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Ministry of Energy and Mines
BC Geological Survey

Assessment Report
Title Page and Summary

TYPE OF REPORT [type of survey(s)]: Metallurgical, Drilling (auger), Environmental, Access R **TOTAL COST:** $1, 534,842

AUTHOR(S): Randal Cullen, P. Geo., pres. RDConsult **SIGNATURE(S):**

NOTICE OF WORK PERMIT NUMBER(S)/DATE(S): MX-1-520-2011 **YEAR OF WORK:** 2014

STATEMENT OF WORK - CASH PAYMENTS EVENT NUMBER(S)/DATE(S): Event # 5565504

PROPERTY NAME: Silvertip

CLAIM NAME(S) (on which the work was done): 509656, 509810, 1038295

COMMODITIES SOUGHT: Ag, Pb, Zn, Au

MINERAL INVENTORY MINFILE NUMBER(S), IF KNOWN: 82963

MINING DIVISION: Liard **NTS/BCGS:** 104-O16W

LATITUDE: 59 ° 55 ' 37 " **LONGITUDE:** 130 ° 20 ' 32 " **(at centre of work)**

OWNER(S):
1) JDS Silver Inc
2)

MAILING ADDRESS:
900 - 999 West Hastings Street Vancouver, BC V6C 2W2

OPERATOR(S) [who paid for the work]:
1) JDS Silver Inc
2)

MAILING ADDRESS:
900 - 999 West Hastings Street Vancouver, BC V6C 2W2

PROPERTY GEOLOGY KEYWORDS (lithology, age, stratigraphy, structure, alteration, mineralization, size and attitude):
Omineca Belt, Canadian Cordillera, Cassiar terrane, Upper Proterozoic, Middle Devonian, carbonate, clastic marine, continental margin, Earn Assemblage, McDame Group, Sylvester allochthon, thrust, folding, Camp Creek fault, skarn, carbonate replacement, manto, Ag, Pb, Zn, Au, poly metallic, 2.4Mt Ind, 0.7Mt Inf, flat lying, vertical feeder

REFERENCES TO PREVIOUS ASSESSMENT WORK AND ASSESSMENT REPORT NUMBERS: 9912,11020,11799,13259,14104,15560, 16899,20735,25495,25791,26073,26240,31919,32976

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TYPE OF WORK IN THIS REPORT	EXTENT OF WORK (IN METRIC UNITS)	ON WHICH CLAIMS	PROJECT COSTS APPORTIONED (incl. support)
GEOLOGICAL (scale, area)			
Ground, mapping			
Photo interpretation			
GEOPHYSICAL (line-kilometres)			
Ground			
Magnetic			
Electromagnetic			
Induced Polarization			
Radiometric			
Seismic			
Other			
Airborne			
GEOCHEMICAL (number of samples analysed for...)			
Soil			
Silt			
Rock			
Other			
DRILLING (total metres; number of holes, size)			
Core			
Non-core	10 x Augur Holes	1038295	77,843
RELATED TECHNICAL			
Sampling/assaying			
Petrographic			
Mineralographic			
Metallurgic	Process development and testing, bulk sam	1038295	120,190
PROSPECTING (scale, area)			
PREPARATORY / PHYSICAL			
Line/grid (kilometres)			
Topographic/Photogrammetric (scale, area)			
Legal surveys (scale, area)			
Road, local access (kilometres)/trail	Access Road Repair	509656,509810,1038295	634,237
Trench (metres)			
Underground dev. (metres)			
Other	Environmental baseline study	509656,509810,1038295	702,570
		TOTAL COST:	1,534,842

BC Geological Survey
Assessment Report
35781A

SILVERTIP PROJECT-2014 ASSESSMENT REPORT

Bulk Sampling for Metallurgical Testing on the SILVERTIP PROPERTY,
Liard Mining Division,
NTS 104-O/16W,
British Columbia, Canada

For Owner and Operator
JDS Silver Corp,
Suite 860 – 625 Howe Street
Vancouver, BC
V6C 2T6

Written by R. D Cullen, PGeo, RDConsult
Submitted September , 2015

SUMMARY

The Silvertip deposit is at the early mine development stage. Since the 1955 discovery of an argentiferous galena outcropping on Silvertip Hill by A. Zborovsky, V. Alfody, S. Mezaros and S. Papp working under a government grub staking program, previous owners have defined exhalative sedimentary hosted Zn and Pb deposits in Earn Group sediments and the Silvertip Ag Manto Carbonate Replacement Deposit (CRD) in McDame limestone below the Earn Group sediments. The property has undergone 85,354.8 m of drilling in 531 surface and underground collared holes, completion of 2.8 kilometers of declines and tunnels to support underground drilling and sampling of the deposit in situ, geophysical and geochemical surveys and surface geological mapping. Exploration for extensions to the Silvertip Deposit and additional CRD deposits with surface methods including ground and airborne geophysical surveying, sediment sampling geological and structural mapping has defined additional high priority targets for further surface exploration and drilling.

The property was acquired by JDS Silver (JDSS) in October 2013 for $15.5M from Silvercorp Metals Inc. in an all cash deal. JDSS now owns 100% of the Silvertip property and the mineral deposits therein. Silvercorp Metals Inc retains a 5% NSR on future production.

In preparation for completion of a small mine permit application, JDSS has advanced the design of an underground mine and surface facilities, tailings treatment and storage facilities, roadways and infrastructure as well as completing the archaeological assessment, traditional use assessment, environmental impact assessment and consultation with First Nations and local communities. The mine permit application has been approved as of June 26, 2015. Financing for mine development has been completed and work at the site was begun in late August of 2015.

During 2014 JDS Silver completed repairs to the access road damaged during June 2012 flooding allowing heavy equipment and construction materials to be moved to site. A bulk sample and metallurgical testing of ore recovered from a surface stockpile was completed. The analysis of the ore under a variety of process scenarios resulted in recommendations for innovation on the mill circuit. Further work included completion of an environmental baseline study for inclusion in the mine permit application and a geotechnical study to support mine planning. This program resulted in expenditures of C$1,534,842 which are applicable to expenditures required to maintain the claims comprising the Silvertip property in good standing.

Attachments

Appendices

1.0 INTRODUCTION

1.1 Location and Access

The Silvertip Property is situated in northern British Columbia, just south of the Yukon border, approximately 90 km by air west-southwest of Watson Lake, Yukon (Fig. 1.1). The property is accessible via a 25-km gravel site access road starting from Kilometer 1128 (Mile 701) of the Alaska Highway, about 15 km east of Rancheria. It lies within NTS map sheet 104-O/16W, in the Liard Mining Division.

Watson Lake, the main supply centre for operations on the property, is a 107 km drive east along the Alaska Highway from the end of the property access road. Watson Lake also has an all weather runway accessible by medium sized aircraft and a fixed base helicopter operator. The nearest airport for scheduled flights is in Whitehorse, a 3 hour drive along the Alaska Highway to the northwest, with daily flights to Vancouver and points south varying in regularity and operator on a seasonal basis. Flight time from Vancouver direct to Whitehorse is about 2 hours.

1.2 Physiography and Climate

The property lies on the northeastern flank of the Cassiar Mountains. The terrain is moderately mountainous, with generally rounded peaks and ridges separated by U-shaped valleys. The highest peaks are about 1,950 metres; topographic relief is typically about 300 to 500 metres. Roughly 35% of the property is above tree line, which is at approximately 1,450 metres amsl.

Temperatures on the property normally range from highs of 20^{o}c in summer to the -45^{o}c to -55^{o}c range in mid-winter. Precipitation is moderate with about half of the annual precipitation occurring as snow. Snow accumulations of 1 metre are typical for the area and operations have been carried out on the property 12 months of the year in the past with no significant operational problems.

1.3 Ownership

The Silvertip Property is 100% owned by JDS Silver Inc., through a purchase from Silvercorp Metals Inc in October, 2013. Total consideration was CDN$15.5M with Silvercorp retaining a

5% NPR on future production. There is also a 5% net profit royalty on eight of the mineral claims but none of the known mineral resources are on the claims to which the royalty applies.

1.4 Description of Mineralization and Infrastructure

The property contains both sedimentary exhalative lead zinc deposits (upper zone) in lenses hosted in topmost Earn Group sediments and a carbonate replacement (CRD) silver-lead-zinc manto style deposit hosted in McDame limestone capped by Earn Group sediments (lower zone). The lower zone CRD mineralization is composed of several more or less discreet mineralized zones (named Silver Creek, Discovery, Discovery North, and 65 Zone), centered at UTM coordinates 6,643,900 N and 425,200 E (NAD 27).

About 2,800 metres of underground workings are developed in and near the lower zone mineralization (Figure 1.2). These have been allowed to flood post 2001 exploration drilling and are under care and maintenance at this time. On surface, in the immediate vicinity of the portal, an approx. 6,000 tonne ore stock pile, lime mixing shed and two water settling ponds remain from previous programs. The site also has a network of drill, portal, camp and 'property at large' access roads, a 49 man portable camp, outbuildings for management of core and core drilling programs and 3 core storage laydown areas. A well services the campsite with water for cooking and washing.

1.5 Land Tenure

The property currently comprises 190 tenures, covering an area of approximately 88,238.93 hectares (Figure 1.1 and Attachment One). These claims have not been surveyed by a designated BCLS. The claims and their status at the time work was completed are listed in Appendix 1.

1.6 Environmental Liability

A $79,000 bond has been posted with the BC Minister of Mines to cover the outstanding disturbance on the property as of end of 2011 exploration program.

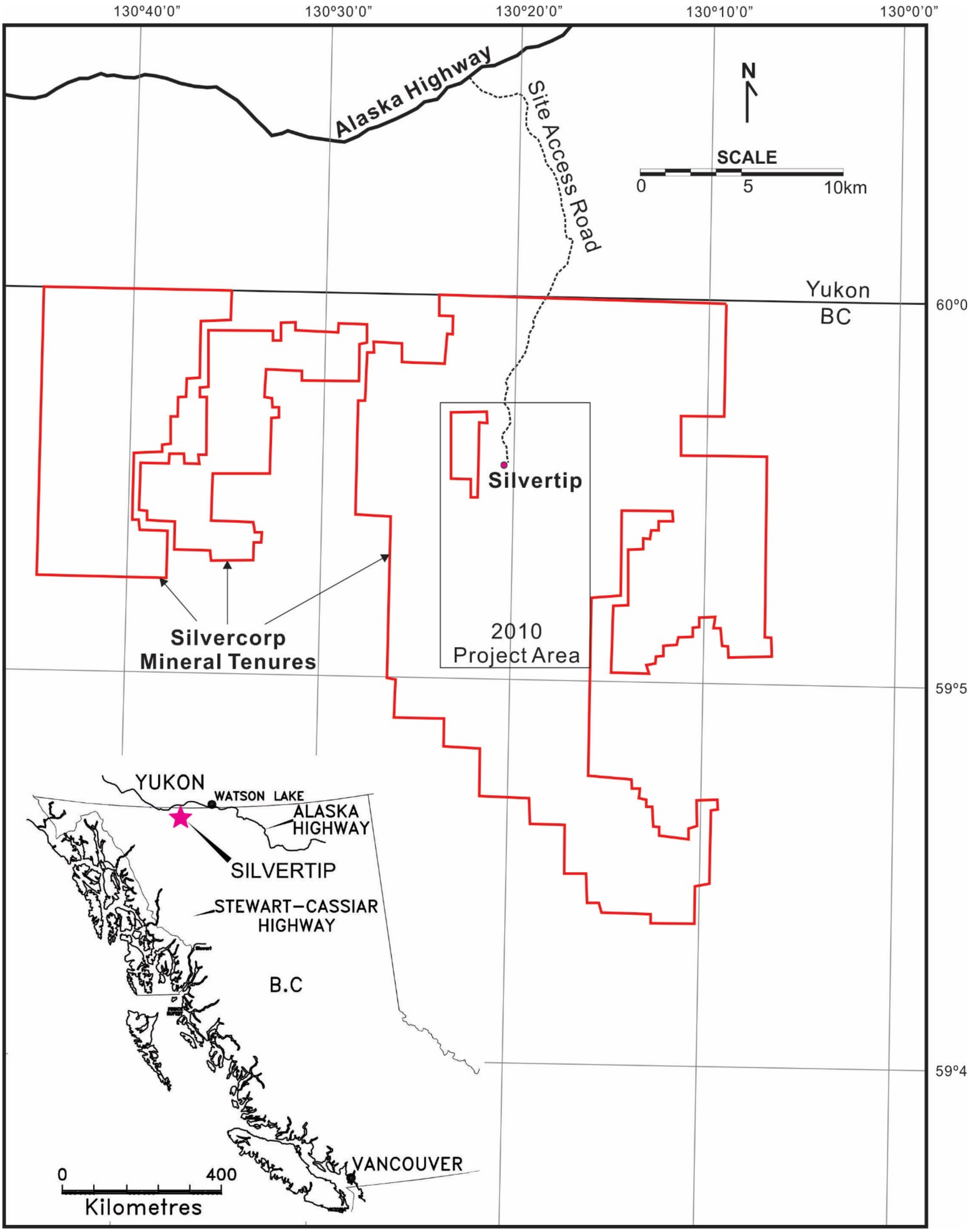


Figure 1.1 Location and Access to the Silvertip Project

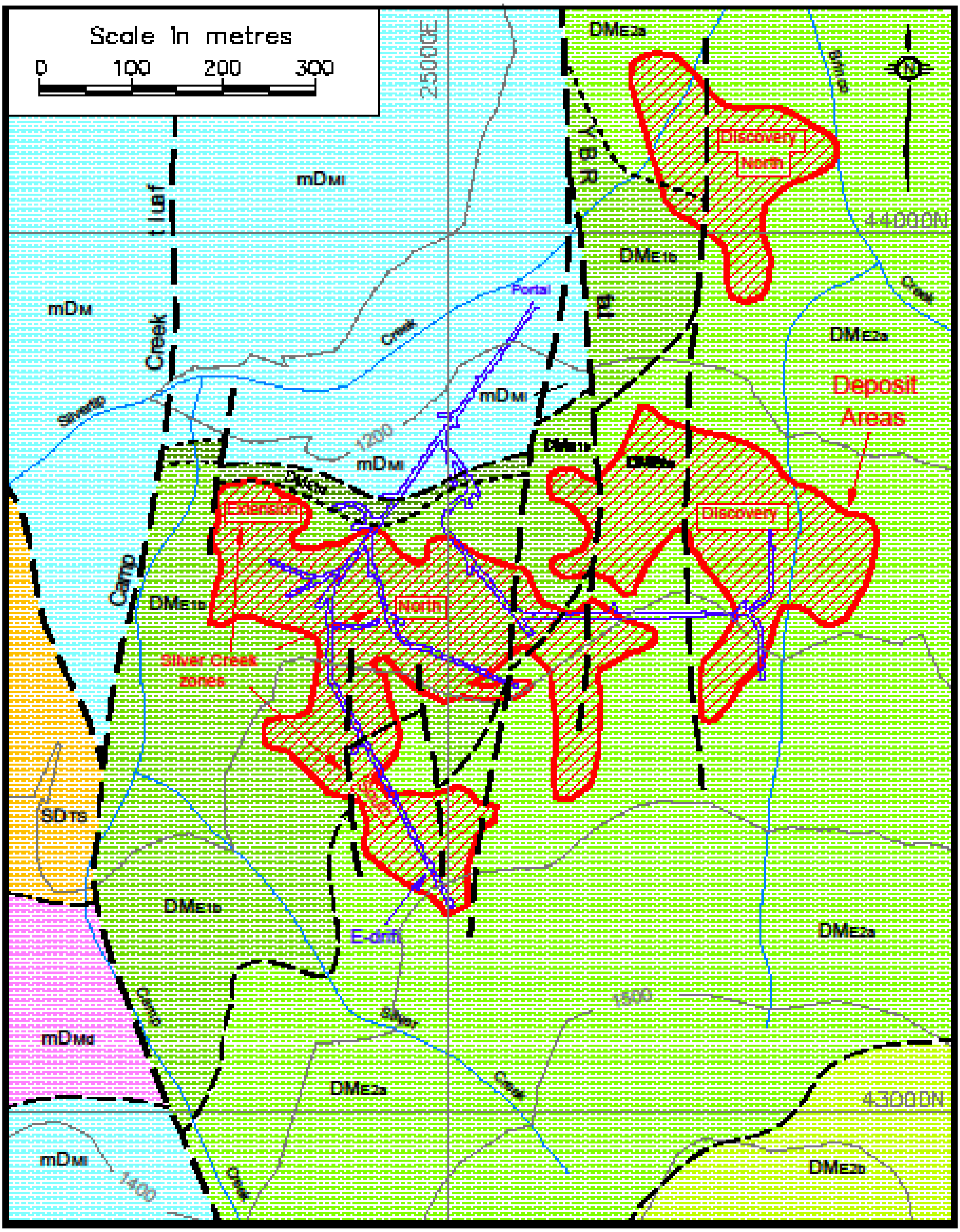


Figure 1.2. Surface geological map of the main Silvertip deposit showing the areas of mineralization and plan view of underground workings. For Legend see Fig. 3.5. (From Rees, Akelaitis and Robertson, 2000)

2.0 PROPERTY HISTORY

(pre-2010 description excerpted from 'NI43-101 Technical Report Resource Update on the Silvertip Property, Northern British Columbia, Canada', Feb. 19, 2010)

The development history of the Silvertip property is summarized in Table 2.1.

Galena-rich float was discovered by prospectors on Silvertip Hill in 1955. In late 1956 and 1957, Conwest Exploration Company explored gossanous zones in the McDame Group limestone by drilling and surface and underground workings. Zones of galena and silver-rich values were found but most of the sulphides were thoroughly oxidized.

In 1958, drilling was continued by a joint venture between Noranda Mines Limited, Canex Aerial Exploration Limited and Bralorne Mines Limited. A number of other companies optioned the property between 1960 and 1966, conducting AFMAG and IP surveys over Silvertip Hill to identify drill targets. Other work included photo and geological mapping, rock and soil sampling, and trenching and stripping. Some good anomalies were found, but follow-up drilling found only deeply oxidized mineralization with generally uneconomic silver grades.

Silverknife Mines Limited owned the Silvertip claims from 1966 until the claims lapsed in the early 1970s. During this time, four rotary holes were drilled (1966) to test IP anomalies, and two diamond drill holes were targeted on EM survey targets (1967). Two diamond drill holes tested geophysical anomalies in 1968. By this time, the idea that silver-lead mineralization was related to replacement of limestone at its contact with overlying 'shale' was the dominant exploration model for the Silvertip Hill area. However, results were still not encouraging due to various drilling problems, weak mineralization, or deep oxidation.

Very little work was done in the 1970s. The main phase of exploration began in 1980 when Cordilleran Engineering, on behalf of subsequent property owner Regional Resources Limited, was conducting regional reconnaissance in search of shale-hosted, lead-zinc sedex deposits. The property was then known as Midway. They found base metal anomalies in soils and stream sediments about 1500 metres northeast of Silvertip Hill, which led to the discovery of baritic and siliceous gossans of exhalite origin within the Earn Group. Regional mapping, soil and EM surveys followed in 1981, with six diamond drill holes around the exhalite showings. Four of these unexpectedly intersected massive sulphide below the base of the Earn Group, at the top of the McDame Group limestone, and by the end of 1982 the exploration focus had again shifted back to limestone-hosted replacement mineralization.

An aggressive surface drill program was conducted between 1982 and 1984, along with geophysics and petrographic and metallurgical research. Two main, blind areas of mineralization were outlined, Silver Creek and Discovery, and a manto-type deposit model was formulated. Encouraged by the apparent size of the mineralized area and the good grade and thickness of sulphides, the company began underground exploration development in the Silver Creek area (1984), followed by 12,383 metres of underground drilling over 170 holes, in fans spaced 20 metres apart. The results showed that the mineralization was more erratic and discontinuous than had been modeled from the widely spaced surface drill pattern, leading to a reduced estimate of the size of the resource.

A new underground development initiative was carried out between 1989 and 1991 by operator Strathcona Mineral Services, with the opening of a decline to the east towards the Discovery area, and completion of 9,620 metres of underground drilling.

In 1996, Imperial Metals Corporation of Vancouver acquired Regional Resources and renamed the company Silvertip Mining Corporation (SMC). A large exploration program in 1997 comprising diamond drilling, seismic surveying, and surface geological mapping resulted in the discovery of a new zone, the Silver Creek Extension, now part of the Silver Creek zone. This added significantly to the total geological resource, which was subsequently recalculated at 2.57 million tonnes grading 325 g/t Ag, 6.4% Pb, 8.8% Zn and 0.63 g/t Au. In 1998, SMC entered the Environmental Assessment review process with the provincial government for project certification. That year, various environmental baseline studies were done and monitoring procedures instigated, along with a reconnaissance CSAMT (Controlled Source Audio Frequency Magneto Telluric) geophysical survey. This survey revealed a large, vertically oriented low-resistivity anomaly between the Silver Creek South area and the Camp Creek fault, suspected of indicating a sulphide chimney.

A more detailed, follow-up CSAMT survey was done in 1999, and the best three geophysical targets were drilled. One hole (99-65) intersected thick, feeder-style mineralization. This intersection prompted the re-opening of a portion of the existing underground workings by de-watering and refurbishment of decline and tunnels in the fall of 1999. This was followed by 3,210 metres of underground diamond drilling in January-February 2000, centred on drill hole 99-65. This drilling identified the Zone 65 area of the deposit in sufficient detail to support a resource calculation for the 65 Zone but this was not done. The 65 Zone had failed to live up to the company's hopes for the addition of significant additional high grade mineralization in the form of a feeder pipe or chimney.

Table 2.1 Development History of the Silvertip claims and deposit

Year	**Operating Company**	**Drilling Company**	**Surface Drill**	**Surface meters**	**Tunnel meters**	**U/G Drill**	**U/G Core(m)**
1957	Conwest Exploration		11	582	548m	6	786
1958	Noranda/Canex Aerial/Bralorne Mines		3	972		3	972
1961/62	Pegasus Exploration		4	495			
1963			1	51			
1966	Silverknife Mines						
1967			2	152			
1968	Northern Comstock Mining		2	388			
1981	Regional Resources	Amity Drilling	6 (NQ)	857			
1982		E Carron Drilling	19 (NQ)	5,283			
1983			32 (NQ)	11,733			
1984/85			50 (NQ)	10,981	1,453	142 (BQ); 29 (NQ)	7,578; 4,805
1986			14 (NQ)	2,660			
1990		Advanced Drilling			765	68 (NQ)	9,620
1997	Imperial Metals	Olympic Drilling	63 (NQ)	8,594			
1999		DJ Drilling	3 (NQ)	1,285			
2000		Advanced Drilling				22 (HQ)	3210
2010	Silvercorp Metals Inc.	Lyncorp Drilling	29	9,209			
		Cabo Drilling	7	1,705			
2011		G4 Drilling	15	3,437			
Total			**261**	**58,383.5**	**2,766**	**270**	**26,971**

A 14 line, 8.85 line-km natural source Audio Frequency Telluric (AMT) geophysical survey was conducted in the summer of 2001. The grid was installed to the north of the previously known mineralization, in an area where the McDame limestone is not under the cover of the typically graphitic Earn Group sedimentary rocks. The AMT survey was successful in defining previously known geological features that confirmed the techniques effectiveness. At least one strong anomaly evident in the survey data was recommended for drilling, that target is as yet untested.

In 2002 Silver Standard acquired the Silvertip Project from Imperial Metals in 2002 and put further work on the property on hold. Silvercorp subsequently purchased the property from Silver Standard in February, 2010 and completed an NI 43-101 compliant report on the resources and work to date on the property. Silvercorp also initiated a review of the previous work programs on the property, planned a surface exploration program, and began the application process to commence surface exploration, dewater the underground workings and acquire a small mines permit.

Silvercorp transferred ownership of the Silvertip project to 0875786 BC Ltd and began the exploration program with a 4113 line km helicopter borne VTEM survey over the entire property, completing this program in mid-August 2010. Results were not ready in time to be incorporated into the 2010 field program. A report on this program is attached as Appendix one. An exploration permit was granted in late July 2010. Four drill rigs were mobilized to the property and a 10,000m drill program was initiated with the option to expand to 20,000m if results were favorable. The surface drilling was planned to extend the known mineralization envelope to the east and north, explore satellite areas to the south where earlier step out drilling had been successful and to further test AMT geophysical anomalies that had been identified by Imperial Metals in 2001 but never drilled. The 2010 program began on the ground on August 6th and completed 10,913.29m of drilling before shutdown on Nov. 16, 2010. A brief mapping and sampling program was undertaken on the Donegal Mountain showings, (renamed the DM Zone) by Silvercorp geologists in September and October, 2010. This program provided detailed mapping of this previously explored anomaly in preparation for drill testing. The Silvertip 2010 field program was shutdown in November 2010 and the onsite camp put on care and maintenance.

In 2011 a drilling program was undertaken beginning on June 28. G4 Drilling of Val D'Or, Quebec completed the 3437.5m drill program on September 1, 2011. Drilling targeted a high grade upper zone exhalite intersection adjacent to the portal, an extensive geophysical

(AMT) anomaly in limestone to the west of the portal and the DM zone of anomalous Ag, Pb and Zn surface showings at the Earn McDame contact on the southeastern flank of Donegal Mt.

Geotechnical work to test the potential for use of insitu glacial till for building materials and foundations for proposed infrastructure and a groundwater well installation program began in late September and was completed on Nov.15, 2011 after completing 11 boreholes and 7 water wells. Following the completion of the water wells the camp was put on care and maintenance with road clearing being the main activity for the winter.

3.0 GEOLOGY

(excerpted from 'NI43-101 Technical Report Resource Update on the Silvertip Property, Northern British Columbia, Canada', Feb. 19, 2010)

3.1 Regional Geology

The Silvertip property is situated in the northern Omineca Belt of the Canadian Cordillera. The most important element of this region is the Cassiar terrane, composed of Upper Proterozoic through Middle Devonian carbonate and clastic sedimentary rocks formed on a marine platform on the ancient continental margin of western North America (Cassiar Platform), and overlying Devono-Mississippian rift-related clastics (Earn Assemblage). Structurally overlying the Cassiar terrane is a tectonic assemblage of marginal basin and island arc sediments and igneous rocks of the Upper Paleozoic Sylvester allochthon (Fig. 3.1).

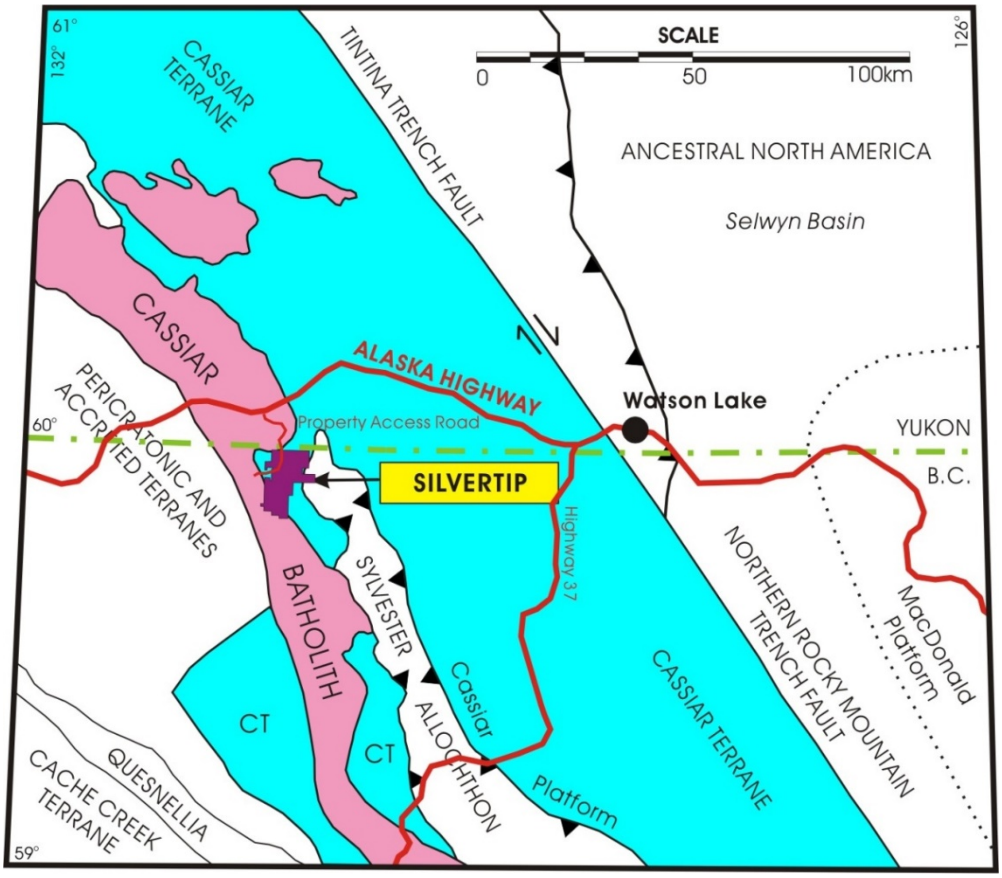


Figure 3.1: Geology Map showing the main tectonic elements of northern British Columbia and southern Yukon showing regional setting of Silvertip. (From Rees, Akelaitis and Robertson, 2000)

The region was moderately deformed by folding and thrust faulting in the Jurassic, and later by extensional and dextral transcurrent faulting in the Late Cretaceous to early Tertiary (Fig. 3.2). The Cassiar Batholith, a large, granite to granodiorite intrusion of mid-Cretaceous age, lies west of the property. Small intrusions and related hydrothermal alteration of possibly Late Cretaceous age are minor but important features of the region.

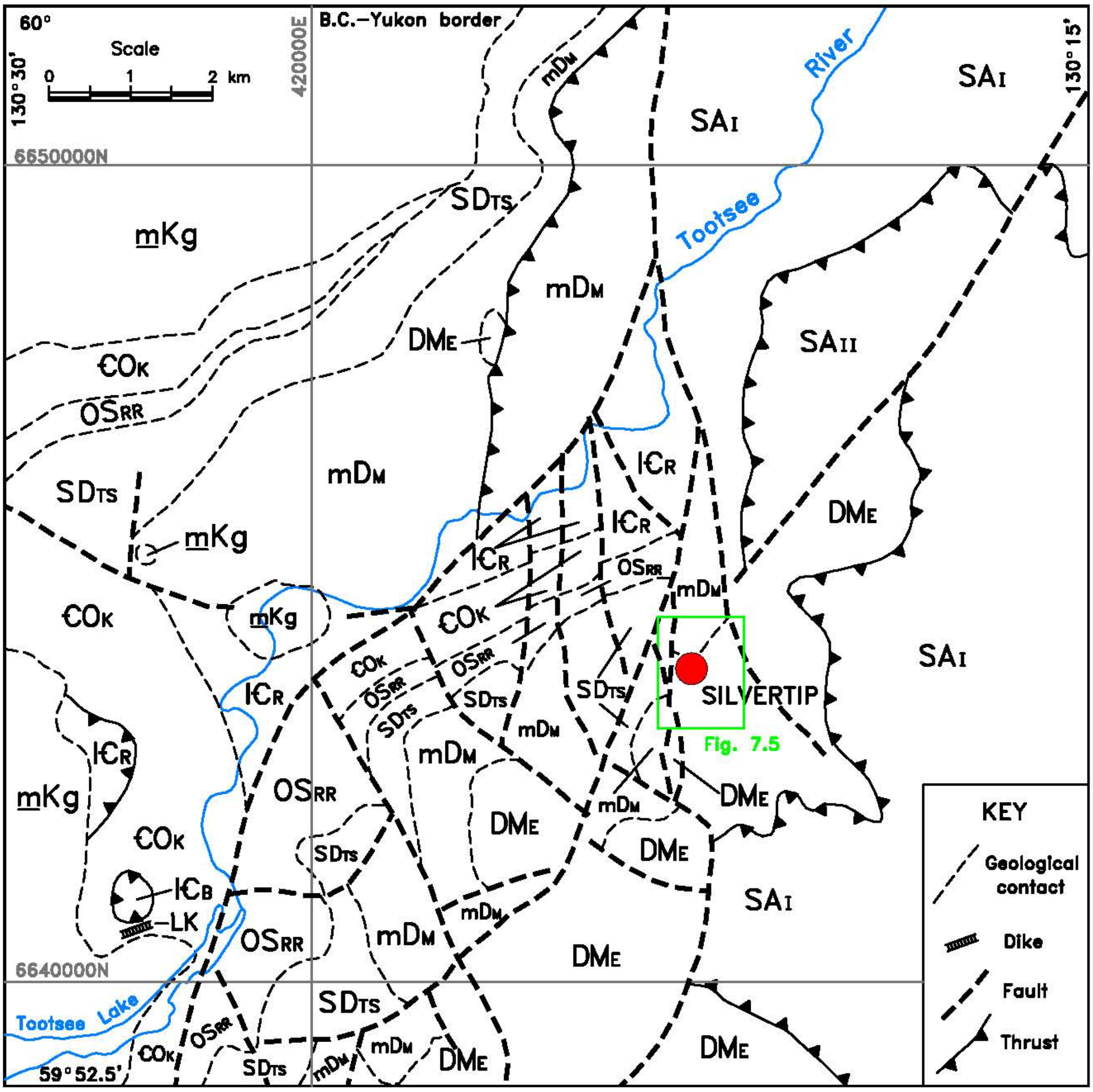
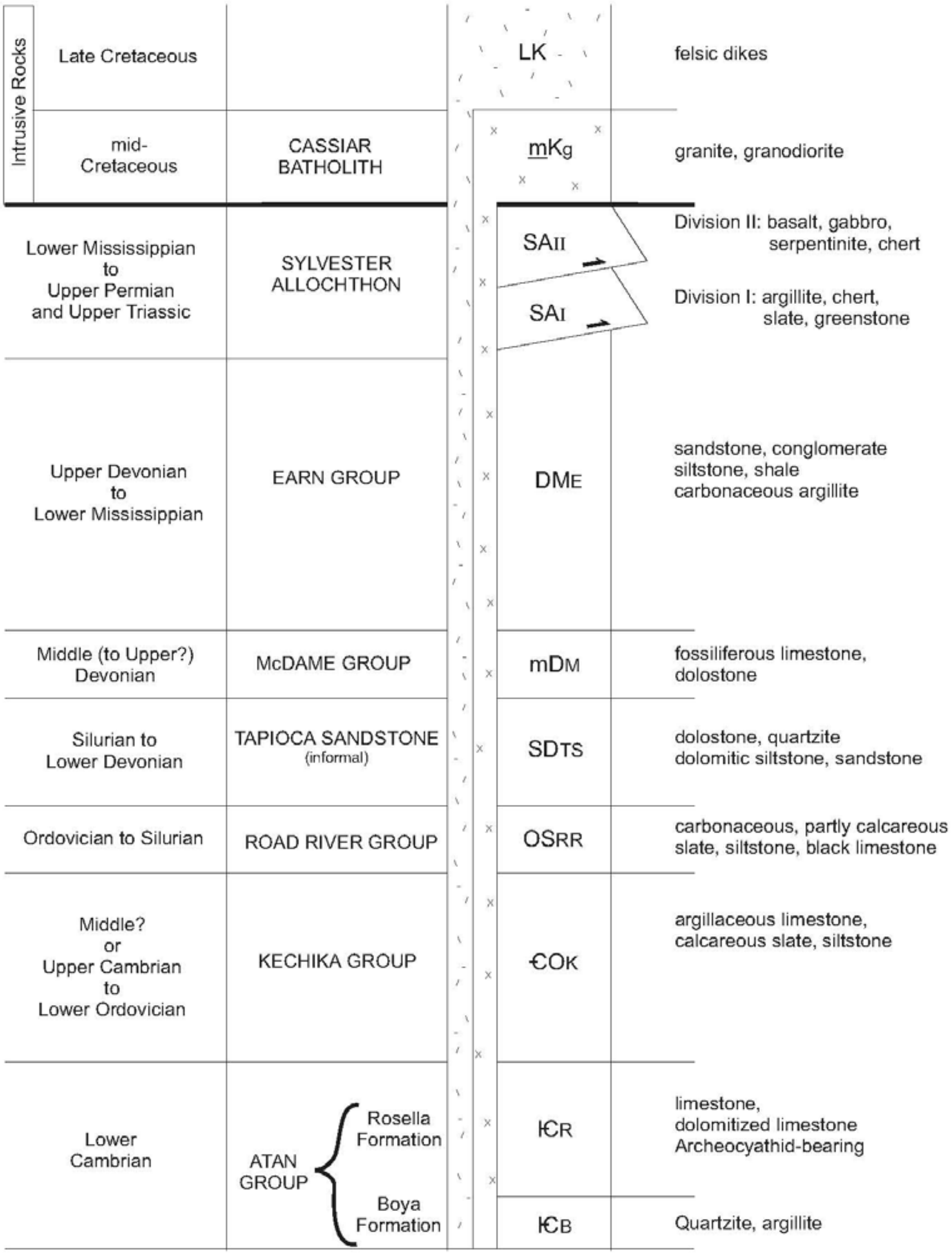


Figure 3.2: Regional Geology Map showing the location of Silvertip with respect to stratigraphic units of the Cassiar Platform, the southern margin of the Cassiar Batholith and the western margin of the Sylvester allochthon. Adapted from Nelson and Bradford (1967). For legend see Figure 3.3. Regional scale geological maps are included at the back of this report in scales of 1:10,000 and 1:25,000.

The main mineral deposits are syngenetic barite +/- lead, zinc prospects in Paleozoic sediments, and skarn and replacement deposits related to Cretaceous intrusive and hydrothermal activity. An account of mineralization in the Rancheria district, including the Silvertip area, is given by Abbott (1983).

The principal sources of regional geology data are Gabrielse (1963), Nelson and Bradford (1993), and Nelson and Bradford's (1987) open file map of the Tootsee Lake area, from which Fig. 3.2 is adapted.



Intrusive Rocks	Late Cretaceous		LK	felsic dikes
	mid-Cretaceous	CASSIAR BATHOLITH	mKg	granite, granodiorite
	Lower Mississippian to Upper Permian and Upper Triassic	SYLVESTER ALLOCHTHON	SAII	Division II: basalt, gabbro, serpentinite, chert
			SAI	Division I: argillite, chert, slate, greenstone
	Upper Devonian to Lower Mississippian	EARN GROUP	DME	sandstone, conglomerate siltstone, shale carbonaceous argillite
	Middle (to Upper?) Devonian	McDAME GROUP	mDM	fossiliferous limestone, dolostone
	Silurian to Lower Devonian	TAPIOCA SANDSTONE (informal)	SDTS	dolostone, quartzite dolomitic siltstone, sandstone
	Ordovician to Silurian	ROAD RIVER GROUP	OSRR	carbonaceous, partly calcareous slate, siltstone, black limestone
	Middle? or Upper Cambrian to Lower Ordovician	KECHIKA GROUP	ЄOK	argillaceous limestone, calcareous slate, siltstone
	Lower Cambrian	ATAN GROUP: Rosella Formation	lЄR	limestone, dolomitized limestone Archeocyathid-bearing
		ATAN GROUP: Boya Formation	lЄB	Quartzite, argillite

Fig. 3.3 Regional Geology Stratigraphic Column. (from Rees, Akelaitis and Robertson, 2000)

3.2 Property Geology

3.2.1 Stratigraphy

The geology of part of the Silvertip property in the vicinity of the identified resource is shown in (Fig. 3.4), and the stratigraphic column in (Fig. 3.5). Essentially, the area comprises easterly to southeasterly dipping Tapioca sandstone and McDame Group carbonates, overlain by the Earn Group. All these rocks are deformed by generally north-trending faults related to the Tootsee River fault system (Nelson and Bradford, 1993), the most important of which is the Camp Creek fault.

3.2.2 Tapioca Sandstone

This is an informal unit, partly equivalent to the (formal) Sandpile Group. The Tapioca is Silurian to Lower Devonian in age, and roughly 475 metres thick. It consists of pale buff-grey dolomitic sandstone to quartzite, silty dolostone and dolostone. The characteristic texture is well-rounded sand grains in dolomitic cement. Good cross-bedding is present locally.

3.2.3 McDame Group

This carbonate unit hosts the massive sulphide mineralization at Silvertip. It consists of a lower dolomitic unit, about 100 metres thick, and an upper limestone unit up to 260 metres thick. The McDame is Middle Devonian, but may extend into the Upper Devonian.

The lower dolomitic unit consists of pale to dark buff-grey or blue-grey, very fine grained dolostone and silty dolostone, grading upwards into dolomitic limestone. The rocks are fairly well bedded, and locally have fine cryptalgal laminations. In contrast to the overlying limestone unit, this unit has a uniform, non-bioclastic texture. It is distinguished from the underlying Tapioca sandstone by the absence of sand grains or siliceous component, and by its colour and less blocky weathering.

The main, upper part of the McDame Group is composed of distinctive bioclastic limestone, noted for its rich fauna of stromatoporoids, corals and brachiopods. The limestone is pale to dark bluish-grey, and fine to medium grained with a crystalline texture. It is moderately to

thickly bedded (up to 1 or 2 metres). Parts of the limestone have been hydrothermally altered to a buff-grey, medium-grained dolostone, or to a pink or white, crystalline 'marble'.
The stromatoporoid *Amphipora* is characteristic of the limestone, as are several forms of massive stromatoporoids. The bioclastic facies are generally not recognizable in surface outcrops because of weathering.

Brecciation is another important feature of the McDame limestone, again most conspicuous in drill core. Some of these are primary depositional breccias related to karst erosion (see below), and others were formed much later by solution collapse processes due to hydrothermal activity accompanying mineralization.

3.2.4 Earn Group

In the Late Devonian, the carbonate platform emerged above sea level for a time, and the McDame limestone was karst eroded. This episode ended with crustal extension, re-submergence, and the deposition of the succeeding Earn Group siliciclastics in the Late Devonian through Early Mississippian. The basal Earn was deposited disconformably on the McDame with little or no angular discordance, but stratigraphic relief due to dissection at the unconformity is up to 165 metres. The top of the Earn is not preserved; the known thickness in the area ranges between 600 and 1000 metres.
The Earn comprises two coarsening-upward cycles (1 and 2) of distal to proximal turbiditic siliciclastics. In each sequence, the lower part is characterized by carbonaceous, siltstone-mudstone and lesser sandstone or greywacke (1A and 2A), and the coarser, upper part by sandstone-greywacke and chert-pebble conglomerate (1B and 2B). The rocks were deposited as intertonguing turbidite fans in extensional basins or half-grabens with restricted circulation.

3.2.4.1 Unit 1A

The basal Earn Group consists of very carbonaceous mudstone to siltstone (1AA), deposited directly on top of the McDame limestone, or in cavities at some depth below the unconformity, due to the muddy sediment infiltrating the karst features. These inclusions of Earn in the McDame are termed 'enclaves'. The rocks are fine grained and finely laminated, and indicate low energy deposition under euxinic conditions. Syngenetic or diagenetic pyrite is present, generally less than 2%. The bottom few metres of 1A are commonly calcareous (1AC). Total thickness is up to 45 metres.

3.2.4.2 Unit 1B

The upper, coarser part of the lower cycle begins with interlaminated siltstone and sandstone, which becomes predominantly medium- to thickly bedded sandstone up-section. The sandstone is grey, medium- to coarse-grained greywacke, characterized by chert-rich detritus. Sandstone beds are generally centimetres to decimetres thick, separated by beds of siltstone or interlaminated sandstone-siltstone. These lithologies may be somewhat calcareous in places. Pyrite, mainly syngenetic or diagenetic, typically varies between 1 and 3%, and is more prominent in the more argillaceous beds or laminae than in the sandstones. Graded

beds of chert-argillite pebble conglomerate are common; they may be two metres thick in the upper part of the unit.

The higher energy conditions implied by Unit 1B suggest increasingly active, fault-controlled block uplifts and erosion in the basin. This mode of formation probably contributes to the wide variation in the thickness of Unit 1B, which ranges from as little as 60 metres to 200 to 300 metres.

