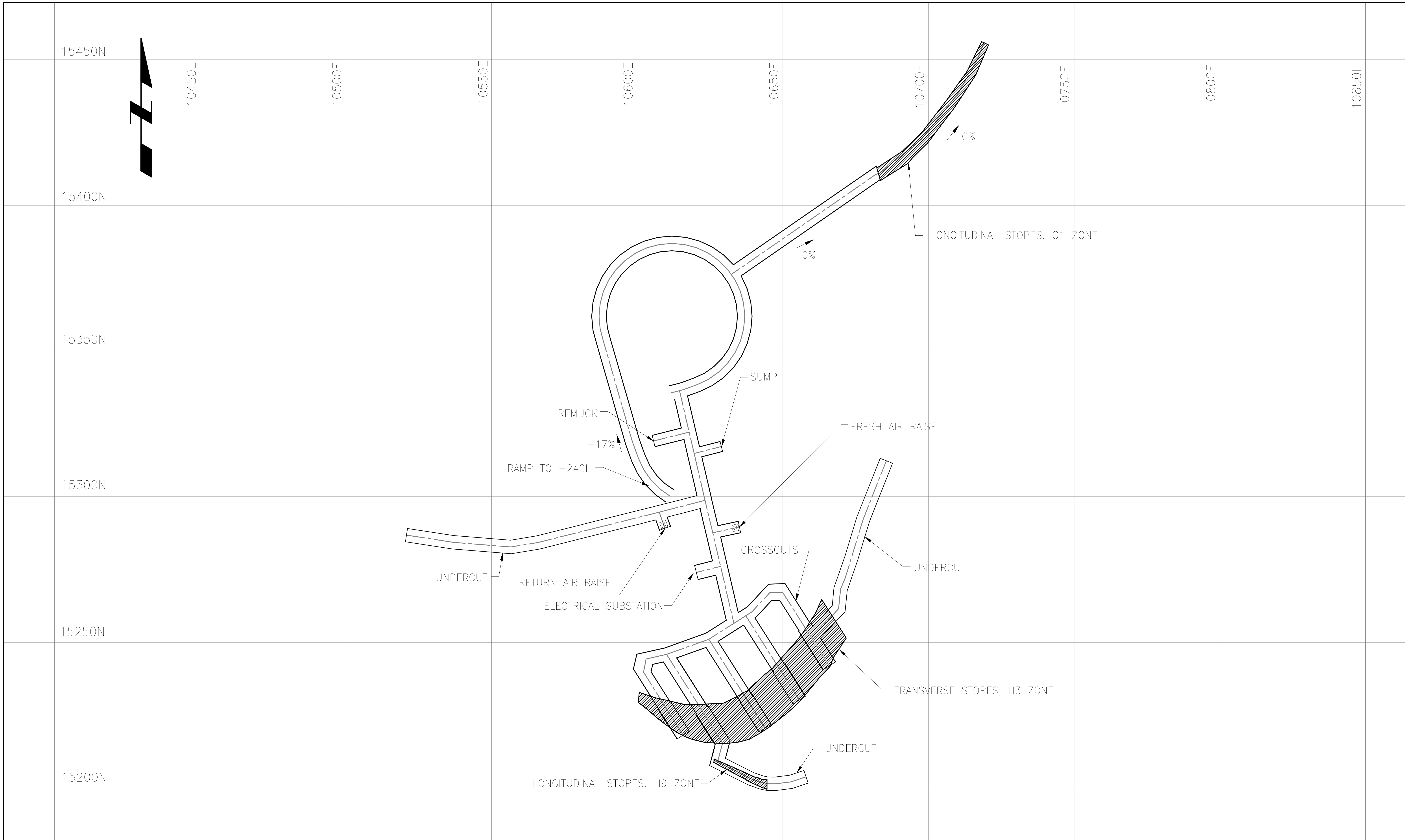
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-240m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-024
REV No. F1



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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

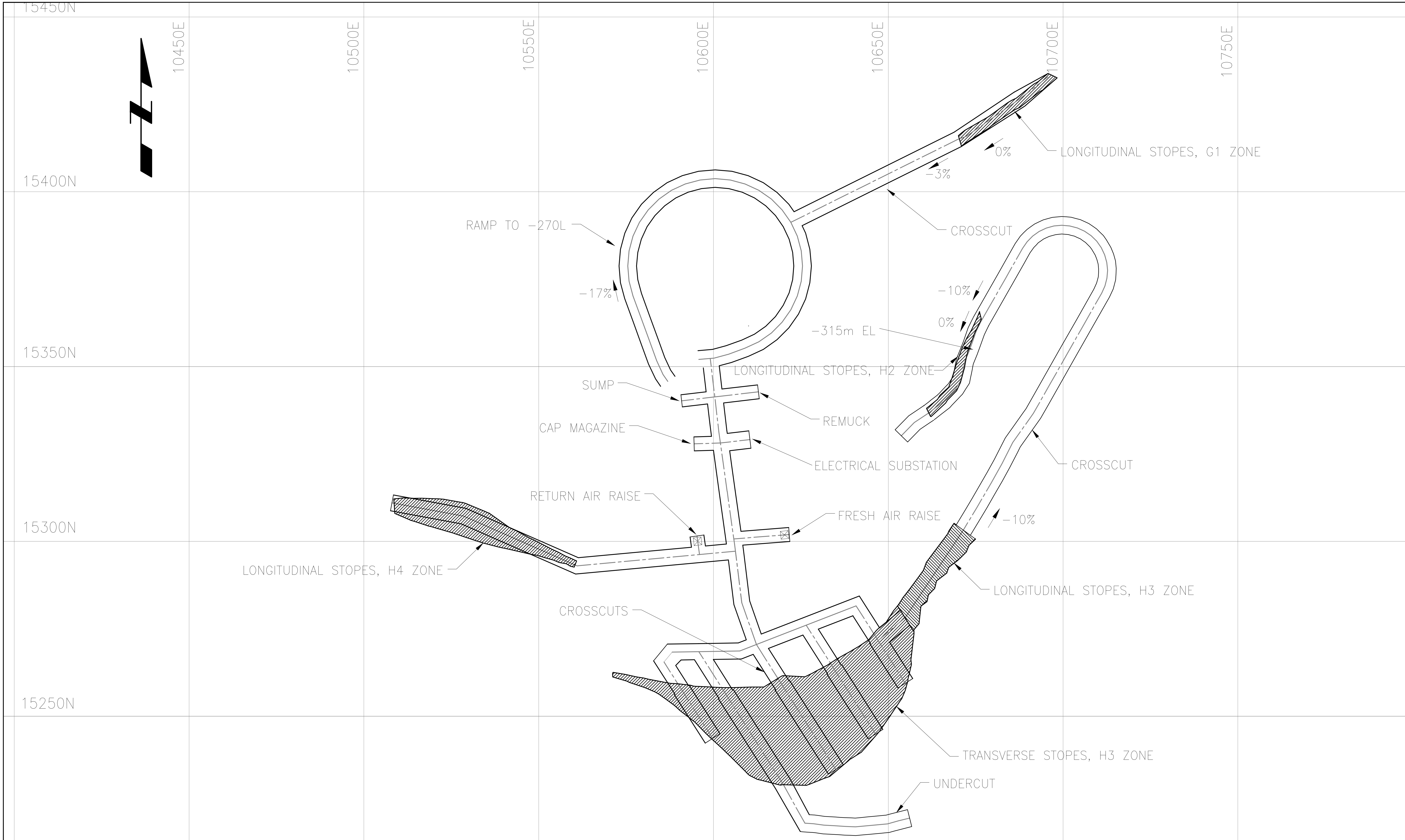
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−270m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500−10−025
REV No. F1



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE

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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

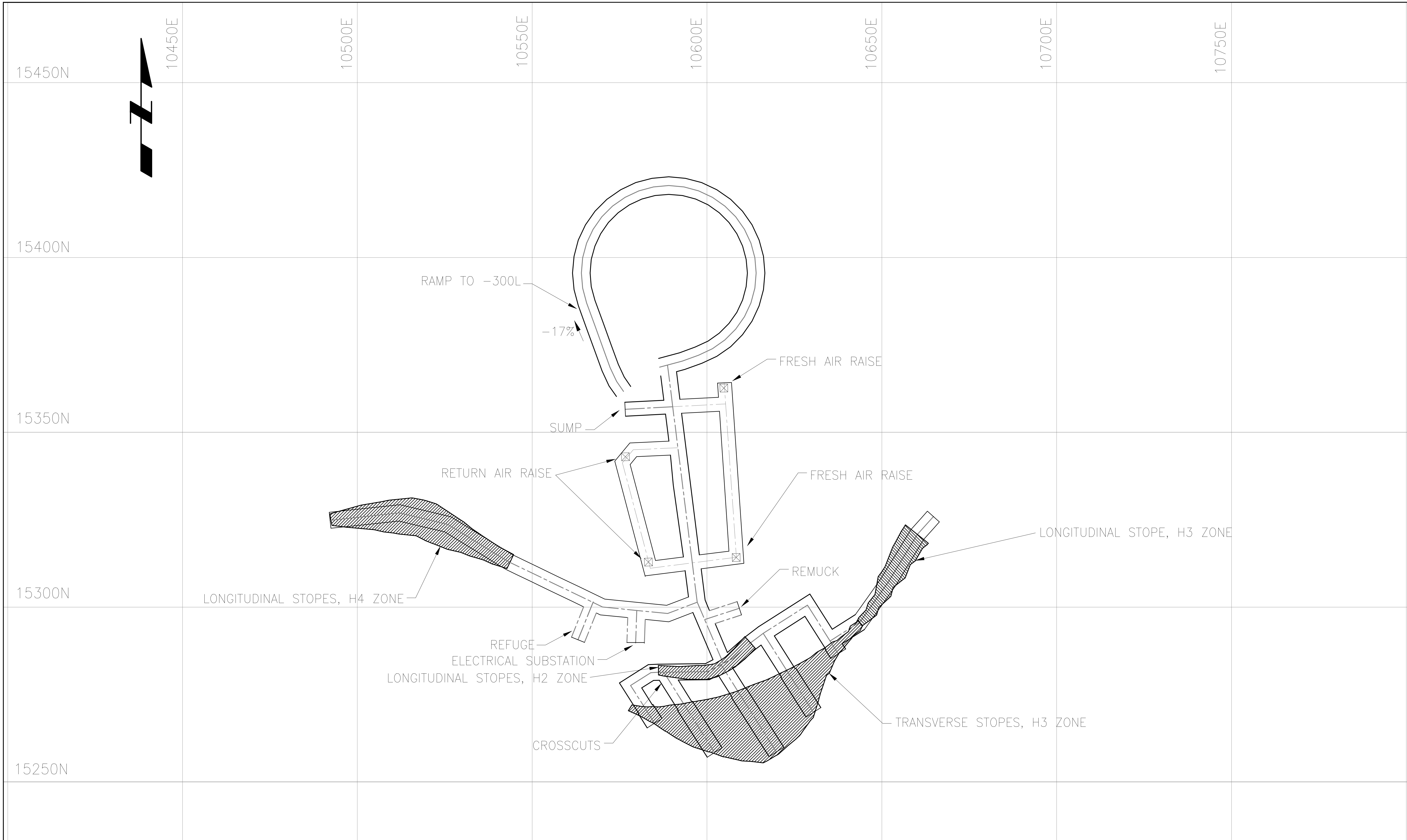
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-300m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-026
REV No. F1



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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