3.2.4.3 Unit 2A

This is the lower, finer grained part of the upper cycle, and is the thickest and most inhomogeneous unit in the Earn Group. It is between 200 and 640 metres thick. Subunit AA at the base is recessive, dark grey to black carbonaceous mudstone to siltstone. Above it is the lowest and generally thickest and most important of the several exhalite subunits that are diagnostic of Unit 2A: the D-zone exhalite. It consists of pale grey to buff, fine-grained, siliceous and pyritic, laminated exhalite. Above the D-zone is 2AC, a calcareous interval comprising interlaminated siltstone, calc-arenite and locally impure limestone; it is 5 to 80 metres thick. This is followed by a more siliceous subunit up to 100 metres thick, 2AS, consisting of thinly laminated siliceous siltstone, slate and fine sandstone. In addition to the D-zone, several other minor exhalites occur within Subunits 2AC and 2AS. They are typically no more than a few metres thick, and some are probably not very laterally continuous. It is not clear if they occur consistently at the same stratigraphic horizons from place to place.

The thickest (up to 450 metres) and most characteristic subunit of Unit 2A is 2AP, which is composed of thinly to thickly interbedded and finely laminated slaty siltstone and fine- to medium-grained sandstone. The main feature of 2AP is the disrupted structure of the sandstone laminae that have been broken into discrete, sheared and rotated lenses millimetres to centimetres in size, due to slumping and soft-sediment deformation of a semi-consolidated turbidite sequence.

3.2.4.4 Unit 2B

The highest unit of the Earn is 2B, which is marked by the abrupt appearance of coarse, chert- and argillite pebble conglomerates above Subunit 2AP. It represents the upper coarse-grained component of the second cycle. These polymictic conglomerates are thickly bedded, and commonly contain subunits of very well bedded greywacke-sandstone. They are typically matrix supported, and the clasts are rounded to subrounded. Unit 2B is at least 200 metres thick. It is quite similar to unit 1B, but is distinguished by its coarser components, thicker bedding, and a lower amount of siltstone.

3.2.5 Structure

The basic structure of the Silvertip area is not complicated. Like the rest of the immediate region, it is dominated by faulting rather than folding. Strata generally strike north to northeast and dip gently to moderately east to southeast. There are no fold closures affecting the local map pattern, which is characterized by a general younging of units eastwards, broken up by faults.

The main regional ductile deformation resulted from crustal shortening in the Jurassic, when the Sylvester allochthon was tectonically emplaced onto the Cassiar stratigraphy and all units were subjected to folding, thrusting and foliation development, accompanied by very low grade metamorphism. The main foliation is generally parallel to bedding. A prominent extension lineation, trending north-northwest, is represented by elongated clasts in the Earn conglomerates, and is kinematically related to the foliation. A north-northwest-striking, moderately dipping crenulation of this foliation is discernible in argillaceous laminae and locally on foliation surfaces. Drilling and mapping in the main Silvertip deposit area indicates that no significant folds are present here, but minor thrusts do occur and larger thrusts have been mapped farther west towards the Cassiar Batholith and elsewhere in the Cassiar terrane.

Faults related to the Tootsee River fault system are Late Cretaceous through early Tertiary in age. The faults are mainly extensional with dominantly dip slip to oblique slip, east-side-down displacement. They strike predominantly north, ranging between northwest and northeast, and dip steeply. The most important fault in the deposit area is the Camp Creek fault, which in cross-section has a vertical separation in the order of several hundred metres, down to the east. Several other faults with the same general geometry are known in the area from drill hole information and surface mapping, but have much smaller, down-to-the-east displacements, in the range of metres to tens of metres.

The main area of mineralization is known in more detail because of the large amount of drilling. Here, reconstruction of the unconformity surface between the Earn and McDame groups shows that it dips gently to the south, but appears to undulate around gently southeast-plunging axes. It is not clear how much of this undulation is due to buckling and how much is the effect of block faulting or even pre-Earn dissection of the McDame.

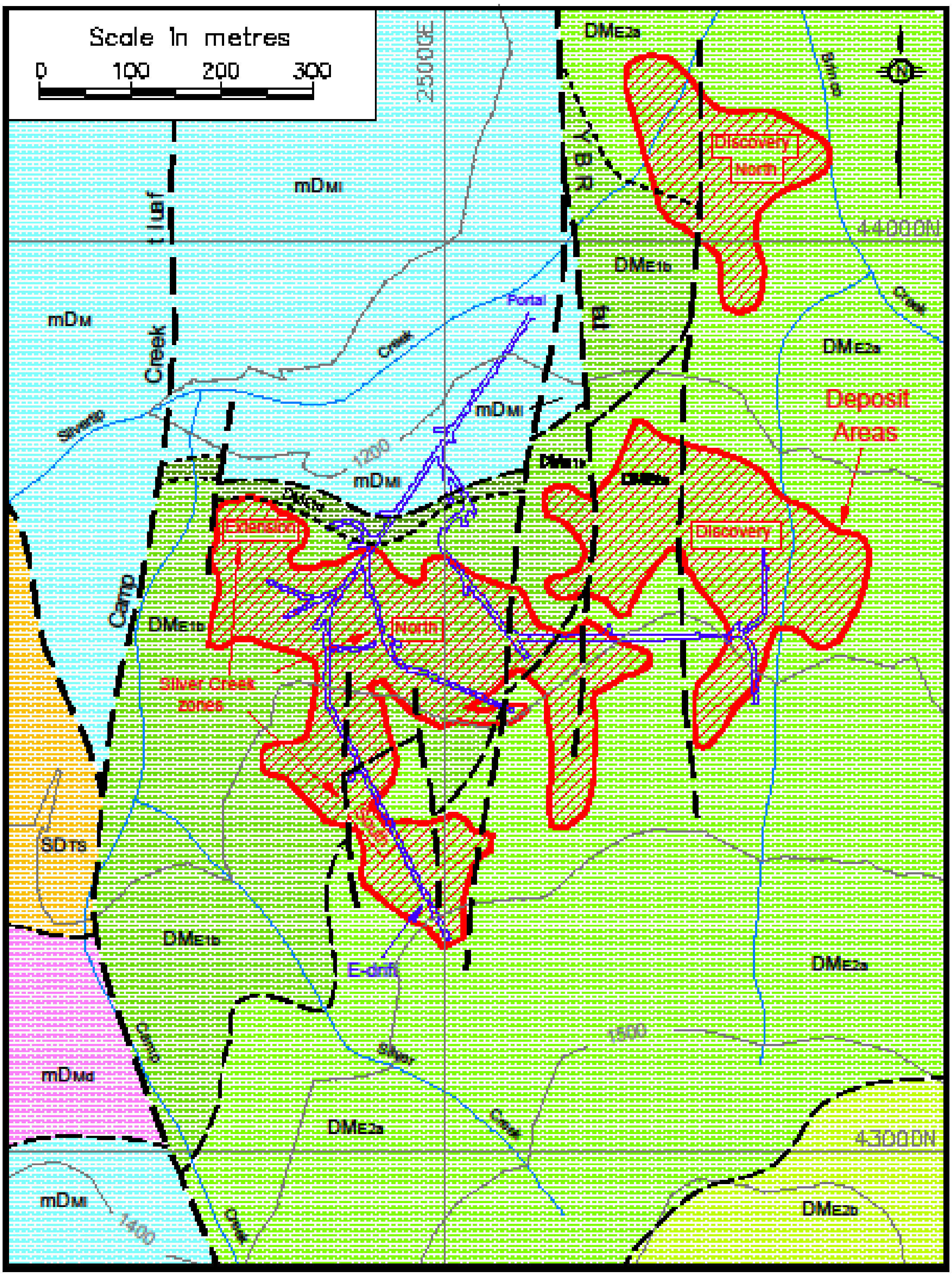


Figure 3.4: Surface geological map of the main Silvertip deposit showing the areas of mineralization and plan view of underground workings. For Legend see Fig. 3.5. (From Rees, Akelaitis and Robertson, 2000). Detailed Geological maps at a scale of 1:1500, 1:5000 and 1:10,000 are included in map pockets at the back of this report.

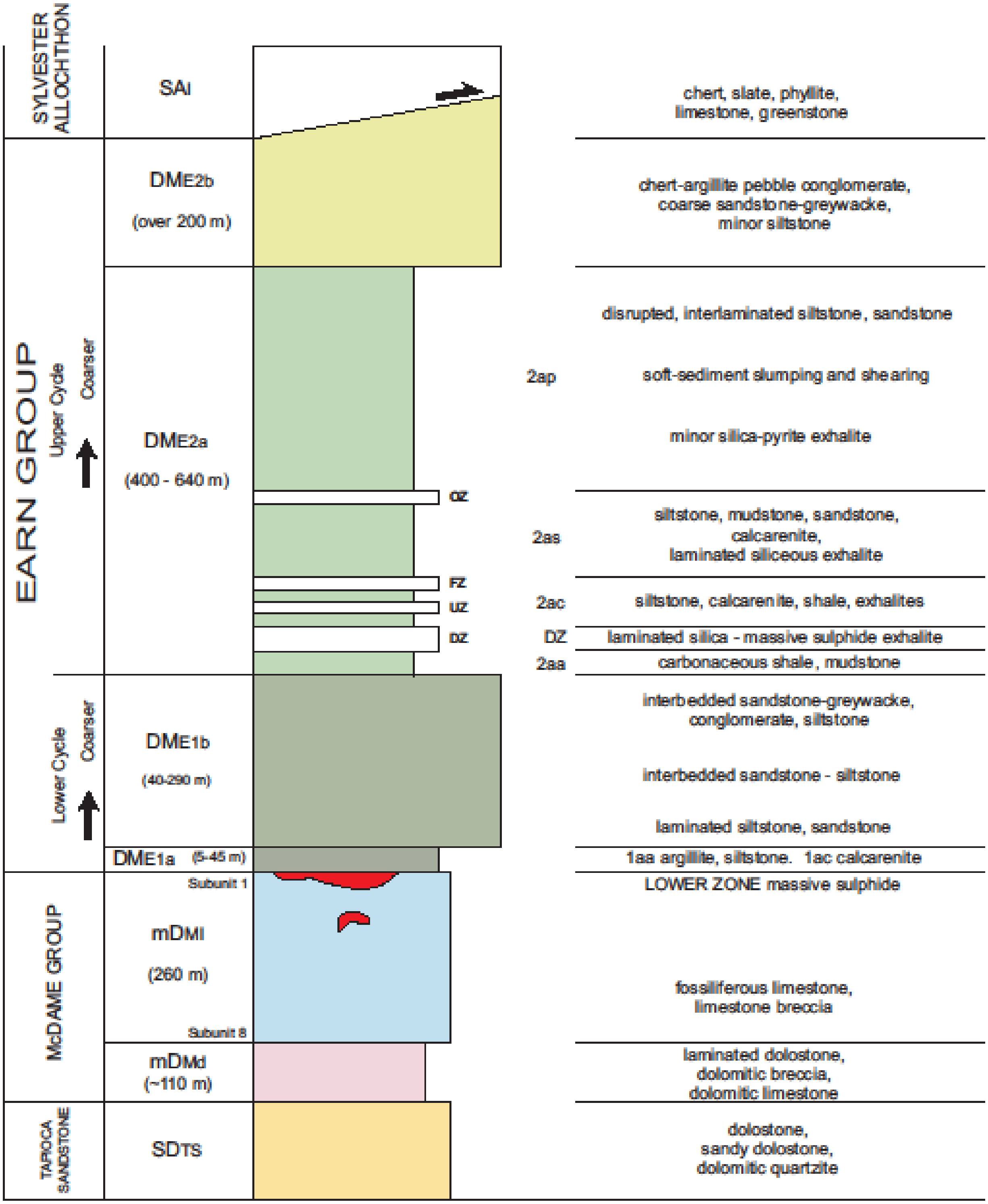


Figure 3.5: Silvertip area stratigraphic column (From Rees, Akelaitis and Robertson, 2000)

4.0 DEPOSIT TYPES

The Silvertip mineralization is a silver-zinc-lead Carbonate Replacement Deposit (CRD) (Fig. 4.1) with metals content, polyphase mineralization, abundant replacement textures, pyrite pseudomorphing pyrrhotite, and wallrock alteration reminiscent of many of the manto-chimney CRD's of Mexico and the western US (Megaw, 1998).

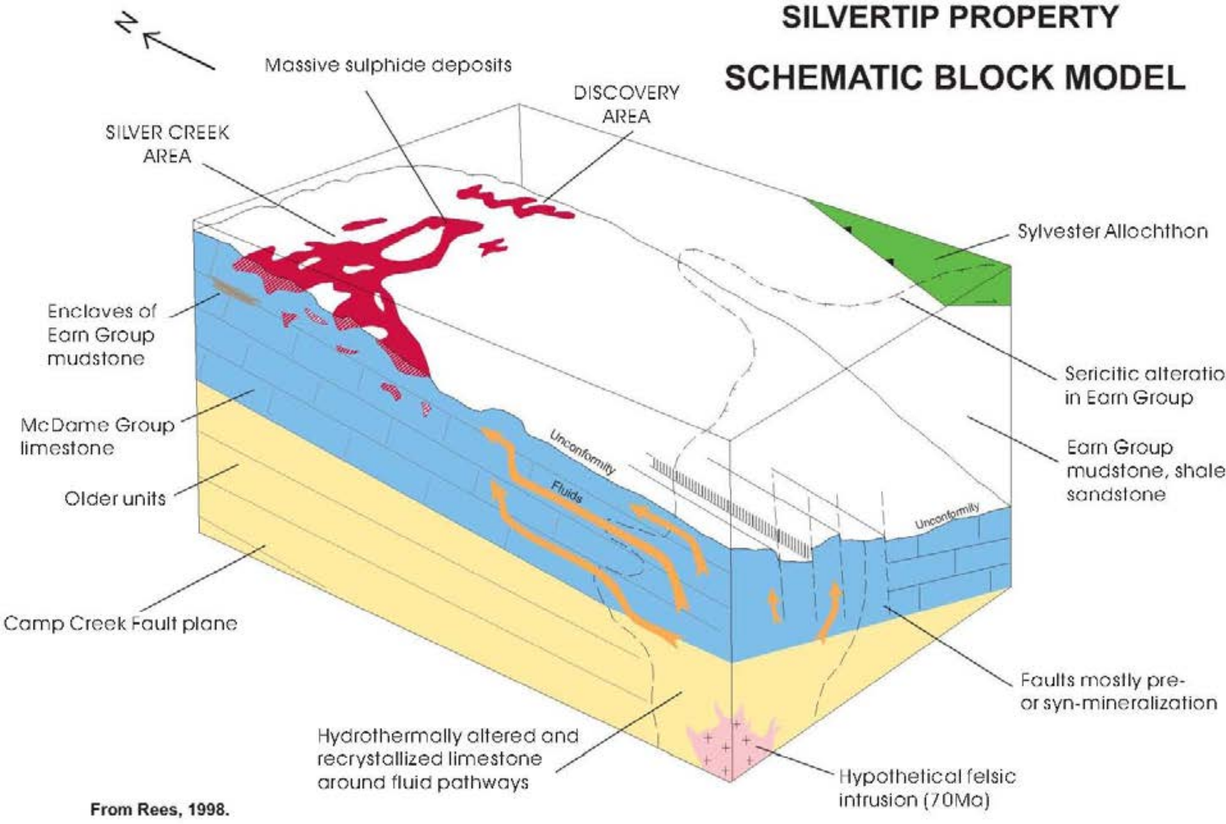


Figure 4.1: Schematic block diagram illustrating the general genetic model of the Late Cretaceous intrusive-hydrothermal system and mineralization at Silvertip. From Rees, 1998

These economically attractive, polymetallic systems can stretch continuously from copper-gold enriched skarns near intrusion contacts in the 'proximal' part of the system, to massive sulfide manto and chimney deposits with no exposed igneous relationship in the 'distal' areas. Traditionally these deposits have been considered difficult exploration targets due to a paucity of peripheral indicators to mineralization such as hydrothermal alteration or consistent relationships to breccias or structures.

In the case of Silvertip, mineralization and geology indicate that the resources identified to date probably represent the distal portions of the CRD system and that the higher grade feeder ‘chimneys’ and the proximal copper-gold skarn portions of the system have not been found. Similar manto style mineralization is found adjacent to Silvertip (Silverknife) to the west. To the northeast in the Yukon Territory, but within 30 kilometres of the Silvertip deposit similar mineralization has been found at Meister River, Jax, Head and Veronica as listed in Yukon MINFILE. These discoveries vary from drilled prospects (Meister River) to showings (Veronica) but, if genetically related to Silvertip and Silverknife, provide evidence for the presence of a larger scale mineralizing system centered in the area.

Silvertip mineralization discovered to date extends over an area 400 metres (east-west) by 600 metres (north-south). The deposit is blind to the surface, intersected by drilling at depths ranging from 75m to 300m below topography. The deposit is shallowest in the western portion and deepest to the east with dips ranging from near zero at the shallow points to in excess of 45 degrees to the east. In general the deposit conforms to the folded contact between Earn Group metasediments and McDame limestones although mineralization has been encountered enclosed entirely within the upper sediments and entirely within the lower limestones demonstrating the ‘planes of weakness’ following and ‘void filling’ nature of the distal parts of CRD style deposits.

5.0 MINERALIZATION

(excerpted from 'NI43-101 Technical Report Resource Update on the Silvertip Property, Northern British Columbia, Canada', Feb. 19, 2010)

The Silvertip mineralization consists of silver-lead-zinc massive sulphide, formed by hydrothermal replacement processes in McDame Group limestone. In Silvertip terminology it is known as 'Lower Zone' (Fig. 4.5). The main mineralized zones are not exposed, lying between about 50 and several hundred metres beneath the surface, and covered by the Earn Group. These zones are mainly north of Silvertip Mountain and east of Camp Creek (Fig. 4.4). The 'Silver Creek' area is in the west and northwest; the 'Discovery' area lies farther east and at greater depth. To the north, the 'Discovery North' area has received relatively little attention to date, but is likely continuous with the other zones.

Another type of lead-zinc sulphide mineralization is present on the property, namely Early Mississippian syngenetic 'sedex' deposits associated with siliceous to baritic exhalite subunits in unit 2A of the Earn Group (see section 'Unit 2A' above). These were the original exploration target on the property in 1980. They are not considered economic, although they are of interest because they contain a sulphide overprint that may be related to the much younger hydrothermal event that mineralized the McDame carbonates structurally below.

The main sulphide deposits formed by the interaction of hot, magmatically derived, metal-enriched hydrothermal fluids with McDame carbonate rocks (see Figure 5.1, above). The source of the fluids has not been found, but an area of quartz-sericite-pyrite alteration on the surface south and southeast of Silvertip Mountain might indicate a buried intrusion below. This alteration has a fluorine signature, and has been dated at around 70 Ma (Late Cretaceous), the same age as felsic intrusions exposed elsewhere in the region. On this basis, the mineralizing event is assumed to be Late Cretaceous in age, although it may be slightly older.

Most of the mineralization so far defined occurs at the top of the McDame limestone, at or near the unconformable contact with the Earn Group, although significant sulphides are also present much deeper in the McDame. The massive sulphides are in the form of gently plunging tubes, or cape shaped mantos, up to about 20 metres thick and 30 metres wide, and in places extending for at least 200 metres. Narrower and thicker bodies of massive sulphide, between 20 and 30 metres thick, have been intersected locally by past drilling, and are probably discordant, vertically oriented (minor) chimneys connecting mantos at different levels. Sulphide intersections deeper in the McDame are much less well defined; most are probably also mantos, but some might be parts of structurally hosted chimneys (no scale implied) or connections between stacked mantos.

Contacts between the massive sulphides and the host limestone can be remarkably sharp, but transitional zones of alteration (silicification, dolomitization), and recrystallization and brecciation are common. (Not all the dolomitization is related to the mineralization event – some is much older.) The mineralization consists of early-formed pyrite, pyrrhotite and sphalerite and lesser galena, and a slightly younger, higher temperature, sulphosalt-sulphide suite of minerals. The latter contain the main silver-bearing phases including pyrargyrite-proustite, boulangerite-jamesonite and tetrahedrite (freibergite), as well as silver-rich galena. Quartz and calcite are the main gangue minerals and locally fill late-stage vugs and cavities. Brecciation of sulphides, mixed with limestone vein quartz and calcite, attest to multiple phases of fluid infusion and intra-mineral, solution collapse processes. Unmineralized, crackle- or rubble brecciated limestone is common, as are tectonic stylolites. Some rubble and matrix breccias are of Late Devonian paleokarst origin, although they too may be infiltrated by sulphide replacement, as they represent suitable ‘ground preparation’. Paleokarst is probably only an indirect controlling factor with respect to mineralization.

The main control on the mineralization in the deposit area is the Earn unconformity which formed a relatively impermeable cap to the upwelling fluids, concentrating the development of mantos at or near the top of the McDame. The Silvertip mantos are believed to have been fed from depth, at some point in the system, by structurally controlled chimney feeders. These feeders were possibly channeled in faults such as the Camp Creek fault and numerous subsidiary fractures, or along other faults such as those in the Discovery area where the unconformity steps down to the east. Many intra-limestone mantos, which occur 100 metres or more vertically below the unconformity, probably formed by lateral fluid flow branching off from the feeders, and were controlled by a combination of structural and stratigraphic permeability contrasts. The main zone of chimney development, if it exists, has not yet been discovered, and is believed to occur closer to the thermal source of the system.

6.0 2014 Work Program

6.1 Introduction

After assuming ownership of the Silvertip property in October 2013, JDS Silver completed repairs to the access road beginning in March, 2014. The road had suffered washout of several culverts and damage to the bridge across the Rancheria River in June 2012. This work was undertaken in to facilitate geo-technical drilling on the mine site and removal of a bulk sample of ore for metallurgical testing. Only that portion of the work which took place within BC was included here. The 250 kg of bulk sample material that JDSS collected for further metallurgical testing was transported to Vancouver for processing by SGS Canada Inc in their Burnaby Laboratories under the direction of B. Cutriss, contracted to JDSS to oversee and report on results. Geotechnical work was completed by Geo-Tech Drilling and supervised by Telford Geotechnical. Also completed in 2014 was an Environmental Baseline Study for inclusion in JDSS's MAPA for a small mine. Total expenditures on this program were as follows:

Summary of 2014 Expenses	
Access Road Repair	C$634,237
Metallurgical Work	C$120,190
Geotechnical Drilling	C$77,843
Environmental Baseline Study	C$702,570
TOTAL	**C$1,534,842**

6.2 Access Road Repair

The 26km access road to the Silvertip property was damaged by flooding in June 2012. The access road joins the Alaska Highway in the Yukon at km 1128 and runs south, crossing the BC/Yukon border at km 17. The heavy rains also cut the Alaska highway just west of the Rancheria Motel and caused the Liard River to overflow it's banks damaging houses at Lower Post as well as threatening the highway bridge across the Liard River at Liard.

Damage to the JDSS access road included washout of 7 culverts, one section of roadway and damage to the southern abutment of the bridge across the Rancheria River. The BC portion of this road, which extends from the from the northern BC border cutline at km 17 to the portal at km 26 had culvert washouts at km 17.4, km 20 and km 22.4. A 600m section of roadbed was also washed out from km 18.4 to km 19.

6.2.1 Permitting for Road Repairs

Repair work for the BC portion of the road was permitted under a NOW that was extended by JDSS to expire on March 31, 2016 by granting of a deemed authorization from the Smithers office of BC Energy and Mines.

Figure 1. Excavating the stream bed for emplacement of culvert at km 22.4.

6.2.2 Description of Work

The work was contracted to JDS Energy and Mines and managed by Clint Able. Design work for the repair began in March, 2014 and the road was operational for use by light equipment at the end of July, 2014.

A plan for management of environmental impact was in place for work in or near streams that included containment for turbid water, limiting loss of equipment fluids into streams, work related erosion control and dust control.

A map illustrating the repair sites is attached as Plate 1. The repair sites included in the expense claim are located between the BC border and the Silvertip Portal. This portion of the access road, from Km 17 to km 26 was washed out in 4 places as follows:

- Km 17.4	Culvert Washout
- Km 18.4 to Km 19	Roadbed Washout
- Km 20	Culvert Washout and Stream Diversion
- Km 22.4	Culvert Washout and Roadbed Washout

Repairs included culvert replacement, roadbed re-profiling and re-establishment of a stream bed blocked by gravel and detritus related to the flooding.

6.3 Metallurgical Sample

A metallurgical sample was collected from surface stock piled ore and submitted to SGS Labs in Vancouver for metallurgical testing. 250 kg of unconsolidated ore, stored at surface from tunneling activity in the late 1980's.

Robert Cutriss supervised the analytical and testing program of this ore. Analysis was carried out by SGS Laboratories in Burnaby, BC. The ore has been stockpiled on surface since the mid 1980's and it's provenance with relation to the various major lobes of the Silvertip deposit has been lost. In addition, surface storage over 30 years has allowed weathering, in the form of oxidation to occur. Oxidation has probably altered the reaction of this ore to process testing so results are not considered definitive as an indication of recovery, but are useful as a guideline for process design. In addition, insufficient ore grade material from the 65 zone was available for complete testing of that zone. Results from the Silver Creek zone and the Discovery zone have provided the bulk of the material for this stage of metallurgical testing.

6.3.4 History of Metallurgical Testing

Metallurgical testwork over a number of years confirms that the Silvertip ore responds well to conventional floatation processing. Imperial Metals 1998 program achieved the following results, focusing on Zn recovery as that was the most valuable metal by volume that was anticipated to be produced.

Ag	2200g/t in Pb concentrate, 65% recovery of Ag
Pb	60% in Pb concentrate, 80% recovery of Pb
Zn	63% in Zn concentrate, 94% recovery of Zn

Imperial's program was terminated as the economics of the deposit were compromised by low metal prices. Their reporting included the observation that these results were

"conservative for the deposit as a whole" as "...some samples/composites consistently produce very high recoveries and concentrate grades..."

6.3.3 Development of the Current Process Flow Sheet*

*(excerpted from Cutriss Contract Metallurgy Ltd Memo, January 14, 2015)

The Silvertip Project as currently proposed by JDS Silver will process ore through the re-located Sa Dena Hes (SDH) plant. The Sa Dena Hes process followed a conventional Pb-Zn flowsheet and indications to date are that the Silvertip ore responds well to this generic approach.
Test work was conducted on four composite samples representing the major zones within the deposit:

> Silver Creek Zone – has the highest tonnage and grades, but more difficult metallurgy. Lead sulfosalts, arsenopyrite, and organic carbon contents are all higher in this zone.
>
> 65 Zone – somewhat lower grades than Silver Creek, but delivers higher metal recoveries.
>
> Discovery and Discovery North Zones – lower Pb and Ag head grades are reflected in lower concentrate grades while recoveries are similar to 65 Zone; intermediate zinc performance.

Two series of metallurgical tests were conducted by SGS Vancouver and were reported in October 2011[1] and November 2013 [2]. The programs focused on the higher grade Silver Creek and 65 Zone samples, but were quite limited in scope (the 2011 program involved only 24 tests, including

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[1] "Metallurgical Testwork on the Pb/Zn/Ag Samples from the Silvertip Property", prepared for Silvercorp Metals Inc. Project 50091-001. Final Report October 19, 2011. SGS Canada Inc, Vancouver, BC

[2] "Preparation of Environmental and UCS Testing Material using 65 Zone and Silver Creek Pb/Zn/Ag Composites from the Silvertip Property", prepared for Silvercorp Metals Inc. Project 13933-001 Final Report November 21, 2013

15 on Silver Creek and three on 65 Zone material; the 2013 program totaled 11 tests, six on Silver Creek and five on 65 Zone). The absence of variability tests is a concern, but the composites themselves cover a range of grades and mineral assemblages. Additional variability testing is proposed when fresh sample becomes available.

The complete memo is located in Appendix 3.

6.4 Geo-Technical Drilling

A short program of geo-technical drilling to evaluate foundations for mine structures was completed between June 21 and June 30, 2014. Evaluation of the ground was based on standard penetration tests (SPT's). A description of the program including hole locations and analysis is included as appendix 4.

6.5 Environmental Impact Assessment and Water Quality Prediction

In support of an application for a Mines Act Permit JDSS completed the Mines Act Permit Application (MAPA) with Klohn Crippen Berger. This document covers the environmental impact assessment of the project and mitigation measures.

Upon review of this document, Mines Department experts required further work on water quality modelling for waters released to the environment from the project site. JDSS commissioned this report on Water Quality Prediction from Lorax Environmental. The report was completed in October 2014 and completed the requirements for the MAPA. Total expenditure on the MAPA Environmental Impact Assessment and the Water Quality Prediction Model was C$670,093.00.

A copy of these reports is attached as appendix 5.

7.0 Conclusions and Recommendations

7.1 Conclusions

The Silvertip Project will focus on mine development during 2015 and 2016, following granting of a mining permit, expected in the 2nd quarter of 2015. Work completed in 2014 provided data and analysis in support of a MAPA submission, established baseline conditions for construction of mine infrastructure on site, and allowed finalization of process design for the mill circuit. JDSS has now compiled sufficient data and analysis to support the first steps toward construction of the mine infrastructure required to support a start-up of mining operations as early as 4th quarter 2015.

7.2 Recommendations

7.2.1 Exploration

Exploration should focus on resource expansion and upgrading of inferred resources to indicated resources over the next two years. This work will follow underground development, taking advantage of mine workings to establish drill stations in areas of highest potential for discovery of additional resources.

Following successful launch of the underground exploration program, step out drilling to assess the potential for drill intersections made during 2010 to indicate satellite deposits should be started. Targets include Discovery North and the 28 zone.

As a third priority, exploration will continue on a property wide basis, particularly along the sediment/limestone contact to the south and north of the Silvertip deposit where surface geochemical and geophysical work indicate the possibility for discovery of further polymetallic, manto style deposits at depth.

7.2.2 Metallurgical Work

Testing of the response of Silvertip ore to metallurgical processes should be an ongoing priority. JDSS's work to date has advanced the understanding of the ore's response to conventional recovery methods to the point where a mill can be designed around the available data. Further work will seek to refine the process further.

Attachments

Attachment 1 – JDS Silver Tenure Map
Attachment 2 – Access Road Repair Sites

JDS Silver Claims with Generalized Geology

JDS SILVER

N

🟥 JDSS

0 1 2 4 Kilometers

NAD83, UTM Zone 9

1:75,000

Drawn By: R D Cullen

Date: April 28, 2015

RD Consult
Exploration and Development

Rock Types (Generalized Geology)

🟪	Granite, alkali feldspar granite, intrusive rocks
🟦	Metamorphic rocks, undivided
🟨	Chert, siliciclastic rocks
🟫	Argillite, greywacke, wacke, conglomerate turbidites
🟦	Limestone, marble, calcareous sedimentary rocks
🟨	Felsic volcanic rocks
🟩	Mafic volcanic rocks
🟪	Ultramafic rocks

Silvertip Access Road Showing Flood Damage Repair Sites

JDS SILVER

NAD83, UTM Zone 9
1:10,000

Drawn By: R D Cullen
Date: April 28, 2015

RD Consult
Exploration and Development

Appendix One

Silvertip Project
2014 Expenses

JDS Silver 2014 Work Programs

Item	Total $
Geotech Drilling	$77,843.44
Road Repair	$634,237.60
Metallurgy	$120,190.00
Environmental Assessment	$702,570.90
Total	**$1,534,841.94**

Total First Nations	$40,575.00

Access Road Repair Work

Provider	Item	Usage	$ total	Rate	Period
JDS	Cessna Citation Bravo	4.33	$10,392.00	$2,400.00	$/hr
JDS	1994 Cat D7H Dozer	2.75	$42,000.00	$15,272.73	$/mo
JDS	2001 Cat 320 CL Excavator	0.33	$5,445.00	$16,500.00	$/mo
JDS	2001 Cat D3C LGP Tractor	3.75	$7,875.00	$2,100.00	$/mo
JDS	2001 Cat IT28G Loader	8.33	$19,921.50	$2,391.54	$/wk
JDS	2003 Cat 330 CL Excavator	3.00	$48,000.00	$16,000.00	$/mo
JDS	2007 Cat 735 Wiggle Wagon	8.33	$59,415.00	$7,132.65	$/wk
JDS	2012 Cat 262C Skid Steer	1.75	$7,312.50	$4,178.57	$/mo
JDS	Drum Packer 80"+(CS563/SD100+)	2.33	$24,570.00	$10,545.06	$/mo
JDS	Goose Neck Utility Trailer	1.75	$1,470.00	$840.00	$/mo
MacPherson Rentals	Generator 4 wk rental	1.00	$562.00	$562.00	$/mo
MacPherson Rentals	Hilti TE 76 Hammer/Drill	1.00	$387.00	$387.00	$/mo
MacPherson Rentals	Bobcat Rent	1.00	$4,586.34	$4,586.34	$/mo
MacPherson Rentals	Boom Rental	1.00	$5,775.10	$5,775.10	$/mo
MacPherson Rentals	Generator/Trash Pump	1.00	$3,972.00	$3,972.00	$/mo
MacPherson Rentals	Straight Boom	1.00	$8,180.86	$8,180.86	$/mo
JDS	Ford F350 Super Duty #16	3.00	$8,100.00	$2,700.00	$/mo
JDS	Ford F350 Super Duty #3	1.00	$2,800.00	$2,800.00	$/mo
JDS	Ford F350 Super Duty #4	1.00	$2,700.00	$2,700.00	$/mo
MacPherson Rentals	Tool Rentals	11.00	$29,430.51	$2,675.50	$/mo
Sub Total			**$292,894.81**		

	Materials	Qnty	$ total		
	Canada Culvert: 1000mm X 2.8mm Galvanized	1	$3,747.60		
	Canada Culvert: 1500mm X 2.8mm Galvanized	1	$14,822.40		
	Canada Culvert: 1800mm X 2.8mm Galvanized	1	$17,822.40		
	Canada Culvert: 2400mm X 2.8mm Galvanized	1	$23,484.00		
	Bluestone: 90 -Lock Blocks	1	$29,700.00		
	Epic Polymer: 1/2" Thick, 600mmx900 Bearing P	4	$4,225.20		
	Epic Polymer: Epic Stongbond Blk Adh. Compou	3	$220.95		
	Epic Polymer: Freight Exp: Mobilization	1	$290.00		
	Shepherd's: Job Materials	1	$12,732.55		
	Nilex: Roll of Nilex	8	$1,920.00		
	Nilex: Uniaxial Geogrid	22	$6,204.00		
	Twilite: Culverts	1	$588.00		
Sub Total			**$115,757.10**		

Personnel	Item	Usage	$ total	Rate	Period
Ben Vorstermans	Operator	156.00	$12,480.00	$80.00	$/hr
Blake Hill	Supervisor	1.00	$170.00	$170.00	$/hr
Bruce Abel	Operator	482.00	$38,560.00	$80.00	$/hr
Clint Stibbard	Laborer	50.00	$3,250.00	$65.00	$/hr
Clinton Abel	Manager	578.00	$119,068.00	$206.00	$/hr
Cody Stibbard	Laborer	154.00	$7,700.00	$50.00	$/hr
Dave Humes	Operator	278.00	$22,240.00	$80.00	$/hr
Dmitry Chouliak	Operator	11.50	$1,092.50	$95.00	$/hr
Doug Van Bibber	Operator	483.00	$38,640.00	$80.00	$/hr
Gerald Taylor	Operator	326.00	$26,080.00	$80.00	$/hr
Gordon Bussieres	Manager	14.00	$4,620.00	$330.00	$/hr
Jerry Warren	Operator	4.00	$320.00	$80.00	$/hr
Joey Wallick	Laborer	376.00	$18,800.00	$50.00	$/hr
Josh Krysta	Laborer	496.00	$24,800.00	$50.00	$/hr
Kevin Mather	Manager	2.00	$660.00	$330.00	$/hr
Kevin Whieldon	Supervisor	14.00	$2,450.00	$175.00	$/hr
Lawrence Cordell	Supervisor	51.00	$6,120.00	$120.00	$/hr
Matt Gilchrist	Laborer	132.00	$6,600.00	$50.00	$/hr
Randy Brodziak	Manager	418.00	$103,664.00	$248.00	$/hr
Sean Meldrum	Supervisor	108.00	$12,175.00	$112.73	$/hr
Steve Wallick	Laborer	2.00	$130.00	$65.00	$/hr
Travis Rance	Operator	84.00	$6,720.00	$80.00	$/hr
Iyon Kechika: Professional Services		invoice	$1,530.00		
Iyon Kechika: Labour - Melanie Sept 7-23		invoice	$405.00		
Sub Total			**$458,274.50**		

JDSS

Silvertip Project - 2014 Assessment Report

	Expenses	Qnty	$ total		
	Accommodations	7	$1,006.29		
	Airfare	22	$5,266.86		
	Ajax Steel: Small Tools & Supplies	4	$2,497.49		
	AllNorth: Prof Serv	1	$25,150.37		
	Baggage Fee	1	$54.39		
	Boreal: Professional Services	1	$119,815.01		
	Dollar Saver: Freight Expense	1	$3,850.00		
	Dollar Saver: Job Supplies	60	$6,900.00		
	Fuel-Petro Pass	2	$426.92		
	Job Office Supplies	1	$5,299.05		
	Living Out Allowance (LOA)	154	$36,850.34		
	Meals	16	$1,739.53		
	Mercer Cont: Hauling	1	$14,350.00		
	Nilex: Freight Expense	1	$4,800.00		
	Northern Hos: Level 3 1st Aid Kits	1	$310.00		
	Paint	1	$43.99		
	Per Diem	2	$123.80		
	Vehicles - Gas & Oil	1	$57,751.12		
	Safety Gear	1	$44.99		
	Shepherd's: Freight	1	$3,400.00		
	Small's Expediting	1	$3,114.74		
	Superpass: Vehicle - Gas & Oil	1	$2,971.88		
	Transportation	11	$949.77		
	WG Davis: Mobilize Equipment	1	$52,820.00		
	Administration Fee	9.00	$52,012.25		
Sub Total			**$401,548.79**		
Grand total		**$1,268,475.20**			
BC Portion			**$634,237.60**		

JDSS

Silvertip Project - 2014 Assessment Report

Geotech Drilling

Mob/Demob	**Usage**	**$ total**	**Rate**	**Period**
June 21 to June 30		$8,366.40		
Sub Total		**$8,366.40**		
Drilling	**Usage**	**$ total**	**Rate**	**Period**
(Odex/Coring) moving, set-up, insallations, etc.	73.5	$20,528.55	266	$/hr
Sub Total		**$20,528.55**		
Equipment	**Usage**	**$ total**	**Rate**	**Period**
Pickup Truck	8.00	$1,260.00	$150.00	$/dy
Support Trailer	8.00	$0.00	$0.00	$/dy
Support Vehicle	8.00	$2,486.40	$296.00	$/dy
Compressor	5.00	$3,018.75	$575.00	$/dy
Consultant Vehicle	1.00	$2,083.47		cst
Sub Total		**$8,848.62**		
Personnel	**Usage**	**$ total**	**Rate**	**Period**
Regular time (Safety Meetings)	4.50	$704.03	$149.00	$/hr
Overtime	30.50	$2,401.88	$75.00	$/hr
Consultant	11.75	$2,467.50	$200.00	$/hr
Consultant	107.00	$16,852.50	$150.00	$/hr
Sub Total		**$22,425.90**		
Materials	**Usage**	**$ total**	**Rate**	**Period**
Bentonite Chips	10	$299.78	$28.55	bag
Sub Total		**$299.78**		
Expenses	**Usage**	**$ total**	**Rate**	**Period**
Accommodations	10	$4,882.50	$465.00	$/dy
Crew Travel	18	$2,816.10	$149.00	$/hr
NSC	1	$733.95	$699.00	$/dy
Bit Wear	193	$1,165.24	$5.75	$/ft
Consultant Accommodation	9	$1,204.35		cst
Sample Shipment		$312.05		cst
Sub Total		**$11,114.19**		
Report Prep.	**Usage**	**$ total**	**Rate**	**Period**
Geotechnical Report Preparation		$5,000.00		flatfee
Test Hole Logs	8.00	$1,260.00	$150.00	$/Log
		$6,260.00		
Grand total		***$77,843.44***		

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Silvertip Project - 2014 Assessment Report

MAPA Environmental Impact Assessment

Date	Provider	Category	Type of Work	Amount
02-28-14	Austin Hitchins	Environmental	Environmental Consulting	$3,000.00
03-31-14	Austin Hitchins	Environmental	Environmental Consulting	$1,000.00
04-30-14	Austin Hitchins	Environmental	Environmental Consulting	$1,125.00
05-31-14	Austin Hitchins	Environmental	Environmental Consulting	$2,875.00
07-31-14	Austin Hitchins	Environmental	Environmental Consulting	$4,875.00
11-30-14	Austin Hitchins	Environmental	Environmental Consulting	$1,375.00
01-31-14	Ecofor	Environmental	Environmental Services	$7,610.00
01-31-14	Hemmera	Environmental	Environmental Services	$36,573.01
01-31-14	KCB	Environmental	Environmental Services	$23,116.05
02-28-14	Hemmera	Environmental	Environmental Services	$6,832.50
03-31-14	Hemmera	Environmental	Environmental Services	$5,651.25
03-31-14	KCB	Environmental	Environmental Services	$179,468.83
04-30-14	Hemmera	Environmental	Environmental Services	$2,835.00
05-31-14	Hemmera	Environmental	Environmental Services	$1,166.25
05-31-14	KCB	Environmental	Environmental Services	$2,114.10
05-31-14	KCB	Environmental	Environmental Services	$310.50
09-30-14	KCB	Environmental	Environmental Services	$19,538.25
10-31-14	Loralee Johnstone	Environmental	Environmental/Permitting	$4,558.00
12-30-14	SGS	Environmental	Process Testwork	$1,952.00
07-31-14	Lorax	Environmental	Water Chemistry	$32,216.81
08-31-14	Lorax	Environmental	Water Chemistry	$39,869.03
09-30-14	Lorax	Environmental	Water Chemistry	$77,010.57
10-31-14	Lorax	Environmental	Water Chemistry	$55,113.66
12-27-14	Lorax	Environmental	Water Chemistry	$58,637.53
11-30-14	Lorax	Environmental	Water Treatment	$76,761.18
01-31-14	Global ARD Testing	Environmental	Water Treatment Plant Design	$1,937.25
02-28-14	Global ARD Testing	Environmental	Water Treatment Plant Design	$15,146.50
04-30-14	Global ARD Testing	Environmental	Water Treatment Plant Design	$1,787.75
10-31-14	Global ARD Testing	Environmental	Water Treatment Plant Design	$5,636.75
11-30-14	Access Mining Consultants	Technical Work	Geochemical Water	$12,983.34
12-04-14	Access Mining Consultants	Technical Work	Geochemical Water	$977.70
01-31-14	Access Mining Consultants	Technical Work	Geochemical Water Samples	$6,653.68
04-30-14	Access Mining Consultants	Technical Work	Geochemical Water Samples	$1,077.57
05-31-14	Access Mining Consultants	Technical Work	Geochemical Water Samples	$1,017.19
07-31-14	Access Mining Consultants	Technical Work	Geochemical Water Samples	$9,768.65
Grand Total				**$702,570.90**

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Silvertip Project - 2014 Assessment Report

Metallurgical Report

Date	Provider	Category	Type of Work	Amount
08-31-14	BGC	Technical Work	Geotechnical	$12,677.50
07-31-14	BGC	Technical Work	Geotechnical Design	$10,895.00
12-27-14	BGC	Technical Work	Geotechnical Design	$2,710.00
03-31-14	McElhanney	Technical Work	LiDAR Survey	$22,500.00
01-31-14	Robert Cuttriss	Technical Work	Metallurgy	$26,062.50
02-28-14	Robert Cuttriss	Technical Work	Metallurgy	$812.50
07-31-14	Robert Cuttriss	Technical Work	Metallurgy	$10,000.00
08-31-14	Robert Cuttriss	Technical Work	Metallurgy	$5,687.50
09-30-14	Robert Cuttriss	Technical Work	Metallurgy	$3,500.00
11-30-14	Process Mineralogical	Technical Work	Mineralogical Studies	$8,585.00
11-30-14	Process Mineralogical	Technical Work	Mineralogical Studies	$2,276.00
10-31-14	SGS	Technical Work	Process Testwork	$7,970.00
12-12-14	SGS	Technical Work	Process Testwork	$3,078.00
12-12-14	SGS	Technical Work	Process Testwork	$3,436.00
Grand TOTAL				**$120,190.00**

Appendix Two

Silvertip Property Tenures

Title Number	Claim Name	Owner	Map Number	Issue Date	Good To Date	Status	Area (ha)
221837	TOOTS 4	278785 (100%)	104O099	1979/jul/06	2015/oct/15	GOOD	500.0
221908	RENEE 1	278785 (100%)	104O099	1979/nov/02	2015/oct/15	GOOD	300.0
222004	BETH 1	278785 (100%)	104O089	1980/aug/08	2015/oct/15	GOOD	300.0
222005	BETH 2	278785 (100%)	104O089	1980/aug/08	2015/oct/15	GOOD	500.0
222006	BETH 3	278785 (100%)	104O089	1980/aug/08	2015/oct/15	GOOD	500.0
222007	BETH 4	278785 (100%)	104O089	1980/aug/08	2015/oct/15	GOOD	450.0
509654		278785 (100%)	104O	2005/mar/24	2015/oct/15	GOOD	1263.252
509655		278785 (100%)	104O	2005/mar/24	2015/oct/15	GOOD	1068.893
509656		278785 (100%)	104O	2005/mar/24	2015/oct/15	GOOD	971.695
509657		278785 (100%)	104O	2005/mar/24	2015/oct/15	GOOD	1264.214
509658		278785 (100%)	104O	2005/mar/24	2015/oct/15	GOOD	421.949
509808		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1053.504
509809		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1297.541
509810		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	761.694
509812		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1171.879
509840		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1138.671
509841		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1170.563
509843		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1220.191
509865		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1299.282
509868		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1249.576
509875		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	649.177
509876		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	1152.875
509883		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	910.231
509885		278785 (100%)	104O	2005/mar/30	2015/oct/15	GOOD	518.718
510224		278785 (100%)	104O	2005/apr/05	2015/oct/15	GOOD	1119.047
708322	SILVERTIP WEST 1	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	404.8321
708362	SILVERTIP WEST 2	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	259.2326
708382	SILVERTIP WEST 3	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	388.9648
708422	SILVERTIP WEST 4	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	356.7382
708442	SILVERTIP WEST 5	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	389.1587
708462	SILVERTIP WEST 6	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	389.3624
708482	SILVERTIP WEST 7	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	405.5663
708522	SILVERTIP WEST 8	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	389.5887
708543	SILVERTIP WEST 9	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	373.331
708562	SILVERTIP WEST 10	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	389.7911
708582	SILVERTIP WEST 11	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	389.9809
708602	SILVERTIP WEST 12	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	390.1709
708622	SILVERTIP WEST 13	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	390.3789

Title Number	Claim Name	Owner	Map Number	Issue Date	Good To Date	Status	Area (ha)
708642	SILVERTIP SOUTH 1	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	390.5986
708662	SILVERTIP SOUTH 2	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	406.9948
708682	SILVERTIP SOUTH 3	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	407.1198
708702	SILVERTIP SOUTH 4	278785 (100%)	104O	2010/feb/26	2015/oct/15	GOOD	390.9192
713762	SILVERTIP WEST 14	278785 (100%)	104O	2010/mar/04	2015/oct/15	GOOD	405.1174
715942	SILVERTIP EAST 1	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.6112
715962	SILVERTIP EAST 2	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.4459
715983	SILVERTIP EAST 3	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.385
716002	SILVERTIP EAST 4	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	390.0148
716062	SILVERTIP EAST 5	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.049
716102	SILVERTIP EAST 6	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.1676
716142	SILVERTIP EAST 7	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	389.7202
716162	SILVERTIP EAST 8	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	389.7157
716202	SILVERTIP EAST 9	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.1475
716222	SILVERTIP EAST 10	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.3264
716242	SILVERTIP EAST 11	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	406.5134
716282	SILVERTIP SOUTH 4	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	391.0016
716362	SILVERTIP SOUTH 5	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.4092
716463	SILVERTIP SOUTH 6	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.4089
716483	SILVERTIP SOUTH 7	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.411
716542	RANCH 1	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.5794
716562	RANCH 2	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.6608
716582	RANCH 3	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.7552
716602	RANCH 4	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	391.4007
716622	RANCH 5	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.6095
716642	RANCH 6	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	407.9162
716663	RANCH 7	278785 (100%)	104O	2010/mar/05	2015/oct/15	GOOD	391.623
766322	HD	278785 (100%)	104O	2010/may/03	2015/oct/15	GOOD	388.5637
766402	HD2	278785 (100%)	104O	2010/may/03	2015/oct/15	GOOD	388.7839
766462	HD3	278785 (100%)	104O	2010/may/03	2015/oct/15	GOOD	388.6751
766483	HD4	278785 (100%)	104O	2010/may/03	2015/oct/15	GOOD	389.0307
1033743		278785 (100%)	104O	2015/jan/29	2016/jan/29	GOOD	16.2241

Title Number	Claim Name	Owner	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Status	Area (ha)
221837	TOOTS 4	278785 (100%)	Mineral	Claim	104O099	1979/jul/06	2022/oct/15	GOOD	500.0
221908	RENEE 1	278785 (100%)	Mineral	Claim	104O099	1979/nov/02	2022/oct/15	GOOD	300.0
222004	BETH 1	278785 (100%)	Mineral	Claim	104O089	1980/aug/08	2022/oct/15	GOOD	300.0
222005	BETH 2	278785 (100%)	Mineral	Claim	104O089	1980/aug/08	2022/oct/15	GOOD	500.0
222006	BETH 3	278785 (100%)	Mineral	Claim	104O089	1980/aug/08	2022/oct/15	GOOD	500.0
222007	BETH 4	278785 (100%)	Mineral	Claim	104O089	1980/aug/08	2022/oct/15	GOOD	450.0
509654		278785 (100%)	Mineral	Claim	104O	2005/mar/24	2022/oct/15	GOOD	1263.252
509655		278785 (100%)	Mineral	Claim	104O	2005/mar/24	2020/oct/15	GOOD	1068.893
509656		278785 (100%)	Mineral	Claim	104O	2005/mar/24	2020/oct/15	GOOD	971.695
509657		278785 (100%)	Mineral	Claim	104O	2005/mar/24	2022/oct/15	GOOD	1264.214
509658		278785 (100%)	Mineral	Claim	104O	2005/mar/24	2022/oct/15	DEMI 2015/sep/01	421.949
509808		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	1053.504
509809		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	1297.541
509810		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	761.694
509812		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2022/oct/15	GOOD	1171.879
509840		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2022/oct/15	GOOD	1138.671
509841		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	1170.563
509843		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2022/oct/15	GOOD	1220.191
509865		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2022/oct/15	GOOD	1299.282
509868		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	1249.576
509875		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	649.177
509876		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	1152.875
509883		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2022/oct/15	GOOD	910.231
509885		278785 (100%)	Mineral	Claim	104O	2005/mar/30	2020/oct/15	GOOD	518.718
510224		278785 (100%)	Mineral	Claim	104O	2005/apr/05	2022/oct/15	DEMI 2015/sep/01	1119.047
708322	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	404.8321
708362	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	259.2326
708382	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	388.9648
708422	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	356.7382
708442	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	389.1587
708462	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	389.3624
708482	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	405.5663
708522	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	389.5887
708543	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	373.331
708562	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	389.7911
708582	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	389.9809
708602	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	390.1709
708622	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	390.3789
708642	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	390.5986
708662	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	406.9948
708682	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	407.1198
708702	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/feb/26	2020/oct/15	GOOD	390.9192
713762	SILVERTIP WEST	278785 (100%)	Mineral	Claim	104O	2010/mar/04	2020/oct/15	GOOD	405.1174
715942	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.6112
715962	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.4459
715983	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.385
716002	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	390.0148
716062	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.049
716102	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.1676
716142	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	389.7202
716162	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	389.7157
716202	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.1475
716222	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.3264
716242	SILVERTIP EAST	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	406.5134
716282	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	391.0016
716362	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.4092
716463	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.4089
716483	SILVERTIP SOUT	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.411
716542	RANCH 1	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.5794
716562	RANCH 2	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.6608
716582	RANCH 3	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.7552
716602	RANCH 4	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	391.4007
716622	RANCH 5	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.6095
716642	RANCH 6	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	407.9162
716663	RANCH 7	278785 (100%)	Mineral	Claim	104O	2010/mar/05	2020/oct/15	GOOD	391.623
766322	HD	278785 (100%)	Mineral	Claim	104O	2010/may/03	2015/oct/15	GOOD	388.5637
766402	HD2	278785 (100%)	Mineral	Claim	104O	2010/may/03	2015/oct/15	GOOD	388.7839
766462	HD3	278785 (100%)	Mineral	Claim	104O	2010/may/03	2015/oct/15	GOOD	388.6751
766483	HD4	278785 (100%)	Mineral	Claim	104O	2010/may/03	2015/oct/15	GOOD	389.0307
1033743		278785 (100%)	Mineral	Claim	104O	2015/jan/29	2016/jan/29	GOOD	16.2241

Appendix Three

Silvertip Metallurgical Report

Cuttriss Contract Metallurgical Ltd.

Cuttriss Contract Metallurgy Ltd. rcuttriss@metnet.ca

MEMO

TO: Kevin Weston

FROM: Bob Cuttriss

DATE 8th January, 2014

SUBJECT: Silvertip Project – Review of SGS Metallurgy Testwork, 2011, 2013.

Summary

The test work conducted by SGS for Silvercorp Metals on samples from the Silvertip Project is reviewed in detail.

Areas where the flowsheet can be improved or optimized are identified. Additional test work is required to optimize the flowsheet.

1 Introduction

The Silvertip Project as currently proposed by JDS Silver will process ore through the re-located Sa Dena Hes (SDH) plant. The Sa Dena Hes process followed a conventional Pb-Zn flowsheet and indications to date are that the Silvertip ore responds well to this generic approach.

Test work was conducted on four composite samples representing the major zones within the deposit:

> Silver Creek Zone – has the highest tonnage and grades, but more difficult metallurgy. Lead sulfosalts, arsenopyrite, and organic carbon contents are all higher in this zone.
>
> 65 Zone – somewhat lower grades than Silver Creek, but delivers higher metal recoveries.
>
> Discovery and Discovery North Zones – lower Pb and Ag head grades are reflected in lower concentrate grades while recoveries are similar to 65 Zone; intermediate zinc performance.

Two series of metallurgical tests were conducted by SGS Vancouver and were reported in October 2011[1] and November 2013 [2]. The programs focussed on the higher grade Silver Creek and 65 Zone samples,

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[1] "Metallurgical Testwork on the Pb/Zn/Ag Samples from the Silvertip Property", prepared for Silvercorp Metals Inc. Project 50091-001. Final Report October 19, 2011. SGS Canada Inc, Vancouver, BC

but were quite limited in scope (the 2011 program involved only 24 tests, including 15 on Silver Ck and three on 65 Zone material; the 2013 program totalled 11 tests, six on Silver Ck and five on 65 Zone). The absence of variability tests is a concern, but the composites themselves cover a range of grades and mineral assemblages. Additional variability testing is proposed when fresh sample becomes available.

The comments below primarily address the Silver Creek test results as this zone holds the greatest potential upside from a metallurgical perspective.

2 Sample head grades

SGS received a total of 229 individual samples that were blended into four composite samples representing the major zones in the deposit. The individual samples were identified by sample number, hole, and intercept (SGS 2011 Report, Appendix A), but individual sample grades were not presented in this report and there is no information concerning the range of grades incorporated into each composite. Presumably these data can be obtained from the geology/exploration database.

The head grades of the metallurgical composites are:

Element	Units	Silver Creek	Discovery	Discovery North	65 Zone
Pb	%	7.16	2.67	3.22	5.56
Zn	%	9.09	7.59	7.59	8.81
Ag	g/t	336	155	214	342
Au	g/t	1.35	0.07	0.35	0.45
Cu	%	0.11	0.09	0.2	0.1
Fe	%	20	15	26	19.1
S	%	26.6	19.8	28.5	26.1
As	%	0.83	0.35	0.46	0.38
Sb	g/t	2100	560	690	380
C-tot	%	2.27	1.3	1.85	2.52
CO3	%	9.77	3.92	8.12	11.6

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[2] "Preparation of Environmental and UCS Testing Material using 65 Zone and Silver Creek Pb/Zn/Ag Composites from the Silvertip Property", prepared for Silvercorp Metals Inc. Project 13933-001 Final Report November 21, 2013

3 Mineralogy

A QEMSCAN analysis indicated that pyrite is the dominant sulfide in the Silvertip deposit, representing30-45% of the mineral assemblage. The economic base metal sulfides are sphalerite (12-15%), galena (2-6%), and lead sulfosalts (1-4%). Cerussite, $PbCO_3$, is also present (<1%). Minor pyrrhotite, arsenopyrite, and copper sulfides were also detected in most zones. The pyrhotite content of Discovery North composite was 5.4%, ten times the level in the other zones. The gangue minerals are carbonates and oxides (7-19%) with quartz and silicates (15-44%).

Mineral	Abundance, %			
	Silver Ck	Discovery	Discov. N	65 Zone
pyrite	40.6	31.1	45	41
sphalerite	14.5	13	12.8	15.3
galena	6.1	2	2.3	4.7
Pb sulfosalts	4.1	1.4	1.7	2.3
cerrusite	0.8	0.2	0.3	0.6
pyrrhotite	0.1	0.1	5.4	0.5
arsenopyrite	0.5	0.5	0.5	0.2
Cu sulfides	0	0.1	0.4	0.1
other sulfides	0	0	0.1	0
carbonates, oxides	15.5	6.9	14	19.1
Qtz & silicates	17.4	44.4	17.3	15.8
other	0.3	0.4	0.2	0.4
Total	99.9	100.1	100	100

Lead sulfosalts and the cerussite represent a high proportion of the total Pb content compared to many base metal deposits. Assuming a Pb content of 65% in the sulfosalts (typically 50 – 70% depending on the particular minerals present) the estimated distribution of Pb values between the minerals is:

Pb present as	Silver Ck	Discovery	Discov. N	65 Zone
galena	61.7	61.9	59.8	67.5
sulfosalts	31.1	32.5	33.2	24.8
cerussite	7.2	5.5	7	7.7
Total	100	100	100	100

The results are remarkably consistent across the four ore zones, indicating that, regardless of head grade, a high recovery of sulfosalts and cerussite will be required to achieve high overall lead recoveries. Sulfosalt flotation is not well understood and can be problematic because these minerals lower the concentrate grade and also carry penalty elements including As and Sb. However they may also carry the bulk of the silver, so their recovery may be required to maximize the value of the deposit.

The reported arsenopyrite content of around 0.5% would account for 0.23% As in the feed. In the Silver Ck and 65 Zone material, arsenopyrite accounts for only 24-28% of the total arsenic. In these zones the majority of the arsenic is presumably present in lead sulfosalts so maximizing lead recovery will also increase the arsenic content of the concentrate. In the Discovery and Discovery North zones the reported arsenopyrite content accounts for 66% and 50% of the totals, respectively.

The observed sphalerite content accounts for virtually all of the zinc in the composite samples.

Silver mineralogy was not reported. Even with QEMSCAN this can be a challenging task due to the large number of possible silver minerals.

4 Gravity gold recovery

As can be seen from the head grades, the Silver Creek sample was the only one with significant gold content (1.35 g/t Au).

A gravity concentration test at 135µm grind using a Knelson concentrator and Mozley table yielded 3% Au recovery into a concentrate assaying only 11 g/t grade.

Such a poor result indicates little scope for conventional recovery by a centrifugal concentrator in the grinding circuit. The low concentrate grade and minimal change between the Mozley concentrate and middling grades (11g/t and 7 g/t respectively) suggest that the gold is associated with sulfides, usually pyrite, with virtually no free gold. This is supported by the results of flotation test F5 in the 2013 SGS program where only 2% of the gold reported in the combined final lead and zinc concentrates and 82% reported to the pyrite rougher concentrate. Grade was 2.87 g/t in the pyrite concentrate. No other Silver Creek tests included data for gold.

Given the significant gold head grade, it may be beneficial to examine the geological model to determine if there are areas with more elevated gold grades within the Silver Creek zone. These should be evaluated individually to determine whether higher grade shoots offer more opportunity for liberation and free gold recovery or if these areas yield higher grade pyrite concentrate that may be further treated to realize the contained gold values.

5 Initial flotation results, Tests F1 and F2, 2011

The initial flotation tests were based on a conventional two-stage (differential) Pb-Zn flotation with a $ZnSO_4$-NaCN depressant for Cu and Zn minerals in the Pb flotation stage and $CuSO_4$ activation of sphalerite in the subsequent Zn flotation stage.

The first test proved to be very unselective with over 90% of the Pb and Ag, together with over 60% of the Cu and Zn reporting in the Pb rougher concentrate. Almost 35% of the total sulfur was also recovered in the Pb concentrate, indicating extensive pyrite flotation (Fe analyses not provided). In the second test the $ZnSO_4$-NaCN depressant addition was increased from 150 g/t to 350 g/t, but a similar, unselective result was obtained. In both tests 1100 g/t of lime was added to the grind and lead flotation was conducted at pH 9.9-10.

The poor selectivity in these initial tests was attributed to the presence of organic carbon, with the suggestion that the carbon may be adsorbing flotation reagents. A carbon pre-flotation stage was introduced in the next test and was accompanied by much improved Pb and Zn flotation results.

However, looking more closely at the test procedure and results, it appears that this conclusion was incorrect and that the test conditions, rather than the presence of carbon, were more likely to be the cause of the poor initial flotation performance. This is apparent from the Pb rougher concentrate results as it was collected in three successive stages:

Pb Ro 1 concentrate - 2 minutes flotation after addition of 15g/t of collectors

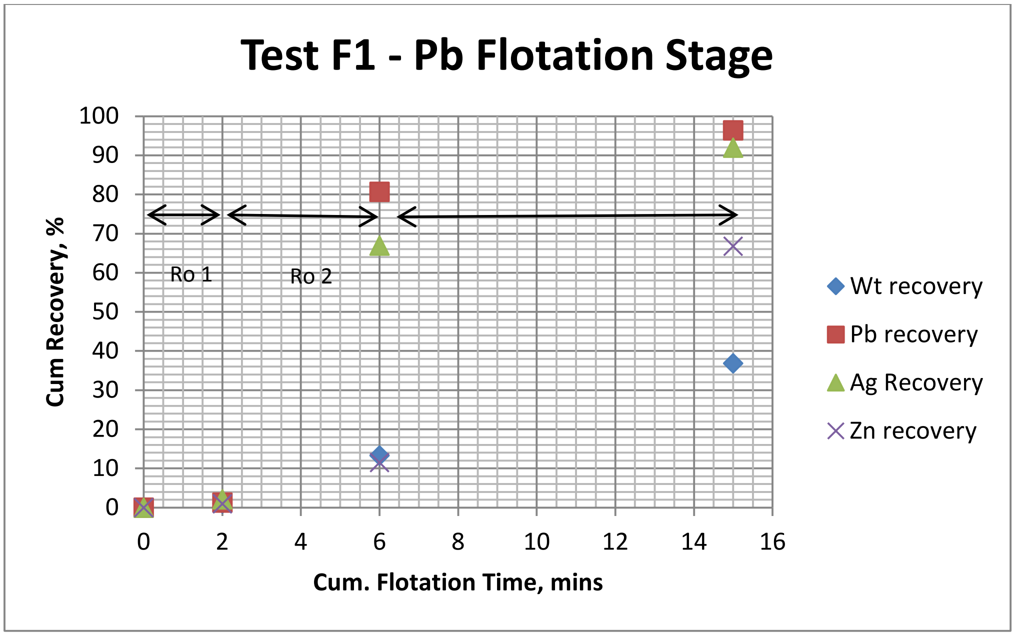
Pb Ro 2 concentrate - 4 minutes flotation after addition of 30 g/t of collectors

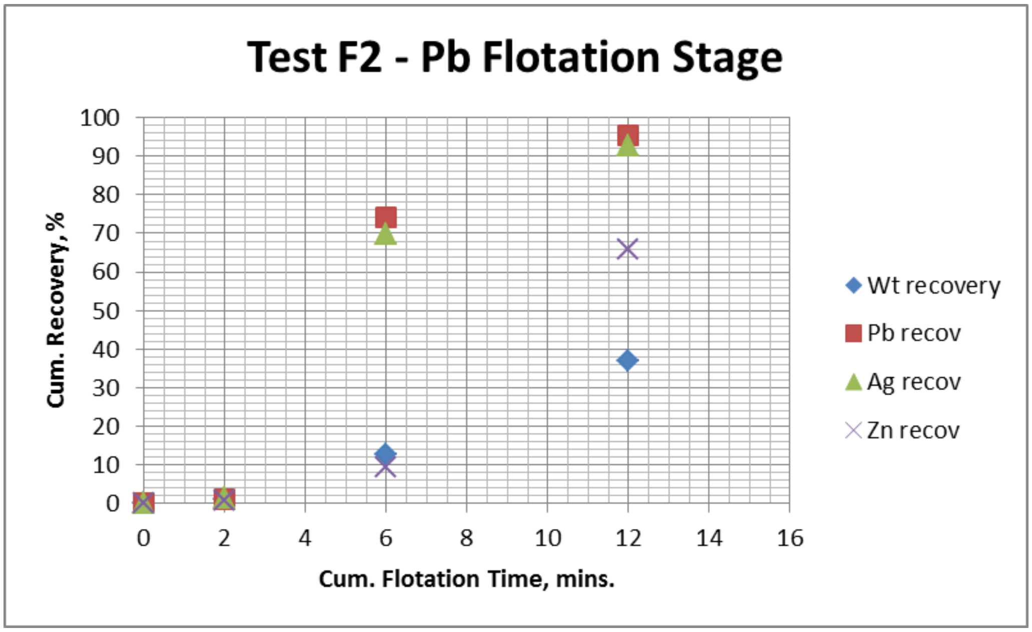
Pb Ro 3 concentrate – F1: 9 minutes, 50 g/t collectors; F2: 6 minutes, 55 g/t collectors.

Figures 1 and 2 are flotation kinetics curves showing the cumulative weight and metal recoveries as a function of time for the first two tests. The separate concentrates are identified on Figure 1; the same divisions apply in Figure 2.

Several features are apparent from the graphs:

1. The initial flotation stage, Ro 1, was strongly inhibited with only around 1% mass and metal recovery – the circuit is strongly over-depressed. This could be due to the high lime addition in grinding (1100 g/t) relative to the low collector addition in this stage. There might also be a contribution from low the redox potential after batch grinding (discussed in the Aeration section, below). In subsequent tests no lime was added to the ball mill and the total addition to the lead circuit was reduced by 30% to an average of 751 g/t. However, it is not possible to make a direct comparison as those later test included the carbon pre-float and the relative effects cannot be disentangled.





2. The second Pb flotation stage, Ro 2, employed higher collector additions and achieved a reasonable mass pull, decent Pb and Ag recoveries and good selectivity against Zn. In both tests this product was an acceptable starting point for optimization:

Test	Pb Ro 2	Assay, %,g/t			Distribution, %		
No	% wt	Pb	Ag	Zn	Pb	Ag	Zn
F1	12.3	49.6	1687	8.6	79.4	64.8	10.5
F2	11.6	47.9	2230	6.9	72.9	68.2	8.6

3. In the third Pb flotation stage, Ro 3, there was an additional heavy collector addition and extended flotation time which pulled so much extra weight into the concentrates that the final product represented 37% of the feed weight. This excessive mass-pull increased recovery, but greatly diluted grades. Zn recovery was elevated above the mass recovery, a sign of unselective collector performance. This third stage overwhelmed the previous acceptable concentrate quality and left so little zinc available for the next stage that it too appeared to be a failure. The collector overdose, not the presence of carbon, is the most likely reason for the unselective overall performance of Tests F1 and F2. In part this was recognized and the collector addition to the third stage was reduced by 85% in subsequent tests.

6 Carbon pre-flotation

After its apparent success in test F3, the majority of tests included the carbon pre-flotation stage. It typically involved conditioning with 20 g/t of fuel oil and 25 g/t of MIBC at natural pH and floating for 2 minutes. The results of 17 tests are shown in Table 1. The mass pull in the pre-float ranged from 1.5% to 4.6% of the feed weight with an average of 2.4%. Metal losses in the pre-float concentrate averaged 3% for Pb, 4% for Ag, and 2% for Zn.

Pre-float concentrate grades show very little upgrading; Pb grades were only 12% higher than head grades, Ag grades were 60% higher and Zn grades were 10% lower. A single test in the 2013 program, F7, included a cleaner stage on the carbon rougher concentrate, but it was essentially a simple mass split with similar grades in cleaner concentrate and tailing. (Similar results were observed for the 65 Zone composite – Tests F1, F2 of the 2013 SGS program).

Despite the references to organic carbon, no analyses were presented to demonstrate the carbon response in the carbon pre-float. The head analysis noted above, indicated 2.27% total carbon and 9.77% carbonate, so the organic carbon content, by difference, is only 0.32%. This is a moderately low value and it would not normally require pre-flotation to manage these levels in the plant. Operations with carbon pre-float stages more commonly see organic carbon levels of 1% or more and the carbon is detrimentally associated with the sulfide minerals (usually pyrite). At Silvertip the carbon comes from the surrounding sediments and is not a component of the sulfide mineralogy.

It is concluded that the carbon pre-flotation stage is an unnecessary complication in this circuit.

7 Aeration effects

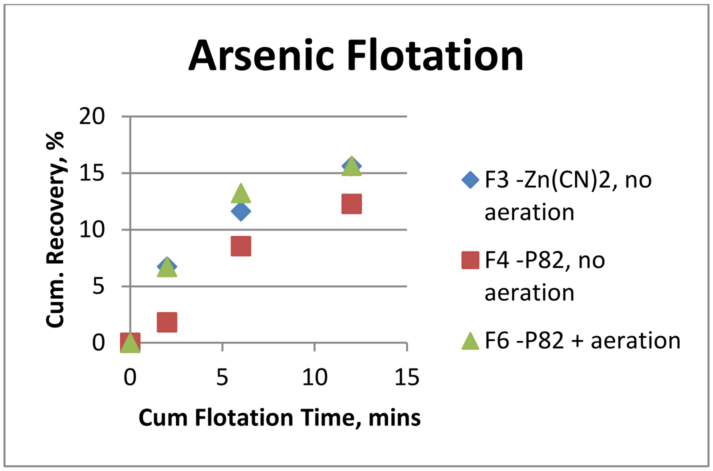
Given the low organic carbon content of the Silver Creek composite, the main effect of the carbon pre-flotation stage may have been to aerate the pulp and raise the redox potential ahead of flotation.

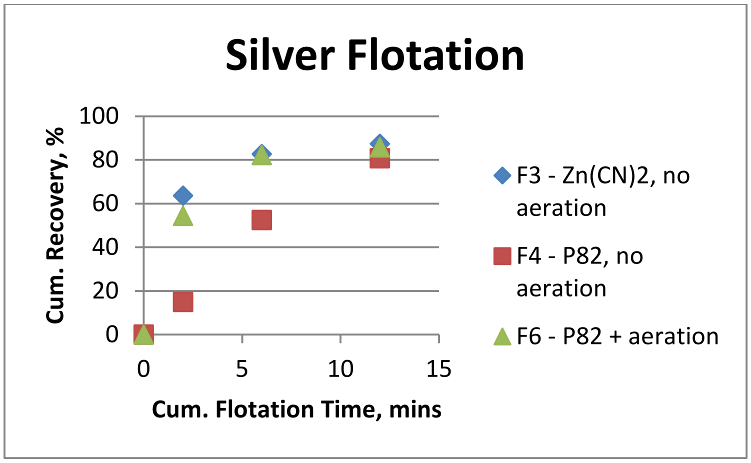
This can be beneficial in some flotation situations where reactive sulfides or abraded steel from grinding media, rapidly deplete oxygen levels and cause low redox potentials in the slurry entering the flotation bank. Sulfide recovery may be depressed until aeration restores redox potentials to levels that support flotation. The effect is more commonly observed in the laboratory where it can be exacerbated by batch grinding in a closed ball mill. In a commercial plant continuous grinding, pumping, and classification all induce aeration of the slurry and additional aeration is usually not required.

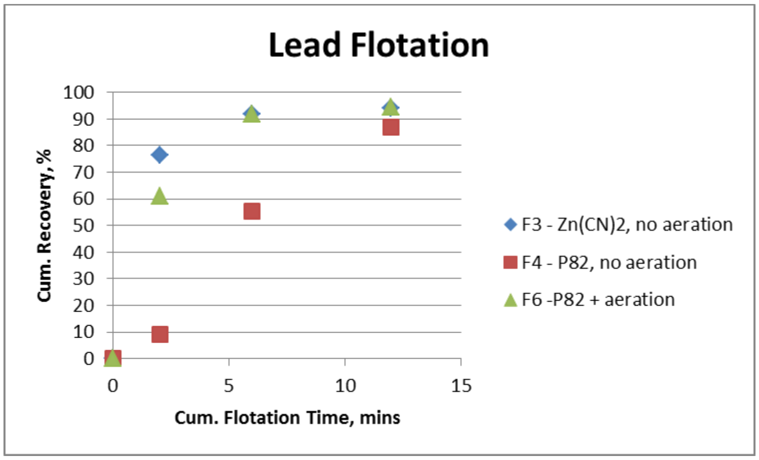
Aeration was introduced in Test F6 of the 2011 program with the specific objective of depressing arsenic. The slurry was aerated for 30 minutes between the carbon pre-float and the Pb flotation stage. The redox potential was increased from -271 mV to -8 mV in the course of aeration, but there was no discernable effect on As recovery.

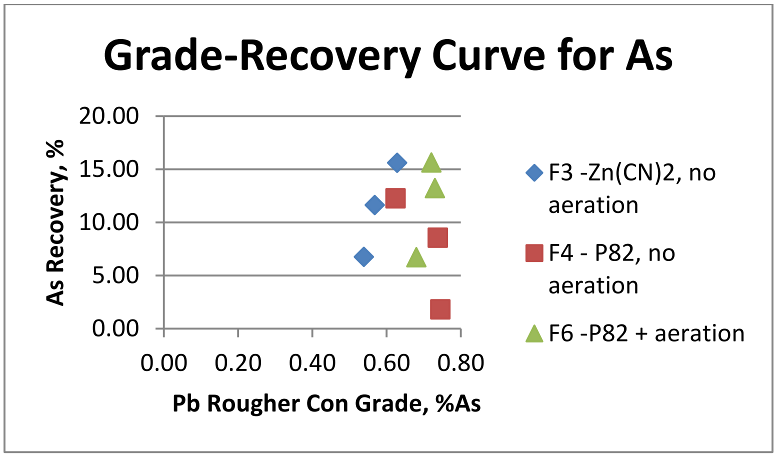
The aeration effect was confused by the simultaneous introduction of a mixed sulfoxy depressant labelled P82, and by the addition of 500g/t of soda ash in the aeration stage. Comparing the flotation kinetic curves and grade recovery curves for tests F3, F4, and F6 reveals that:

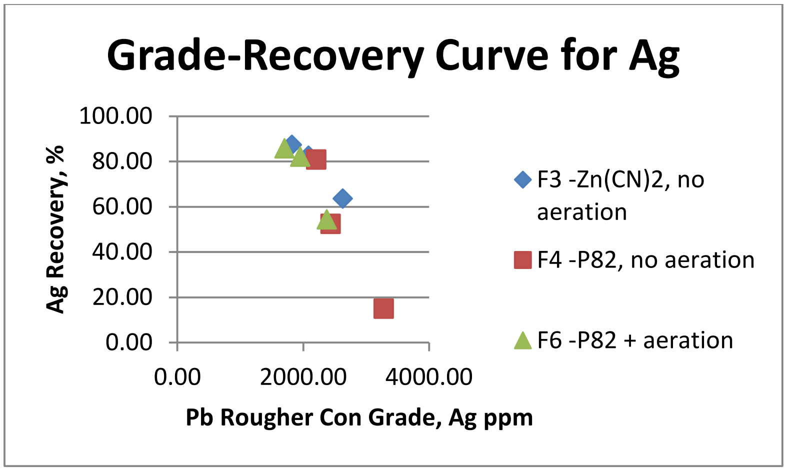
- the depressant P82 detrimentally affected flotation in Test F4 compared to the standard zinc cyanide depressant used in Test F3
- aerating for 30 minutes in Test F6 appears to have restored flotation to levels achieved in the absence of P82
- arsenic flotation kinetics were comparable in both F3 and F6
- comparing grade recovery curves for F6 and F3, for a given recovery the As grade in the Pb concentrate was higher after aeration than without.

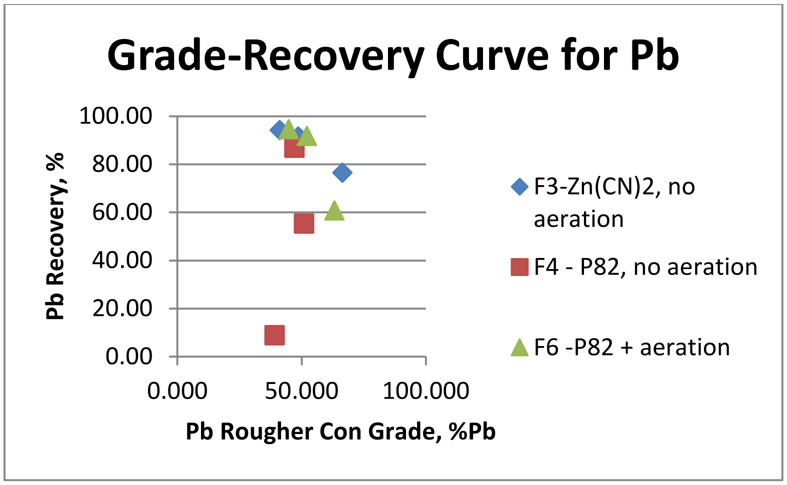



Silver Flotation
Cum. Recovery, %
100
80
60
40
20
0
0
5
10
15
Cum. Flotation Time, mins
◆ F3 - Zn(CN)2, no aeration
■ F4 - P82, no aeration
▲ F6 - P82 + aeration

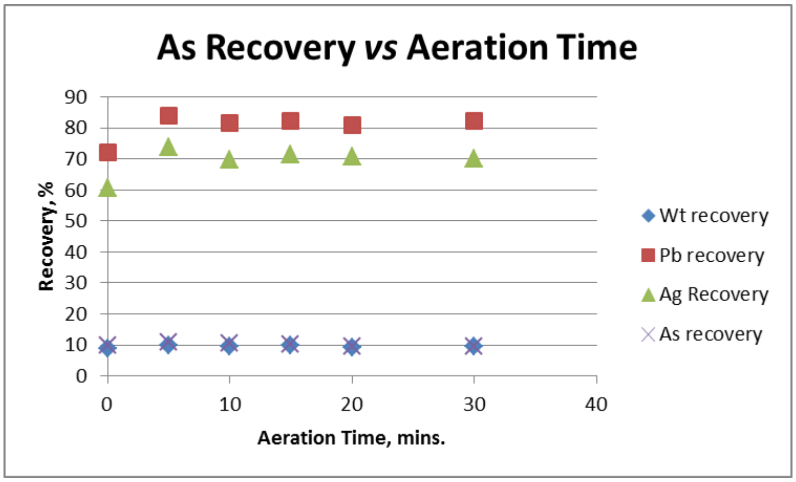

Lead Flotation
Cum. Recovery, %
100
90
80
70
60
50
40
30
20
10
0
0
5
10
15
Cum. Flotation Time, mins
◆ F3 - Zn(CN)2, no aeration
■ F4 - P82, no aeration
▲ F6 -P82 + aeration


Grade-Recovery Curve for As
As Recovery, %
20.00
15.00
10.00
5.00
0.00
0.00
0.20
0.40
0.60
0.80
Pb Rougher Con Grade, %As
◆ F3 -Zn(CN)2, no aeration
■ F4 - P82, no aeration
▲ F6 -P82 + aeration





Aeration was also tested in an unusual manner in locked cycle test LCT2 in the 2013 program. This test followed the standard flotation conditions with zinc cyanide depressant added to the grind and did not include soda ash during aeration. The aeration period was successively increased from zero in the first cycle to 30 minutes by the sixth cycle. Comparing the final Pb concentrate recoveries from the individual cycles with increasing aeration again confirms that aeration did not reduce arsenic recovery, as shown in the following graph



It is concluded that aeration had no significant effect on arsenic flotation. Despite the negative results, aeration was included in the 2013 tests, even extending to 40 minutes in LCT3. There is no justification for such a stage, particularly of such duration, in a commercial operation.

8 Overall Metallurgical Performance

Excluding the initial scoping tests F1 and F2, and those with clearly inappropriate conditions (F4,F5,F6) the combined SGS programs delivered final concentrates in nine batch tests and three locked cycle tests.

Results are summarized in the tables below. The average and 95% confidence limits are shown to indicate the spread of results in the batch tests; results of the three locked cycle tests are presented individually for comparison.

- Representative samples – the narrow range of head grades (seen in the 95% confidence limits for the calculated heads) is indicative of good sample preparation and representative splitting of test samples. There was no detectable bias between the two programs. The locked cycle tests generally fell within the range of the batch tests with the notable exception of the As head in the 2011 LCT 1. This is likely an erroneous assay and is discussed further in Section 9.
- The carbon pre-float results were quite reproducible in the batch tests with an average 2.2% weight recovery and average losses of 1.9% Zn, 2.4% Pb, and 3.6% Ag. Weight and metal losses in the locked cycle tests were somewhat higher and more erratic, the worst being LCT 3 of 2013 with 4.3 % mass pull and 5.4% Pb and 6.9% Ag losses. The recorded pH after grinding was lower in that particular test, but whether that is accurate or significant is not known.
- Lead final concentrate grades averaged 70% Pb and 2613 g/t Ag at recoveries of 77.4% and 66% respectively in the batch tests. The locked cycle tests had around 1% higher mass pull to the final Pb concentrate and grades average 66% Pb and 2566 g/t Ag, but recoveries were higher at 82.7% for Pb and 71.9% for Ag.

- The average arsenic response in the lead batch concentrates was 0.8% As grade at 8.9% recovery. The As content of the 2011 LCT 1 test was very high at 2.2% and 22.9% recovery, whereas the other two locked cycle test results were similar to the batch tests. The high result in 2011 LCT 1 appears to be spurious - see comments in Section 9.

SGS 2011, 2013		Head Grades (Calc)				Carbon Pre-float			
Silver Ck composite						Rougher Recovery			
9 batch flotation tests		Pb	Zn	Ag	As	% wt	Pb	Zn	Ag
average		7.44	9.13	326	0.73	2.2	2.4	1.9	3.6
95% lower limit		7.29	8.90	316	0.70	2.0	2.2	1.8	3.3
95% upper limit		7.58	9.36	336	0.76	2.4	2.6	2.1	4.0
locked	LCT1, 2011	7.27	8.86	331	0.89	2.6	2.1	4.9	3.5
cycle	LCT2, 2013	7.22	9.39	330	0.75	2.3	3.0	2.1	4.3
tests	LCT3, 2013	7.31	9.34	324	0.73	4.3	5.4	3.9	6.9

SGS 2011, 2013		Pb Cleaner 2								
Silver Ck composite		Con	Grades				Recoveries			
9 batch flotation tests		% wt	Pb	Zn	Ag	As	Pb	Zn	Ag	As
average		8.3	70.0	3.4	2613	0.78	77.4	3.4	66.0	8.9
95% lower limit		7.2	66.0	1.6	2505	0.69	71.4	1.3	60.1	7.0
95% upper limit		9.5	74.0	5.2	2722	0.88	83.3	5.5	72.0	10.8
locked	LCT1, 2011	9.4	66.5	3.6	2642	2.18	86.0	3.8	75.0	22.9
cycle	LCT2, 2013	9.7	61.9	4.5	2442	0.76	82.3	4.9	71.5	9.9
tests	LCT3, 2013	8.4	69.9	3.2	2614	0.75	79.8	2.9	69.1	8.6

SGS 2011, 2013		Zn Cleaner 2								
Silver Ck composite		Con	Grades				Recoveries			
9 batch flotation tests		% wt	Pb	Zn	Ag	As	Pb	Zn	Ag	As
average		10.6	0.8	56.6	180.1	0.1	1.1	64.3	5.4	2.7
95% lower limit		8.0	0.6	51.8	134.5	0.0	0.9	52.7	4.6	0.0
95% upper limit		13.2	1.0	61.3	225.7	0.3	1.3	75.9	6.2	6.1
locked	LCT1, 2011	10.1	0.97	59.2	130	0.05	1.36	67.6	3.98	0.58
cycle	LCT2, 2013	8.9	1.79	58	262	0.08	2.18	58.6	7.01	11.2
tests	LCT3, 2013	14.9	2.27	53.7	250	0.19	4.62	86	11.7	3.93

- The final zinc concentrate grades were in broad agreement between the batch and locked cycle tests with values in the range 52-61% Zn. The LCT silver assays were higher; the cause is not known. Zinc recoveries varied greatly in both the batch and locked cycle tests, largely driven by the mass-pull into the various stages of zinc flotation. Mass pull of 12-15% into final concentrate was required for marketable Zn grades at acceptable recoveries.
- Examining the stage performance of the zinc circuit:

- In most tests 85% to 89% of the zinc in the feed carried through to the zinc flotation stage
- Zinc stage recoveries were generally quite high:
  - Zn rougher 94.1– 97.1% stage recovery
  - Zn Cleaner 1 91.8 – 95.5% stage recovery
  - Zn Cleaner 2 81 – 88.6% stage recovery
- In the 2013 program tests F5 and F6 had excessive collector additions in the Zn rougher and, although it appears that flotation was "throttled back" in the rougher stage, the rougher concentrates yielded 52.7% Zn grade at 77.3% recovery and 59.4% Zn grade at 74.2% recovery, respectively. Upgrading these concentrates was unsuccessful, resulting in a recovery loss of 30% for a small improvement in grade. The reported final zinc results for these tests tend to disguise the fact of successful flotation.
- It appears that the zinc grade-recovery curve is very steep for grades above 50% Zn, but it is not clear if this is an artifact of the test procedure or is dictated by mineral liberation effects. (See the Zn Grade- Recovery curves in Section 10.)

9 Response of Penalty Elements, As and Sb

Arsenic and antimony head grades were determined in the 2011 Program by direct analysis of one sub-sample from each of the four test composites. Results were:

Composite	As, %	Sb, ppm
Silver Creek	0.83	2100
65 Zone	0.38	380
Discovery	0.35	560
Discovery North	0.46	690

No other direct head grade analyses were reported.

Arsenic was reasonably well tracked through the test program, enabling a calculated As head grade to be determined for each test. For the Silver Creek composite a total of 15 batch tests gave:

Average calculated head grade	0.73% As
Range	0.63 – 0.83
95% confidence interval	0.70 – 0.76% As

It is noteworthy that the only direct arsenic head assay for Silver Ck is outside the 95% confidence limits developed for the calculated head grades. Normally that would flag poor sampling, but it is noted that the value falls within the range of calculated values (just).

Antimony results were reported only on the final Pb and Zn concentrates from the 2011 locked cycle tests and for 2013 batch tests F1 and F2 (65 Zone Composite).