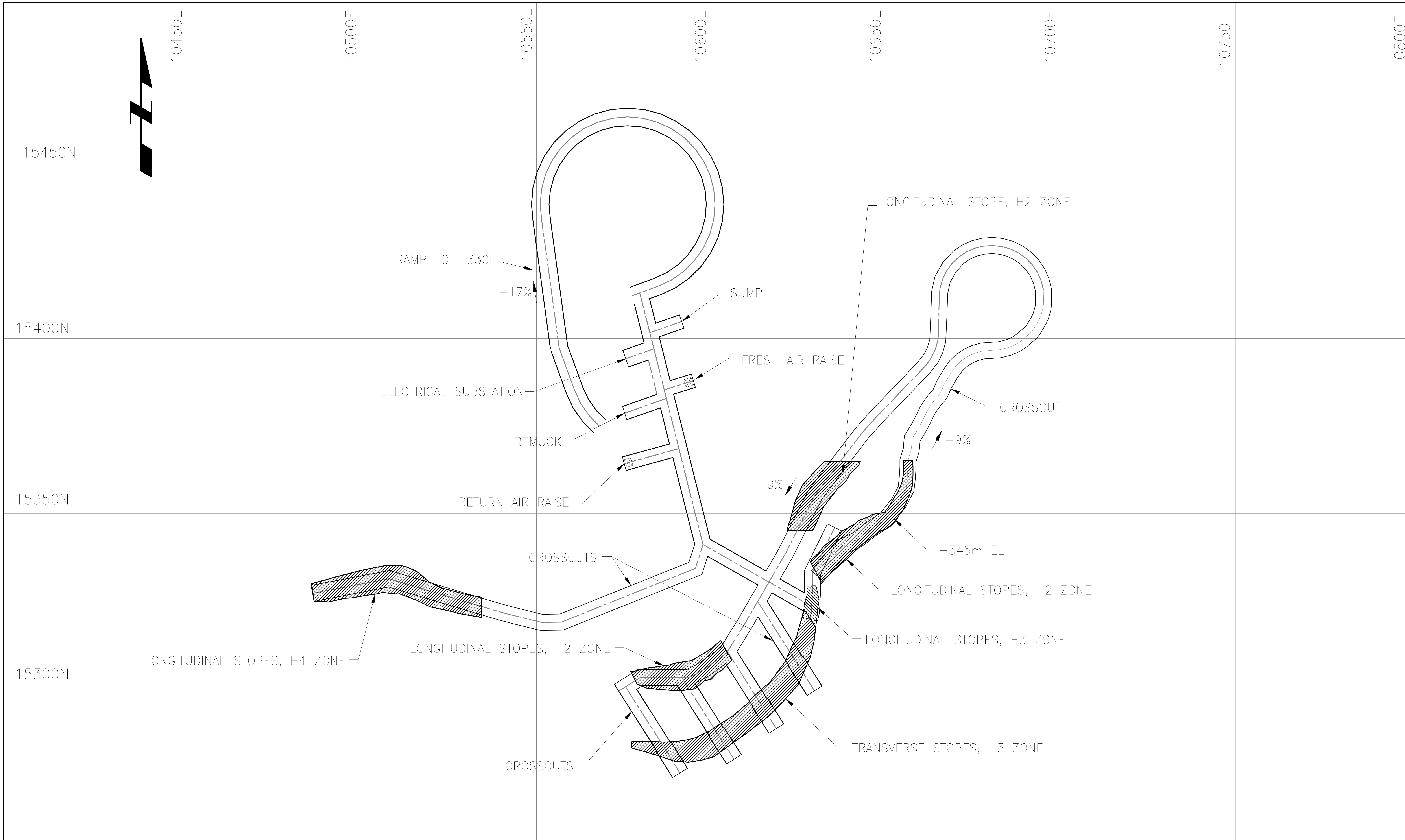
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−330m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500−10−027
REV No. F1



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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

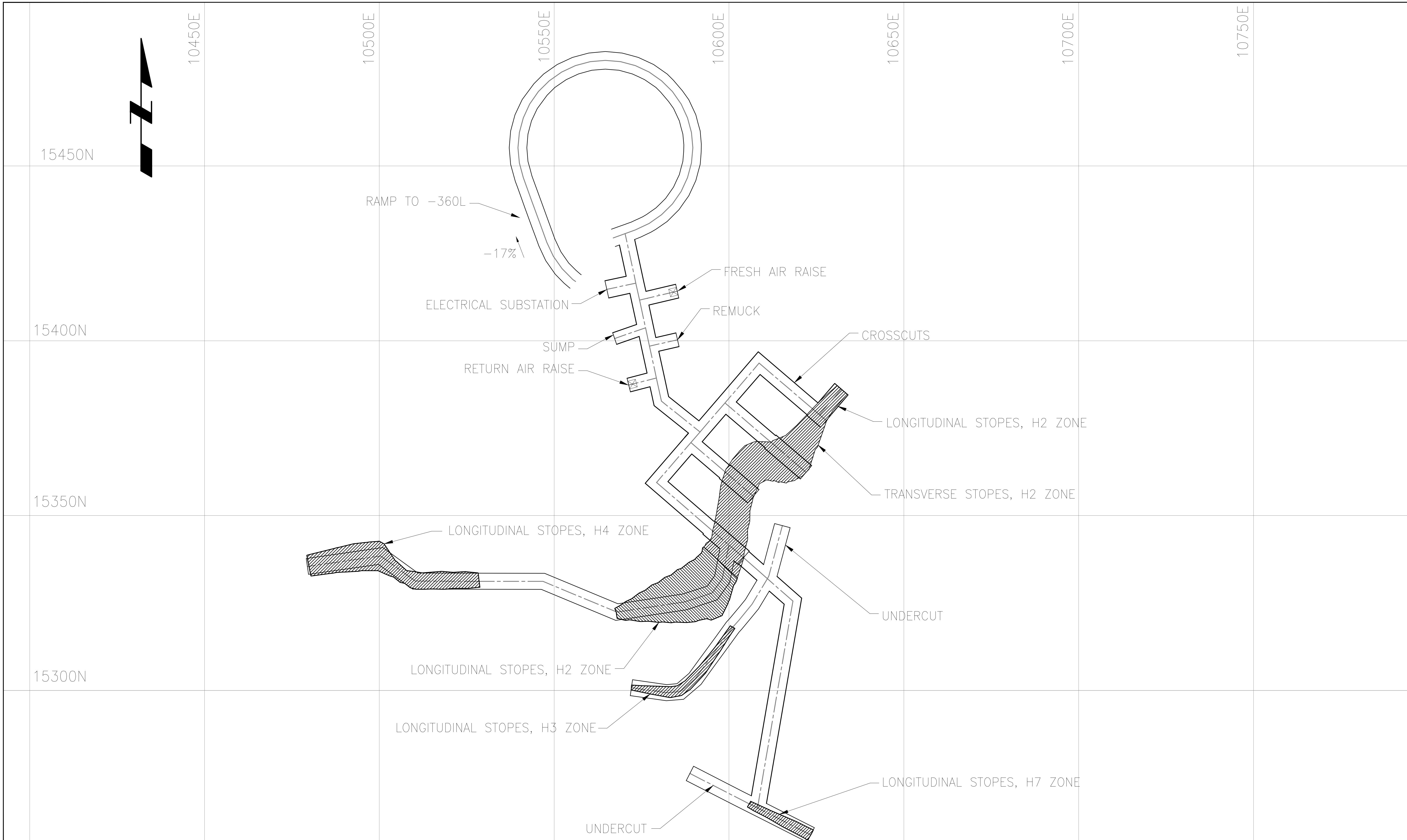
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−360m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500−10−028
REV No. F1



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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500 DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

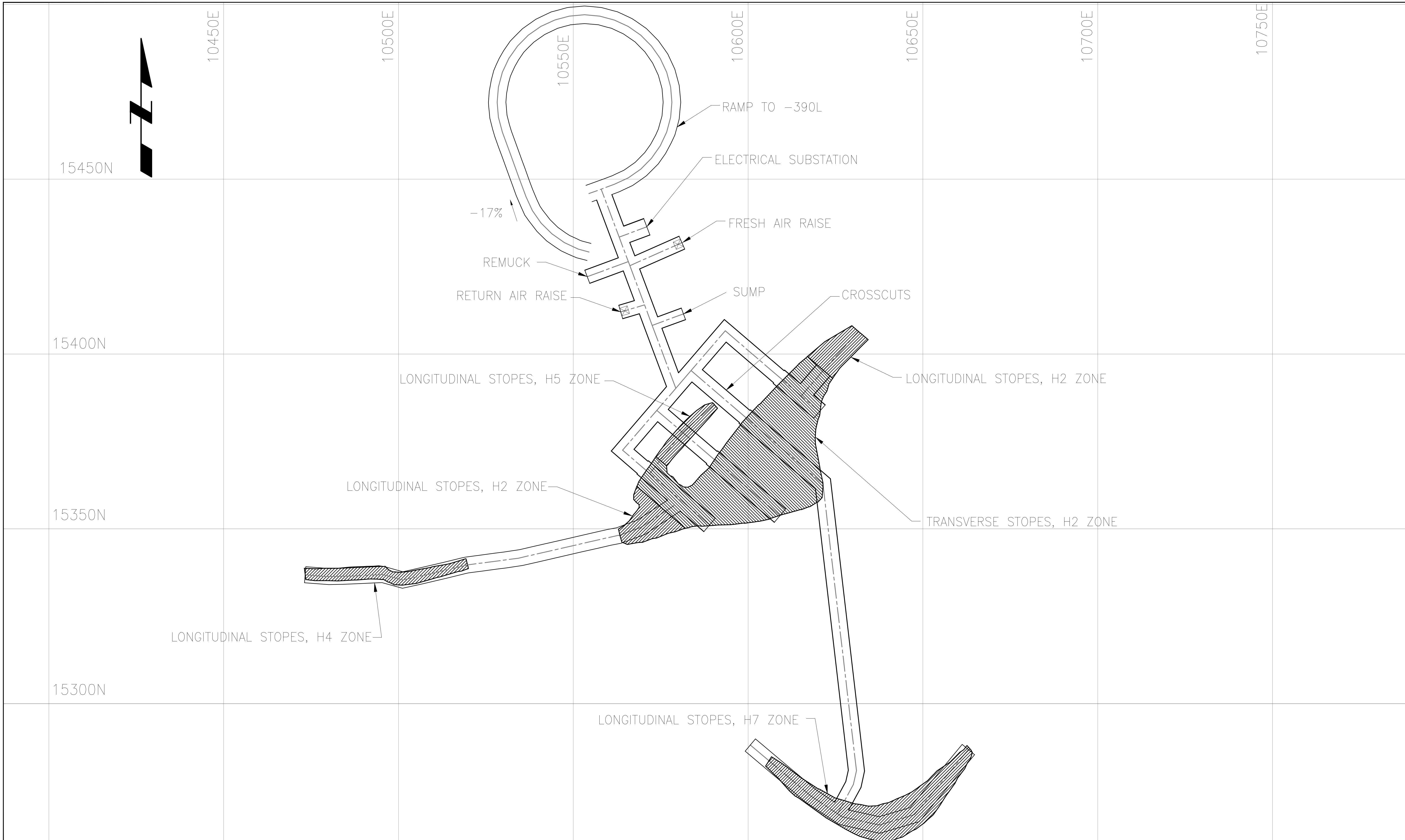
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−390m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No. | DRAWING No. 1500-10-029 | REV No. F1



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REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

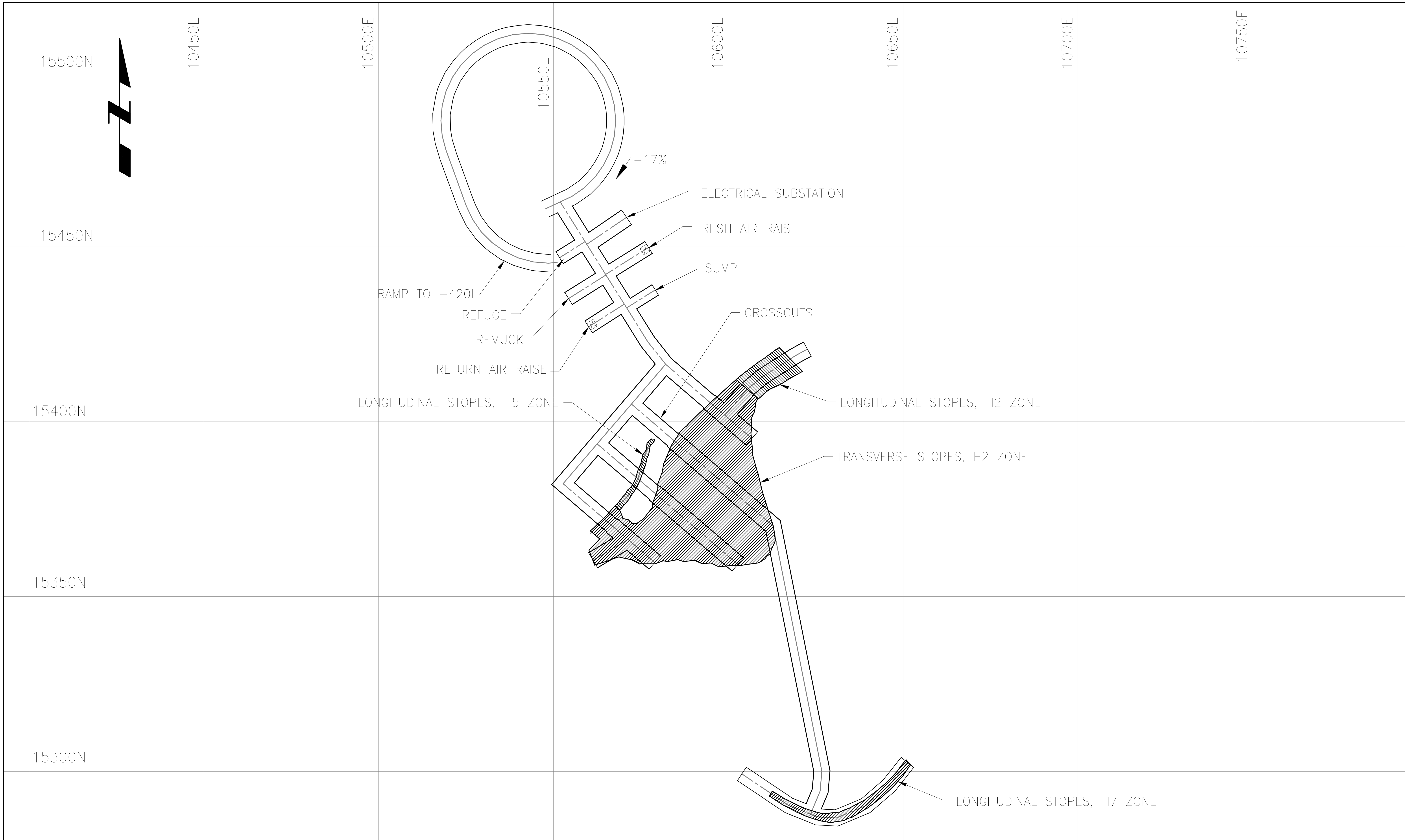
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
−420m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-030
REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:500
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