Results of all tests with Sb data are tabulated below:

			Head Grade		**Pb Cleaner 2 Concentrate**					**Zn Cleaner 2 Concentrate**				
Program & Sample		**Test No**	**As (calc.)**	**Sb (assay)**	**Con**	**Grade**		**Recovery****		**Con**	**Grade**		**Recovery**	
			%	**ppm**	**% wt**	**As, %**	**Sb,ppm**	**As, %**	**Sb, %**	**% wt**	**As, %**	**Sb, ppm**	**As, %**	**Sb, %**
2011	Silver Ck	LCT 1	0.89	2100	9.4	2.18	14500	22.9	64.9	10.1	0.05	552	0.6	2.7
2013	Silver Ck	LCT 2	0.75		9.51	0.79	na	10.1	na	8.72	0.08	na	0.98	na
2013	Silver Ck	LCT 3	0.73		7.9	0.76	na	8	na	14	0.19	na	4	na
2011	65 Zone	LCT 4	0.39	380	7.3	1.15	3840	22	73.8	13.2	0.01	93	0.23	3.2
2013	65 Zone	LCT 1	0.29		7.69	0.1	na	2.5	na	13.4	0.02	na	1	na
2013	65 Zone	F1	0.3		6.2	0.1	4350	2	71.0	11.7	0.02	117	0.7	3.6
2013	65 Zone	F2	0.31		5.1	0.08	4630	1.3	62.1	13.2	0.02	117	1	4.1
2011	Discovery	LCT 2	0.35	560	4.01	1.76	12300	20.3	88.1	9.4	0.004	175	0.12	2.9
2011	Disc. North	LCT 3	0.44	690	4.7	1.79	9840	19.1	67.0	11	0.02	170	0.5	2.7

** Sb recovery calculated from product weights and assays, and direct head assay

Arsenic results were inconsistent between the 2011 and 2013 programs. The earlier program reported As values of 1-2.2% in the final Pb concentrate, in the later program values were substantially reduced. It has been suggested that superficial oxidation during storage may explain the decline in arsenic mineral flotation. In contrast, the 65 Zone Sb data was not affected by the timing of the tests (note that the comparison is inexact between 2011 LCT 4 and 2013 batch tests F1,F2).

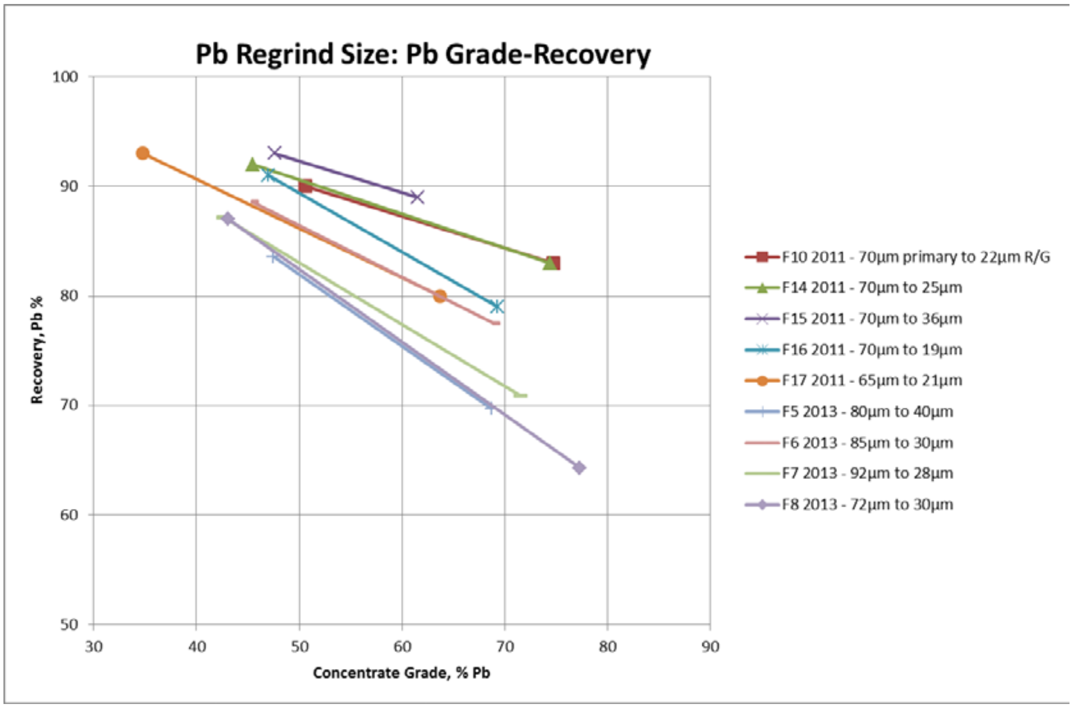
Based on these results, the Silver Creek final Pb concentrates could contain as much as 1.5 – 2% As and 1 – 1.5% Sb. The extreme arsenic values were not observed in the batch tests where, regardless of test date, those concentrates typically assayed around 0.8% As. However, there is insufficient data to determine which is more likely to be correct.

Given that the As and Sb, as well as the silver values, occur primarily in the sulfosalts, it is likely that significant Ag losses would accompany any attempt to depress the penalty elements. On the limited available data it is concluded that it will be more economic to maximize sulfosalt recovery and that the resulting silver values will more than compensate for the penalties incurred for As and Sb.

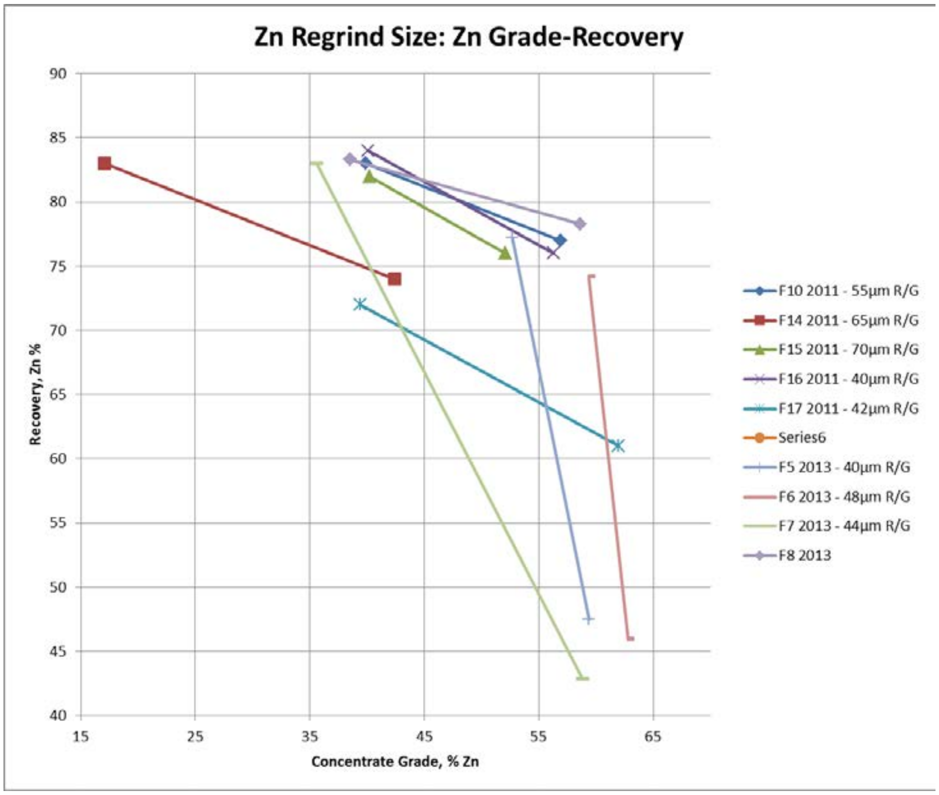
10 Concentrate Regrinding

It is not possible to determine with confidence whether a regrind stage is required in either the Pb or Zn circuits. All tests with cleaning stages also included a regrind step so there was not a zero-regrind point for comparison. In the tests with regrinds the final concentrate grade and recovery results were not systematically related to regrind size:

- The Pb circuit data provide no basis for choosing between 40 micron and 19 micron regrind, other factors appear to control the final cleaner grade and recovery.



- Four zinc circuit results were grouped around the desirable grade recovery position, but their regrind sizes ranged from 44µm to 70 µm. The primary grind for those tests had a P80 of 70µm suggesting that regrinding may not be required in the zinc circuit.



11 Projected Metallurgical Performance

The overall metallurgical performance was projected from locked cycle tests on each composite. These followed standard industry procedures:

Composite/Zone	Pb Concentrate				Zinc Concentrate	
	Pb Grade %	**Pb Recovery %**	**Ag Grade g/t**	**Ag Recovery %**	**Zn grade**	**Zn Recovery**
Silver Ck - 2013 LCT3	69.9	79.8	2614	69.1	53.7	86.0
65 Zone - 2011 LCT4	68.9	95.2	3954	87.6	58.1	93.3
Discovery - 2011 LCT2	57.6	91.4	3066	84.2	63.2	83.2
Discov. N - 2011 LCT3	57.4	91.3	3139	80	57.7	88.4

The projections do not include any of the potential improvements or changes noted previously in this review and there appears to be considerable scope for further optimization. In particular the Silver Creek composite underperforms in comparison with the metallurgical results achieved for the other ore zones and further work is recommended to improve this response.

Table 1- Calculated Head Grades and Carbon Pre-Float Recoveries

Test							Primary	Carbon Pre-float			
Description	Test	Head Grades (Calc)					Grind	Rougher Recovery			
	No	Cu	Pb	Zn	Ag	As	P80	% wt	Pb	Zn	Ag
2011 Program											
C-prefloat (note pH's)	F3	0.14	6.89	8.57	331	0.63	70	1.91	1.93	1.62	2.84
F3 overdep with P82	F4	0.14	7.14	9.05	361	0.67	70	1.49	1.33	1.28	1.94
F3 + Na_2S + calgon	F5	0.15	7.18	9.56	351		70	4.57	9.04	1.52	7.48
Soda ash,P82,30m aera	F6	0.14	7.57	9.47	320	0.73	70	1.7	1.67	1.5	3.19
F3 -finer grind +ZnCN	F7	0.15	7.61	9.67	311	0.74	50	2.44	2.34	1.83	3.69
F3 - coarser grind	F8	0.14	7.66	9.08	347	0.76	90	2.17	2.31	1.85	2.87
F3 - SIPX in Zn circuit	F9	0.14	7.32	9.27	301	0.74	70	2.3	2.55	2.06	3.83
F3 with R/G & cleaning	F10	0.15	7.69	8.9	323	0.83	70	2.2	2.8	1.9	4
F10 - soda ash not lime	F14	0.14	7.62	9.43	332	0.64	70	2.6	2.8	2.2	4.5
F10 - no R/G, hi 3418A	F15	0.13	7.68	8.94	330	0.78	70	2	2	1.8	3.5
F10 -5100 in Pb ro	F16	0.14	7.41	8.67	336	0.75	70	1.8	2	1.7	3.5
F10- lime, NaCN-Pb Clnr	F17	0.13	7.56	9.47	337	0.73	65	2.6	3	2.4	4.5
F10 - locked cycle	LCT1	0.14	7.27	8.86	331	0.89	70	2.62	2.07	4.89	3.48
2013 Program											
2011 F? - C-float with Pb	F5		7.49	9.82	351	0.69	80				
repeat F5 (grinds?)	F6		7.31	8.8	324	0.75	85				
2011 F10 - C-cleaner	F7		7.2	8.92	302	0.73	92	2.25	2.53	2.08	3.88
	F8		7.23	9.13	313	0.75	72	1.92	2.13	1.65	2.97
	LCT2		7.22	9.39	330	0.75	82	2.3	3.01	2.13	4.33
	LCT3		7.31	9.34	324	0.73	72	4.26	5.4	3.91	6.9
	count	0	19	19	19	18		17	17	17	17
	deg freedom	0	18	18	18	17		16	16	16	16
	average		7.39	9.18	329.21	0.74		2.42	2.88	2.14	3.96
	std dev		0.22	0.37	16.55	0.02		0.82	1.82	0.91	1.38
	min		6.89	8.57	301	0.63		1.49	1.33	1.28	1.94
	max		7.69	9.82	361	0.89		4.57	9.04	4.89	7.48
	$t_{(n-1),0.05}$		2.101	2.101	2.101	2.110		2.120	2.120	2.120	2.120
	95% conf int		0.11	0.18	7.98	0.01		0.42	0.93	0.47	0.71
	95% lower limit		7.28	9.00	321.23	0.73		2.00	1.94	1.67	3.25
	95% upper limit		7.50	9.35	337.19	0.75		2.84	3.81	2.61	4.68

Table 2 - Pb Rougher Flotation Results

Test		Primary	Pb Rougher								
Description	Test	Grind	Con	Grades				Recoveries			
	No	P80	% wt	Pb	Zn	Ag	As	Pb	Zn	Ag	As
2011 Program											
C-prefloat (note pH's)	F3	70	**15.4**	**41.2**	7.12	1819	0.63	**92.4**	12.8	84.9	15.4
F3 overdep with P82	F4	70	13	47.1	5.74	2205	0.62	85.7	8.23	79.3	12.2
F3 + Na_2S + calgon	F5	70	18.3	32.8	16	1571		83.7	30.7	81.9	
Soda ash,P82,30m aera	F6	70	15.7	44.8	7.68	1701	0.72	93	12.8	83.5	15.5
F3 -finer grind +ZnCN	F7	50	16.5	42.5	7.68	1598	0.84	92.3	13.1	84.8	18.7
F3 - coarser grind	F8	90	14.9	47	6.73	1915	0.84	91.4	11.1	82.3	16.5
F3 - SIPX in Zn circuit	F9	70	10.1	59.4	4.17	2050	0.6	81.9	4.54	68.7	8.13
F3 with R/G & cleaning	F10	70	13.7	50.6	6.98	1903	0.82	90	11	81	14
F10 - soda ash not lime	F14	70	15.4	45.5	7.29	1792	0.82	92	12	83	20
F10 - no R/G, hi 3418A	F15	70	15	47.6	7.55	1884	0.84	93	13	86	16
F10 -5100 in Pb ro	F16	70	14.4	47	6.53	1940	0.77	91	11	83	15
F10- lime, NaCN-Pb Clnr	F17	65	20.2	34.8	10.6	1457	0.73	93	23	87	20
F10 - locked cycle	LCT1	70									
2013 Program											
2011 F? - C-float with Pb	F5	80	13.2	47.5	5.48	2005	0.56	83.6	7.4	75.2	10.6
repeat F5 (grinds?)	F6	85	14.3	45.4	6.6	1838	0.65	88.6	10.7	81.1	12.4
2011 F10 - C-cleaner	F7	92	14.7	42.6	6.26	1589	0.76	87.2	10.4	77.5	15.5
	F8	72	14.6	43.1	6.39	1691	0.74	87	10.2	78.8	14.4
	LCT2	82									
	LCT3	72									
	count		16	16	16	16	15	16	16	16	15
	deg freedom		15	15	15	15	14	15	15	15	14
	average		14.96	44.93	7.43	1809.88	0.73	89.11	12.62	81.13	14.96
	std dev		2.24	6.04	2.65	199.41	0.10	3.79	6.16	4.56	3.27
	min		10.1	32.8	4.17	1457	0.56	81.9	4.54	68.7	8.13
	max		20.2	59.4	16	2205	0.84	93	30.7	87	20
	$t_{(n-1),0.05}$		2.131	2.131	2.131	2.131	2.145	2.131	2.131	2.131	2.145
	95% conf int		1.19	3.22	1.41	106.26	0.05	2.02	3.28	2.43	1.81
	95% lower limit		13.77	41.71	6.01	1703.62	0.68	87.09	9.34	78.69	13.14
	95% upper limit		16.15	48.15	8.84	1916.13	0.78	91.13	15.91	83.56	16.77

Table 3 – Pb Cleaner 2 Flotation Results

Test		Primary	Pb	Pb Cleaner 2								
Description	Test	Grind	R/G	Con	Grades				Recoveries			
	No	P80	P80	% wt	Pb	Zn	Ag	As	Pb	Zn	Ag	As
2011 Program												
C-prefloat (note pH's)	F3	70										
F3 overdep with P82	F4	70										
F3 + Na_2S + calgon	F5	70										
Soda ash,P82,30m aera	F6	70										
F3 -finer grind +ZnCN	F7	50										
F3 - coarser grind	F8	90										
F3 - SIPX in Zn circuit	F9	70										
F3 with R/G & cleaning	F10	70	22	8.6	74.7	2.76	2630	0.87	83	2.7	70	9
F10 - soda ash not lime	F14	70	25	8.46	74.4	2.44	2630	0.9	83	2.2	67	12
F10 - no R/G, hi 3418A	F15	70	36	11.2	61.5	6.08	2370	0.88	89	7.6	80	12.6
F10 -5100 in Pb ro	F16	70	19	8.42	69.3	2.84	2720	0.86	79	2.8	68	9.7
F10- lime, NaCN-Pb Clnr	F17	65	21	9.5	63.7	8.41	2490	0.79	80	8.5	70	10
F10 - locked cycle	LCT1	70	26	9.4	66.5	3.57	2642	2.18	86	3.78	75	22.9
2013 Program												
2011 F? - C-float with Pb	F5	80	40	7.6	68.7	2.3	2810	0.53	69.7	1.8	60.8	5.8
repeat F5 (grinds?)	F6	85	30	8.2	68.9	2.8	2630	0.64	77.5	2.6	66.8	7
2011 F10 - C-cleaner	F7	92	28	7.14	71.5	1.86	2490	0.83	70.9	1.49	58.9	8.17
	F8	72	30	6.02	77.3	1.26	2750	0.74	64.3	0.83	52.9	5.94
	LCT2	82	35	9.7	61.9	4.45	2442	0.76	82.3	4.94	71.5	9.9
	LCT3	72	23	8.37	69.9	3.23	2614	0.75	79.8	2.91	69.1	8.64
	count			12	12	12	12	12	12	12	12	12
	deg freedom			11	11	11	11	11	11	11	11	11
	average			8.55	69.03	3.50	2601.50	0.89	78.71	3.51	67.50	10.14
	std dev			1.33	5.03	1.99	130.63	0.42	7.15	2.37	7.24	4.54
	min			6.02	61.5	1.26	2370	0.53	64.3	0.83	52.9	5.8
	max			11.2	77.3	8.41	2810	2.18	89	8.5	80	22.9
	$t_{(n-1),0.05}$			2.201	2.201	2.201	2.201	2.201	2.201	2.201	2.201	2.201
	95% conf int			0.84	3.20	1.26	83.00	0.27	4.55	1.51	4.60	2.88
	95% lower limit			7.71	65.83	2.24	2518.50	0.63	74.16	2.00	62.90	7.26
	95% upper limit			9.40	72.22	4.76	2684.50	1.16	83.25	5.02	72.10	13.02

Table 4 – Zn Rougher Flotation Results

Test Description	Test No	Primary Grind P80	Zn Rougher Con % wt	Grades Pb	Zn	Ag	As	Recoveries Pb	Zn	Ag	As
2011 Program											
C-prefloat (note pH's)	F3	70	19.4	0.74	36.2	121.8	0.48	2.09	81.7	7.13	14.7
F3 overdep with P82	F4	70	19.8	1.49	39.4	152.2	0.37	4.11	86.1	8.33	10.8
F3 + Na_2S + calgon	F5	70	19	1.55	32.2	116.3		4.1	64	6.3	
Soda ash,P82,30m aera	F6	70	18.8	0.91	41.4	126	0.43	2.27	82.2	8.89	11.2
F3 -finer grind +ZnCN	F7	50	17.9	0.65	44	114.7	0.48	1.53	81.4	6.59	11.5
F3 - coarser grind	F8	90	19.5	0.83	39	153.8	0.46	2.12	83.9	8.64	11.8
F3 - SIPX in Zn circuit	F9	70	19.5	2.29	42.4	235	0.52	6.11	89.3	15.2	13.5
F3 with R/G & cleaning	F10	70	18.6	0.93	39.9	139.3	0.48	2.2	83	8	11
F10 - soda ash not lime	F14	70	46.1	0.55	17.1	79.1	1	3.4	83	11	72
F10 - no R/G, hi 3418A	F15	70	18.2	0.74	40.2	121.4	0.48	1.8	82	6.7	11
F10 -5100 in Pb ro	F16	70	18.1	0.88	40.1	146	0.43	2.2	84	7.9	11
F10- lime, NaCN-Pb Clnr	F17	65	17.2	0.68	39.4	108.5	0.48	1.6	72	5.5	11
F10 - locked cycle	LCT1	70									
2013 Program											
2011 F? - C-float with Pb	F5	80	14.4	1.93	52.71	283	0.23	3.79	77.8	11.7	4.7
repeat F5 (grinds?)	F6	85									
2011 F10 - C-cleaner	F7	92	20.8	1.32	35.6	158	0.48	3.82	83	10.9	13.6
	F8	72	19.8	1.4	38.5	155	0.44	3.83	83.3	9.8	11.7
	LCT2	82									
	LCT3	72									
	count		15	15	15	15	14	15	15	15	14
	deg freedom		14	14	14	14	13	14	14	14	13
	average		20.47	1.13	38.54	147.34	0.48	3.00	81.11	8.84	15.68
	std dev		7.24	0.52	7.45	50.97	0.16	1.29	6.05	2.54	16.37
	min		14.4	0.55	17.1	79.1	0.23	1.53	64	5.5	4.7
	max		46.1	2.29	52.71	283	1	6.11	89.3	15.2	72
	$t_{(n-1),0.05}$		2.145	2.145	2.145	2.145	2.160	2.145	2.145	2.145	2.160
	95% conf int		4.01	0.29	4.12	28.22	0.10	0.71	3.35	1.41	9.45
	95% lower limit		16.46	0.84	34.42	119.12	0.39	2.28	77.76	7.43	6.23
	95% upper limit		24.48	1.41	42.66	175.56	0.58	3.71	84.46	10.25	25.13

Table 5 – Zn Cleaner 2 Flotation Results

Test		Zn	Zn Cleaner 2								
Description	Test	R/G	Con	Grades				Recoveries			
	No	P_{80}	% wt	Pb	Zn	Ag	As	Pb	Zn	Ag	As
2011 Program											
C-prefloat (note pH's)	F3										
F3 overdep with P82	F4										
F3 + Na_2S + calgon	F5										
Soda ash,P82,30m aera	F6										
F3 -finer grind +ZnCN	F7										
F3 - coarser grind	F8										
F3 - SIPX in Zn circuit	F9										
F3 with R/G & cleaning	F10	55	12.1	0.69	56.9	159	0.09	1.1	77	6	1.3
F10 - soda ash not lime	F14	65	16.5	0.63	42.4	116	0.55	1.4	74	5.8	14
F10 - no R/G, hi 3418A	F15	70	13	0.56	52.1	126	0.25	0.9	76	5	4.2
F10 -5100 in Pb ro	F16	40	11.6	0.73	56.3	166	0.06	1.1	76	5.7	0.9
F10- lime, NaCN-Pb Clnr	F17	42	9.3	0.63	62	140	0.05	0.8	61	3.9	0.6
F10 - locked cycle	LCT1	38	10.1	0.97	59.2	130	0.05	1.36	67.6	3.98	0.58
2013 Program											
2011 F? - C-float with Pb	F5	40	7.8	1.45	59.4	306	0.1	1.5	47.5	6.8	1.1
repeat F5 (grinds?)	F6	48	6.4	0.91	62.8	223	0.05	0.8	46	4.4	0.5
2011 F10 - C-cleaner	F7	44	6.52	0.83	58.8	213	0.05	0.75	42.9	4.59	0.44
	F8	44	12.2	0.94	58.6	172	0.09	1.58	78.3	6.7	1.4
	LCT2	64	8.9	1.79	58	262	0.08	2.18	58.6	7.01	11.2
	LCT3	51	14.9	2.27	53.7	250	0.19	4.62	86	11.7	3.93
	count		12	12	12	12	12	12	12	12	12
	deg freedom		11	11	11	11	11	11	11	11	11
	average		10.78	1.03	56.68	188.58	0.13	1.51	65.91	5.97	3.35
	std dev		3.18	0.53	5.42	61.32	0.15	1.06	14.44	2.11	4.54
	min		6.4	0.56	42.4	116	0.05	0.75	42.9	3.9	0.44
	max		16.5	2.27	62.8	306	0.55	4.62	86	11.7	14
	$t_{(n-1),0.05}$		2.201	2.201	2.201	2.201	2.201	2.201	2.201	2.201	2.201
	95% conf int		2.02	0.34	3.45	38.96	0.09	0.68	9.18	1.34	2.89
	95% lower limit		8.75	0.70	53.24	149.62	0.04	0.83	56.73	4.63	0.46
	95% upper limit		12.80	1.37	60.13	227.55	0.23	2.18	75.08	7.30	6.23

Table 6 – Zinc Rougher Stage Recovery and Tailings Grades

Test Description	Test No	Zn distn in Zn Feed	Zn Ro Stage Recov, %	Final Tail (no pyrite flotation stage)								
					Grades				Distribution, %			
				% wt	Pb	Zn	Ag	As	Pb	Zn	Ag	As
2011 Program												
C-prefloat (note pH's)	F3	85.58	95.5	63.3	0.39	0.52	26.7	0.68	3.58	3.84	5.11	68.7
F3 overdep with P82	F4	90.49	95.1	65.8	0.96	0.6	57.6	0.77	8.84	4.36	10.5	76.1
F3 + Na_2S + calgon	F5	67.78	94.4	58.1	0.39	0.63	25.9		3.16	3.83	4.29	
Soda ash,P82,30m aera	F6	85.7	95.9	63.7	0.36	0.52	32.6	0.83	3.03	3.5	7.82	72.3
F3 -finer grind +ZnCN	F7	85.07	95.7	63.2	0.46	0.56	24.1	0.8	3.82	3.66	4.89	68.2
F3 - coarser grind	F8	87.05	96.4	63.4	0.5	0.46	30	0.84	4.14	3.21	5.5	70.3
F3 - SIPX in Zn circuit	F9	93.4	95.6	68.1	1.01	0.56	54.3	0.84	9.4	4.11	12.3	76.9
F3 with R/G & cleaning	F10	87	95.4	66	0.58	0.55	35.5	0.94	4.9	4.1	7.2	74
F10 - soda ash not lime	F14	86	96.5	35.9	0.34	0.65	10.9	0.1	1.6	2.5	1.2	5.6
F10 - no R/G, hi 3418A	F15	86	95.3	64.8	0.42	0.51	21.7	0.86	3.5	3.7	4.3	71
F10 -5100 in Pb ro	F16	87	96.6	65.7	0.53	0.46	29.6	0.84	4.7	3.5	5.8	74
F10- lime, NaCN-Pb Clnr	F17	75	96.0	60	0.34	0.54	16.3	0.82	2.7	3.4	2.9	67
F10 - locked cycle	LCT1			66.7	0.58	0.62	41	0.86	5.31	4.64	8.34	64
2013 Program												
2011 F? - C-float with Pb	F5	92.6		72.6	1.32	2.05	64	0.81	12.8	15.2	13.3	84.9
repeat F5 (grinds?)	F6	89.3		74.6	0.97	1.71	50	0.86	9.9	14.5	11.5	86.1
2011 F10 - C-cleaner	F7	87.6		68.4	0.81	0.78	39	0.83	7.72	6.06	8.81	78.7
	F8	88.1		68.7	0.86	0.79	45	0.86	8.16	5.94	9.87	79.7
	LCT2			77.8	1.05	3.63	67	0.85	12.2	29.7	16	87.6
	LCT3			72.4	0.99	0.96	62	0.85	10.1	7.1	12	84.3
	count	16	0	19	19	19	19	18	19	19	19	18
	deg freedom	15		18	18	18	18	17	18	18	18	17
	average	85.85		65.22	0.68	0.90	38.59	0.79	6.29	6.68	7.98	71.63
	std dev	6.29		8.62	0.30	0.78	16.75	0.18	3.44	6.59	3.94	17.88
	min	67.78		35.9	0.34	0.46	10.9	0.1	1.6	2.5	1.2	5.6
	max	93.4		77.8	1.32	3.63	67	0.94	12.8	29.7	16	87.6
	$t_{(n-1),0.05}$	2.131		2.101	2.101	2.101	2.101	2.110	2.101	2.101	2.101	2.110
	95% conf int	3.35		4.15	0.15	0.38	8.07	0.09	1.66	3.18	1.90	8.89
	95% lower limit	82.50		61.07	0.53	0.52	30.52	0.70	4.63	3.50	6.08	62.74
	95% upper limit	89.21		69.38	0.82	1.28	46.66	0.88	7.95	9.85	9.88	80.53

Appendix Four

Silvertip Geotechnical Drilling and Report

Telford Geotechnical Ltd.

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

p 250-828-2466
1215 Rockcress Drive, Kamloops, BC V2E 0B4

JDS Energy & Mining Inc.
860-625 Howe Street
Vancouver, BC
V6C 2T6

February 17, 2015
File: 132

Attention: Mr. Maz Mohaseb, P.Eng.

Re: Geotechnical Investigation Report - Proposed Silvertip Mine Plant
Silvertip Mine Site, British Columbia

1.0 INTRODUCTION

As requested, Telford Geotechnical Ltd. has been retained by JDS Energy & Mining Inc. (JDS) to carry out a geotechnical investigation for the proposed plant site buildings. The scope of work was conducted in general accordance with our proposal dated June 2, 2014. We understand that it is proposed to construct a new crusher building, concentrator building, paste plant, truck shop, power house, water treatment plant and a pump house.

This report presents the results of a geotechnical investigation of the soil and groundwater conditions at the proposed development site and makes recommendations for the design and construction of the new process buildings. The report has been prepared exclusively for JDS Energy & Mining Inc., for their use, the use of others on their design team.

2.0 SITE DESCRIPTION

The Silvertip mine is located approximately 85 km southwest of Watson Lake in northern British Columbia, just south of the Yukon border. The mine site is accessed off a gravel road starting from kilometer 1128 of the Alaska Highway, approximately 15 km east of Rancheria, Yukon.

The entrance to the mine is approximately 24 km south of the turn off from the Alaska Highway. The Site is generally flat and is situated at the base of Silvertip Mountain. The Site is flat due to the addition of waste rock and colluvium fill which has been placed on terrain that is assumed to be originally sloping down-gradient towards the north.

The proposed plant site infrastructure is located east of the mine portal on the eastern side of Silvertip Creek. At the time of the investigation, the site was comprised of three water collection ponds (one currently in use), an abandoned one-story lime treatment building adjacent to one of the collection ponds, and a laydown area for drilling supplies. The only visible bedrock outcrops are sedimentary rocks located at the entrance of the mine portal and on the slopes of Silvertip Mountain above the site. The mine camp (currently not in use) is located a few kilometers to the north of the site.

Two additional test holes (TH-13 and TH-14) were advanced on the western side of Silvertip Creek. The terrain on the western side of Silvertip Creek is heavily forested and steep with limited access.

3.0 FIELD INVESTIGATION

The subsurface ground conditions at the site were investigated between June 23 and June 29 using a track mounted Fraste odex/auger drill rig that was supplied by Geotech Drilling of Prince George, BC. 12 test holes were advanced between depths of 9 and 49 feet below the existing site grades. 10 of the test holes were drilled within the plant area and 2 test holes were drilled north of the river as requested by JDS. Disturbed samples were collected from the test holes for future laboratory testing if required. The investigation was supervised by a geologist from Cassiar Geoscience Consulting Ltd.

Standard Penetration Test (SPT) was advanced in some of the test holes to provide an indication of the in-situ density of the strata. The SPT measures the number of blows by a 63.5 kg hammer falling freely through a height of 760 mm that are required to drive a standard split-spoon sampling tube (50 mm OD, 35 mm ID) to a 300 mm depth after an initial seating drive of 150 mm.

The approximate location of the test holes are shown on the attached Silvertip Mine Site Plans following the text of this report.

4.0 SUBSURFACE CONDITIONS

4.1 Test Hole Locations

A summary of the test hole locations is shown in Table 1 below.

Table 1: Test Hole Locations

Area	Test Hole	Easting	Northing
Plant Site	TH-2	425223	6644042
Plant Site	TH-3	425255	6644104
Plant Site	TH-4	425223	6644098
Plant Site	TH-5	425261	6644136
Plant Site	TH-7	425210	6644167
Plant Site	TH-8	425171	6644115
Plant Site	TH-9	425168	6644167
Plant Site	TH-10	425132	6644103
Plant Site	TH-11	425259	6644318
Plant Site	TH-12	425213	6644390
North of River	TH-13	424747	6644355
North of River	TH-14	425031	6644247

4.2 Soil and Rock Conditions

The site is generally underlain by fills and colluvium soils, over dense glacial till, then limestone bedrock.

Within the main plant site area (TH-2 to TH-5 and TH-8 to TH-10), the site is generally underlain by mine waste rock and/or colluvium soils over a layer of clay, then glacial till. At the location of TH-2 along the south edge (crusher location), limestone bedrock was encountered at a depth of 5 feet below the existing site grades. The other test holes generally encountered a loose to dense mine waste rock, and/or loose colluvium soils of varying mixtures of sand and silt with gravel up to depths of 38.5 feet below the existing site grades. A layer of soft to firm glacio-lacustrine clayey silt was encountered between the fills and colluvium and the glacial till. Glacial till was encountered in TH-3, 4, 5, 8 and TH-10. The glacial till generally consists of a compact becoming very dense clayey silt with some sand and gravel.

At the location of the proposed water treatment plant (TH-11 and TH-12) compact to medium dense sandy gravel to gravelly sand was encountered in TH-11 up to a depth of 19', then compact to dense glacial till was encountered to a final depth of 34 feet. In TH-12, loose to firm sandy to silty soils were encountered becoming wet at 15 feet. A very soft to soft clayey silt was noted between depths of 19 to 29 feet. The glacial till was not encountered due to poor and wet drilling conditions.

In TH-13 the glacial till was encountered at the surface and extends to the maximum depth of exploration of 19.5 feet. In TH-14, compact gravelly sand was encountered up to a depth of 29 feet, then compact to dense glacial till was encountered below this depth to 34 feet.

Table 2: Summary of Soil Bearing Layer Depths Below Grade (feet)

Test Hole	Bearing Layer Depth	Test Hole	Bearing Layer Depth
TH-2	5	TH-9	40
TH-3	41	TH-10	22
TH-4	23	TH-11	19
TH-5	22	TH-12	> 39
TH-7	> 10	TH-13	3
TH-8	> 19	TH-14	29

The detailed test hole logs are presented in Appendix A of this report.

4.3 Groundwater Conditions

The groundwater table was encountered generally overlying the soft to firm clayey silt layer within the main plant site area and overlying the limestone bedrock at the location of TH-2. It is expected that groundwater table is perched on the relatively impervious soil and would likely fluctuate during the wetter months of the year or after periods of prolonged precipitation.

5.0 DISCUSSION

As noted, it is proposed to construct a new crusher building, concentrator building, paste plant, truck shop, power house, water treatment plant and a pump house. The site is generally underlain by thick mine waste rock fills and loose colluvium soils over glacial till then limestone bedrock. Recent design meetings have concluded that due to the depth of the fills and loose soils, over-excavation of the materials to a competent bearing soil is not considered economical or practical.

It is our understanding that the service loads from the coarse ore bin, sag mill and ball mill range from 10,100 to 11,700 kips. Column loads for the concentrator building vary from 224 to 650 kips

The lightly loaded pump house, power house and truck shop buildings could be constructed on a raft foundation over the existing fills to reduce differential settlement. The crusher building is expected to be founded on limestone bedrock (TH-2). The mine waste rock fills and loose colluvium soils and soft clays are not considered suitable for the support of the heavy concentrator building, paste plant and water treatment plant using conventional foundations, therefore the structures are recommended to be supported on piles founded in the dense glacial till.

Following our review, we are of the opinion that the proposed development is feasible from a geotechnical standpoint provided that our recommendations are followed.

6.0 DESIGN RECOMMENDATIONS

6.1 Foundation Types

As noted, different foundation types have been recommended for the proposed plant structures at the Silvertip Mine Site. Table 3 below provides a summary of the recommended foundation types.

Table 3: Recommended Foundation Types

Structure	Foundation Type
Crusher Building	Conventional foundation
Concentrator Building	Raft foundation
Paste Plant	Pile foundation
Water Treatment Plant	Pile foundation
Truck Shop, Power House and Pump House	Raft foundation

6.2 Conventional Foundations

Prior to the construction of foundations for the crusher building, all loose soils and weathered bedrock should be removed to expose a subgrade of competent limestone bedrock. Any grade reinstatement beneath the crusher foundations should consist of self-leveling concrete having a compressive strength of at least 15 MPa.

Foundations can be designed for a Service Limit State bearing pressure of 30 ksf and a factored Ultimate Limit State bearing pressure of 60 ksf. Footings should not be less than 24 inches and 36 inches in width for strip footings and pads, respectively. The exterior foundations should be buried at least 3 feet and foundations founded on bedrock are not considered susceptible to frost protection.

The limestone bedrock as defined in Table 4.1.8.4.A. of the 2012 British Columbia Building Code are classified as Site Class B.

6.3 Pile Foundations

As noted, the most practical means of supporting the heavy concentrator building, paste plant and water treatment plant is on piles. The structural engineer has indicated that 12 and 16 inch diameter pipe piles would be suitable for this project based on the design loads. It is recommended that the piles contain a wall thickness of 0.375 inches or greater to reduce damage when driving through the mine waste rock. Based on the test holes advanced at this site, the glacial till would provide a suitable stratum for the piles to be driven to. We would expect that the piles would meet effective refusal within the top 10 feet of the dense to very dense glacial till.

Our preliminary analysis indicates that a 12 and 16 inch pipe pile when driven into the dense glacial till should contain a factored axial geotechnical resistance at Ultimate Limit State (ULS) of 100 and 140 kips respectively. For preliminary design, uplift capacities of 20 and 30 kips could be used for 12 and 16 inch piles. The post-construction settlement of properly installed piles would be less than 1 inch total. The pile caps are recommended to be protected from frost penetration as noted in Section 6.4. The subsurface soils as defined in Table 4.1.8.4.A. of the 2012 British Columbia Building Code are classified as Site Class C.

Due to presence of boulders in the mine waste, it is likely that pile driving shoes will be necessary. The potential contractors should consider the expected soil conditions and judge whether their installation methods will be satisfactory.

It is expected that the piles could be driven open ended and normally the piles are filled with concrete after driving. A dynamic load testing of the piles using a Pile Driving Analyzer (PDA) would need to be done on a select number of test piles to determine the pile capacity, pile driving criteria, driving stresses during driving, hammer efficiency and shaft integrity. If PDA testing is performed a geotechnical resistance factor of 0.5 can be used.

Provided piles are installed at least 3 pile diameters, centre to centre from each other it can be assumed for design purposed that they will act as individual piles with no group effects.

6.4 Raft Foundation

The recommended site preparations for the thickeners truck shop, pump house and power house buildings include the removal of 3 feet of the surficial fills and reinstating grade beneath the raft foundations with engineered fill. We understand that the up to 12 feet of material is to be removed from the thickener area prior to construction, thus unloading the underlying soils.

In the context of this report, engineered fill, is locally borrowed mine waste rock fill that is free from debris and organics. The rock fills would be required to be moisture conditioned to achieve the desired level of compaction. The fills are recommended to be compacted in 300 mm loose lifts with a vibratory drum roller to a minimum of 100% Standard Proctor (ASTM D698) maximum dry density (SPMDD). The fills should also be re-compacted and wetted if allowed to dry out. TGL would be required to monitor the excavation re-compaction of the fills beneath the buildings to ensure that the fills are being compacted properly.

The average raft contact stresses on the compacted engineered fill may be designed for a Service Limit State bearing pressure of 2,000 psf and a factored Ultimate Limit State bearing pressure of 5,000 psf. A raft subgrade modulus of 50 pci can be used for raft design. For raft foundations designed as recommended, we would expect that the post construction would be limited to less than 1 inch at the recommended bearing pressures. The subsurface soils as defined in Table 4.1.8.4.A. of the 2012 British Columbia Building Code are classified as Site Class C.

Due to the deep frost penetration depth at this site (~ 10 to 12 feet), it is recommended that the footings be located a minimum of 3 to 4 feet below final grades and protected with rigid insulation for frost protection. The rigid insulation is recommended to extend horizontally from the base of the footing for a distance of 8 feet outside of the buildings and extend up the edge of the footing wall until it meets grade. The minimum insulation thickness for a heated building is 4 inches.

6.5 Grade Supported Floor Slabs

To provide suitable support for any concrete slabs-on-grades, we recommend that any grading fills placed under the slab should be compacted in 200 mm loose lifts to a minimum of 100% SPMDD. The floor slab should be underlain by 150 mm of compacted 19 mm minus crushed gravel.

6.6 Site and Foundation Drainage Systems

For at-grade supported structures, with no below grade construction, we expect that perimeter drainage would not be required provided the following recommendations are incorporated into the design by the civil and mechanical designers:

1. The top of the finished floor slab is constructed a minimum of 100 mm above the finished outside grades.
2. The site is graded such that surface water drains away from the building.
3. The building floors are underlain by a minimum of 150 mm of free draining granular fill.

If any of the above requirements cannot be met then perimeter drainage should be installed for the building.

6.7 Earth Pressures on Foundation Walls

We recommend that the foundation walls be designed to resist a static triangular soil pressure distribution of 4.7 H (kPa), or 30 H (psf), where H is equal to the total wall height in metres and feet, respectively. The dynamic loading induced by the 2012 British Columbia Building Code design earthquake should be added to the static loads and should be taken as 1.5 H (kPa), or 10 H (psf) inverted triangular. The dynamic earth pressure is based upon unfactored soil parameters and that the wall is backfilled with compacted free draining sand and gravel. Any surcharge loads from equipment and vehicles would have to be included in the wall design.

7.0 FIELD REVIEWS

As required by the 2012 BC Building Code "Letters of Assurance", Telford Geotechnical Ltd. will carry out sufficient field reviews during construction to ensure that the geotechnical design recommendations contained within this report have been adequately communicated to the design team and to the contractors implementing the design. These field reviews are not carried out for the benefit of the contractor's; therefore they do not in any way effect the contractor's obligations to construct the works in accordance with the design.