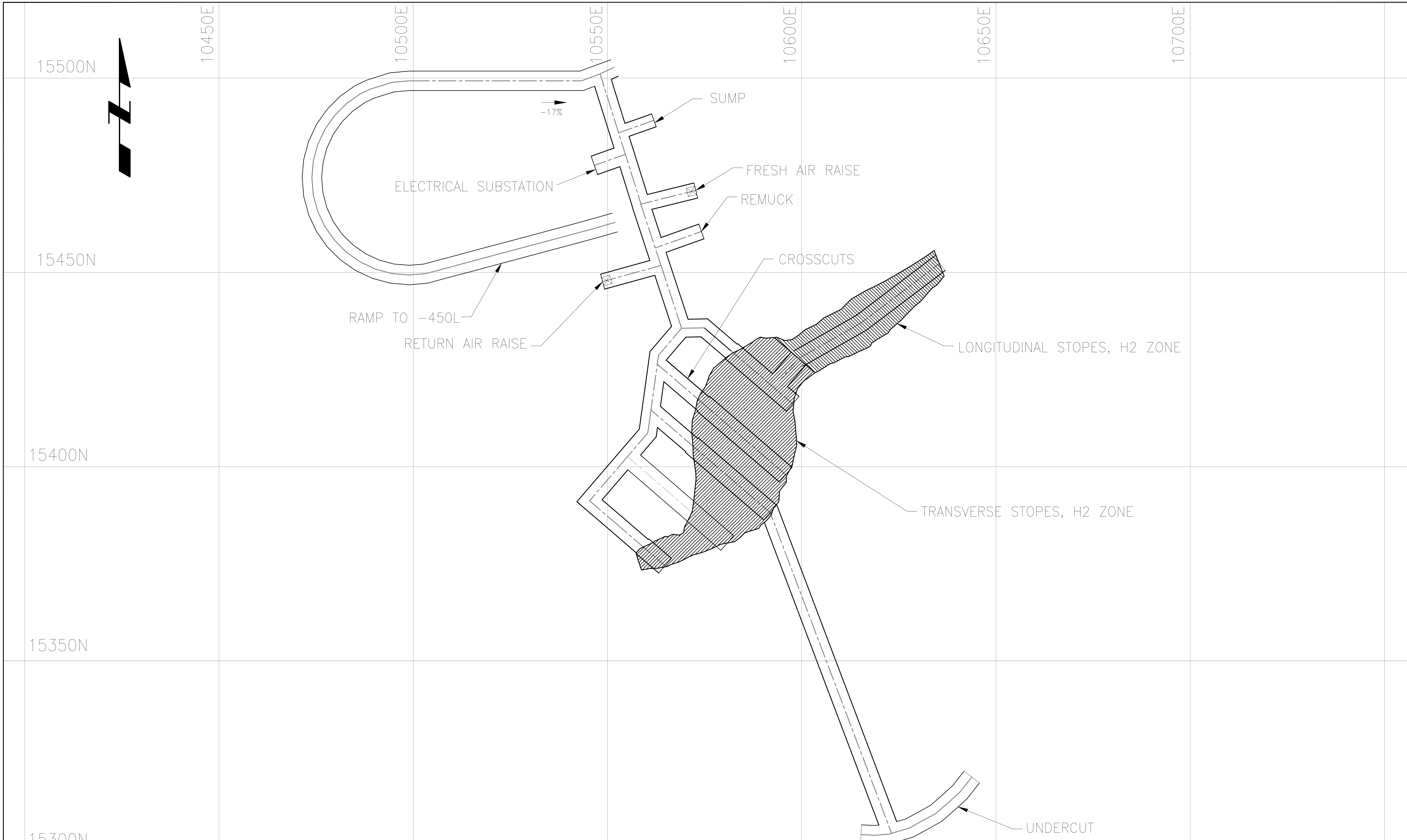
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-450m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-031
REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

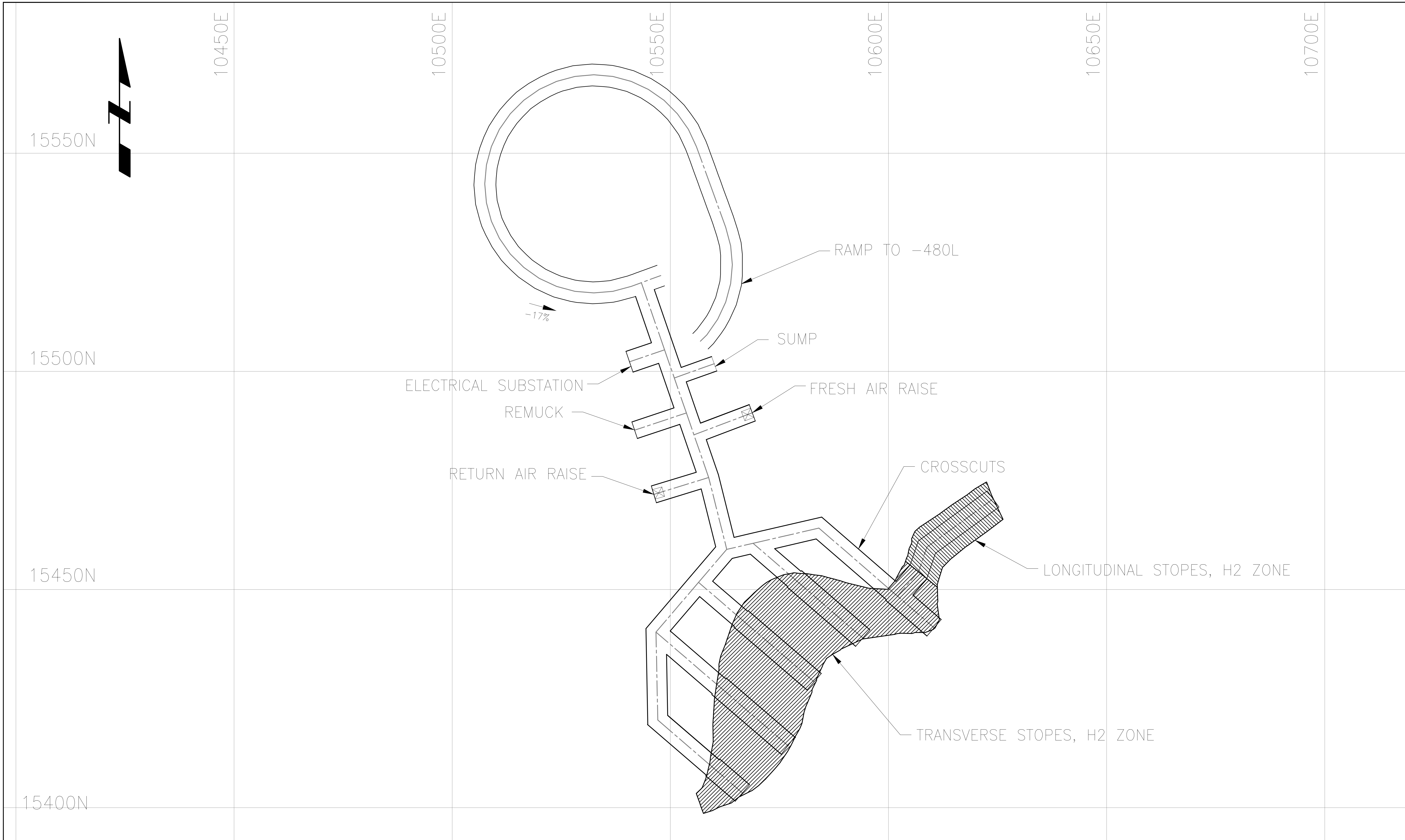
SCALE: 1:450
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-480m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-032
REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

SCALE: 1:400
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

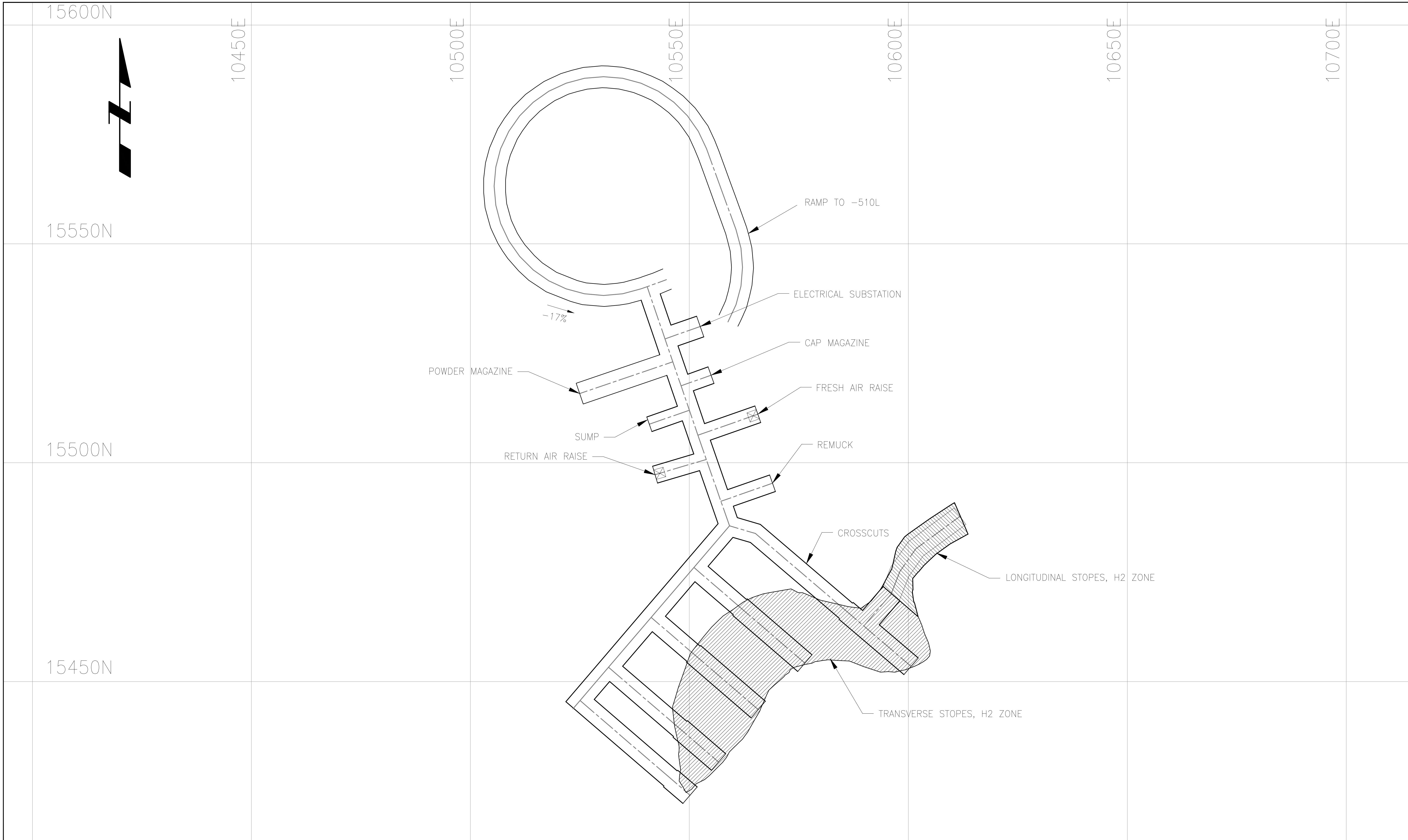
TULSEQUAH CHIEF PROJECT

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-510m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-033
REV No. F1