It is the contractors' responsibility to advise Telford Geotechnical Ltd. (a minimum of 5 days in advance) that a field review is required. Geotechnical field reviews are normally required at the time of these activities:

1.	Excavation	Excavation and site preparations noted
2.	Fill	Review of placement and compaction of engineered fills
3.	Piles	Review of pile installation
4.	Slab-on-Grade	Subgrade and under slab fill

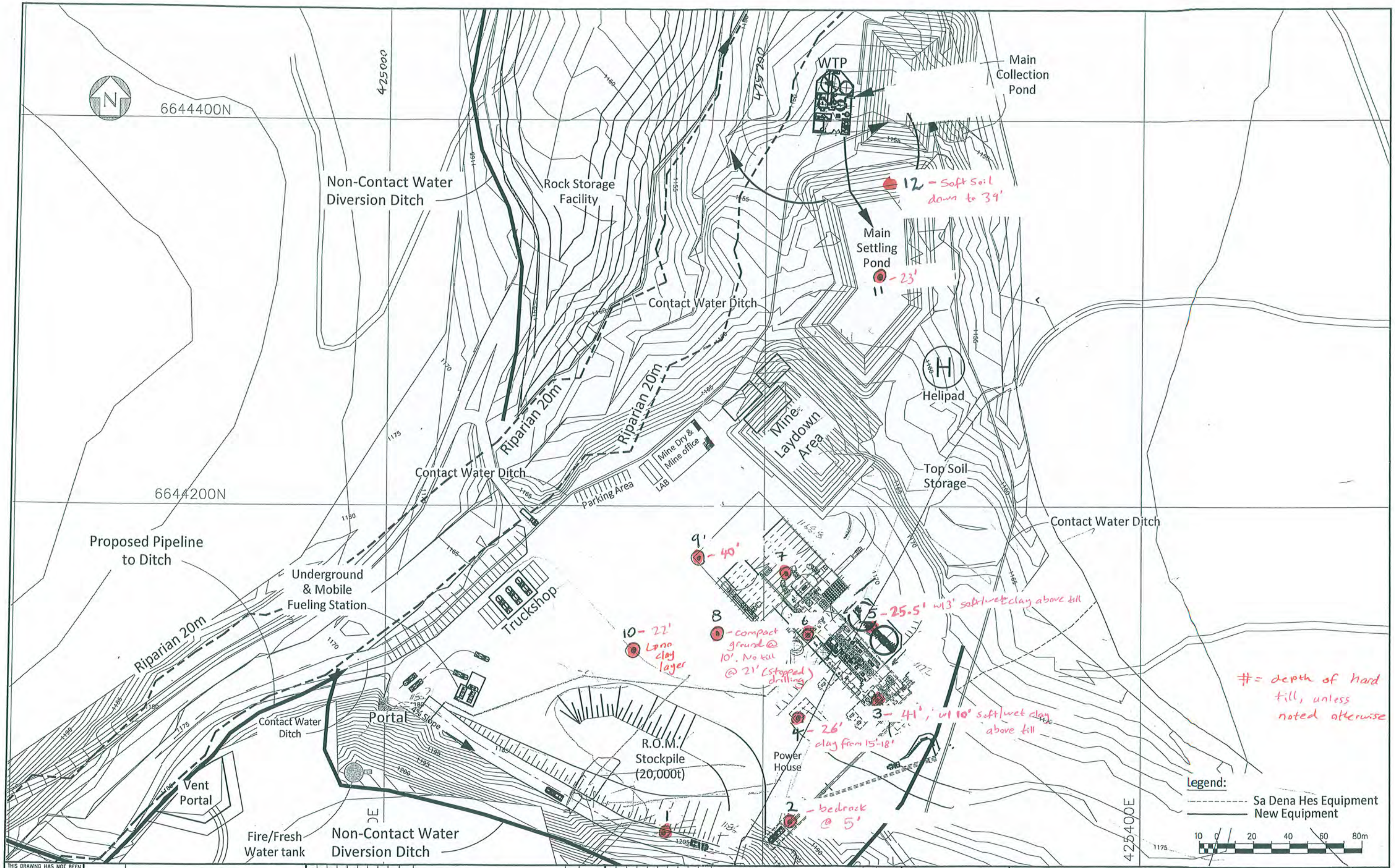
8.0 CLOSURE

This report has been prepared exclusively for JDS Energy and Mining Inc.., for the purpose of providing geotechnical recommendations for the design and construction of the proposed Silvertip Mine Plant Site as described in this report.

We are pleased to be of assistance to you on this project and we trust that our comments and recommendations are both helpful and sufficient for this project. If you would like further details or require clarification, please do not hesitate to contact the undersigned.

For:
Telford Geotechnical Ltd.

Bill Telford, M.Eng., P.Eng.
Geotechnical Engineer



12 - Soft soil down to 39'

11 - 23'

9' - 40'

7

8 - compact ground @ 10'. No till @ 21' (stopped drilling)

10 - 22' Lono clay layer

6

5 - 25.5' w/3' soft/wet clay above till

3 - 41', w/ 10' soft/wet clay above till

4 - 26' clay from 15'-18'

2 - bedrock @ 5'

1

= depth of hard till, unless noted otherwise

THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY JDS FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF JDS

DWG. NO.	REFERENCE DRAWINGS
	6644000

PROJECT | PROCESS | CIVIL | MECH. | STRUCT. | PIPING | SERVICES | ELECT. | INSTR.

NO	DESCRIPTION	BY	DATE
D	ISSUE FOR TENDER	RB	3/26/14
C	ISSUE FOR MAPA	RB	2/14/14
B	ISSUE FOR MAPA	RB	9/1/14
A	ISSUE FOR PROJECT DESCRIPTION	RB	12/9/13

ISSUE

PROJECT | PROCESS | CIVIL | MECH. | STRUCT. | PIPING | SERVICES | ELECT. | INSTR.

NO	DESCRIPTION	BY	DATE

REVISIONS

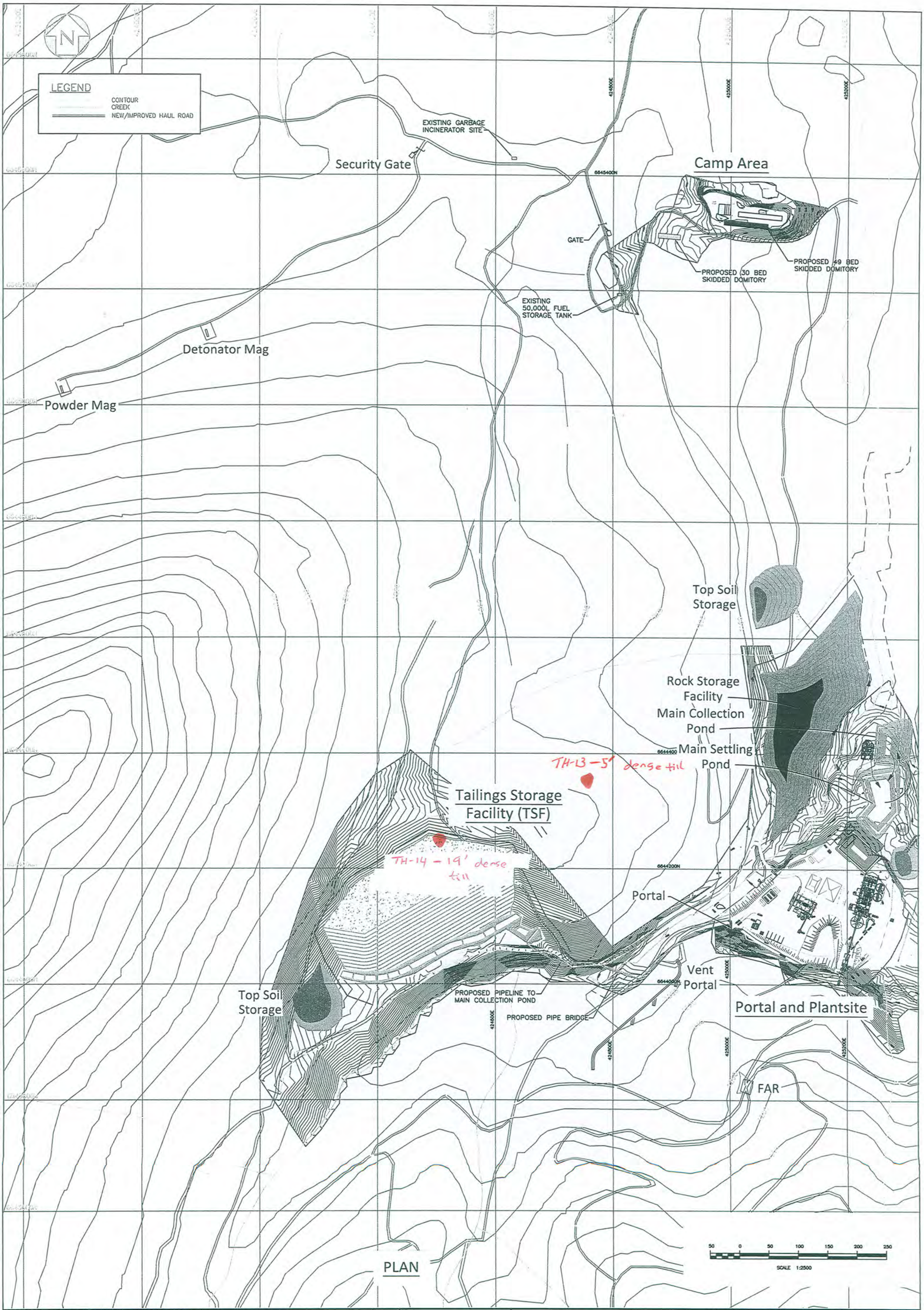
SECTION:
SCALE:
DESIGN. BY J.D.S. Oct 27 13
DRAWN BY: B. Wong Oct 27 13
CHECK. BY
APP. BY:

JDS Energy & Mining Inc.

JDS Silver Inc.

SILVERTIP MINE
PORTAL AREA SITE PLAN

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
		10-10-002	D



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY JDS FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF JDS

PROJECT PROCESS CIVIL MECH. STRUCT. PIPING SERVICES ELECT. INSTR.

NO	DESCRIPTION	BY	DATE
D	ISSUE FOR TENDER	RB	3/26/14
C	ISSUE FOR MAPA	RB	2/14/14
B	ISSUE FOR MAPA	RB	9/1/14
A	ISSUE FOR PROJECT DESCRIPTION	RB	12/9/13

REVISIONS

SECTION:
SCALE: 1:30 000
DESIGN. BY JDS Nov 17 13
DRAWN BY: B.Wong Nov 17 13
CHECK. BY
APP. BY:

JDS Energy & Mining Inc.

JDS Silver Inc.

SILVERTIP MINE
PROJECT OVERALL SITE PLAN

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
		10-10-001	D

APPENDIX A - TEST HOLE LOGS

Boring Location: TH-2 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425223.0 E	INCLIN/H: Vertical	START: 06/27/2014
DRILLER: Caleb	DRILLING METHOD: ODEX	AUGER O.D.: N.A.	6644042.0 N	CASING DPT:	FINISH: 06/27/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.: 4.2 IN	1182.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT): 5.0	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 9.5		

		FIELD SAMPLE RECOVERY DATA							LABORATORY TEST DATA						
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0		☒ 2"				(0 - 5.0') SAND- Silty Fine Sand, some angular gravel, dry to moist, loose, orange/brown	·								
1		☒		2 3 4			·								
2	BR		10				·								
3							·								Soil Sample TH-2 (0-4')
4		☒ 2"		22 50			·								
5						(5.0 - 9.0') BEDROCK- Dark Grey fine grained Limestone. Weathered from 5.0 to 8.0 (orange).	▨								
6							▨								
7							▨								
8							▨								Soil Sample TH-2 (7-9')
9		☒ 2"		50											SPT from 9' - 9.5' was 50 for 1"
10															
11															
12															
13															
14															
15															
16															
17															
18															

Boring Location: TH-3 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425255.0 E	INCLIN/H: Vertical	START: 06/24/2014
DRILLER: Caleb	DRILLING METHOD:Solid Stem Auger	AUGER O.D.: 3.5 IN	6644104.0 N	CASING DPT:	FINISH: 06/25/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1169.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 42.5		

DEPTH (FT)	COLOR	FIELD SAMPLE RECOVERY DATA SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	LABORATORY TEST DATA WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0		☒ 2"	12	6 7 8 7		(0.0' - 15.0') Silt, some sand and angular gravel, dry, firm, grey (Colluvium)	🟩								
1							🟩								
2							🟩								Soil Sample TH-3 (2-3')
3							🟩								
4							🟩								
5							🟩								
6						*from 5'-10' - slight increase in moisture and silt content*	🟩								Soil Sample TH-3 (6-7')
7					ML		🟩								
8							🟩								
9	GY					*at 9' - color changes to brown*	🟩								Soil Sample TH-3 (9')
10		☒ 2"	4	2 3 2 2			🟩								
11							🟩								Soil Sample TH-3 (11')
12							🟩								
13							🟩								
14							🟩								Soil Sample TH-3 (14')
15						(15.0' - 18.0') Sand (coarse-grained), some angular gravel and silt, wet, loose, grey	⁚								
16					SM	*from 15'-17' - sloughing in the hole so we did not retrieve an SPT*	⁚								Soil Sample TH-3 (16')
17							⁚								
18							🟩								

Boring Location: TH-3 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
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		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(18.0' - 20.0') Clay/Silt, trace to some gravel, low plasticity, wet, soft, brown	🟩								
19		2"													Soil Sample TH-3 (19')
20			0	0 2 3		(20.0' - 24.5') Clay, some silt, trace gravel, moderate plasticity, wet, soft, brown	🟩								
21		2"		6											Soil Sample TH-3 (21')
22			2	4 3	CL										
23				5 7											
24															Soil Sample TH-3 (24')
25	BR					(24.5' - 30.0') Sandy Silt (sand is fine-grained), trace gravel, wet, loose, brown	🟩								
26															
27					ML	*at 27' - drilling became denser (large cobble at 29')*									
28															
29						*from 25'-30' - very poor recovery on the auger*									Soil Sample TH-3 (29.5')
30	GY	2"				(30' - 41.0') Clay, some silt, trace gravel, high plasticity, wet, soft to firm, grey	🟩								
31			12	7 7											Soil Sample TH-3 (31-32')
32				15 13											
33															
34						*from 30'-35' - very easy drilling*									Soil Sample TH-3 (34')
35		2"			CH										
36			19	3 5											Soil Sample TH-3 (36')
37				6 7		*from 35'-37' - plenty of sloughing in the hole before the SPT*									
38															
39															
40															
41						(41.0' - 42.5') Silt matrix, trace to some clay, some angular gravel, trace fine sand, low plasticity, dry to	🟩								Soil Sample TH-3 (41')

Boring Location: TH-3 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

DEPTH (FT)	COLOR	FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
		SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
		☒ 2"	0	50	ML	(41.0' - 42.5') Silt matrix, trace to some clay, some angular gravel, trace fine sand, low plasticity, dry to moist, stiff (Glacial Till)	🟩								SPT from 42' to 42.5' was 50 blows for 2"
43															
44															
45															
46															
47															
48															
49															
50															
51															
52															
53															
54															
55															
56															
57															
58															
59															
60															
61															
62															
63															
64															
65															

Boring Location: TH-4 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425223.0 E	INCLIN/H: Vertical	START: 06/24/2014
DRILLER: Caleb	DRILLING METHOD: Solid Stem Auger	AUGER O.D.: 3.5 IN	6644098.0 N	CASING DPT:	FINISH: 06/24/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.: 4.2 IN	1174.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT): N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 35.0		

		FIELD SAMPLE RECOVERY DATA										LABORATORY TEST DATA			
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0						(0.0' - 0.5') Sand (fine grained), trace silt and gravel, dry, loose, brown (Waste Rock)	·								
1						(0.5' - 11.0') Sand/Silt (sand is fine-grained), some angular gravel, dry, soft, brown (Colluvium)	·								
2							·								Soil Sample TH-4 (2-3')
3							·								
4							·								
5		☒ 2"					·								
6			4	2 4 3 3	ML	*at 5.0' - silt content increases, trace to some clay*	·								
7							·								Soil Sample TH-4 (6-8')
8	BR						·								
9							·								Soil Sample TH-4 (9')
10		☒ 2"					·								
11			8	2 6 5 5		(11.0' - 18.0') Clay/Silt, moderate plasticity, wet, soft, brown	🟩								Soil Sample TH-4 (11')
12							🟩								
13							🟩								
14					CL	*at 14.5' - color changes to grey*	🟩								Soil Sample TH-4 (14')
15		☒ 2"					🟩								
16			17	1 3 4 4			🟩								
17	GY						🟩								Soil Sample TH-4 (17-18')
18															

Boring Location: TH-4 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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		FIELD SAMPLE RECOVERY DATA								LABORATORY TEST DATA					
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
19						(18.0' - 23.0') Gravel, some silt and some sand, wet, medium dense, orange/brown	🟩								
20		2"					🟩								Soil Sample TH-4 (19-20')
21	OR		6	4 5	GM		🟩								
22				4 10			🟩								
23						(23.0' - 35.0') Silt/Clay matrix, some fine sand and angular gravel, low plasticity, dry to moist, hard, grey (Glacial Till)	🟩								Soil Sample TH-4 (23')
24							🟩								Soil Sample TH-4 (24-25')
25		2"				*at 23' - hard drilling from 23' to 35'*	🟩								
26			6	1 1 50			🟩								SPT from 26' to 26.5' was 50 blows for 5"
27						*at 25' - matrix of till is sandier with some gravel and trace to some silt, dense*	🟩								Soil Sample TH-4 (25-30')
28							🟩								
29					SM		🟩								
30	BR	2"					🟩								SPT from 30.5' to 31' was 50 blows for 3"
31			9	25 50			🟩								
32							🟩								
33						*at 32.5' - color changes to grey*	🟩								
34							🟩								Soil Sample TH-4 (33-35')
35															
36															
37															
38															
39															
40															
41															

Boring Location: TH-5 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425261.0 E	INCLIN/H: Vertical	START: 06/23/2014
DRILLER: Caleb	DRILLING METHOD: Solid Stem Auger	AUGER O.D.: 3.5 IN	6644136.0 N	CASING DPT:	FINISH: 06/23/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.: 4.2 IN	1167.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT): N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 35.0		

DEPTH (FT)	COLOR	FIELD SAMPLE RECOVERY DATA SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	LABORATORY TEST DATA WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0						(0.0' - 12.0') Sand (fine grained), trace silt and gravel, dry, loose, brown (Waste Rock)									
1															Soil Sample TH-5 (2')
2	BR														
3						*at 3.5' - fine sandy silt matrix, some gravel, dry, soft to stiff, grey*									
4															
5	GY	2"	16	3 5 7 6		*at 5' - silt, trace to some fine sand, trace angular black gravel, moist, light brown*									Soil Sample TH-5 (4-5')
6															Soil Sample TH-5 (6-7')
7						*at 7' - sand (fine-grained), some angular gravel, trace silt, moist, dark brown*									
8															
9															Soil Sample TH-5 (8-10')
10	BR	2"	16	3 2 2 4		*from 10'-11.5' - silt, some fine sand, wet, dark brown*									
11															
12						(12.0' - 18.0') Sand (fine grained), some gravel, trace silt, moist, compact to dense, brown (Colluvium)									
13															Soil Sample TH-5 (12-14')
14						*at 14.5' - hard drilling from 14.5' to 19'*									
15		2"	15	10 28 30 50											
16															
17															
18															

Boring Location: TH-5 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(18.0' - 19.0') Silt, trace gravel, moist, soft to stiff, brown	🟩								Soil Sample TH-5 (18-19')
19						(19.0' - 22.0') Clay, some silt, moderate plasticity, moist, medium stiff to stiff, grey	🟩								Soil Sample TH-5 (19-20')
20		2"		1	CL		🟩								
21			14	3 5			🟩								Soil Sample TH-5 (22-23')
22				9		(22.0' - 35.0') Silt/Clay matrix, some fine sand and angular gravel, low plasticity, dry to moist, hard, grey (Glacial Till)	🟩								
23							🟩								
24							🟩								Soil Sample TH-5 (23-25')
25	GY	2"		13			🟩								
26			12	35 37	ML		🟩								Soil Sample TH-5 (26')
27				38			🟩								Soil Sample TH-5 (26-28')
28							🟩								Soil Sample TH-5 (28')
29							🟩								
30		2"		31		*at 30' - slight increase in the sand content within the till*	🟩								SPT from 30.5.0'-31' was 50 blows for 5"
31			12	50			🟩								
32							🟩								
33							🟩								
34							🟩								Soil Sample TH-5 (33-35')
35															
36															
37															
38															
39															
40															
41															

Boring Location: TH-7 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425210.0 E	INCLIN/H: Vertical	START: 06/24/2014
DRILLER: Caleb	DRILLING METHOD:Solid Stem	AUGER O.D.: 3.55 IN	6644167.0N	CASING DPT:	FINISH: 06/24/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1171 a msl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 10.0		

DEPTH (FT)	COLOR	FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
		SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0						(0 - 10.0') Fine to medium SAND, some angular gravel, dry, loose, grey- (WASTE ROCK)									
1															
2															
3															
4															
5	GY														
6															
7															Hole abandoned due to sloughing
8															
9															
10															
11															
12															
13															
14															
15															
16															
17															
18															

Boring Location: TH-8 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425171.0 E	INCLIN/H: Vertical	START: 06/26/2014
DRILLER: Caleb	DRILLING METHOD:ODEX	AUGER O.D.: N.A.	6644115.0 N	CASING DPT:	FINISH: 06/26/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1169.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 21.0		

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0		2" ☒				(0.0' - 4.0') Sandy Silt, trace to some angular gravel, dry, medium stiff to stiff, brown (Fill/Colluvium)	🟩								
1			16	3 5 7 6			🟩								Test hole drilled 10 meters south of the proposed location
2							🟩								
3							🟩								Soil Sample TH-8 (0-4')
4		2" ☒				(4.0' - 9.0') Sand (fine to medium grained), trace gravel, dry, loose to medium dense, dark brown (Fill/Colluvium)	·								
5	BR		12	2 4 6 7			·								
6							·								
7							·								Soil Sample TH-8 (7-9')
8						*at 8.5' - increase in the gravel content*	·								
9		2" ☒				(9.0' - 19.0') Sandy Gravel (mostly angular), dry, medium dense to dense, brown (Waste Rock)	🟩								Soil Sample TH-8 (9-11')
10			12	9 11 11 24			🟩								
11							🟩								
12							🟩								Soil Sample TH-8 (11-14')
13							🟩								
14		2" ☒				*at 14' - increase in the sand content (well-graded sand) and density*	🟩								
15	BR		16	10 20 18 33			🟩								
16							🟩								Soil Sample TH-8 (14-18')
17						*from 18-19' - hard drilling, dry, dusty, cuttings are angular limestone fragments (cobble or boulder)*	🟩								
18							🟩								

Boring Location: TH-8 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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		FIELD SAMPLE RECOVERY DATA									LABORATORY TEST DATA				
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(9.0' - 19.0') Sandy Gravel (mostly angular), dry, medium dense to dense, brown (Waste Rock)	🟩								Soil Sample TH-8 (18-19')
19		☒ 2"	0	37 20 26 24		*at 19' - drilling abandoned (spoon dropped down the hole)*									Drillers could not retrieve spoon from 19'-21'
20															
21															
22															
23															
24															
25															
26															
27															
28															
29															
30															
31															
32															
33															
34															
35															
36															
37															
38															
39															
40															
41															

Boring Location: TH-9 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425168.0 E	INCLIN/H: Vertical	START: 06/26/2014
DRILLER: Caleb	DRILLING METHOD:ODEX	AUGER O.D.: N.A,	6644167.0 N	CASING DPT:	FINISH: 06/27/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1176.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 49.0		

		FIELD SAMPLE RECOVERY DATA								LABORATORY TEST DATA					
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0		2" ☒	12	13 20 40 33		(0.0' - 38.5') Gravel with some fine to medium grained sand, dry, dense, grey (Waste Rock)	🟩								
1							🟩								
2							🟩								
3							🟩								
4		2" ☒	0	5 3 2 1		*from 4' - 6' - material is loose*	🟩								
5							🟩								
6							🟩								
7							🟩								Soil Sample TH-9 (6-9')
8							🟩								
9		2" ☒	16	2 4 7 16		*from 5'-12' - Sand (fine to medium grained), trace gravel, dry, medium dense, brown*	🟩								
10	BR						🟩								
11							🟩								
12							🟩								
13						*from 12'-14' - Gravel, with some well-graded sand, dry, brown*	🟩								Soil Sample TH-9 (12-14')
14		2" ☒	10	10 30 26 24			🟩								
15						*from 14'-16' - Sand (well-graded), some gravel, dry, dense, grey/brown*	🟩								
16							🟩								Soil Sample TH-9 (16')
17							🟩								
18							🟩								

		FIELD SAMPLE RECOVERY DATA							LABORATORY TEST DATA						
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(0.0' - 38.5') Gravel with some fine to medium grained sand, dry, dense, grey (Waste Rock)	🟩								Soil Sample TH-9 (17-19')
19		2"													
				11											
20	BR		12	29											Soil Sample TH-9 (21-24')
				22											
21				19											
22						*from 16'-29' - Gravel, with some well-graded sand, dry, dense, grey/brown*									
23															
24		2"	10	11											
25				20											SPT from 25'-25.5' was 30 blows for 3" (bouncing on a rock)
				30											
26															
27															
28															
29		2"													
				13											
30	BR		12	18											
				16											
31				28											
32															
33						*from 29'-38' - Sand (well-graded), some gravel, dry, dense, grey/brown*									
34															
35															
36															
37															
38						(38.5' - 40.0') Silt, some clay, trace fine sand and gravel, low plasticity, wet, medium stiff to stiff, grey	🟩								
39		2"			0										from 39'-49' - drilling rate is slow - dense soil
				3											
40			18	8		(40.0' - 49.0') Silty Sand (fine-grained sand), trace to some gravel, dry to moist, dense to very dense, dark grey (Glacial Till)	· · ·								
				17											
41				27											

Boring Location: TH-9 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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		FIELD SAMPLE RECOVERY DATA								LABORATORY TEST DATA					
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(40.0' - 49.0') Silty Sand (fine-grained sand), trace to some gravel, dry to moist, dense to very dense, dark grey (Glacial Till)									Soil Sample TH-9 (42-44')
43	GY														
44		☒ 2"	6	28											
45				50	0										SPT from 44.5'-45' was 50 blows for 4"
46															
47						*at 47' - increase in the silt content within the soil matrix*									Soil Sample TH-9 (47-49')
48															
49															
50															
51															
52															
53															
54															
55															
56															
57															
58															
59															
60															
61															
62															
63															
64															
65															

Boring Location: TH-10 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425132.0 E	INCLIN/H: Vertical	START: 06/25/2014
DRILLER: Caleb	DRILLING METHOD:Solid Stem Auger	AUGER O.D.: 3.5 IN	6644103.0 N	CASING DPT:	FINISH: 06/25/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1170.0 m asl	AZIMUTH:	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 40.0		

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0						(0.0' - 11.0') Silt, some fine sand and angular gravel, dry to moist, soft, dark grey (Fill/Colluvium)	🟩								
1							🟩								Test hole drilled 20 meters south of the proposed location
2							🟩								Soil Sample TH-10 (2')
3	GY					*at 3.5' - there is a very thin layer of black organic silt*	🟩								
4							🟩								Soil Sample TH-10 (4')
5		2" ☒					🟩								
6		☒	16	4 11 6 6			🟩								Soil Sample TH-10 (6')
7						*from 5'-10' - very easy drilling (silt, trace to some fine sand, dry, soft to firm, light brown)*	🟩								
8	BR					*at 8' - silt content increases*	🟩								
9							🟩								Soil Sample TH-10 (9')
10		2" ☒					🟩								
11		☒	12	1 4 6 10		(11.0' - 20.0') Sand (fine to medium grained), some gravel, trace silt, dry, loose to medium dense, grey (Waste Rock)	⋰								
12							⋰								Soil Sample TH-10 (11-15')
13							⋰								
14							⋰								
15	GY						⋰								
16						*from 15'-20' - tough rocky drilling. SPT's not conducted after 12' due to large amount of sloughing*	⋰								
17							⋰								Soil Sample TH-10 (16-19')
18							⋰								

Boring Location: TH-10 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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		FIELD SAMPLE RECOVERY DATA								LABORATORY TEST DATA					
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
19						(11.0' - 20.0') Sand (fine to medium grained), some gravel, trace silt, dry, loose to medium dense, grey (Waste Rock)	· · ·								
20						(20.0' - 22.0') Gravel, some sand, trace silt, wet, loose to medium dense, brown	🟩								
21					GM										Soil Sample TH-10 (21')
22						(22.0' - 40.0') Silt matrix, some fine sand and angular gravel, trace clay, moist, soft to firm, dark grey (Glacial Till)	🟩								
23															
24						*from 20'-30' - drilling is relatively easier than from 15'-20'*									Soil Sample TH-10 (22-24')
25															
26															Soil Sample TH-10 (26')
27															
28															Soil Sample TH-10 (28')
29															
30	GY				ML										
31															
32															
33															
34															
35						*from 30'-40' - drilling was difficult and the rate slowed down. Very poor recovery on the auger.*									
36															
37															Soil Sample TH-10 (35-40')
38															
39															
40															
41															

Boring Location: TH-11 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

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CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425259.0 E	INCLIN/H: Vertical	START: 06/27/2014
DRILLER: Caleb	DRILLING METHOD:ODEX	AUGER O.D.: N.A.	6644318.0 N	CASING DPT:	FINISH: 06/28/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1156.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 34.0		

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0		2"		4 12 12 11		(0.0' - 6.0') Sandy Gravel (well-graded sand), gravel is angular, dry, medium dense, grey/brown	🟩								
1			10				🟩								
2					GW		🟩								
3	BR						🟩								
4		2"		8 8 12 14		*at 4.0' - sand content increases*	🟩								
5			12				🟩								
6						(6.0' - 19.0') Gravelly Sand (well-graded sand), dry, dense, brown	·								
7							·								
8							·								Soil Sample TH-11 (7-9')
9		2"		28 27 19 21		*from 9'-11' -density of the sand increases, dry and dusty drilling (gravel/cobbles)*	·								
10			0				·								
11					SW	*from 11-12' - fine to medium sand, brown*	·								
12						*at 12' - back into gravelly well-graded sand, dry, brown*	·								
13							·								Soil Sample TH-11 (12-14')
14		2"		12 11 18 14		*at 14' - slight increase in moisture content*	·								
15	BR		8				·								
16							·								
17							·								
18							·								

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(6.0' - 19.0') Gravelly Sand (well-graded sand), dry, dense, brown									Soil Sample TH-11 (17-19')
19		2"		4		(19.0' - 34.0') Silty Sand matrix, some gravel, wet, loose to medium dense, brown (Glacial Till)									
20			4	4 8											Soil Sample TH-11 (20-21')
21				7											
22						*at 22' - till is dry to moist, dark brown*									
23															Soil Sample TH-11 (22-24')
24		2"		16											
25			1	21 22		*at 25' - dry and dusty drilling*									Soil Sample TH-11 (24-26')
26				21	SM	*at 26' - increase in the silt content and decrease in the gravel content*									
27	GY														Soil Sample TH-11 (26-29')
28															
29		2"		11		*from 29-34' - very dense drilling*									
30			12	26 27											
31				42											
32															
33															Soil Sample TH-11 (32-34')
34															
35															
36															
37															
38															
39															
40															
41															

Boring Location: TH-12 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425213.0 E	INCLIN/H: Vertical	START: 06/28/2014
DRILLER: Caleb	DRILLING METHOD:ODEX	AUGER O.D.: N.A.	6644390.0 N	CASING DPT:	FINISH: 06/28/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1142.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 39.0		

DEPTH (FT)	COLOR	FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
		SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0		☒ 2"				(0.0' - 5.0') Sand (fine to medium grained), some gravel, dry, medium dense to dense, brown	· · ·								
1		☒	2	4 2			· · ·								Test hole is approximately 70 meters west of the proposed location (dense forest)
2				9 40	SP		· · ·								
3							· · ·								Soil Sample TH-12 (0-2')
4		☒ 2"					· · ·								
5	BR	☒	10	5 5		(5.0' - 9.0') Sandy Silt, wet, medium stiff, light brown	🟩								
6				3 4			🟩								
7					ML		🟩								
8							🟩								
9		☒ 2"				(9.0' - 19.0') Gravelly Sand (well-graded sand), trace silt, wet, medium dense, orange/brown	· · ·								
10	BR	☒	10	11 10			· · ·								
11				7 5			· · ·								
12						*from 11-14' - fine to medium grained sand, trace gravel, dry, loose, orange/brown*	· · ·								Soil Sample TH-12 (11-14')
13					SW		· · ·								
14		☒ 2"					· · ·								
15		☒	0	0 0		*14-16' -spoon sank 2 feet without the use of the hammer. A lot of water in the hole.*	· · ·								
16				0 0		*from 14-19' - fine sand, orange/brown. Barely any cuttings coming out of the hole - just water*	· · ·								
17							· · ·								
18							· · ·								

Boring Location: TH-12 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

		FIELD SAMPLE RECOVERY DATA							LABORATORY TEST DATA						
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
						(9.0' - 19.0') Gravelly Sand (well-graded sand), trace silt, wet, medium dense, orange/brown	.								
19						(19.0' - 29.0') Sandy Silt (fine-grained sand), wet, very soft to soft, grey	🟩								
20							🟩								
21						*No SPT conducted from 19-21' as the ground is too soft*	🟩								Soil Sample TH-12 (19-24')
22	GY				ML		🟩								
23							🟩								
24							🟩								
25						*at 25' - drilling is slightly harder - cuttings consist of wet silt*	🟩								
26							🟩								
27							🟩								
28							🟩								
29		2"				(29.0' - 39.0') Clay, some silt, moderate plasticity, wet, soft, grey	🟩								
30	GY		18	0 0 0 6		*from 29-39' - very poor recovery, wet clay*	🟩								
31							🟩								
32					CH		🟩								
33							🟩								
34		2"				*from 34-39' - clay is highly plastic*	🟩								
35			24	0 0 2 6			🟩								
36							🟩								
37							🟩								
38							🟩								
39															
40															
41															

Boring Location: TH-13 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	424747.0 E	INCLIN/H: Vertical	START: 06/29/2014
DRILLER: Caleb	DRILLING METHOD:ODEX	AUGER O.D.: N.A.	6644355.0 N	CASING DPT:	FINISH: 06/29/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1247.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 19.5		

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0						(0.0' - 4.0') Silty fine sand matrix (sand is predominantly fine grained), some silt, some gravel and cobbles, dry to moist, compact to dense, dark grey (Glacial Till)									
1		2"	4	6 11 50											SPT from 1.0'-1.5' was 50 blows while hitting a cobble/boulder
2															Soil Sample TH-13 (0-4')
3															
4						(4.0' - 19.5') Gravelly sand (sand is well-graded), trace silt, dry, dense to very dense, dark grey (Glacial Till)									
5		2"	6	11 15 17											
6															Soil Sample TH-13 (4-9')
7															Very dense at 7'
8															
9	GY	2"	4	26 50											
10															
11															
12															
13															Soil Sample TH-13 (12-14')
14															
15															
16															Soil Sample TH-13 (14-19')
17															
18															

Boring Location: TH-13 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

		FIELD SAMPLE RECOVERY DATA							LABORATORY TEST DATA						
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
19		☒ 2"	0	50		(4.0' - 19.5') Gravelly sand (sand is well-graded), trace silt, dry, dense to very dense, dark grey (Glacial Till)	. . .								SPT from 19'-19.5' was 50 blows for 3"
20															
21															
22															
23															
24															
25															
26															
27															
28															
29															
30															
31															
32															
33															
34															
35															
36															
37															
38															
39															
40															
41															

Boring Location: TH-14 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

CONTRACTOR INFORMATION	DRILL RIG INFORMATION		BOREHOLE INFORMATION		
DRILLING COMPANY Geotech	DRILLING RIG: Track (Fraste MDML)	BIT TYPE: N.A.	425031.0 E	INCLIN/H: Vertical	START: 06/29/2014
DRILLER: Caleb	DRILLING METHOD:ODEX	AUGER O.D.: N.A.	6644247.0 N	CASING DPT:	FINISH: 06/29/2014
DRILLER'S HELPER: Cody	HAMMER TYPE: Auto.	HOLE DIAM.:4.2 IN	1168.0 m asl	UTM ZONE 9U	LOGGED BY: CAC
DEPTH TO BEDROCK (FT):N.A.	HAMMER WT: 140 LB	CORE DIAM.:	TOTAL DEPTH (FT): 36.0		

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
0						(0.0' - 29.0') Gravelly Sand (well-graded sand), dry, medium dense, brown									
1															Test hole is approximately 140 meters downslope (to the east) of the proposed location
2															
3															Soil Sample TH-14 (0-4')
4		2"													
5			3	7 10 8 9											
6															
7															Soil Sample TH-14 (4-9')
8															
9		2"													
10	BR		4	7 11 19 11	SW										
11															
12															Soil Sample TH-14 (9-14')
13															
14		2"				*from 14-16' - fine to medium grained sand, trace silt and gravel, moist, medium dense, brown*									
15			4	7 8 8 10											
16															
17															Soil Sample TH-14 (14-19')
18															

Boring Location: TH-14 Client - JDS Energy & Mining Inc. Job # 132 - Silvertip Mine

TELFORD GEOTECHNICAL
Geotechnical Engineering | Materials Testing

		FIELD SAMPLE RECOVERY DATA						LABORATORY TEST DATA							
DEPTH (FT)	COLOR	SOIL SAMPLE	RECOVERY (inches)	BLOW COUNT	USCS CLASS	DESCRIPTION	GRAPHIC	WATER CONTENT (%)	DRY DENSITY (PCF)/ SATURATION (%)	SPECIFIC GRAVITY	ATTERBERG LIMITS (LL/PL/PI)	% GRAVEL	% SAND	% FINE	REMARKS
					SW	(0.0' - 29.0') Gravelly Sand (well-graded sand), dry, medium dense, brown	·								
19		2" ☒		41			·								
20	BR		2	50		*from 19-21' -fine to medium grained sand, trace to some silt and gravel, water observed in the spoon*	·								SPT from 19.5-20' was 50 blows for 3" (cobble?)
21							·								
22						*from 21-24' - increase in the moisture content*	·								Soil Sample TH-14 (21-24')
23							·								
24						*from 24-29' - dry and dusty drilling even though there was water in the hole before drilling this run*	·								
25							·								
26						*from 25-29' -sand is fine to medium grained and dry*	·								
27							·								Soil Sample TH-14 (25-29')
28							·								
29		2" ☒		11		(29.0' - 34.0') Sand/Silt (medium to coarse grained sand), trace to some gravel, wet, medium dense to dense, brown **(Glacial Till)**	🟩								
30	BR		12	14 15			🟩								Soil Sample TH-14 (29-31')
31				22	SM/ ML		🟩								
32						*from 31-34' - silt content increases, wet, density increases, brown*	🟩								
33							🟩								Soil Sample TH-14 (31-34')
34		2" ☒		23											SPT from 34.5-35' was 50 blows
35			4	50											
36															
37															
38															
39															
40															
41															

Appendix Five

Environmental Impact Assessments

Lorax Environmental – 1 Volume

Klohn Crippen Berger – 3 Volumes

JDS SILVER

Addendum
Silvertip Mine MAPA
Water Quality Predictions

Project No. A348-1
14 October 2014

LORAX ENVIRONMENTAL

2289 Burrard Street
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Executive Summary

Executive Summary

JDS Silver Inc. (JDSS) is proposing to develop and operate the Silvertip Project (the Project) as a 74,000 tonne per year, seasonal, underground Ag-Pb-Zn mine. The Project is located in northern British Columbia, near the Yukon border. At present, the tunnels are flooded and passively discharging water. Water from the portal discharge infiltrates to ground before reporting to Silvertip Creek as a diffuse source. A Mines Act Permit Application (MAPA) for the Project was prepared in February 2014. A screening review of the MAPA generated a number of comments from the BC Ministry of Environment and BC Ministry of Energy and Mines regarding water quality and the prediction of water quality during operations and closure in the receiving environment. Specifically, comments included requests for clarification on the water quality modeling, including source term inputs to the model, mine water treatment, and the determination of background water quality in the receiving environment.

Lorax Environmental Services Ltd. (Lorax) was retained by JDSS to prepare an addendum to the MAPA related to water quality that addressed the initial screening level comments and concerns raised by MEM and MOE. Lorax developed revised contact water source terms and integrated these predictions with an updated Goldsim® water balance and mass loading model. The updates to these model inputs and the model results presented in this addendum supersede those presented in the MAPA (KCB 2014).

The revised model was used to generate predictions of water quality in Silvertip Creek downstream of the Project. The mass balance model uses inputs of flow and concentration to calculate loadings of parameters of interest. The model combines loadings from mine contact water (source terms) with background loading in the receiving environment to derive water quality predictions in receiving environment.

The receiving environment for treated mine effluent will be Silvertip Creek. Silvertip Creek is a non-fish bearing tributary of the Tootsee River. Background levels of sulphate, cadmium, zinc and other metals are naturally elevated in Silvertip Creek due to natural loadings from surface water (primarily Camp Creek) and groundwater in the vicinity of the Silvertip deposit. Due to these naturally high loadings to Silvertip Creek at the Project location, upstream Silvertip Creek water quality is not representative of background water quality downstream of the Project. Baseline monitoring of Silvertip Creek downstream of the Project was initiated at the same time as excavation and dewatering of exploration tunnels began in 1984. Therefore baseline water quality data from downstream of the Project includes both natural loadings and loadings from the underground exploration. In order to estimate background Silvertip Creek water quality for input to the water quality model, the loading of each parameter of interest in water flowing from the underground exploration workings was subtracted from the total loading measured downstream of the portal discharge. The resultant loading was divided by the total flow of Silvertip Creek to derive estimates of background water quality (Table E-1).

Source terms have been developed to predict drainage chemistry from mine components that will exist during mine operations and closure. Base case and upper case source terms are developed for each mine component to reflect the inherent uncertainty in drainage chemistry predictions. Most source terms are calculated from scaling of laboratory kinetic testwork, with the exception of the mine portal source term which is primarily based on observations from historical dewatering events of the existing underground mine workings. A summary of source term concentrations is provided in Table E-2. Estimates of annual flow produced from mine components during operations and closure scenarios are provided in Figure E-1. The portal is expected to produce considerably higher flows than all the other mine components during all mine phases. As such, the chemistry associated with portal water will largely dictate water treatment requirements and post-closure water quality.

The Goldsim® mass loading model was calibrated through a trial-and-error approach by comparing the model predictions to the collected baseline data (including flows and water quality concentrations) in the receiving environment until a reasonably accurate model performance was achieved; particularly for the water quality predictions.

Water quality was modeled for the following conditions:

- base case and upper estimates of source term chemistry;
- monthly mean flows;
- no treatment of mine water, treatment of mine water to levels equal to effluent quality standards, and treatment of mine water to levels expected for the proposed high density sludge MWTP; and
- operation and closure phases of the mine life.

Table E-1:
Background water quality estimates (mg/L) for Silvertip Creek at site WQ8

Parameter	**Jan**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**
pH	8.1	8.1	7.9	8.1	7.8	8.1	8.2	8.2	8.1	8.2	8.2	8.1
SO_4	68	68	81	82	37	43	49	54	52	55	61	67
As-T	0.0002	0.0001	0.0003	0.0004	0.0037	0.0004	0.0004	0.0003	0.0004	0.0002	0.0003	0.0002
Ca-T	57	62	33	33	43	46	44	46	50	58	57	62
Cd-T	0.00076	0.00071	0.00100	0.00109	0.00100	0.00067	0.00061	0.00072	0.00069	0.00062	0.00065	0.00080
Cd-D	0.00082	0.00074	0.00093	0.00112	0.00060	0.00064	0.00059	0.00070	0.00061	0.00055	0.00061	0.00089
Cr-T	0.0005	0.0005	0.0005	0.0005	0.0009	0.0005	0.0003	0.0004	0.0006	0.0005	0.0005	0.0005
Cu-T	0.0005	0.0005	0.0006	0.0006	0.0072	0.0019	0.0008	0.0006	0.0013	0.0006	0.0005	0.0005
Fe-T	0.032	0.030	0.030	0.033	1.339	0.100	0.055	0.026	0.224	0.030	0.073	0.034
Fe-D	0.030	0.030	0.030	0.030	0.066	0.030	0.020	0.024	0.020	0.030	0.030	0.030
Hg-T	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001
Mg-T	15	15	8	7	10	11	10	11	13	14	15	15
Mn-T	0.005	0.005	0.006	0.006	0.060	0.018	0.006	0.006	0.017	0.004	0.005	0.006
Mo-T	0.00045	0.00045	0.00047	0.00049	0.00039	0.00034	0.00043	0.00043	0.00055	0.00045	0.00046	0.00045
Ni-T	0.0028	0.0027	0.0031	0.0043	0.0059	0.0033	0.0022	0.0023	0.0027	0.0022	0.0024	0.0029
Pb-T	0.0005	0.0003	0.0007	0.0003	0.0201	0.0025	0.0009	0.0004	0.0024	0.0004	0.0006	0.0006
Ag-T	0.00001	0.00001	0.00001	0.00001	0.00013	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Sb-T	0.0003	0.0003	0.0004	0.0004	0.0007	0.0002	0.0003	0.0003	0.0003	0.0003	0.0003	0.0003
Se-T	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
U-T	0.0010	0.0010	0.0012	0.0012	0.0006	0.0006	0.0007	0.0008	0.0008	0.0009	0.0010	0.0010
Zn-T	0.229	0.236	0.256	0.302	0.199	0.128	0.146	0.180	0.160	0.157	0.198	0.223

Table E-1:
Summary of predicted drainage chemistries from mine facilities during operations and closure scenarios

Mine Facility:		Paste Feed Stockpile	Ore Stockpile	PAG Stockpile	Mine Facilities Area	NPAG-RSF		TMF		Mine Portal	
Scenario:		Operations	Operations	Operations	Operations	Operations	Closure	Operations	Closure	Operations	Closure
Estimate:		Base Case									
pH	s.u.	7.6	7.1	7.4	7.5	7.7	7.7	7.7	7.7	7.7	7.1
SO_4	mg/L	1610	403	489	35	438	218	764	1720	1920	902
As	mg/L	0.155	0.00453	0.00219	0.000157	0.00158	0.000397	0.00257	0.0154	0.0214	0.00953
Ca	mg/L	594	192	158	11.3	153	65	311	565	589	343
Cd	mg/L	0.154	0.0227	0.0511	0.00366	0.00898	0.000641	0.0264	0.0751	0.0466	0.0199
Cr	mg/L	0.0189	0.00172	0.001	7.17E-05	0.00209	0.00131	0.000567	0.00856	0.00964	0.0029
Cu	mg/L	0.149	0.041	0.00694	0.000497	0.011	0.0124	0.00882	0.0488	0.0328	0.00704
Fe	mg/L	0.0973	0.0211	0.0103	0.000739	0.0124	0.00794	0.0227	0.0685	1.07	0.955
Hg	mg/L	0.00964	4.77E-05	1.31E-05	9.34E-07	2.72E-05	1.74E-05	1.88E-05	6.65E-05	0.00243	0.000144
Mg	mg/L	40.8	1.2	37	2.65	33.8	25.1	4.74	33.4	170	57.1
Mn	mg/L	11.7	1.39	1.2	0.0859	0.598	0.00816	1.37	16.8	3.77	1.06
Mo	mg/L	0.0344	0.0125	0.0472	0.00337	0.243	0.112	0.0148	0.0626	0.0133	0.00186
Ni	mg/L	0.343	0.0103	0.0274	0.00196	0.00433	0.00137	0.0325	0.21	0.259	0.0792
Pb	mg/L	0.0225	0.0449	0.000665	4.76E-05	0.00388	0.00115	0.0176	0.0182	0.0187	0.0198
Sb	mg/L	0.13	0.0142	0.0104	0.000748	0.0212	0.00922	0.0263	0.112	0.0904	0.0268
Se	mg/L	0.0298	0.0034	0.00371	0.000265	0.0026	0.000817	0.00192	0.0158	0.0104	0.00294
U	mg/L	0.0117	0.00017	0.0027	0.000193	0.0129	0.00557	0.000549	0.00252	0.035	0.0108
Zn	mg/L	9.48	3.14	1.18	0.0847	2.5	0.404	2.65	4.63	5.15	4.86
Estimate:		Upper Case									
pH	s.u.	7.5	6.9	7.3	7.3	7.4	7.4	7.6	7.6	7.5	6.5
SO_4	mg/L	1780	1620	1850	140	1750	871	1530	1960	2060	1470
As	mg/L	0.31	0.00905	0.00875	0.000626	0.00631	0.00159	0.0103	0.0465	0.0355	0.0111
Ca	mg/L	580	384	587	45.3	614	260	608	577	576	612
Cd	mg/L	0.273	0.0453	0.204	0.0146	0.0359	0.00257	0.106	0.266	0.151	0.167
Cr	mg/L	0.0378	0.00345	0.00401	0.000287	0.00836	0.00523	0.00227	0.026	0.013	0.00323
Cu	mg/L	0.298	0.082	0.0278	0.00199	0.0442	0.0495	0.0353	0.148	0.158	0.0832
Fe	mg/L	0.195	0.0422	0.0413	0.00296	0.0498	0.0318	0.0907	0.208	7.79	42.1
Hg	mg/L	0.0193	9.53E-05	5.22E-05	3.74E-06	0.000109	6.94E-05	7.52E-05	0.0002	0.00864	0.000163
Mg	mg/L	81.6	2.39	148	10.6	135	101	19	111	204	121
Mn	mg/L	23.4	2.77	4.8	0.344	2.39	0.0326	5.48	50.6	5.97	2.72
Mo	mg/L	0.0688	0.0251	0.189	0.0135	0.974	0.446	0.0592	0.228	0.0418	0.00203
Ni	mg/L	0.686	0.0206	0.11	0.00784	0.0173	0.00548	0.13	0.632	0.323	0.118
Pb	mg/L	0.028	0.104	0.00266	0.00019	0.0155	0.00459	0.0219	0.0227	0.0249	0.0389
Sb	mg/L	0.26	0.0284	0.0418	0.00299	0.0849	0.0369	0.105	0.34	0.162	0.0299
Se	mg/L	0.0596	0.00679	0.0148	0.00106	0.0104	0.00327	0.00769	0.0477	0.0177	0.0042
U	mg/L	0.0234	0.00034	0.0108	0.000772	0.0518	0.0223	0.00219	0.00968	0.0432	0.0174
Zn	mg/L	16.7	6.28	4.74	0.339	9.99	1.61	9.12	9.92	13.1	13.1

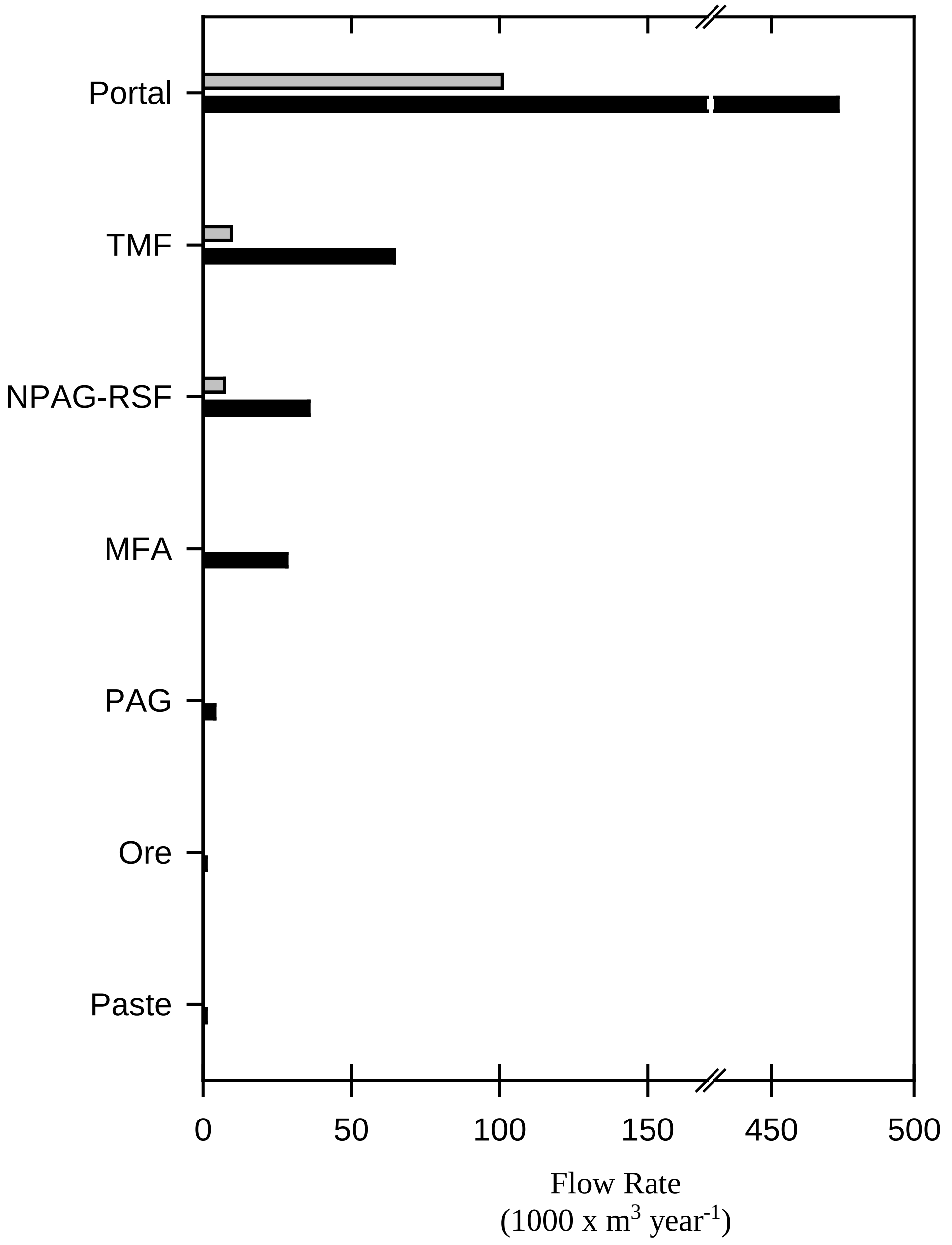

Figure E-1: **Maximum annual flow rates that will be produced from mine facilities during operations (black) and closure (grey).**

Modeled water quality for the no-treatment cases indicated that treatment of mine contact water will be required in order to meet BC water quality guidelines for protection of freshwater aquatic life (BCWQG) in the receiving environment at monitoring site WQ8.

The Goldsim water quality model was also run in a "hindcast" mode to derive effluent quality standards (EQS) that, if met in the discharge from the mine waste treatment plant (MWTP), would result in water quality in the receiving environment that meets BCWQG (Table E-3). Cadmium and zinc are naturally present in Silvertip Creek at levels that exceed BCWQG, so even if these metals were completely removed by the MWTP the concentration at WQ8 would be above BCWQG. Therefore effluent quality standards for Cd and Zn were determined as levels that would not degrade water quality from background levels. Non-degradation benchmarks were estimated as 10% above the maximum mean monthly background concentration.

Table E-2:
MWTP EQS Benchmarks

Parameter	***Proposed Effluent Quality Standards (mg/L)***
pH	**6.5 to 8**
TSS	**15**
SO_4	**1800**
Nitrate-N	**15**
Nitrite-N	**0.09**
NH_3-N	**4.3**
CN_{WAD}	**0.024**
Al (diss)	**0.23**
Sb	**0.1**
As	**0.024**
Cd	**0.003**
Co	**0.02**
Cu	**0.03**
Fe	**4**
Pb	**0.045**
Hg	**0.00006**
Mn	**6**
Mo	**5**
Ni	**0.5**
Se	**0.006**
Ag	**0.007**
Zn	**0.5**

The MWTP is expected to treat most parameters to levels lower than EQS (see Table 3-56). Water quality for operation phase of the Project, using the expected contaminant levels in effluent from the MWTP, was below BCWQG for all parameters except nitrite, Cd, and Zn. In BC, water quality guidelines are developed to be protective of all species of aquatic life, and all aquatic stages of their life cycle, during indefinite exposure. Therefore, water quality predictions that are below BCWQG indicate that the predicted levels of contaminants will not harm aquatic life in the receiving environment. Predicted cadmium and zinc levels were approximately equal to background concentrations of these parameters at WQ8. Background concentrations were determined as the concentration of contaminants in Silvertip Creek at station WQ8 without any loading from the existing portal discharge. Therefore the results of modeled cases using expected levels of Cd and Zn in treated effluent indicate that concentrations of these parameters will be approximately equal to their concentration prior to mine development.

The maximum modeled nitrite level during operation phase is 0.08 mg/L, which is 4 times higher than the BCWQG for nitrite. The model assumed that the MWTP has no effect on nitrite. Nitrite is a parameter that is derived primarily from estimating the residue from explosives. The model results indicate that mitigation of nitrogen species in mine contact water should be addressed through best management practices of explosives in the underground during operations.

The MWTP will be shut down at the closure phase of the Project. Mine contact water will be directed to a constructed wetland, which will be designed to produce a discharge that meets the EQS derived for the MWTP. The Goldsim model was run to determine the removal efficiency necessary meet these EQS in the water discharged from the wetland. Removal of 61 – 74% of the estimated Cd levels in water flowing into the wetland, and 65 – 79% removal of Zn, would produce levels in water exiting the wetland that meet proposed EQS and therefore would not degrade Silvertip water quality compared to background levels.

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JDS SILVER

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1. Introduction

1. Introduction

JDS Silver Inc. (JDSS) is proposing to develop and operate the Silvertip Project (the Project) as a 74,000 tonne per year, seasonal, underground Ag-Pb-Zn mine. The Project is located in northern British Columbia, near the Yukon border. Exploration and development of the Silvertip deposit has occurred periodically since 1957. Tunnel excavation occurred in 1984, 1985, and 1990. At present, the tunnels are flooded and passively discharging water. Water from the portal discharge infiltrates to ground before reporting to Silvertip Creek as a diffuse source. Silvertip Creek is a non-fish bearing tributary of the Tootsee River. In addition to the contribution of mine contact water to water quality in Silvertip Creek, surface water and groundwater in the vicinity of the Silvertip deposit contribute naturally elevated loadings of sulphate and some metals to Silvertip Creek. Natural loadings of cadmium and zinc to Silvertip Creek produce background water quality that exceeds current BC water quality guidelines for the protection of freshwater aquatic life.

A Mines Act Permit Application (MAPA) for the Project was prepared in February 2014. The MAPA included a description of proposed Project facilities, which include additional tunneling, a process plant, mine camp, dry stack tailings facility, mine rock facility, and wastewater treatment plant. The MAPA presented results of a Goldsim® model that predicted water quality in Silvertip Creek downstream of the Project. A screening review of the MAPA generated a number of comments from the BC Ministry of Environment and BC Ministry of Energy and Mines regarding water quality and the prediction of water quality in the receiving environment. Specifically, comments included requests for clarification on the water quality modeling, including source term inputs to the model, mine water treatment, and the determination of background water quality in the receiving environment.

Lorax Environmental Services Ltd. (Lorax) was retained by JDSS to prepare an addendum to the MAPA related to water quality that addressed the initial screening level comments and concerns raised by MEM and MOE. Lorax developed revised contact water source terms and integrated these predictions with an updated Goldsim® water balance and mass loading model. The model was used to generate predictions of water quality in Silvertip Creek downstream of the Project.

An overview of the water balance and mass loading water quality model is described in Chapter 2 of this report. Inputs of flows and water quality to the Goldsim® mass balance model are described in Chapter 3. New estimates of background Silvertip Creek water quality are described which include natural sources of contaminants and excludes contributions from water discharged from the existing main portal. New estimates of contact water chemistry from mine facilities are also described in Chapter 3, including details of their derivation. Finally, Chapter 4 presents the revised water quality model results for operations and closure.

2. Model Description

2. Model Description

The site-wide water balance and water quality model was developed to simulate the conveyance of water and mass communications between mine site components and discharges to the receiving environment downstream of the mine site.

The project specific mine production schedule (Appendix A) and mill operation plan as provided by JDS forms the basis for the model approach. To accommodate the complexity of mine production and operation schedule and project objectives, the model was constructed to generate monthly flow and water quality predictions with the use of the Goldsim® platform. The Goldsim® software has been widely-used as a tool for developing water balance and water quality models. It provides a flexible programming environment for exploring inter-related time-dependent processes (flow, storage, and load) and their effect on the receiving environment, with the ability to export results to spreadsheets for further analysis and reporting.

The water input into the model system is comprised of the mine portal discharge (associated with the underground workings) and the meteoric precipitation. The portal discharge has been estimated externally and provided by JDS (MAPA Appendix 9-IV-A). Annual precipitation values as reported in the February 2014 Mines Act Permit Application (MAPA) will be directly used in this work. Monthly flows and discharges in response to the precipitation input were estimated as the product of the precipitation and the runoff coefficient. To properly approximate the flow hydrograph from the snowmelt-dominated catchment, the annual precipitation was presumably redistributed with the use of the regional monthly stream distribution percentages of the Rancheria Watershed.

A schematic of the Goldsim® model structure for the mine stockpile is shown in Figure 2-1, illustrating the communications amongst three balance model blocks of an individual stockpile. The material balance model forms the base for the water balance model and mass loading model. During the operation phase, all three balance model blocks are characterized by dynamic variables that vary on both temporal and spatial scales. At closure, the pile material balance model variables including the pile volume and pile tonnage will become either constants (for TMF and NAG-RSF) or zeros (for PAG rock pile, Paste-Feed pile, and Ore pile).

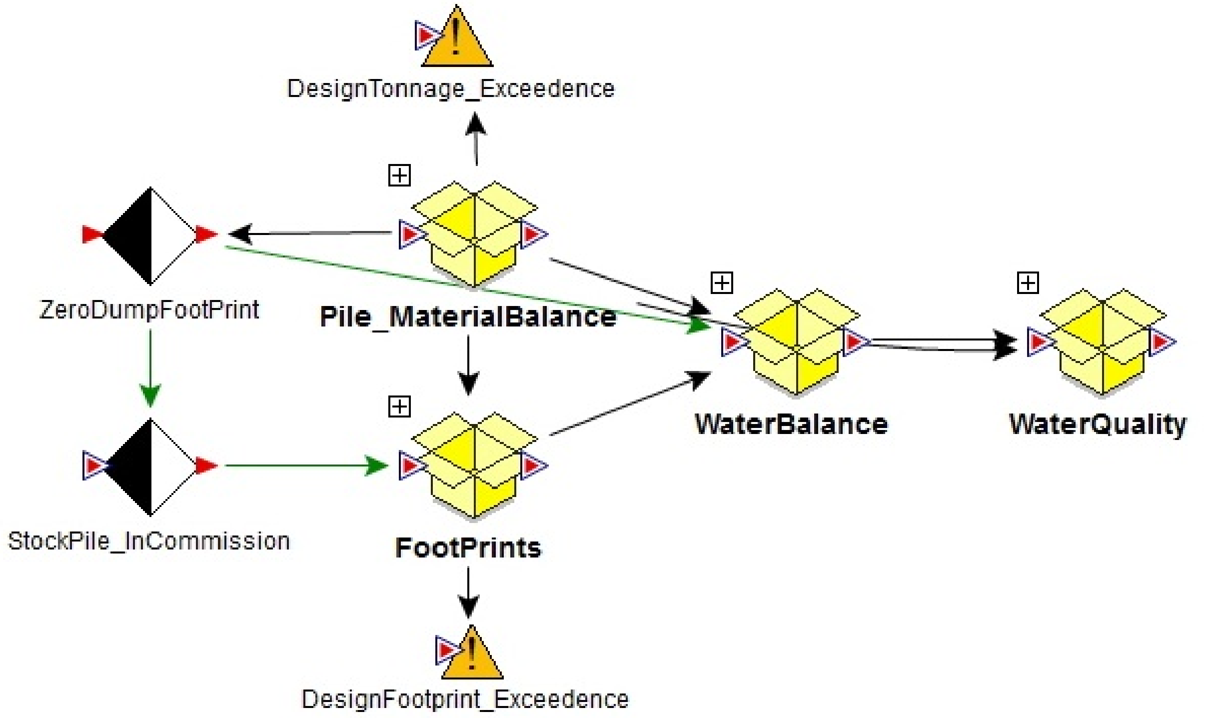


Figure 2-1: Goldsim® Model Schematic Showing the Implementation of the pile water balance associated with the pile material balance and the loading balance

Construction of the pile material balance model requires the incorporation of the mine production schedule as provided by JDS (Appendix A), which interprets the pile material generation and removal rates throughout the operation phase. In Goldsim®, this accumulative balance process for an individual stockpile is represented or simulated by a Reservoir element. Determination of the pile footprints was based on the pile volume and will be described in their respective subsections.

In the model, the total load from all mine contact water sources is fed to the main collection pond (MCP) which was represented by a Goldsim® transport cell behaving as a fully-mixed tank (Figure 2-2). Discharge water from the main collection pond (MCP) was treated by the proposed water treatment plant and then released to the settling pond, which was also simulated with the use of the Goldsim® transport cell. The load released from the settling pond to the receiving Silvertip Creek was calculated as the product of the overflow rate of the settling pond and the instantaneous concentration of each parameter of interest in the settling pond cell.

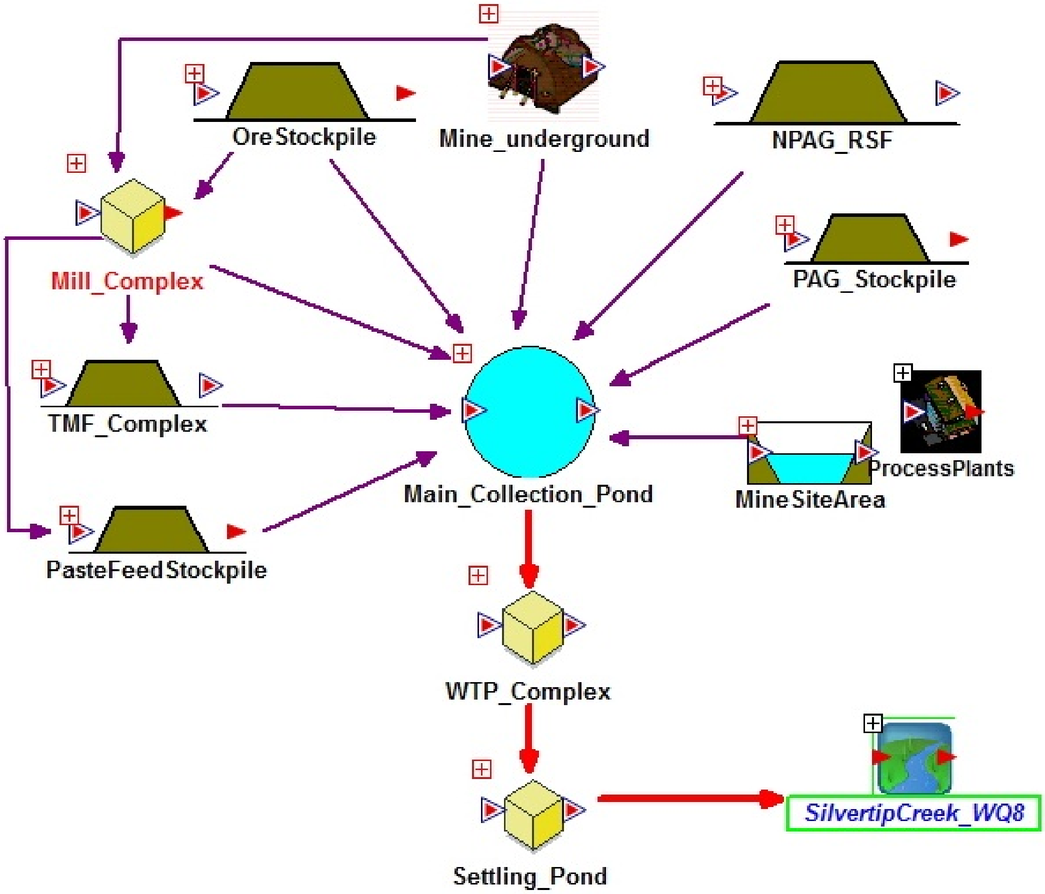


Figure 2-2: **Schematic showing the Goldsim® model layout for water quality loadings**

The load discharged from the settling pond is propagated downstream to the mixing node in the receiving creeks. The model assumed that completion of fully mixing of source waters with the background flow has been achieved at the compliance point. As such, water quality concentrations at the compliance location were computed using conservation of mass according to Equation (2.1-1). Of note, the background concentrations for all water quality parameters were derived externally to the model and presented in Section 3.3.

$$C_{CP} = \frac{\sum_{i=1}^{N} Q_i C_i}{\sum_{i=1}^{N} Q_i} \qquad (2.1)$$

where C_{CP} is the contaminant concentration at the compliance point (*i.e.*, WQ8) and C_i and Q_i represent the concentrations and flows of all loading sources (including background) to the compliance point, respectively.

3. Model Inputs

3. Model Inputs

The water quality model is a mass balance model that estimates the concentration of a parameter of interest by combining loadings from all mine contact sources with loadings that are in the background receiving environment (Section 2). Loadings (mg/s) are calculated as the product of the flow rate (L/s) and concentration (mg/L) of each parameter of interest. This chapter provides a description of input terms of the water balance (Section 3.1), receiving water background concentrations (Section 3.2) and contact water source term inputs (Section 3.3) to the water quality model.

3.1 Water Balance

Inputs to the water quality model of monthly average flows in the receiving environment and mine contact flows are described below.

3.1.1 Receiving Environment Flows

The receiving environment is Silvertip Creek downstream of the main sedimentation pond, represented by monitoring site WQ8. WQ8 hydrology is described in MAPA Vol.2 Appendix 2-II-B. This includes pressure transducer measurements in 1998-2000 and 2011, and a stage-discharge curve derived from manual flow measurements made throughout the baseline monitoring program (1984-2011). Monthly average discharges at WQ8, as presented in Table 6 of the MAPA Appendix 2-II-B, are specific to the years with pressure transducer measurements. In order to derive flows typical of the long-term average, a regression analysis was conducted between site specific data and daily flow measurements at Water Survey of Canada hydrometric station WSC 10AA004 at the mouth of the Rancheria River. The regression derived was used to rectify monthly average flows at WQ8 (Table 3-1). Rectified flows at WQ4 (Tootsee River) were also used in calibration of the Goldsim® modeled flows.

Table 3-1:
Rectified long-term average discharge at WQ8 and WQ4 used in the updated water quality model

	Discharge by Month (m^3/s)											
Location	**Jan**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**
WQ8	0.060	0.056	0.057	0.067	0.387	0.744	0.356	0.241	0.256	0.189	0.123	0.090
WQ4	2.26	1.66	1.52	1.03	7.45	12.34	5.83	4.07	5.68	3.80	3.88	2.44

3.1.2 Contact Water Flows

Contact water from the various mine facilities was derived from mine production schedule (Appendix A) and precipitation estimates presented in MAPA Vol. 2. Monthly average flows were calculated as the product of precipitation and a runoff coefficient. Monthly average hydrographs from the Rancheria watershed were used to distribute annual precipitation into monthly precipitation estimates. Monthly average flows in contact with mine infrastructure are described below. Flows were estimated coming from the tailings management facility (TMF); from non-acid generating (NAG) rock storage facility; temporary stockpiles of ore, paste feed material, and potentially acid generating (PAG) rock; and from underground dewatering.

Contact water emanating from each stockpile was quantified from a pile water balance developed for each source. Precipitation falling on the pile surface was partitioned into infiltration and runoff according to a runoff coefficient and infiltration coefficient. The existence of preferential paths in mine stockpiles (MEND, 1995) was assumed, therefore infiltration coefficients (Table 3-2) were assumed to be high (≥80%) for all the uncovered piles. The model also assumes no water loss through evaporation from the pile. As a result of this, the model will conservatively estimate the flow from the stockpiles. The effect of pile covers placed on the TMF and NAG-RSF during closure is represented by a reduction in infiltration coefficient in the model (Table 3-2).

Table 3-2:
Infiltration Coefficients for Mine Material Stockpiles

Stock Pile	Infiltration Coefficient	Remark
TMF	0.80	Operation phase only
TMF with the Cover	0.20	Closure Phase
NAG-RSF	0.90	Operation phase only
NAG-RSF with the Cover	0.20	Closure Phase
PAG Rock Pile	0.90	Operation phase only
Ore Stock Pile	0.90	Operation phase only
Paste Feed Stock Pile	0.85	Operation phase only

Equation 3.1 is the water balance equation used in the Goldsim® model for each stockpile.

$$(WI_{material} + WI_{infiltration}) - (WO_{material} + W_{seep}) = V_s \qquad (3.1)$$

where:

$WI_{material}$= the water volume input associated with the mine material deposition rate as determined from the pile material balance model,

$WI_{infiltration}$ = the water volume due to precipitation infiltration,

$WO_{material}$ = the water volume associated with the pile material removal rate that is determined from the pile material balance model

W_{seep} = the seepage from the stock pile, and

V_s = the storage volume within the pile.

In calculating the stockpile water balance with Equation 3.1, an excess volume of water was defined when the maximum storage void volume (*i.e.*, water storage capacity or maximum pile moisture) was reached. Generation of the seepage flow process was modeled by routing this excess water volume and the water storage. In this modeling work, the routing transfer function proposed by Fenicia et al. (2006) was employed, which consists of two linear reservoir filters, namely fast reacting reservoir and slow reacting reservoir. Physically and conceptually, the fast flow can be attributed to water infiltration through preferential flow channels and the slow flow might be from the matrix-zone storage release (Mend, 1995; Fenicia et al., 2006). Specific to the Silvertip model, the fast flow reacting time was assumed to range between 5 to 10 days and the slow reacting time was assumed to have a time scale of months.

Flow estimates for each facility are described below. Also described are water balance models for the mine plant and mine site surface runoff.

3.1.2.1 Tailings Management Facility (TMF)

The TMF footprint area was modeled as largely the same area as its design footprint area (~ 5.3 ha) in accordance to its construction plan. Liner performance was modeled by assuming an efficiency of 100% (*i.e.*, no leakage). In this regard, all the water produced from the TMF area was modeled as contact water.

The model assumes that placement of a cover on the TSF is effective immediately at the end of the operation phase. A reduced infiltration coefficient of 20% was also simulated for the TMF water balance estimate at closure.

3.1.2.2 NAG Rock Storage Facility

The phased NAG-RSF footprint schedule (Appendix A) was a direct input into the model. Accordingly, the “bare” footprint area was calculated as the difference between the design and the phased footprint areas.

Water emanating from the NAG-RSF pile itself was modeled following the procedure as described above. Runoff generated from the bare footprint area was estimated by multiplying the precipitation by the runoff coefficient. In support of a conservative flow estimate, the runoff coefficient was set to be 0.70, nearly twice of the value (*i.e.*, 0.37) for the background area (obtained through model calibration and validation as described in Section 4.1.1).

The model assumed an efficiency of 100% for the NAG-RSF collection and diversion system which collect and direct all NAG-RSF waters to the main collection pond (MCP).

According to the mine plan, a soil cover is put in place at the closure phase. The effect of the soil cover on the pile water balance was represented by a reduced infiltration coefficient (*i.e.*, ~20%, Table 3.1-2). The model also assumed that the cover would be placed over a sufficiently short time period (*e.g.*, less than one day) and is fully effective immediately at the end of the operation phase.

3.1.2.3 Stockpile Materials

Compared to TMF and NAG-RSF, three temporary stockpiles will be removed at the end of the operation phase. Aside from this, another notable departure for these three temporary stockpiles is the pile footprint for each of them is expected to vary dynamically with the pile volume. In the model, this dynamic footprint was calculated with the use of Equation 3.2, assuming a conical geometry of the pile. In the absence of input values, the design footprint (*i.e.*, the maximum deposition area) of each of the stockpiles as provided by JDS was used:

$$A_{footprint} = \frac{V\sqrt[3]{\pi}}{\left(\frac{1}{3}Slope\right)^{2/3}} \qquad (3.2)$$

where: $A_{footprint}$ is the pile footprint area,

V is the pile volume as determined through the pile material balance model, and

the pile slope was set equal to 1/3 for all three piles.

Runoff generation over the "bare" footprint area was determined as the product of the precipitation and the runoff coefficient. Similar to the mine site surface catchment (to be addressed in Section 3.1.2.6), the runoff coefficient for the bare pile footprint was set equal to 0.70 to achieve a conservative approach. Also for the purpose of conservative water quality prediction, the runoff from the bare ground within the design footprint area was modeled as "contact water", characterized by similar chemistry as the water from the mine site surface area.

Specific considerations in the model construction for each of these three temporary piles are addressed below.

Ore Stockpiles

The Run of Mine (ROM) ore stockpile is a temporary stockpile to hold ores according to the production schedule that feeds the mill when the mill is on. According to the mine schedule, the ore pile is expected to remain on the ground surface from May through September of each calendar year during the mine operation phase.

PAG Mine Rock Stockpiles

The PAG mine rock stockpile is a transient material reservoir to hold PAG mine rock. It receives rock material from the mine and delivers materials in response to CRF and rock fill demands. It receives water only from climatic sources and loses water via seepage and with the material removal (as the rock supply for Rock Fill material and Cement Rock Fill).

At closure, any remaining PAG material will be directed underground. The time required to remove the PAG pile at closure was assumed to be sufficiently short and therefore would not affect the model construction.

Paste-Feed Stockpiles

The Paste Feed Stockpile is intended to store the pyrite tailings and a fraction of de-sulphidized tailings feed for paste. According to the mine schedule, paste Feed material is supplied from the mill and is drawn by the Paste Plant.

Similar to the Ore stockpile, the paste feed pile will only remain on the ground surface over the months from May to November within each calendar year during operation phases.

3.1.2.4 Underground Dewatering

At present the underground workings are flooded. The model assumes that dewatering of the underground will be carried out within the first month of the operation phase.

The groundwater influx to the mine has been defined from the groundwater model (MAPA Appendix 9-IV-A). Accordingly, the phased average flows from the underground workings used in the model were presented in Table 3-3. Of note, it was assumed the portal discharge rates at closure were modeled to remain at the pre-development levels (~3.2 L/s).

Table 3-3:
Phased Average Flows from Underground Workings

Phase or year	Pre-development
Pre-development	3.2
0	10
5	10
7	15
19	8
Closure	3.2

The dewatering flow rates presented in Table 3-3 are average flow rates over the respective phase periods. Flow from the underground varies seasonally, as described in MAPA Vol. 1 Section 2.5.3.2. Monthly average flow for each phase was determined by multiplying the annual flow from each phase by the monthly proportion of the annual flow (Table 3-4), which were determined from flow measurements at the portal discharge (WQ9) in 2010 and 2011 (MAPA Table 2.5.7). The seasonal distribution of flow from the portal follows the same pattern as the monthly distribution for the stream flow as derived from WSC Rancheria River near the Mouth Station, with a less pronounced spring peak flow.

Table 3-4:
Monthly Distribution Percentages of Portal Discharges Compared to Streams

	% of Annual Flow	
Month	Portal	Stream
January	6.6	2.3
February	6.5	2.1
March	4.7	2.2
April	2.9	2.6
May	10.1	14.7
June	14	28.3
July	12.7	13.6
August	8.6	9.2
September	9.2	9.7
October	9.9	7.2
November	8.1	4.7
December	6.8	3.4

3.1.2.5 Mill Plant Operation

The mill water balance reflects the mill operation plan as provided by JDS (Figure 3-1). The mill draws ore from the Run of Mine (ROM) ore stockpile and generates zinc and lead concentrate, de-sulphidized tailings and pyrite tailings. The de-sulphidized tailings are mixed with the pyrite concentrate to form the feed for the paste feed stockpile for supply to the paste plant. The remaining de-sulphidized tailings are dewatered and disposed of in the Tailings Storage Facility (TSF). The mill will be operated on a 25 days on/off cycle from May throughout September during the operation phase.

The water balance model of the mill was constructed on the requirement to maintain water contents in material flows through the mill process and end products, and accounting for specific demands on designated water sources. As illustrated in Figure 3-1 the process water is received via an internal tank from ore and mine portal discharge with additional make-up water supply from the main collection pond (MCP) (calculated as a deficit in response to the process water requirement). Water is lost with the material products, de-sulphidized tailings and pyrite tailings. Water is also lost for flushing paste backfilling pipelines. Any excess water output is directed to the MCP.

Since the mill is scheduled to run in 25day on/off cycles, it is important to note that the mill water balance model and the flow chart were developed only for the "on" periods.

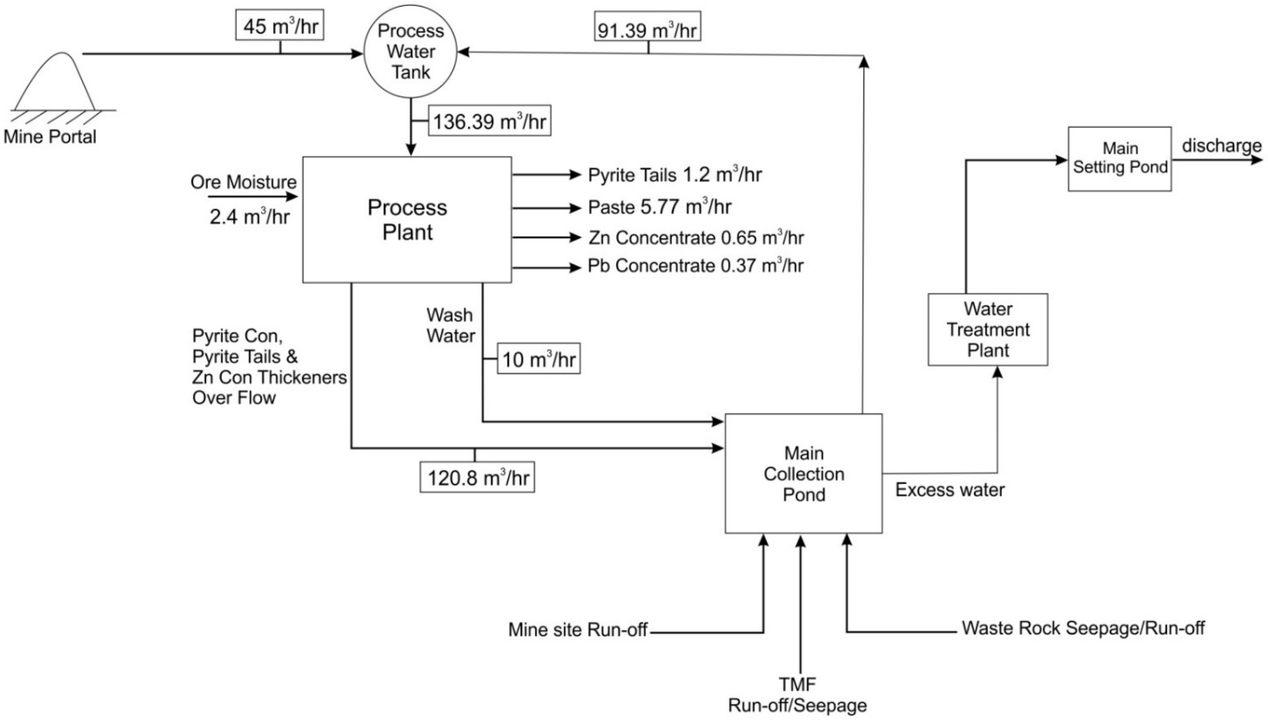


Figure 3-1: Mill Water Balance Schematic

3.1.2.6 Mine Site Surface Area Runoff

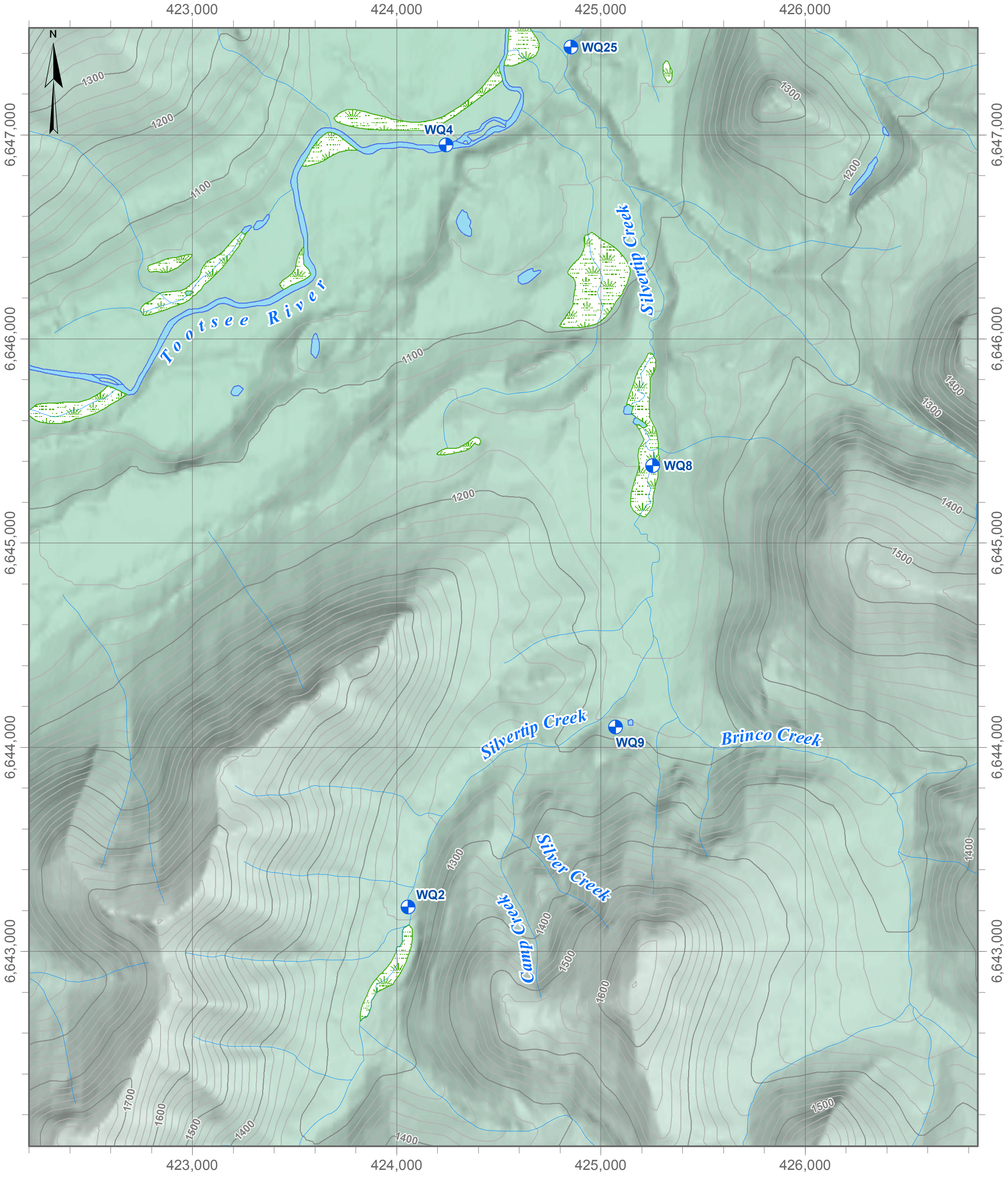
The runoff generated from the mine site surface area was quantified by multiplying the precipitation by the runoff coefficient. The area of the mine site catchment was calculated to reflect the dynamic response to the footprint changes of the mine stockpiles including ore stockpile, PAG rock stockpile, and paste feed stockpile during the operation phase. In the model, the phased runoff coefficient was assigned as below.

- Pre-mine phase, the mine site area represents natural ground surface, and therefore the runoff coefficient was modeled to be 0.37, which was determined through model calibration and validation (Section 4.1.1)
- Operation phase, a runoff coefficient of 0.70 was used, which is nearly twice of that for the natural catchment (*i.e.*, 0.37). The use of this considerably high runoff coefficient will assure a conservative estimate on the mine site contact flow generation.
- Closure Phase, the processing mine facilities on the surface area will be decommissioned and the surface area will be reclaimed. Accordingly, the runoff coefficient was assumed to drop from 0.70 for the mine operation phase to 0.50. Note that this runoff coefficient is still conservatively higher than that of the natural catchment.
- Post closure, the runoff coefficient was set equal to that of the pre-mine phase (*i.e.*, 0.37).

3.2 Receiving Environment Water Quality

The receiving environment for the Project is Silvertip Creek (Figure 3-2). Baseline monitoring of Silvertip Creek downstream of the Project, at site WQ8, began on September 4, 1984. Underground excavation began in October, 1984 and ended in November 1985 with 1450 m of underground tunnelling completed. Water from active dewatering of tunnels during exploration and passive dewatering after tunnels were allowed to flood was directed to a settling pond near the portal. The settling pond does not have a surface discharge because water in the pond infiltrates to ground. This water will flow to Silvertip Creek near the settling pond, and contribute to the water quality measured at WQ8. Therefore only 1 out of the 113 baseline water quality samples collected at WQ8, between 1984 and 2012, is representative of background (pre-development) water quality downstream of the Project.