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F1	FEASIBILITY	FINAL FOR FEASIBILITY STUDY	B.W.	C.H.		29/09/14

REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

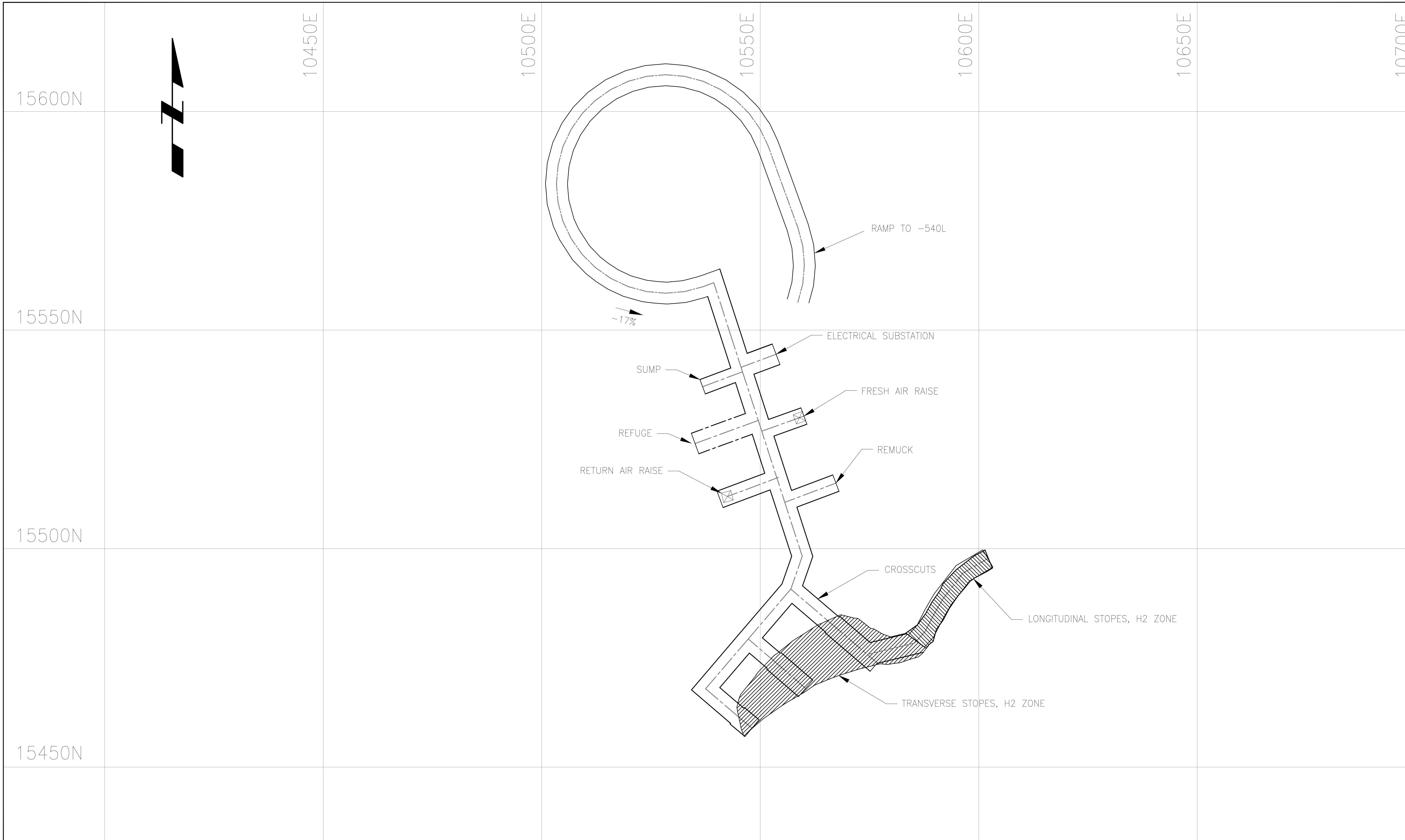
SCALE: 1:400	DATE:	
DESIGNED:		
DRAWN:		
CHECKED:		
APPROVED:		

Chieftain Metals Inc.

JDS Energy & Mining Inc.

TITLE
-540m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.	DRAWING No.	REV No.
	1500-10-034	F1



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REFERENCE DRAWING	REF. DWG No.

TULSEQUAH CHIEF PROJECT

SCALE: 1:400
DATE:
DESIGNED:
DRAWN:
CHECKED:
APPROVED:

Chieftain Metals Inc.

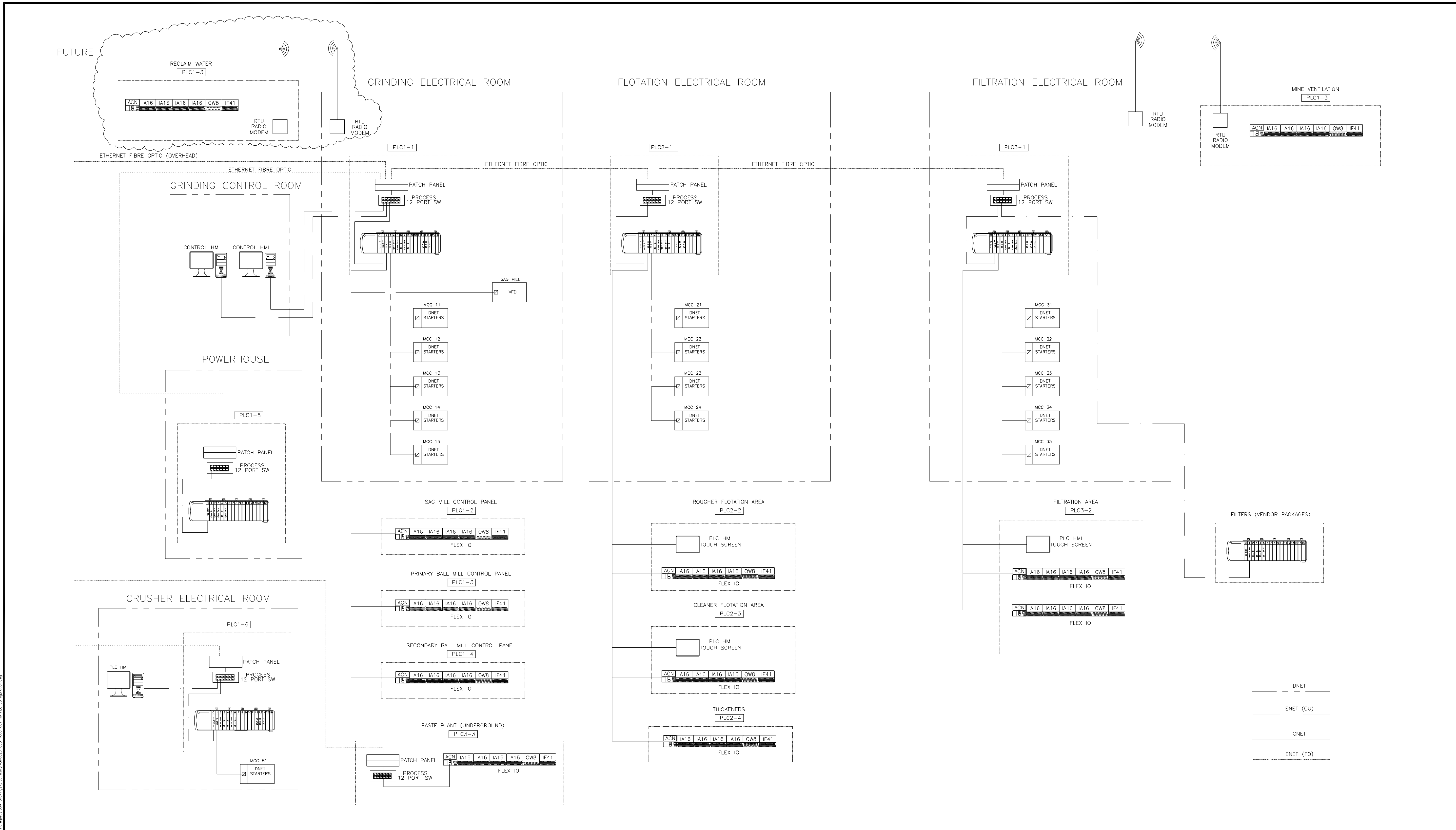
JDS Energy & Mining Inc.