Water quality at WQ8 is markedly influenced by natural drainage from the area of the Silvertip deposit. Camp Creek and Silver Creek are tributaries to Silvertip Creek, located upstream of the portal, which have acidic water with elevated sulphate and metal levels. For example, dissolved Zn levels ranged from 3.3 to 41.3 mg/L in baseline samples from these creeks. A 2012 survey of springs and seeps in the Project area found a maximum zinc concentration of 13.5 mg/L (Table 2.6.12 of MAPA Vol. 1). The September 1984, pre-development monitoring of Silvertip Creek water quality illustrates that these sources naturally elevate the concentration of some parameters downstream of the Silvertip deposit (WQ8) compared to upstream Silvertip Creek (WQ2) (Table 3-5).



LEGEND

- ◐ Water Sampling Locations
- — Watercourses
- ▭ Waterbodies
- ▨ Wetlands
- — Contours (20m)
- — Contours (100m)

DATE SAVED:	Oct 14, 2014
DRAWN BY:	GM
REVIEWED:	DF
VERSION:	1

Coordinate System: *NAD 1983 UTM Zone 9N*
Projection: *Transverse Mercator*
Datum: *North American 1983*
Units: *Meter*

1:25,000

0 200 400 Meters

CLIENT:

JDS Energy & Mining Inc.

LORAX ENVIRONMENTAL

PROJECT:

Silvertip MAPA Addendum

TITLE:

Surface Water Quality
Monitoring Sites

PROJECT #: A379-1

FIGURE: 3-2

P:\@Drafting\Silvertip\Drafting Figures\MXD\Figure 3_Regional Sampling Locations_20141014.mxd

Table 3-5:
September 4th, 1984 sulphate and dissolved zinc concentration in Silvertip Creek upstream (WQ2) and downstream (WQ8) of the Silvertip deposit

Parameter	WQ2	WQ8
Sulphate	19	37
Zn-D	0.020	0.250

Because these natural sources enter Silvertip Creek at approximately the same location as drainage from the portal, upstream Silvertip Creek water quality cannot be used to represent background water quality in the receiving environment. As well, because contaminant loadings from portal dewatering are included in all but one baseline sample from downstream Silvertip Creek, baseline monitoring at WQ8 is not representative of background water quality. Therefore, in order to derive background loadings in the receiving environment for input to the water quality model, loadings contributed from the portal (monitoring site WQ9) were subtracted from baseline loadings at WQ8. Loadings were calculated as the product of monthly average flow (Section 3.1.1) and monthly average concentration.

Data used in the model for generating water quality predictions or validation of predictions are listed in Table 3-6 to Table 3-9. The full water quality database and statistical summaries are provided in Appendix F.

The following conditions were used for generating monthly mean water quality data:

- baseline data from the 2010-1012 monitoring program were used.
- concentrations less than detection limit were set equal to the detection limit.
- months with no baseline data were estimated as follows: Jan. data was set equal to the average of Dec. and Feb. monthly means. Months with no nitrate or nitrite were set equal to the median of months with data. Months with no data are shown as dark red font in data tables.
- WQ2 was not monitored in Dec., Jan, or Feb. therefore the model assumes that Jan. and Feb. concentrations were equal to Mar., and Dec. was the average of Nov. and Mar. monthly mean.
- concentrations of Cl, Sb, Cr, Co, Fe, Mn, and K were near or below detection limit in baseline WQ8 samples, therefore no loadings from the portal were subracted for these parameters to derive background water quality in lower Silvertip Creek. Background alkalinity and pH were also set equal to baseline.

Table 3-6:
Current (2010-2012) monthly mean baseline concentration (mg/L) at site WQ2

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Hardness	183	183	183	186	166	127	132	148	140	150	164	164
Alkalinity	142	142	142	134	124	91	93	102	96	109	112	127
pH	8.1	8.1	8.1	7.9	8.1	8.2	8.2	8.2	8.0	8.2	8.2	8.2
NO_3-N	0.06	0.06	0.06	0.07	0.09	0.09	0.03	0.03	0.04	0.06	0.06	0.06
NO_2-N	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.005	0.005	0.001	0.001	0.001
NH_4-N	0.005	0.005	0.005	0.009	0.005	0.005	0.005	0.005	0.017	0.005	0.005	0.005
SO_4	43	43	43	42	37	34	37	40	41	45	48	45
F^-	0.05	0.05	0.05	0.07	0.07	0.07	0.06	0.07	0.06	0.08	0.08	0.07
Cl^-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Al-D	0.003	0.003	0.003	0.003	0.014	0.012	0.007	0.005	0.006	0.007	0.004	0.004
As-T	0.0003	0.0003	0.0003	0.0003	0.0004	0.0003	0.0003	0.0003	0.0003	0.0003	0.0004	0.0003
Cd-T	0.00013	0.00013	0.00013	0.00012	0.00018	0.00021	0.00018	0.00016	0.00017	0.00017	0.00020	0.00017
Cd-D	0.00010	0.00010	0.00010	0.00011	0.00014	0.00018	0.00016	0.00013	0.00015	0.00014	0.00008	0.00009
Cr-T	0.0005	0.0005	0.0005	0.0005	0.0004	0.0004	0.0002	0.0004	0.0003	0.0005	0.0008	0.0007
Co-T	0.0001	0.0001	0.0001	0.0001	0.0002	0.0001	0.0001	0.0001	0.0001	0.0001	0.0002	0.0001
Cu-T	0.0006	0.0006	0.0006	0.0005	0.0011	0.0009	0.0005	0.0005	0.0005	0.0005	0.0006	0.0006
Fe-T	0.048	0.048	0.048	0.039	0.150	0.081	0.039	0.041	0.041	0.039	0.172	0.110
Fe-D	0.030	0.030	0.030	0.030	0.033	0.024	0.019	0.023	0.021	0.030	0.030	0.030
Pb-T	0.0003	0.0003	0.0003	0.0002	0.0007	0.0003	0.0001	0.0001	0.0001	0.0001	0.0004	0.0004
Mn-T	0.008	0.008	0.008	0.006	0.012	0.009	0.007	0.011	0.007	0.008	0.015	0.011
Mo-T	0.00073	0.00073	0.00073	0.00074	0.00066	0.00054	0.00062	0.00066	0.00057	0.00065	0.00074	0.00073
Ni-T	0.0010	0.0010	0.0010	0.0007	0.0013	0.0015	0.0010	0.0009	0.0009	0.0010	0.0016	0.0013
Ag-T	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Se-T	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Zn-T	0.031	0.031	0.031	0.029	0.038	0.028	0.019	0.017	0.017	0.021	0.026	0.028

Table 3-7:
Current (2010-2012) monthly mean baseline concentration (mg/L) at site WQ8

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Hardness	211	211	216	227	120	126	150	165	156	163	185	224
Alkalinity	121	121	138	126	75	82	93	101	98	101	111	120
pH	8.1	8.1	7.9	8.1	7.8	8.1	8.2	8.2	8.1	8.2	8.2	8.1
NO_3-N	0.05	0.03	0.03	0.04	0.07	0.03	0.01	0.02	0.02	0.03	0.03	0.08
NO_2-N	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.005	0.005	0.001	0.001	0.001
NH_4-N	0.005	0.005	0.005	0.006	0.005	0.005	0.005	0.005	0.013	0.005	0.005	0.005
SO_4	79	82	92	88	41	45	53	58	57	61	70	76
F^-	0.08	0.08	0.06	0.11	0.07	0.07	0.07	0.09	0.07	0.09	0.10	0.08
Cl^-	0.5	0.5	0.7	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Al-D	0.004	0.004	0.005	0.005	0.059	0.028	0.010	0.009	0.012	0.005	0.004	0.003
As-T	0.0004	0.0004	0.0005	0.0005	0.0037	0.0005	0.0005	0.0004	0.0005	0.0003	0.0005	0.0004
Cd-T	0.00122	0.00127	0.00142	0.00134	0.00113	0.00076	0.00080	0.00093	0.00091	0.00085	0.00099	0.00117
Cd-D	0.00124	0.00124	0.00132	0.00134	0.00074	0.00073	0.00077	0.00090	0.00082	0.00079	0.00094	0.00123
Cr-T	0.0005	0.0005	0.0005	0.0005	0.0009	0.0005	0.0003	0.0004	0.0006	0.0005	0.0005	0.0005
Co-T	0.0001	0.0001	0.0001	0.0001	0.0009	0.0004	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
Cu-T	0.0005	0.0005	0.0007	0.0006	0.0072	0.0019	0.0009	0.0007	0.0013	0.0006	0.0005	0.0005
Fe-T	0.032	0.030	0.030	0.033	1.339	0.100	0.055	0.026	0.224	0.030	0.073	0.034
Fe-D	0.030	0.030	0.030	0.030	0.066	0.030	0.020	0.024	0.020	0.030	0.030	0.030
Pb-T	0.0005	0.0004	0.0007	0.0003	0.0201	0.0025	0.0009	0.0004	0.0024	0.0004	0.0007	0.0006
Mn-T	0.005	0.005	0.006	0.006	0.060	0.018	0.006	0.006	0.017	0.004	0.005	0.006
Mo-T	0.00047	0.00048	0.00050	0.00050	0.00040	0.00034	0.00043	0.00044	0.00056	0.00046	0.00048	0.00047
Ni-T	0.0038	0.0039	0.0041	0.0048	0.0062	0.0036	0.0026	0.0028	0.0032	0.0028	0.0032	0.0037
Ag-T	0.00001	0.00001	0.00001	0.00001	0.00013	0.00002	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Se-T	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Zn-T	0.326	0.353	0.338	0.350	0.226	0.147	0.185	0.219	0.205	0.204	0.267	0.298

Table 3-8:
Current (2010-2012) monthly mean baseline concentration (mg/L) at site WQ9

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Hardness	510	497	488	495	488	504	501	482	489	499	504	541
Alkalinity	164	167	171	166	170	174	170	167	166	176	170	160
pH	7.4	7.2	7.5	7.8	7.6	7.8	7.6	7.8	7.8	7.7	7.7	7.6
NO_3-N	0.07	0.08	0.08	0.05	0.08	0.08	0.15	0.14	0.10	0.08	0.08	0.05
NO_2-N	0.001	0.001	0.001	0.001	0.001	0.001	0.005	0.005	0.005	0.001	0.001	0.001
NH_4-N	0.011	0.011	0.011	0.013	0.011	0.011	0.014	0.014	0.034	0.019	0.013	0.012
SO_4	309	308	305	303	309	320	326	311	294	319	333	309
F^-	0.52	0.51	0.33	0.49	0.52	0.57	0.55	0.45	0.34	0.55	0.53	0.53
Cl^-	0.5	0.5	0.5	0.6	0.5	0.5	4.3	0.5	0.5	0.5	0.5	0.5
Al-D	0.005	0.003	0.006	0.004	0.026	0.010	0.004	0.003	0.002	0.003	0.006	0.006
As-T	0.0070	0.0076	0.0050	0.0046	0.0038	0.0061	0.0061	0.0058	0.0061	0.0055	0.0071	0.0063
Cd-T	0.01280	0.01300	0.01145	0.01258	0.01178	0.01327	0.01680	0.01543	0.01485	0.01230	0.01300	0.01260
Cd-D	0.01155	0.01150	0.01080	0.01150	0.01197	0.01277	0.01625	0.01467	0.01420	0.01240	0.01240	0.01160
Cr-T	0.0005	0.0005	0.0005	0.0005	0.0004	0.0001	0.0001	0.0001	0.0001	0.0005	0.0005	0.0005
Co-T	0.0022	0.0022	0.0021	0.0021	0.0020	0.0022	0.0023	0.0023	0.0022	0.0020	0.0022	0.0021
Cu-T	0.0008	0.0005	0.0010	0.0008	0.0005	0.0007	0.0009	0.0006	0.0003	0.0005	0.0010	0.0010
Fe-T	1.095	1.220	0.817	0.642	0.482	0.952	0.888	0.806	0.815	1.160	1.260	0.970
Fe-D	0.093	0.030	0.124	0.078	0.778	0.142	0.020	0.022	0.132	0.030	0.038	0.156
Pb-T	0.0004	0.0005	0.0003	0.0008	0.0003	0.0006	0.0009	0.0006	0.0006	0.0005	0.0007	0.0004
Mn-T	0.341	0.361	0.335	0.328	0.313	0.364	0.362	0.358	0.342	0.337	0.375	0.320
Mo-T	0.00063	0.00064	0.00062	0.00063	0.00065	0.00061	0.00056	0.00061	0.00056	0.00062	0.00073	0.00062
Ni-T	0.0284	0.0285	0.0280	0.0294	0.0265	0.0302	0.0359	0.0339	0.0338	0.0277	0.0302	0.0283
Ag-T	0.00002	0.00001	0.00002	0.00002	0.00002	0.00002	0.00003	0.00002	0.00001	0.00001	0.00002	0.00002
Se-T	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Zn-T	2.645	2.720	2.270	2.504	2.333	2.713	3.440	2.970	2.975	2.460	2.640	2.570

Table 3-9:
Monthly mean background concentration (mg/L) at site WQ8 after subtraction of loadings from WQ9

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Hardness	200	197	202	216	114	112	149	159	152	160	176	203
Alkalinity	121	121	138	126	75	82	93	101	98	101	111	120
pH	8.1	8.1	7.9	8.1	7.8	8.1	8.2	8.2	8.1	8.2	8.2	8.1
NO_3-N	0.05	0.02	0.02	0.04	0.07	0.03	0.01	0.02	0.02	0.03	0.03	0.08
NO_2-N	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.005	0.005	0.001	0.001	0.001
NH_4-N	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.012	0.005	0.005	0.005
SO_4	68	68	81	82	37	43	49	54	52	55	61	67
F^-	0.06	0.06	0.05	0.10	0.07	0.06	0.06	0.08	0.06	0.08	0.08	0.07
Cl^-	0.5	0.5	0.7	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Al-D	0.003	0.004	0.004	0.005	0.058	0.028	0.010	0.008	0.012	0.005	0.003	0.003
As-T	0.0002	0.0001	0.0003	0.0004	0.0037	0.0004	0.0004	0.0003	0.0004	0.0002	0.0003	0.0002
Cd-T	0.00076	0.00071	0.00100	0.00109	0.00100	0.00067	0.00061	0.00072	0.00069	0.00062	0.00065	0.00080
Cd-D	0.00082	0.00074	0.00093	0.00112	0.00060	0.00064	0.00059	0.00070	0.00061	0.00055	0.00061	0.00089
Cr-T	0.0005	0.0005	0.0005	0.0005	0.0009	0.0005	0.0003	0.0004	0.0006	0.0005	0.0005	0.0005
Co-T	0.0001	0.0001	0.0001	0.0001	0.0009	0.0004	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
Cu-T	0.0005	0.0005	0.0006	0.0006	0.0072	0.0019	0.0008	0.0006	0.0013	0.0006	0.0005	0.0005
Fe-T	0.032	0.030	0.030	0.033	1.339	0.100	0.055	0.026	0.224	0.030	0.073	0.034
Fe-D	0.030	0.030	0.030	0.030	0.066	0.030	0.020	0.024	0.020	0.030	0.030	0.030
Pb-T	0.0005	0.0003	0.0007	0.0003	0.0201	0.0025	0.0009	0.0004	0.0024	0.0004	0.0006	0.0006
Mn-T	0.005	0.005	0.006	0.006	0.060	0.018	0.006	0.006	0.017	0.004	0.005	0.006
Mo-T	0.00045	0.00045	0.00047	0.00049	0.00039	0.00034	0.00043	0.00043	0.00055	0.00045	0.00046	0.00045
Ni-T	0.0028	0.0027	0.0031	0.0043	0.0059	0.0033	0.0022	0.0023	0.0027	0.0022	0.0024	0.0029
Ag-T	0.00001	0.00001	0.00001	0.00001	0.00013	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Se-T	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Zn-T	0.229	0.236	0.256	0.302	0.199	0.128	0.146	0.180	0.160	0.157	0.198	0.223

3.3 Contact Water Source Terms

In this section water quality is predicted for various mine components that will exist during mine operations and remain on site upon mine closure. For each mine component an upper case and a base case source term is predicted. The base case is meant to reflect a best estimate, while the upper case is meant to reflect a reasonably conservative upper estimate.

Mine components for which operations source terms are derived include; NPAG stockpile, PAG stockpile, mine facilities area, ore stockpile, paste feed stockpile, the tailings management facility and the underground mine workings. Note that during mine life and for a time period afterwards drainage from all of these mine components will be captured and treated with an HDS lime treatment plant. Mine components that will exist upon mine closure when no active treatment is taking place include: NPAG stockpile, TMF, and the underground mine workings. Source terms are generally developed in the following steps:

1. Selection of kinetic test data which best represents a given mine facility;
2. application of scaling factors to scale laboratory loading rates to field conditions;
3. calculation of scaled mine facility loading rates;
4. predict concentration based on facility size and drainage volume; and
5. identification of potential solubility controls.

The Goldsim® water balance model uses the scaled mine facility loading rates to predict minesite water quality, with the notable exception of portal drainage. For some parameters solubility controls are identified. The Goldsim® model only applies these solubility controls if the concentrations calculated in the Goldsim® model exceed the concentration limit. Portal drainage is treated differently than other mine site components, in that the source term is applied as a constant concentration rather than loading rate. The advantage of applying a loading rate is that source terms can evolve over mine life as the size of mine facilities change. However, considering the number of components contributing to portal drainage chemistry, a dynamic concentration calculation is considered overly complex for the purposes of the water balance model. As such, a concentration of portal drainage at the end of mine life when backfill volumes and wall rock exposures are at their maximum extent is provided. This concentration is applied to portal drainage throughout operations as a conservative assumption. A closure concentration is calculated which reflects portal drainage after mine workings have flooded and the water table has rebounded up to the portal elevation.

Nitrogen loadings are calculated separately from other components of mine drainage chemistry. Unlike other parameters, nitrogen release is associated with mining/blasting practices rather than intrinsic geochemical properties of mine rock. Mine site nitrogen loadings are calculated based on assumptions regarding blasting residues on ore and waste rock brought to the surface, and residual concentrations released from wall rock and tailings. The calculations and assumptions used to estimate nitrogen loadings are provided in Appendix B.

The relative impact that mine facilities will have on site water quality is related to both drainage chemistry and volumes. Estimates of annual flow produced from mine components during operations and closure scenarios are shown in Figure 3-3 and Figure 3-4, respectively. The portal is expected to produce considerably higher flows than other mine components during both scenarios, as shown in (Figure 3-3 and Figure 3-4). As such, the chemistry associated with portal water will largely dictate water treatment requirements and post-closure water quality.

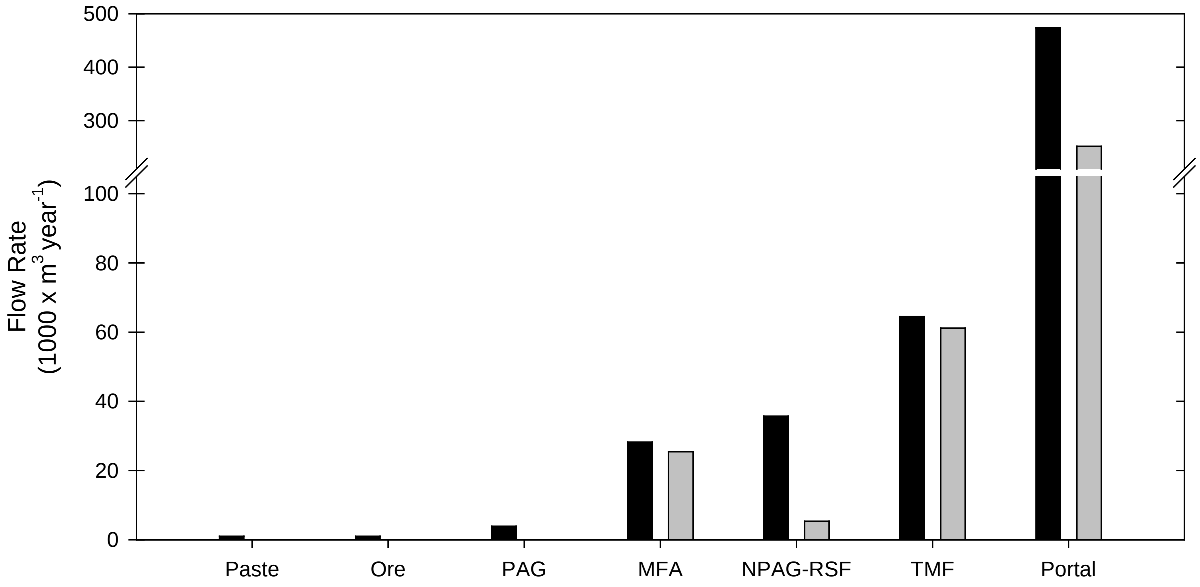


Figure 3-3: Average annual flow produced from mine facilities present during mine operations. The black bars represent maximum annual flow rates while the grey bars represent minimum annual flow rates during mine operations

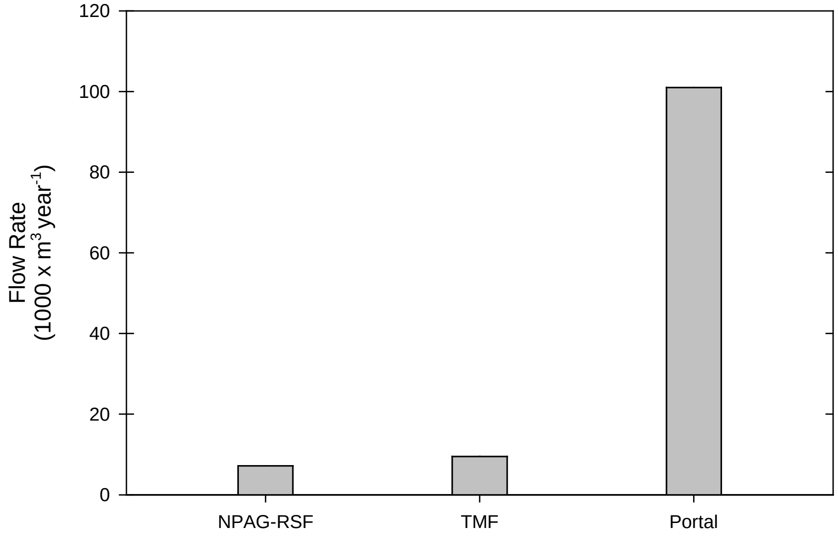


Figure 3-4: Average annual flow produced from mine facilities present at mine closure

3.3.1 Tailings Management Facility

The tailings management facility is a lined facility which will permanently store approximately 160,000 tonnes of desulphurized (DeS) tailings and 60,000 tonnes of NPAG waste rock. Tailings will be dry stacked, hence, no tailings pond will develop and tailings will remain unsaturated during mine life and upon closure. The style of construction is as follows:

- Excavate and line the base of the facility (approximately 5.3 ha);
- Place 0.5 m of dry stacked tails over entire liner as protection layer;
- Place 1.0 m layer of NAG waste rock to act as filter/drain layer over initial layer of filtered tailings;
- Construct TMF in 0.3 m compacted lifts; and
- Place NPAG waste rock on the final surfaces to provide long term erosion protection.

Note that both DeS tailings and NPAG waste rock stored in the TMF are non-acid generating.

3.3.1.1 Data Sources

Metal leaching rates for the DeS tailings is derived from upscaling a DeS tailings humidity cell constructed with tailings produced in 2012 metallurgical testwork (DeS HC3). A summary of 90th percentile and final loading rates for DeS HC3 are provided in Table 3-10.

The 90th percentile loading rates are used to predict operations source terms, while the final loading rates are used to predict closure source terms.

Metal loading rates for the NPAG waste rock placed in the TMF are based on modified loading rates observed in Earn waste rock humidity cells. The scaling approach and scaled rates are described in section 3.3.2 (Rock Storage Facility). Similar to DeS tailings, the 90th percentile loading rates are used to predict operations source terms, while the final loading rates are used to predict the closure source term (Table 3-10).

Table 3-10:
Humidity cell loading rates for desulphurized tailings humidity cell (DeS HC3). Final loading rates and 90th percentile loading rates are used to predict TMF closure and operations, respectively

Parameter	units	Final Loading Rate (median of cycles 24-40)	90th Percentile Loading Rate
SO_4	mg/kg/wk	168	402
As	mg/kg/wk	0.000574	0.00154
Ca	mg/kg/wk	69.14841	153.105
Cd	mg/kg/wk	0.00594	0.0161
Cu	mg/kg/wk	0.00177	0.00477
Mg	mg/kg/wk	0.622	2.83
Ni	mg/kg/wk	0.00730	0.0213
Pb	mg/kg/wk	0.0392	0.0814
Sb	mg/kg/wk	0.00578	0.0110
Se	mg/kg/wk	0.000419	0.0016
U	mg/kg/wk	0.0000243	0.0000391
Zn	mg/kg/wk	0.595	2.05

Note that final loading rate calculated from median of final 5 cycles that metals were analyzed for.

3.3.1.2 Scaling Factors

A series of physical scaling factors are applied to adjust loading rates to account for differences between lab-based kinetic tests and tailings in the field. These include but are not limited to hydrogeological pathways, water/rock ratio, temperature, grain size distribution, and reaction mechanisms. Values and rationale for these correction factors are described below. A summary of the scaling factors used to predict TMF source terms are presented in Table 3-11.

Table 3-11:
Summary of scaling factors used to predict source term concentrations

		Operations		Closure	
Scaling Factors	Unit	Base Case	Upper Case	Base Case	Upper Case
Temperature Correction	-	0.3	0.3	0.3	0.3
Contact Water Factor	-	0.25	0.5	0.075	0.15
Grain Size Correction Factor	-	1.0	1.0	1.0	1.0
Thickness of oxidation zone	m	1.0	1.5	0.5	1.0
Mass of oxidation zone/area*	Tonnes/m	1.7	3.4	0.85	1.7

*Calculated assuming bulk density of 1.7 kg/tonne.

Humidity cells are designed to achieve complete flushing of mineral surfaces, ensured by the small sample size (1 kg) and high flushing rate (0.5 L/week). It is assumed that 100% of mineral surfaces in humidity cells are rinsed. In the field, it is assumed that between 25% and 50% of waste rock surfaces are flushed. This range in contact water scaling factors is used for the upper case and base case source term predictions. At closure, drainage from the tailings facility will be reduced by approximately 66% (75% MAP during operations and 25% MAP during closure). This will result in a reduction in the percentage of tailings surfaces which are rinsed by infiltrating water. It is assumed that this reduction is proportional to the reduction in contact water, thus, the contact water scaling factor is adjusted accordingly.

Sulphide oxidation rates are temperature sensitive; therefore, the temperature difference between laboratory conditions and average temperature at the silvertip site must be taken into account. In order to derive an appropriate temperature correction weathering rates are scaled by an Arrhenius relationship, calculations for which are presented in Appendix C. As a conservative assumption, it is assumed that the interior of the TMF weathers at the average temperature between May and September (7°C). This results in a temperature scaling factor of 0.3. Note that this is considered conservative as average annual temperature at silvertip is near 0°C. Furthermore, the average monthly temperature is below freezing for approximately 6 months of the year. Sulphide oxidation will be inhibited entirely if pore water is frozen. Potential reductions in metal leaching rates due to freezing of tailings pore water are not incorporated into the source term calculation.

The mass of oxidizing tailings in the TMF will be limited by the rate of oxygen ingress. The depth of oxygen ingress will be controlled by tailings permeability, rate of oxygen consumption and water content along with other physical processes (*e.g.*, barometric pumping, temperature and pressure gradients etc.). As a conservative assumption it is assumed that the oxidation zone is 1.0 m to 1.5 m in thickness during operations. Assuming an oxidative thickness of 1.5 m across the tailings footprint area of 5.2 ha, equates to 133,000 tonnes of oxidizing tailings. Accordingly, approximately 80% of the 160,000 tonnes of tailings of tailings is considered oxidizing under the base case-operations scenario.

Upon closure, the surface layer of tailings will gradually become deplete in reactive sulphide minerals, and the zone of oxidation will move deeper into the tailings profile. The closure scenario assumes a reduced oxidative thickness of 0.5 m to 1.0 m. Applying the scaling factors presented in Table 3-11 to the loading rates presented in Table 3-10 results in the scaled loading rates presented in Table 3-12.

Table 3-12:
Scaled desulphurized tailings loading rates

		Operations		**Closure**	
	unit	**Base Case**	**Upper Case**	**Base Case**	**Upper Case**
SO4	mg/m2/year	2670000	8000000	167000	670000
As	mg/m2/year	10.2	30.6	0.57	2.28
Ca	mg/m2/year	1020000	3050000	68800	275000
Cd	mg/m2/year	107	320	5.91	23.6
Cu	mg/m2/year	31.6	94.9	1.76	7.04
Mg	mg/m2/year	447	1790	618	2470
Ni	mg/m2/year	141	423	7.26	29
Pb	mg/m2/year	540	1620	39	156
Sb	mg/m2/year	73.2	220	5.74	23
Se	mg/m2/year	10.3	31	0.417	1.67
U	mg/m2/year	0.259	0.777	0.0242	0.0966
Zn	mg/m2/year	13600	40800	591	2370

Loading rates for NPAG waste rock placed in the TMF are scaled separately from the tailings. In the initial stages of TMF construction, a 1.0 m layer of NPAG waste rock to act as filter/drain layer over the initial 0.5 m layer of filtered tailings. A final layer of NPAG waste rock is placed over the surface of the TMF at closure. In total, 60,000 tonnes of NPAG waste rock will be stored in the TMF. The same scaling factors applied to the NPAG waste rock in the NPAG storage facility are applied to waste rock in the TMF. These scaling factors are presented in Table 3-19 and the scaled loading rates are applied in Table 3-20. The sum of tailings loadings and waste rock loadings are presented as area normalized loading rates for the TMF in Table 3-13. Note that loading rates presented in Table 3-13 are generally similar to those presented in Table 3-10 which only consider the presence of tailings in the TMF, with the notable exception of U and Mg. The similarity in most parameters is due to the relatively small mass of NPAG waste rock compared to DeS tailings, and the relatively low metal leaching rate estimated for in NPAG waste rock compared to DeS tailings.

Table 3-13:
Humidity cell loading rates for desulphurized tailings humidity cell (DeS HC3). Final loading rates and 90th percentile loading rates are used to predict TMF closure and operations, respectively

		Operations		Closure	
	unit	Base Case	Upper Case	Base Case	Upper Case
SO_4	mg/m^2/year	2720000	8190000	171000	685000
As	mg/m^2/year	10.4	31.3	0.577	2.31
Ca	mg/m^2/year	1030000	3110000	69900	280000
Cd	mg/m^2/year	108	324	5.92	23.7
Cu	mg/m^2/year	32.9	99.8	1.98	7.92
Mg	mg/m^2/year	22500	71200	1060	4260
Ni	mg/m^2/year	142	425	7.29	29.1
Pb	mg/m^2/year	540	1620	39	156
Sb	mg/m^2/year	75.5	229	5.91	23.6
Se	mg/m^2/year	10.6	32.2	0.432	1.73
U	mg/m^2/year	1.69	6.52	0.123	0.493
Zn	mg/m^2/year	13900	41900	599	2390

Note that final loading rate calculated from median of final 5 cycles that metals were analyzed for.

3.3.1.3 Cyanide Species

Cyanide release cannot be predicted from DeS HC3 results because cyanide species were not monitored in humidity cell leachate. Due to the lack of direct evidence of cyanide release from DeS tailings, a highly conservative source term approach is adopted. Cyanide concentrations measured in pyrite rougher tailings supernatant produced during metallurgical testwork is used to estimate upper case-operations CN concentrations (WAD-CN of 0.476 mg/L and T-CN 0.522 mg/L). Cyanide concentrations observed in a rougher/cleaner tailings composite humidity cell T1 are used to estimate base case-operations cyanide leaching. For the base case scenario TMF drainage is assigned maximum WAD-CN and T-CN concentrations observed in humidity cell T1 of 0.01 and 0.18 mg/L, respectively. Upon closure it is assumed that cyanide species will decline to below detection limits (0.01 mg/L).

3.3.1.4 Secondary Mineral Controls

Up-scaling laboratory kinetic tests often results in concentrations that exceed thermodynamic solubility constraints. To account for mineral solubility controls that can reasonably be expected to occur in the field, the USGS geochemical modeling program PHREEQC was applied using the Lawrence Livermore National Laboratory (llnl.dat) database (Parkhurst and Apello, 1999). Select solubility controls are applied by speciating the up-scaled solution in PHREEQC, and allowing select minerals to precipitate if the saturation indices (SI) is greater than zero.

In order to apply solubility controls the scaled loading rates presented in Table 3-12 must first be converted into a concentration and an estimate of pH and alkalinity must be derived. In order to convert the loading rates into a concentration, the loading rates are divided into the annual surplus water (infiltration + runoff) of the TMF. TMF surplus water is assumed to be 75% of mean annual precipitation during operations (675 mm), and 25% of mean annual precipitation during closure (225 mm). The base case source terms are assigned the average pH observed in DeS HC3 (pH 7.72), while the upper case source terms are assigned the 10th percentile (pH 7.56). Alkalinity is predicted by equilibrating the solution with pCO_2 set at -3.0. Note that this pCO_2 is within the range typically observed in headwater streams (Butman and Raymond, 2011). Setting pCO_2 at this value results in alkalinities of 73 $mgCaCO_3/L$ and 41 $mgCaCO_3/L$ in the base case and upper case scenarios. Minerals allowed to precipitate are listed in Table 3-14. The PHREEQC input files are presented in Appendix D. Note that the Mn-oxide mineral manganite was oversaturated in the up-scaled solution. However, supersaturation was also identified in humidity cell leachate, where concentrations as high as 2.0 mg/L have been observed at pH values above 7.5. Note that Mn is only soluble at a neutral pH in its reduced oxidation state (+2), which will oxidize and precipitate in the presence of atmospheric oxygen. The elevated Mn concentration at a neutral pH is interpreted as the weathering of minerals containing Mn^{2+} and kinetically slow oxidation of Mn^{2+}. No specific Mn minerals have been identified in mineralogical testwork, however, Mn^{2+} commonly occurs as a replacement for Mg^{2+} and Ca^{2+} in carbonate minerals.

Table 3-14:
Secondary mineral controls applied to TMF source term

		Operations		Closure	
Mineral Phase	**Chemical Formula**	**Base Case**	**Upper Case**	**Base Case**	**Upper Case**
Gypsum	$CaSO_4:2H_2O$	x	x		x
ZnCO3:H2O	ZnCO3:H2O	x	x		x
Cerusite	$PbCO_3$	x	x	x	x
Otavite	$CdCO_3$	x	x		

3.3.1.5 Predicted Concentrations

Predicted concentrations for the Silvertip TMF during operations and closure are presented in Table 3-15. A number of these concentrations represent thermodynamic solubility controls, as outlined in Table 3-14 while others represent concentrations predicted by upscaling kinetic tests. Note that closure source terms are derived from a relatively young humidity cell (40 cycles) and not all parameters reflect stabilized loading rates. Further stabilization of loading rates would presumably result in lower source term predictions for the closure scenario.

Table 3-15:
Predicted source term concentration for the TMF during operations and closure scenarios

Parameter	Unit	Closure		Operations	
		Base Case	**Upper Case**	**Base Case**	**Upper Case**
pH	s.u.	7.72	7.56	7.72	7.56
T-Alkalinity	mgCaCO$_3$/L	67.3	50.4	73.5	53.9
N-NO$_3$	mg/L	1.4	2.3	1.4	2.3
N-NO$_2$	mg/L	0.3	0.6	0.3	0.6
N-NH$_3$	mg/L	2.2	2.6	2.2	2.6
WAD-CN	mg/L	0.01	0.01	0.01	0.476
T-CN	mg/L	0.01	0.01	0.18	0.522
SO$_4$	mg/L	764	1530	1720	1960
F	mg/L	0.953	3.81	2.11	6.45
Cl	mg/L	0.202	0.81	4.04	12.4
Ag	mg/L	0.0000888	0.000355	0.000554	0.00168
Al	mg/L	0.00674	0.0269	0.0307	0.104
As	mg/L	0.00257	0.0103	0.0154	0.0465
Ba	mg/L	0.05	0.2	0.164	0.503
Ca	mg/L	311	608	565	577
Cd	mg/L	0.0264	0.106	0.0751	0.266
Co	mg/L	0.0145	0.058	0.0663	0.2
Cr	mg/L	0.000567	0.00227	0.00856	0.026
Cu	mg/L	0.00882	0.0353	0.0488	0.148
Fe	mg/L	0.0227	0.0907	0.0685	0.208
Hg	mg/L	0.0000188	0.0000752	0.0000665	0.00020
K	mg/L	1.46	5.84	13.7	41.6
Mg	mg/L	4.74	19	33.4	111
Mn	mg/L	1.37	5.48	16.8	50.6
Mo	mg/L	0.0148	0.0592	0.0626	0.228
Na	mg/L	0.659	2.64	26.5	79.9
Ni	mg/L	0.0325	0.13	0.21	0.632
Pb	mg/L	0.0176	0.0219	0.0182	0.0227
Sb	mg/L	0.0263	0.105	0.112	0.34
Se	mg/L	0.00192	0.00769	0.0158	0.0477
Si	mg/L	1.21	4.83	7.19	21.8
Sr	mg/L	0.128	0.51	0.487	1.49
Tl	mg/L	0.0023	0.00921	0.00989	0.0299
U	mg/L	0.000549	0.00219	0.00252	0.00968
Zn	mg/L	2.65	9.12	4.63	9.92

3.3.2 Rock Storage Facility

Approximately 580,000 tonnes of NAG waste rock mined from the McDame formation will be produced over mine life. Most of the waste rock (520,000 tonnes) will be placed in the rock storage facility (RSF), with the rest being stored in the TMF as discussed in the previous section. The RSF facility has a planned footprint of 4 ha and will be located immediately north of the plant site area. The facility will be developed progressively in 5 m layers beginning at the south end moving north. The RSF will have a maximum thickness of 25 m and an overall slope of 2.5H:1V at the end of mine life.

3.3.2.1 Data Sources

Over 100 static test samples have been collected from the McDame formation. However, data regarding metal leaching potential from this unit is limited as kinetic testwork on this rock unit has only recently been initiated. Existing SFE and solid phase metal abundance data indicate that Cd and Zn are the primary parameters of concern, with average SFE concentrations of 0.00094 and 0.394 mg/L, respectively.

In order to estimate a metal leaching rate for this rock unit, lower Earn group humidity cells were scaled by the ratio of SFE concentrations observed in the two rock units. This approach is adopted because it is assumed that historical weathering products which have accumulated on drill core samples are indicative of relative weathering rates of the two rock units. This approach will be re-considered once sufficient kinetic test data is available on McDame limestone samples. Average SFE concentrations for McDame limestone and Earn Group samples are presented in Table 3-16, alongside the ratio of the values used to scale humidity cell results. A summary of averaged 90th percentile and final loading rates from the two Lower Earn formation humidity cells and SFE scaled values used to represent McDame limestone are presented in Table 3-17. Moving forward, the SFE scaled humidity cell loading rates are used in NAG source term derivation. The 90th percentile loading rates are used to calculate operations source terms, while the final loading rates are used to calculate closure source terms.

Table 3-16: Mean SFE concentrations for McDame group and Earn group waste rock with concentration ratio used to scale humidity cell tests

Parameter	McDame SFE	Earn SFE	McDame SFE:Earn SFE
	mg/L	*mg/L*	*Ratio*
SO_4	252	586	0.43
As	0.00113	0.00327	0.35
Ca	96.7	208	0.47
Cd	0.0009406	0.0112	0.08
Cu	0.00089	0.00117	0.76
Mg	7.62	17.4	0.44
Ni	0.00825	0.108875	0.08
Pb	0.00150	0.0005375	2.79
Sb	0.0130	0.0134	0.97
Se	0.00140	0.00415	0.34
U	0.00795	0.00346	2.30
Zn	0.394	0.390	1.01

Table 3-17: Loading rates for Lower Earn Group humidity cells (HC2 and HC4) alongside SFE scaled loading rates used to represent McDame formation waste rock. Final loading rates and 90th percentile loading rates are used to predict NAG waste rock closure and operations respectively

Parameter	units	Final Loading Rate	90th Percentile	SFE Scaled Final Loading rate	SFE Scaled 90th Percentile Loading
SO_4	mg/kg/wk	66.7	145	28.7	62.3
As	mg/kg/wk	0.000151	0.000649	0.0000523	0.000224
Ca	mg/kg/wk	18.4	46.9	8.6	21.8
Cd	mg/kg/wk	0.00100	0.01515	0.0000844	0.00128
Cu	mg/kg/wk	0.00213	0.00206	0.00163	0.00157
Mg	mg/kg/wk	7.56	11.0	3.31	4.81
Ni	mg/kg/wk	0.00238	0.00812	0.000180	0.000615
Pb	mg/kg/wk	5.407E-05	0.000197333	0.000151	0.000551
Sb	mg/kg/wk	0.00125	0.00310	0.00121	0.00302
Se	mg/kg/wk	0.000320	0.00110002	0.000108	0.000370
U	mg/kg/wk	3.19E-04	8.00E-04	0.000733	0.00184
Zn	mg/kg/wk	0.053	0.351	0.0531	0.355

Note that final loading rate calculated from median of cycle 86 to 102, which are the final 5 cycles that metals were analyzed for.

3.3.2.2 Scaling Factors

A series of physical scaling factors are applied to adjust loading rates to account for differences between lab-based kinetic tests and tailings in the field. These include but are not limited to hydrogeological pathways, temperature, and grain size distribution. Values and rationale for these correction factors are described below. A summary of the scaling factors used to predict the NAG waste rock source terms are presented in Table 3-18.

Table 3-18:
Summary of scaling factors used to predict source term concentrations

		Operations		Closure	
Scaling Factors	**Unit**	**Base Case**	**Upper Case**	**Base Case**	**Upper Case**
Contact Water Factor	-	0.25	0.50	0.15	0.075
Grain Size Correction	-	0.1	0.2	0.1	0.2
Temperature Correction	-	0.3	0.3	0.3	0.3

Humidity cells are designed to achieve complete flushing of mineral surfaces, ensured by the small sample size (1 kg) and high flushing rate (0.5 L/week). The high flushing rate ensures nearly all available of mineral surfaces in humidity cells are rinsed. In the field, it is assumed that between 25% and 50% of waste rock surfaces are flushed. This range in contact water scaling factors is used for the upper case and base case-operations source term predictions. At closure surplus water from the NPAG facility will be reduced from 90% MAP to approximately 25% MAP. This will result in a reduction in the percentage of NPAG surfaces which are rinsed by infiltrating water. The contact water correction factor is assumed to reduce to 0.15 and 0.075 for upper case and base case-closure scenarios, respectively.

Drill core samples are grinded to a relatively fine grain size (< ¼“) for humidity cell testing, hence, material used in humidity cells have a greater surface area to mass ratio than is expected for waste rock. Kinetic reaction rates are partly a function of exposed surface area and thus, grain size. As a conservative approximation, it is assumed that 10% to 20% of the mass of the ore pile has a grain size of kinetic test material (< ¼“) and that all metal leaching is associated with this grain size fraction. This range of grain size correction factors is applied to upper case and base case the RSF source terms, respectively.

A temperature correction is applied to Kinetic test loading rates to reflect the lower temperatures that are present at the Silvertip site compared to laboratory temperatures. In order to derive an appropriate temperature correction weathering rates are scaled by an Arrhenius relationship, calculations for which are presented in Appendix C. As a conservative assumption, it is assumed that NAG waste rock weathers at the average summer temperature of 7.0°C year round, and potential reductions in metal leaching rates due to freezing of pore water is not considered. Applying the scaling factors presented in Table 3-18 to the loading rates presented in Table 3-17 results in the scaled loading rates shown in Table 3-19.