TITLE
-570m EL
GENERAL ARRANGEMENT
PLAN

PROJECT No.
DRAWING No. 1500-10-035
REV No. F1

APPENDIX D

INFRASTRUCTURE



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DRAWING NO	DRAWING DESCRIPTION/TITLE	REF
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		5
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		8

REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
C	14/10/02	RELOCATE RECLAIM WATER,		
		DELETE WATER TREATMENT		
		REV. PASTE PLANT CONNECTION.		OP
B	14/08/29	GENERAL REVISION		OP
A	14/07/31	ISSUED FOR REVIEW	-	OP

CLIENT:

Allnorth

TITLE:

PROCESS CONTROL SYSTEM PLC CONFIGURATION

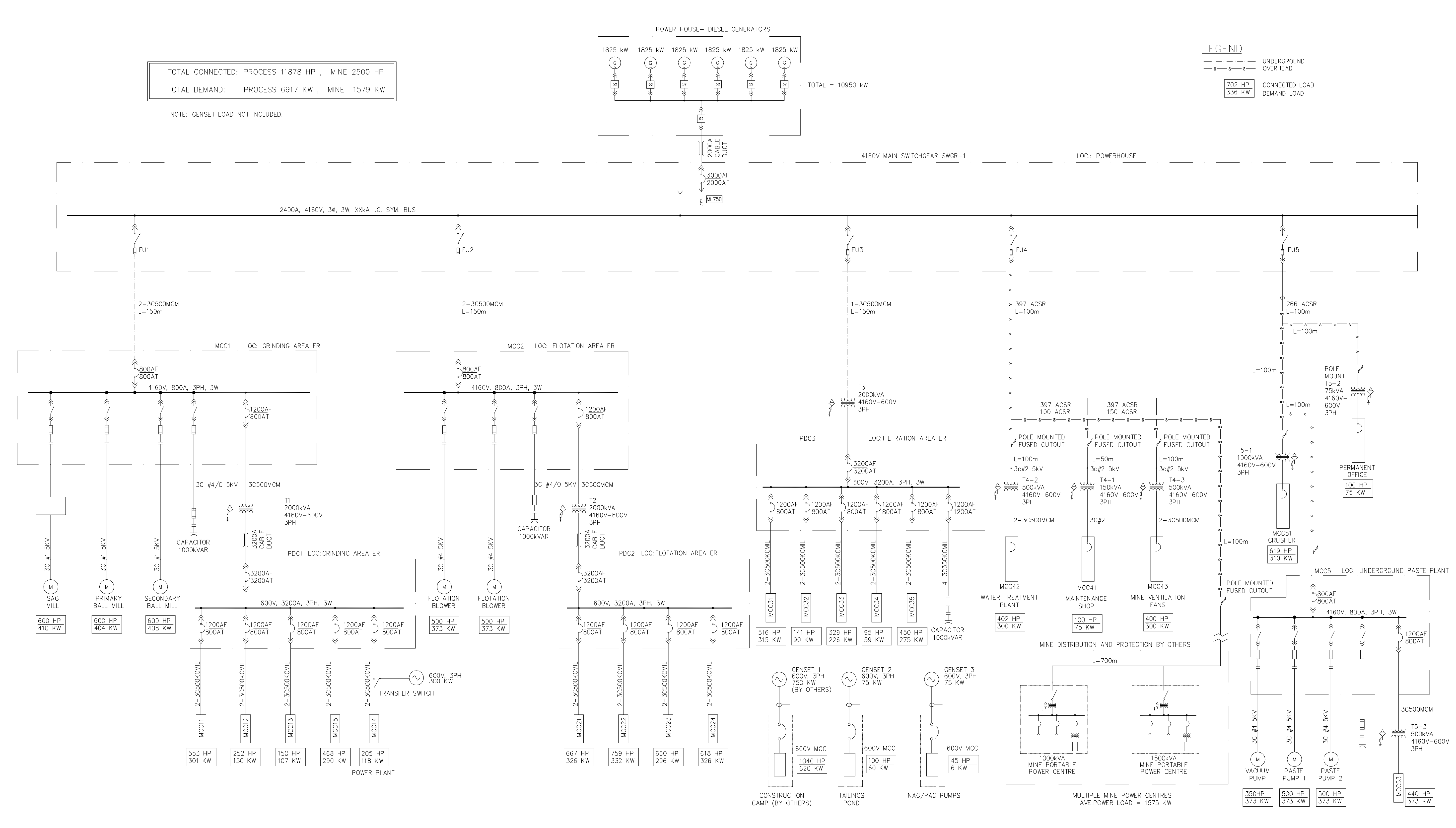
CLIENT NO:	-	DRWN:	-	DATE:	-
PROJECT NO:	12SU0009	DSGN:	OP	DATE:	12/07/31
DRAWING SIZE:	22x34	CHKD:	RT	DATE:	-
SCALE:	AS NOTED	APVD:	-	DATE:	-

PROJECT:

TULSEQUAH CHIEF FEASIBILITY STUDY

DWG NO: **14SU0026-000-1800-001**

REV: **C**



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		4
		5
		6
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		8

REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
F	14/11/04	REV.BALL & SAG MILL LOADS.		OP
E	14/10/06	REVISED FEEDER SIZE.		OP
D	14/10/02	REVISED PASTE PLANT LOAD		
D		AND DISTRIBUTION		OP
C	14/09/03	ADDED MINE VENT FANS,PERM.		
		OFFICE.REVISED CABLE LENGTH.		OP
B	14/08/29	GENERAL REVISION		OP
A	14/07/31	ISSUED FOR REVIEW	-	OP

CLIENT:

Allnorth

TITLE:

POWER DISTRIBUTION SIMPLIFIED SINGLE LINE DIAGRAM

CLIENT NO:	-	DRWN:	-	DATE:	-
PROJECT NO:	12SU0009	DSGN:	OP	DATE:	12/07/31
DRAWING SIZE:	22x34	CHKD:	RT	DATE:	-
SCALE:	AS NOTED	APVD:	-	DATE:	-

PROJECT:

TULSEQUAH CHIEF FEASIBILITY STUDY

DWG NO: **14SU0026-000-1600-001** REV: **F**