Table 3-19:
Scaled NPAG waste rock loading rates

		Operations		Closure	
	unit	**Base Case**	**Upper Case**	**Base Case**	**Upper Case**
SO_4	mg/kg/year	24.3	97.2	3.35	13.4
As	mg/kg/year	0.0000875	0.00035	0.00000612	0.0000245
Ca	mg/kg/year	8.51	34	1	4.01
Cd	mg/kg/year	0.000116	0.000464	2.01E-05	8.05E-05
Cu	mg/kg/year	0.000612	0.00245	0.000191	0.000762
Mg	mg/kg/year	1.88	7.5	0.387	1.55
Ni	mg/kg/year	0.00024	0.00096	2.11E-05	8.43E-05
Pb	mg/kg/year	0.000215	0.00086	1.77E-05	7.07E-05
Sb	mg/kg/year	0.00118	0.00471	0.000142	0.000568
Se	mg/kg/year	0.000144	0.000577	1.26E-05	5.03E-05
U	mg/kg/year	0.000718	0.00287	8.58E-05	0.000343
Zn	mg/kg/year	0.138	0.554	0.00622	0.0249

3.3.2.3 Predicted Concentration and Secondary Mineral Controls

In order to identify potential solubility controls, and place predicted loading rates in the context of water quality objectives, the scaled loading rates presented in Table 3-19 are converted into a concentration. Concentrations are estimated by dividing the annual loading produced by the RSF at its maximum mass by the volume of water that will interact with the waste rock on an annual basis. During operations it is assumed that both infiltration and runoff will interact with waste rock. The volume of water is estimated by assuming RSF surplus water is a combination of infiltration and runoff during operations. Upon mine closure a soil cover will be constructed reducing infiltration and preventing surface runoff from interacting with waste rock, reducing the surplus water in contact with waste rock surfaces to 25% MAP. Parameters used to estimate the volume of water used to predict concentrations are presented in Table 3-20.

Table 3-20:
Values used to convert RSF loading rate into concentration

Scaling Factor	**value**	**unit**
Waste rock mass	520,000	tonnes
footprint	4	ha
Surplus water (operations)	808	mm
Surplus water (closure)	224	mm

Note: Contact water assumed to be 90% MAP for operations and 25% MAP for closure.

The application of solubility controls requires an estimate of pH and alkalinity. For base case scenarios, the RSF is assigned the average pH and alkalinity observed in McDame limestone SFE tests, while the upper case source term is assigned the minimum pH and alkalinity observed in SFE testwork. The resulting solution did not result in supersaturation of any metal hydroxides or carbonates which could be expected to control metal concentrations. Therefore no solubility controls are considered for the RSF source term. As such, the loading from NAG waste rock stored on the surface will be directly proportional to the mass of NAG waste rock.

Predicted concentrations for the RSF during operations and closure are presented in Table 3-21. Concentrations presented in this table represent peak concentrations that will occur once the NPAG facility reaches its maximum mass (520,000 tonnes) assuming diluted into expected infiltration and runoff.

Table 3-21:
Predicted RSF source term concentrations

Parameter	**Unit**	**RSF-Operations**		**RSF-Closure**	
		Base Case	**Upper Case**	**Base Case**	**Upper Case**
pH	s.u.	7.7	7.36	7.7	7.36
T-Alkalinity	mgCaCO$_3$/L	48.2	36.5	48.2	36.5
SO$_4$	mg/L	438	1750	218	871
F	mg/L	0.00278	0.0111	0.00111	0.00445
Cl	mg/L	1.75	6.99	0.699	2.79
Ag	mg/L	0.000116	0.000465	0.000145	0.00058
Al	mg/L	0.0701	0.28	0.0293	0.117
As	mg/L	0.00158	0.00631	0.000397	0.00159
Ba	mg/L	0.0679	0.272	0.059	0.236
Ca	mg/L	153	614	65	260
Cd	mg/L	0.00898	0.0359	0.000641	0.00257
Co	mg/L	0.00396	0.0158	0.000454	0.00182
Cr	mg/L	0.00209	0.00836	0.00131	0.00523
Cu	mg/L	0.011	0.0442	0.0124	0.0495
Fe	mg/L	0.0124	0.0498	0.00794	0.0318
Hg	mg/L	0.0000272	0.000109	0.0000174	6.94E-05
K	mg/L	2.59	10.4	0.279	1.12
Mg	mg/L	33.8	135	25.1	101
Mn	mg/L	0.598	2.39	0.00816	0.0326
Mo	mg/L	0.243	0.974	0.112	0.446
Na	mg/L	1.79	7.15	0.168	0.674
Ni	mg/L	0.00433	0.0173	0.00137	0.00548
Pb	mg/L	0.00388	0.0155	0.00115	0.00459
Sb	mg/L	0.0212	0.0849	0.00922	0.0369
Se	mg/L	0.0026	0.0104	0.000817	0.00327
Si	mg/L	1.48	5.91	0.569	2.27
Sr	mg/L	0.157	0.629	0.137	0.546
Tl	mg/L	0.00126	0.00503	0.000343	0.00137
U	mg/L	0.0129	0.0518	0.00557	0.0223
Zn	mg/L	2.5	9.99	0.404	1.61

3.3.3 PAG Stockpile

Approximately 38,000 tonnes of potentially acid generating Earn group waste rock will produced over mine life. Approximately 7,700 tonnes PAG rock will be returned to the underground as cemented coarse rock fill early in the mine life. The remaining PAG rock (approximately 30,000 tonnes) will be temporarily stored in a surface stockpile until the end of mine life at which point it will be disposed of into the underground mine workings. The PAG rock stockpile will exist on the mine surface for approximately 20 years. The lag time for ARD development from PAG waste rock at exceeds the mine life (19 years), therefore, ARD is not expected to develop before the PAG stockpile is backfilled in the underground mine workings and saturated at the end of mine life (Appendix C).

3.3.3.1 Data Sources

Metal leaching rates for the PAG stockpile are estimated from two Lower Earn group humidity cells (HC1 and HC2). The 90th percentile rates of HC1 and HC2 are used to calculate the PAG stockpile source term (Table 3-17). Note that this facility only exists during mine operations, so a closure source term is not developed.

3.3.3.2 Scaling Factors

The same scaling factors presented in Table 3-18 for the RSF operations source term are applied to the PAG stockpile. Applying the scaling factors presented in Table 3-18 to the 90th percentile loading rates for HC1 and HC2 shown in Table 3-17 results in the scaled loading rates shown in Table 3-22. Note that assumptions regarding the contact water and grain size correction factor are used to differentiate the upper case from the base case source term.

Table 3-22:
Scaled loading rates used to calculate PAG stockpile source term

Parameter	unit	Base Case	Upper Case
SO_4	mg/kg/year	56.6	226
As	mg/kg/year	0.000253	0.00101
Ca	mg/kg/year	18.3	73.2
Cd	mg/kg/year	0.00591	0.0236
Cu	mg/kg/year	0.000803	0.00321
Mg	mg/kg/year	4.28	17.1
Ni	mg/kg/year	0.00317	0.0127
Pb	mg/kg/year	0.000077	0.000308
Sb	mg/kg/year	0.00121	0.00483
Se	mg/kg/year	0.000429	0.00172
U	mg/kg/year	0.000312	0.00125
Zn	mg/kg/year	0.137	0.548

3.3.3.3 Predicted Concentration and Secondary Solubility Controls

In order to identify potential solubility controls, scaled loading rates presented in Table 3-20 are converted into a concentration. Concentrations are estimated by multiplying scaled loading rates presented in Table 3-22 by the maximum mass of the PAG stockpile that may be reached at the end of mine life (Table 3-23). The resulting total load is then divided by the volume of surplus water to produce a concentration estimate.

Table 3-23:
Values used to convert PAG stockpile loading rate into concentration

Scaling Factor	value	unit
Waste rock mass	30,400	tonnes
Footprint	740	m^2
Surplus water (operations)	808	mm

Note: Contact water assumed to be 90% MAP for operations.

In order to apply solubility controls to the predicted metal concentrations an estimate of pH and alkalinity is required. For base case scenarios, the PAG storage facility is assigned the median pH and alkalinity observed in the two Lower Earn Group humidity cells used to predict metal leaching rates (HC2 and HC4). For upper case predictions, the 10th percentile alkalinity and pH observed in these two humidity cells is applied. The resulting solution is speciated in PHREEQC using the llnl.dat database to check for potential secondary solubility controls (Parkhurst and Apello, 1999). No potential solubility controls were identified and were therefore not considered in the base case source term. For the upper case scenario, gypsum was supersaturated in the upscaled solution and a solubility control was applied (Table 3-24). Note that the Mn-oxide mineral manganite was found to be considerably supersaturated in HC4 leachate, with concentrations greater than 5 ppm at a pH greater than 7, similar to results for the desulphurized tailings humidity cell DeS HC3 as described in section 3.3.1.4. The PHREEQC input file is provided in Appendix D. The predicted solution with this solubility control is presented in Table 3-25.

Table 3-24:
Secondary mineral controls applied to TMF source term

		Operations	
Mineral Phase	Chemical Formula	Base Case	Upper Case
Gypsum	$CaSO_4{:}2H_2O$		x
ZnCO3:H2O	$ZnCO_3{:}H_2O$		
Cerusite	$PbCO_3$		
Otavite	$CdCO_3$		

Table 3-25:
Predicted PAG stockpile concentrations

Parameter	Unit	Base Case	Upper Case
pH	s.u.	7.44	7.31
T-Alkalinity	mg$CaCO_3$/L	22.8	17.6
SO_4	mg/L	489	1850
F	mg/L	0.779	3.12
Cl	mg/L	0.838	3.35
Ag	mg/L	0.0000163	0.0000653
Al	mg/L	0.0493	0.197
As	mg/L	0.00219	0.00875
Ba	mg/L	0.0191	0.0765
Ca	mg/L	158	587
Cd	mg/L	0.0511	0.204
Co	mg/L	0.0144	0.0577
Cr	mg/L	0.001	0.00401
Cu	mg/L	0.00694	0.0278
Fe	mg/L	0.0103	0.0413
Hg	mg/L	0.0000131	0.0000522
K	mg/L	2.48	9.91
Mg	mg/L	37	148
Mn	mg/L	1.2	4.8
Mo	mg/L	0.0472	0.189
Na	mg/L	1.09	4.37
Ni	mg/L	0.0274	0.11
Pb	mg/L	0.000665	0.00266
Sb	mg/L	0.0104	0.0418
Se	mg/L	0.00371	0.0148
Si	mg/L	1.0	4.01
Sr	mg/L	0.221	0.883
Tl	mg/L	0.000366	0.00147
U	mg/L	0.0027	0.0108
Zn	mg/L	1.18	4.74

3.3.4 Ore Stockpile

Ore produced during mine life will either be fed directly into the crusher or temporarily stored on a stockpile adjacent to the crusher. The temporary stockpile will only hold ore during months of active mining (*i.e.*, May to October). The ore stockpile will reach a peak mass of 31,000 tonnes in the first year of mine operations, in subsequent years the stockpile is not expected to hold more than 12,000 tonnes at any given time. The ore stockpile area is sized to contain a maximum of 50,000 tonnes in the event that there is an interruption in the milling schedule. Note that ore is classified as PAG material, however, acid generation is not expected during mine life as described in Appendix C.

3.3.4.1 Data Sources

The ore stockpile source term is primarily derived from the silver creek ore sample HC1 which was ran for 108 cycles. Results for the 90th percentile are presented in Table 3-26. In addition to this humidity cell, a field pad was established in 1998 using ore from the stockpile established in the 1985 during underground exploration. The ore field pad results are of limited value as leachate volumes have not been historically recorded and the ore was at a relatively advanced weathering state when the field pad was established.

Table 3-26:
Humidity cell loading rates for ore humidity cell HC1
The 90th percentile loading rates are used in source term derivation

	unit	90th Percentile
SO_4	mg/kg/week	71.2
As	mg/kg/week	0.000799
Ca	mg/kg/week	33.9
Cd	mg/kg/week	0.000304
Cu	mg/kg/week	0.00724
Mg	mg/kg/week	0.211
Ni	mg/kg/week	0.00182
Pb	mg/kg/week	0.042
Sb	mg/kg/week	0.0025
Se	mg/kg/week	0.0006
U	mg/kg/week	0.00003
Zn	mg/kg/week	0.555

3.3.4.2 Scaling Factors

The same scaling factors assumed for the PAG stockpile and RSF are applied to the Ore Stockpile (Table 3-18). Note that assumptions regarding the contact water and grain size correction factor are used to differentiate upper case and base case source terms. Applying the scaling factors presented in Table 3-18 to the 90th percentile loading rates presented in Table 3-26, results in the scaled loading rates shown in Table 3-27.

Table 3-27:
Scaled loading rates used to calculate ore stockpile source term

	unit	Base Case	Upper Case
SO_4	mg/kg/year	27.8	111
As	mg/kg/year	0.000312	0.000623
Ca	mg/kg/year	13.2	26.5
Cd	mg/kg/year	0.000119	0.000237
Cu	mg/kg/year	0.00282	0.00565
Mg	mg/kg/year	0.0824	0.165
Ni	mg/kg/year	0.000709	0.00142
Pb	mg/kg/year	0.0164	0.0328
Sb	mg/kg/year	0.000976	0.00195
Se	mg/kg/year	0.000234	0.000468
U	mg/kg/year	0.0000117	0.0000234
Zn	mg/kg/year	0.216	0.433

3.3.4.3 Predicted Concentrations and Secondary Solubility Controls

In order to identify potential solubility controls, loading rates must be converted into concentrations. To do this, the loading rates presented in Table 3-27 are multiplied by the maximum mass that can be stored on the ore stockpile pad and divided into the volume of contact water that will interact with the ore stockpile on an annual basis Table 3-28). Note that the stockpile will only exist for the months of May to October. It is assumed that 90% of precipitation during this time period reports as surplus water from the stockpile (infiltration + runoff).

Table 3-28:
Values used to convert Ore Stockpile loading rate into concentration

Scaling Factor	value	unit
Maximum capacity	50,000	tonnes
footprint	4500	m^2
Contact water (operations)	270	mm

Note: Contact water assumed to be 90% Mean precipitation during months that stockpile exists (May to October).

The application of solubility controls requires an estimate of pH and alkalinity. For base case scenarios, the up-scaled solution is assigned the average pH and alkalinity observed in the HC1 ore humidity cell, while for the upper case the upscaled solution is assigned the 10th percentile pH and alkalinity observed HC1. The resulting solution is speciated in PHREEQC using the llnl.dat database to identify secondary mineral solubility constraints. The only potential solubility constraints identified is cerusite in both the upper case and base case scenarios (Table 3-29). The predicted solution with this solubility control is presented in Table 3-30.

Table 3-29:
Secondary mineral controls applied to Ore Stockpile source term

		Operations	
Mineral Phase	**Chemical Formula**	**Base Case**	**Upper Case**
Gypsum	$CaSO_4{:}2H_2O$		
ZnCO3:H2O	$ZnCO_3{:}H_2O$		
Cerusite	$PbCO_3$	x	x
Otavite	$CdCO_3$		

Table 3-30:
Predicted Ore Stockpile source term concentrations

Parameter	unit	Base Case	Upper Case
pH	s.u.	7.12	6.93
T-Alkalinity	$mgCaCO_3/L$	17.5	14.4
SO_4	mg/L	403	1620
F	mg/L	0.642	2.57
Cl	mg/L	0.542	1.08
Ag	mg/L	0.000105	0.00021
Al	mg/L	0.0894	0.179
As	mg/L	0.00453	0.00905
Ba	mg/L	0.0146	0.0291
Ca	mg/L	192	384
Cd	mg/L	0.0227	0.0453
Co	mg/L	0.00917	0.0183
Cr	mg/L	0.00172	0.00345
Cu	mg/L	0.041	0.082
Fe	mg/L	0.0211	0.0422
Hg	mg/L	0.0000477	0.0000953
K	mg/L	0.162	0.323
Mg	mg/L	1.2	2.39
Mn	mg/L	1.39	2.77
Mo	mg/L	0.0125	0.0251
Na	mg/L	0.512	1.02
Ni	mg/L	0.0103	0.0206
Pb	mg/L	0.0449	0.104
Sb	mg/L	0.0142	0.0284
Se	mg/L	0.0034	0.00679
Si	mg/L	1.13	2.26
Sr	mg/L	0.0371	0.0743
Tl	mg/L	0.00125	0.0025
U	mg/L	0.00017	0.00034
Zn	mg/L	3.14	6.28

3.3.5 Paste Feed Stockpile

Desulphurized tailings and pyrite concentrate will be stored in a temporary stockpile which feeds the paste plant. This Paste Feed Stockpile (PSF) will reach a peak mass of 8,000 tonnes and consist of a blend of approximately 40% desulpherized tailings and 60% pyrite tailings. The stockpile will be stored on the mine surface adjacent to the paste feed plant while active mining is taking place (May to October). During inactive months, any material remaining in the Paste Feed Stockpile will be moved underground to minimize interaction with meteoric water.

3.3.5.1 Data Sources

The paste feed stockpile consists of a mxi of desuphurized tailings and pyrite tailings. Data is available for two desulphurized tailings humidity cells. Unfortunately, no kinetic testwork has been initiated on pyrite concentrate. However, kinetic testwork has been initiated on Zn rougher tailings (Zn Ro Tails). These are the tailings produced by the Zn Rougher floation process. Zn Ro tails are essentially compose the feed to the pyrite flotation circuit along with other ore processing by products (Zn cleaner tailings, Pb rougher tailings and Pb cleaner tailings). The Zn Ro tailings are considered the best proxy for the PFS as the PFS receives both end products of the Zn Ro tailings after pyrite flotation.

The two relatively short term humidity cells (20 cycles) were conducted on Zn Ro tailings produced during metallurgical testwork. The median loading rate of the two humidity cells is used to predict the PSF source term Table 3-31. Note that other operations source terms developed from humidity cell loading rates have used 90th percentile values. In this case, median values are used due to the short term nature of the two Zn Ro tailings humidity cell tests.

Table 3-31:
Median loading rate of Zn Ro tailings humidity cells

Parameter	unit	Median Loading Rate
SO_4	mg/kg/week	327
As	mg/kg/week	0.000685
Ca	mg/kg/week	142
Cd	mg/kg/week	0.00396
Cu	mg/kg/week	0.00367
Mg	mg/kg/week	1.06
Ni	mg/kg/week	0.00843
Pb	mg/kg/week	0.00513
Sb	mg/kg/week	0.0032
Se	mg/kg/week	0.000733
U	mg/kg/week	0.000287
Zn	mg/kg/week	1.08

3.3.5.2 Scaling Factors

The same scaling factors described for the TMF are applied to the PFS (Table 3-11). Note that assumptions regarding the contact water correction factor are used to differentiate upper case and base case source terms. These scaling factors are applied to the loading rates presented in Table 3-31 to produce the scaled loading rates presented in Table 3-32.

Table 3-32:
Scaled loading rates used for Paste Feed Stockpile source term development

Parameter	unit	Base Case	Upper Case
SO_4	mg/kg/year	1270	2550
As	mg/kg/year	0.00267	0.00534
Ca	mg/kg/year	553	1110
Cd	mg/kg/year	0.0155	0.0309
Cu	mg/kg/year	0.0143	0.0286
Mg	mg/kg/year	4.12	8.24
Ni	mg/kg/year	0.0329	0.0657
Pb	mg/kg/year	0.02	0.04
Sb	mg/kg/year	0.0125	0.025
Se	mg/kg/year	0.00286	0.00572
U	mg/kg/year	0.00112	0.00224
Zn	mg/kg/year	4.2	8.4

3.3.5.3 Cyanide Species

Cyanide is used in the milling process and there is potential for cyanide release from the PSF. Cyanide release will likely be similar to that of DeS tailings in the TMF during mine operations, and the same concentrations developed for TMF are assigned to PSF drainage.

3.3.5.4 Predicted Concentrations and Secondary Solubility Controls

In order to identify potential solubility controls the loading rates are converted into concentrations. The maximum annual concentration that can be reached in a year is calculated by multiplying the scaled loading rate by the maximum PFS mass and dividing it by the volume of surplus water (Table 3-33). In order to identify potential mineral solubility controls estimates of pH and alkalinity are required. For the base case scenarios, the up-scaled solution is assigned the average pH and alkalinity observed in Zn Ro-HC1 and Zn Ro-HC2, while for the upper case the up-scaled solution is assigned the 10th percentile pH and alkalinity observed in these humidity cells.

Table 3-33:
Values used to convert loading rate into concentration for Paste Feed Facility

Scaling Factor	value	unit
Paste Feed Stockpile	8,000	tonnes
Footprint	2,230	m^2
Contact Water (operations)	343	mm

Note: Paste Feed Stockpile only present on mine surface for May to October. During this time period 90% of precipitation is assumed to report as drainage (contact water).

The predicted solution chemistry is speciated in PHREEQC using the llnl.dat database to identify potential solubility constraints. The secondary minerals otavite, cerusite, gypsum and $ZnCO_3$:H_2O were found to be supersaturated and allowed to precipitate to equilibrium as outlined in Table 3-34 . The PHREEQC input files are presented in Appendix D. Predicted concentrations for the PFS are presented in Table 3-35. A number of these concentrations represent thermodynamic solubility controls, as outlined in Table 3-34 while others represent concentrations predicted by up-scaling kinetic tests. Predicted concentrations of nitrogen species in the PSF drainage are discussed in Appendix B.

Table 3-34:
Secondary mineral controls applied to Paste Feed Stockpile source term

		Operations	
Mineral Phase	**Chemical Formula**	**Base Case**	**Upper Case**
Gypsum	$CaSO_4$:$2H_2O$	x	x
ZnCO3:H2O	ZnCO3:H2O	x	x
Cerusite	$PbCO_3$	x	x
Otavite	$CdCO_3$	x	x

Table 3-35:
Predicted source term concentration for Paste Feed Stockpile

Parameter	Unit	Base Case	Upper Case
pH	s.u.	7.61	7.49
T-Alkalinity	mgCaCO$_3$/L	45	55.8
N-NO$_3$	mg/L	0.6	2.3
N-NO$_2$	mg/L	0.3	0.6
N-NH$_3$	mg/L	2.2	2.6
WAD-CN	mg/L	0.01	0.476
T-CN	mg/L	0.18	0.522
SO$_4$	mg/L	1610	1780
F	mg/L	2.82	5.63
Cl	mg/L	1.63	3.27
Ag	mg/L	0.0018	0.00361
Al	mg/L	0.0279	0.0557
As	mg/L	0.155	0.31
Ba	mg/L	0.429	0.858
Ca	mg/L	594	580
Cd	mg/L	0.154	0.273
Co	mg/L	0.0468	0.0937
Cr	mg/L	0.0189	0.0378
Cu	mg/L	0.149	0.298
Fe	mg/L	0.0973	0.195
Hg	mg/L	0.00964	0.0193
K	mg/L	8.67	17.3
Mg	mg/L	40.8	81.6
Mn	mg/L	11.7	23.4
Mo	mg/L	0.0344	0.0688
Na	mg/L	9.22	18.4
Ni	mg/L	0.343	0.686
Pb	mg/L	0.0225	0.028
Sb	mg/L	0.13	0.26
Se	mg/L	0.0298	0.0596
Si	mg/L	46.9	93.8
Sr	mg/L	2.5	5
Tl	mg/L	0.0107	0.0215
U	mg/L	0.0117	0.0234
Zn	mg/L	9.48	16.7

3.3.6 Mine Facilities Area

Mine facilities will be hosted on a pad constructed with NPAG waste rock and borrow material. An existing pad is located near the mine portal, where existing mine facilities are located. This pad will be expanded using NPAG waste rock and local borrow material. Waste rock and borrow material used to construct the mine facilities area (MFA) is assumed to have an ML potential similar to other NPAG waste rock at the minesite. Due to the potential for metal leaching from this construction material, runoff from the MFA will be captured and directed to the HDS treatment plant. In this section, a surface runoff source term for the MFA is developed. Upon mine closure, the mine facilities will be removed and a soil cover will be placed over the MFA footprint. Therefore, a source term is only developed for the operations scenario.

3.3.6.1 Data Sources

It is assumed that the ML potential of the MFA is similar to other NPAG waste rock at site. Therefore the same mass loading rate developed for the NPAG waste rock facility is applied to rock used to construct the MFA (Table 3-17).

3.3.6.2 Scaling Factors

The same scaling factors regarding grain size, contact water and temperature effects on sulphide oxidation rates developed for the NPAG waste rock facility are applied to the MFA. In addition to these scaling factors, it is assumed that oxidation in only the top 0.5 m of the MFA pad will contribute to metal loading in surface runoff. Considering the relatively thin oxidative thickness, it is assumed that this surface layer is entirely frozen during months when the average monthly temperature is less than 0°C (approx. 50% of the year). A summary of scaling factors used to produce the MFA source term are provided in Table 3-36. Applying these scaling factors to the operations-NPAG loading rates developed in Table 3-17 results in the per unit area loading rates presented in Table 3-37.

Table 3-36:
Mine Facilities Area scaling factors used in source term development

		Operations	
Scaling Factors	Unit	Base Case	Upper Case
Contact Water Factor	-	0.25	0.50
Grain Size Correction	-	0.1	0.2
Thickness	m	0.5	0.5
Percent of year frozen		50%	50%
Temperature Correction	-	0.3	0.3

Table 3-37:
Scaled loading rates used to calculate the Mine Facilities Area source term

Parameter	unit	Base Case	Upper Case
SO_4	mg/m^2/year	28300	113000
As	mg/m^2/year	0.127	0.506
Ca	mg/m^2/year	9150	36600
Cd	mg/m^2/year	2.95	11.8
Cu	mg/m^2/year	0.401	1.61
Mg	mg/m^2/year	2140	8570
Ni	mg/m^2/year	1.58	6.33
Pb	mg/m^2/year	0.0385	0.154
Sb	mg/m^2/year	0.604	2.42
Se	mg/m^2/year	0.215	0.858
U	mg/m^2/year	0.156	0.624
Zn	mg/m^2/year	68.5	274

3.3.6.3 Predicted Concentrations and Solubility Controls

Loading rates developed in the previous section must be converted into concentrations in order to identify potential solubility controls. The MFA loading rates are converted into a concentration by assuming that 90% of MAP reports as surplus water (*e.g.*, 808 mm) from the MFA. The same assumptions regarding pH and alkalinity applied to the NPAG RSF are applied to the MFA source term, as described in Section 3.3.2.3. The resulting solution was speciated in PHREQC using the llnl.dat database. No potential solubility controls were identified. The predicted MFA source term concentrations are provided in Table 3-38.

Table 3-38:
Predicted source term concentrations for the Mine Facilities Area.

Parameter	Unit	Base Case	Upper Case
pH	s.u.	7.47	7.31
T-Alkalinity	mg$CaCO_3$/L	48.2	36.5
SO_4	mg/L	35	140
F	mg/L	0.0000955	0.000382
Cl	mg/L	0.06	0.24
Ag	mg/L	0.00000117	0.00000467
Al	mg/L	0.00353	0.0141
As	mg/L	0.000157	0.000626
Ba	mg/L	0.00137	0.00548
Ca	mg/L	11.3	45.3
Cd	mg/L	0.00366	0.0146
Co	mg/L	0.00103	0.00413
Cr	mg/L	0.0000717	0.000287
Cu	mg/L	0.000497	0.00199
Fe	mg/L	0.000739	0.00296
Hg	mg/L	0.000000934	0.00000374
K	mg/L	0.177	0.709
Mg	mg/L	2.65	10.6
Mn	mg/L	0.0859	0.344
Mo	mg/L	0.00337	0.0135
Na	mg/L	0.0781	0.312
Ni	mg/L	0.00196	0.00784
Pb	mg/L	0.0000476	0.00019
Sb	mg/L	0.000748	0.00299
Se	mg/L	0.000265	0.00106
Si	mg/L	0.0718	0.287
Sr	mg/L	0.0158	0.0632
Tl	mg/L	0.0000262	0.000105
U	mg/L	0.000193	0.000772
Zn	mg/L	0.0847	0.339

3.3.7 Portal Drainage Source Term

The portal drainage source term represents the cumulative metal loading from wall rock, ground water recharge and backfill stored in the underground mine workings. Metal loading rates are developed for each of these components to estimate metal loading from the mine adit. Considering the complexity of the underground source term, no attempt is made to scale loading rates for various stages of mine life. Rather, source terms are only developed for the end of operations when peak loadings and concentrations are expected, and for the closure scenario when the water table has rebounded to the portal elevation.

Water quality data is available for an existing mine portal at Silvertip. This water quality data along with historical records of flow rates and dewatering events are used to estimate metal loading rates from wall rock. Kinetic testwork is used to estimate loading rates from backfill material.

3.3.7.1 Wall Rock Source Term

In this section the wall rock source term is developed. Data sources for this source term consist of site water quality data and historical dewatering records. As such, the available data at WQ9 and past dewatering events are discussed. This data is then used to produce scaled wall rock loading rates normalized to per unit area of wall rock exposures.

Excavation and Dewatering Events

The Silvertip property has a long history of mineral exploration dating back to the 1950's. Most underground excavation on the Silvertip property took place from October 1984 to May 1985. The mine workings remained dewatered until November, 1985. This underground exploration consisted of 696 m of large headings (4.56 m x 4.27 m) and 906 m of small headings (3.66 m x 3.66 m). A groundwater inflow rate of 17 L/s was reported during initial decline development. Dewatering rates stabilized in March, 1985 at approximately 9 L/s and declined to 7 L/s during the final month of excavation (May, 1985). Note that dewatering continued until November, 1985, however, no flow rate data is available for this period. Available monthly records of flow rates for these dewatering events are presented in Table 3-39. Concentrations of D-Cd, D-Cu, D-Fe,
D-Pb, D-Zn pH, SO_4 and NO_2 were monitored on roughly a monthly basis from March 12th, 1985 until October 13th, 1985.

Table 3-39:
Average monthly pumping rates reported during the 1984 excavation and dewatering event.

Month	Year	Development (m)	Pumping Rate (L/s)
November	1984	188	17
December	1984	118	16
January	1985	210	10
February	1985	409	9
March	1985	335	9
April	1985	218	9
May	1985	87	7

Additional underground excavation occurred in 1990, when Strathcona Resources excavated an additional 765 m of tunnels towards the Discovery Zone ore body. There are no available dewatering records for this exploration event. No drilling or tunnelling was reported to have occurred in 1991, therefore it is assumed that this dewatering event was limited to the year of 1990.

In 1997, the Silvertip property was acquired by Imperial Metals Corporation. In 1999, the mine workings were again dewatered to allow additional exploration drilling. Dewatering took place from October 25th, 1999 to February 7th, 2000. During this event, peak dewatering rates were up to 109 L/s initially and then declined to 13 L/s by the beginning of January, 2000. The flow rate remained between 13 L/s and 8 L/s for the final two months of dewatering (January and February, 2010). For complete records of flow rates during this time period see Appendix 9-III-A of Klohn, (2014). During this time period dissolved metals and pH were monitored on roughly a weekly basis. A full suite of metals were analyzed during this time period, however, detection limits were relatively high (*e.g.*, 0.2 mg/L for As, Sb and Se) and anions such as SO_4 were not monitored.

WQ9 Water Quality Data

Historical adit (WQ9) water quality for select parameters is presented in Figure 3-5. For complete results see Klohn (2013). Water quality during monitoring at WQ9 can provide an indication of the geochemical response to mine dewatering, excavation, and subsequent flooding.

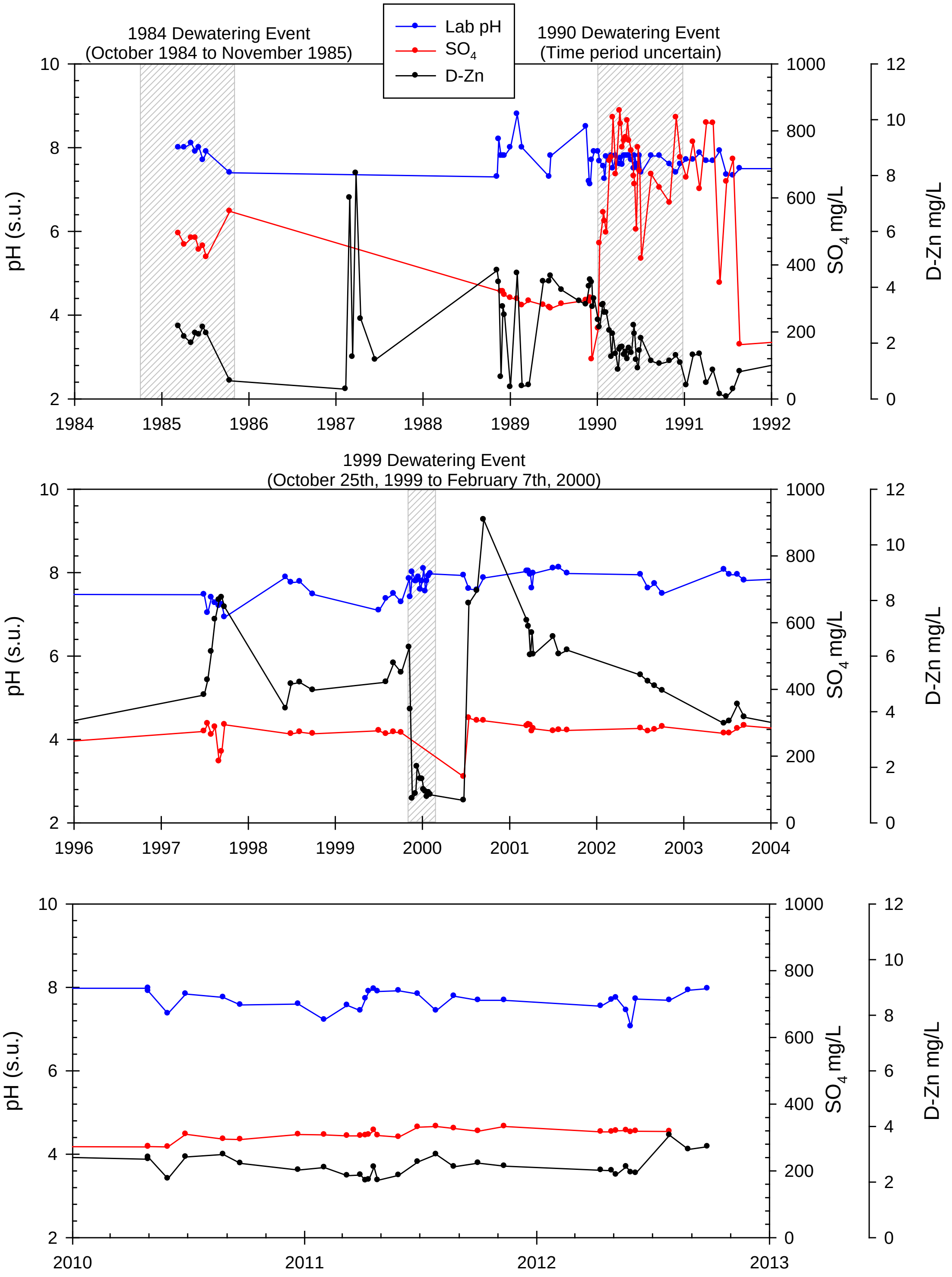


Figure 3-5: **Time series of pH, SO_4 and D-Zn at WQ9 (mine adit). Shaded time periods denote historical dewatering events. Note that no water quality samples were collected during the ommitted time periods (1992-1996 and 2004-2010).**

The three historic dewatering events as described in the previous section are highlited in Figure 3-5. The first dewatering event began in October 1984 when the initial adit excavation took place. Water quality monitoring began in March of the following year. Water quality results during this time period show Zn concentrations generally between 2 mg/L and 2.7 mg/L, while SO_4 ranges were generally between 420 mg/L to 490 mg/L. The last measurement during this dewatering event occurred in October 13th, 1985. This final water quality sample shows a rise in SO_4 concentrations to 560 mg/L and a sharp decline in Zn concentrations to 0.6 mg/L compared to earlier times of the dewatering event. Dewatering ceased in November, 1985, and was not resumed until 1990 and the mine workings were allowed to flood. During this time period, Zinc concentrations were highly variable (0.4 mg/L to 8 mg/L), but were generally higher than during the 1984 dewatering event. Conversely, SO_4 concentrations generally lower than during the 1984 dewatering event, ranging from 290 mg/L to 320 mg/L.

Dewatering of the mine workings in 1990 lead to a rise in SO_4 concentrations and a decline in Zn concentrations as illustrated in Figure 3-5. Unfortunately the exact dates of the 1990 dewatering event are not certain, however considering that 700 m of tunnels were excavated the dewatering event likely lasted for several months. No drilling or excavation are reported for 1991, therefore it is assumed that dewatering had ceased by the end of 1990. Sulphate concentrations remained elevated (820 mg/L to 340 mg/L) and Zn concentrations remain relatively low (<1.7 mg/L) throughout 1991 when dewatering is presumed to have ceased. Water quality was not regularly monitored again until 1997, at which point Zn concentrations had increased and SO_4 had declined to the range observed prior to the 1990 dewatering event.

A third dewatering event occurred in 1999. Dewatering in October 25th, 1999 coincided with a sudden decline in Zn concentrations from 6.3 mg/L to 0.9 mg/L. Zinc concentrations remained below 2 mg/L throughout this dewatering event. Unfortunately SO_4 concentrations were not monitored during this time period. Approximately 7 months after dewatering is ceased, Zn concentrations rebounded to the maximum value observed at WQ9 of 10.9 mg/L. Zinc concentrations gradually declined from this point until September, 2003 when regular water quality monitoring is temporarily stopped (Figure 3-5).

Regular water quality monitoring of the portal was re-established in 2010. Since 2010 concentrations have remained relatively constant, with Zn ranging from 2.1 mg/L to 3.7 mg/L and SO_4 ranging from 272 to 310 mg/L. Flow rates at WQ9 have been regularly monitored over this time period. Flow measurements range from 0.7 L/s to 5.4 L/s with no apparent relationship to drainage chemistry (Table 3-40).

Table 3-40:
Adit (WQ9) flow rates in 2010 and 2011

Date	L/s
30-Apr-10	1.5
31-May-10	3.9
28-Jun-10	5.4
22-Dec-10	2.6
1-Feb-11	2.5
9-Mar-11	1.7
30-Mar-11	1.9
26-Apr-11	0.7
27-Jul-11	4.9
24-Aug-11	3.3
1-Oct-11	3.8
11-Nov-11	3.1

Laboratory pH measurements of WQ9 adit water have remained neutral to alkaline in all water quality samples, ranging from pH 7.0 to pH 8.8, with no clear trend in pH values between 1984 and 2012 (Figure 3-5). This may, in part, be due to relics of laboratory pH measurements. The pH of a sample can evolve during the holding time between sample collection and measurement in a laboratory. Water quality samples from closed systems (cutoff from atmosphere) are particularly sensitive to this storage time, as ingress of O_2 gas drives Fe and Mn oxidation and precipitation lowering the pH, while outgassing of CO_2 creates OH^- ions increasing the pH. Iron oxide staining is observed at the outlet of WQ9, and therefore it is assumed that some degree of Fe oxidation and precipitation is occurring. A comparison of laboratory pH values which have been regularly monitored with available field pH measurements is presented in Table 3-41.

This table shows that field pH values are generally lower than laboratory pH measurements, indicating that pH values presented in Figure 3-5 are likely an overestimate of the actual drainage pH. This also indicates that the process of CO_2 degassing is capable of driving up the pH, even as Fe oxidation and precipitation is occurring which consumes OH^- ions. Note that no field pH measurements were collected during time periods when the mine workings were dewatered.

Table 3-41:
Comparison of field and laboratory pH measurements of adit water (WQ9)

Date	Field-pH	Lab-pH
4-Jun-97	7.4	-
14-Jun-97	7.0	7.19
30-Jun-97	7.3	7.47
15-Aug-97	6.91	7.27
1-Sep-97	6.77	7.2
12-Sep-97	7.01	7.23
23-Sep-97	7.55	6.93
28-Sep-98	7.16	7.48
05-Oct-99	7.79	7.29
05-Mar-01	7.51	-
22-Mar-01	7.38	8.03
29-Mar-01	7.25	7.95
05-Apr-01	6.97	7.62
11-Apr-01	7.20	7.98
05-Jul-02	6.7	7.95
05-Aug-02	6.27	7.62
02-Sep-02	6.42	7.73
04-Oct-02	6.34	7.49
09-Jul-04	6.90	-
06-Aug-04	6.95	-
10-Sep-04	7.10	-
30-Apr-10	7.70	7.98
31-May-10	7.20	7.37
28-Jun-10	6.95	7.84

In summary, dewatering of mine workings generally results in an increase in SO_4 concentrations and decrease in Zn concentrations compared to saturated conditions. The decline in Zn concentrations can be observed in both the 1990 and 1999 dewatering events. An increase in SO_4 concentrations is observed during the 1990 dewatering event. Sulphate concentrations were relatively high during the 1984 dewatering event compared to subsequent time periods when the mine workings were flooded as well. It is unclear why SO_4 and Zn have divergent behavior. The only Zn mineral which has been identified in the mine rock is sphalerite; as such, a correlation between Zn and SO_4 would be expected. This divergent behavior could potentially be related to pH. The pH increases between field and laboratory measurements, presumably due to degassing of CO_2 (Table 3-41). While mine workings are dewatered, a greater degree of degassing will occur below ground within the mine workings, leading to an increase in pH. This response would be consistent with a carbonate mineral control of pH, in that carbonate minerals will buffer the pH to a higher value in an open system where CO_2 degassing can occur versus a closed system where CO_2 accumulates in solution. However, any pH related effect on Zn solubility while mine workings are dewatered remains speculative as no field pH measurements were collected during the three historical dewatering events. Recent data shows that dissolved concentrations are not sensitive to adit flow rates; this indicates that metal release is not affected by a kinetic loading rate, but rather is a result of equilibrium processes.

Scaled Wall Rock Loadings

The wall rock source term is calculated by calibrating observed loading rates to wall rock exposures during time periods when mine workings were dewatered. The two lithologies which will form the wall rock at silvertip (McDame limestone and Earn mudstone) are not differentiated in this calibration process. Not differentiating the McDame limestone an Earn mudstone is considered appropriate as both units were excavated by historical mine workings. That is, existing excavations have produced Earn:McDame waste rock at a 1:7 ratio, while future workings are expected to produce these two units at a ratio of 1:14. Existing static testwork shows that Earn and McDame units have similar metal leaching potential under neutral pH conditions. Therefore this variation in the relative exposures of these two rock units is not considered substantive to the overall source term calculation. The Earn and McDame wall rock units are only distinguished in upper case-closure scenario, where development of ARD from Earn group wall rock is considered.

Water quality, flow rates, and wall rock exposures from the 1984 and 1999 dewatering events are used to calculate the wall rock loading source term. The 1990 dewatering event is not incorporated due to the lack of data regarding pumping rates and exact dates of mine dewatering. The range in concentrations observed in the 1984 and 1999 dewatering events is provided in Table 3-42 and Table 3-43, respectively. This range of concentrations is converted into loading rates by multiplying the concentrations by the stabilized flow rates of 7L/s and 13 L/s during the 1984 and 1999 dewatering events, respectively. These loading rates are then loading rates is then divided by the area of wall rock exposure to estimate an area normalized loading rate. During the 1984 dewatering event approximately 1600 m of tunnels had been excavated by May, 1985, while 2367 m of tunnels existed during the 1999 dewatering event. Assuming tunnel dimensions of 4 m by 4 m, results in a wall rock exposure area of 25600 m^2 and 37900 m^2 for the 1985 and 1999 dewatering events, respectively. The area normalized loading rates are presented in Table 3-42 and Table 3-43. Note that area normalized loading rates for Zn and Cd are similar between the two dewatering events, providing some confidence in the approach.

Table 3-42:
Range of concentrations observed once flow rates had stabilized during the 1984 dewatering event (May, 1985 to October, 1985)

Description	statistic (n = 6)	unit	D-Cd	D-Cu	D-Fe	D-Pb	D-Zn	SO4
Observed Concentrations	max	mg/L	0.010	0.010	0.9	0.164	2.57	560
	median	mg/L	0.006	0.0075	0.303	0.053	2.33	468
	min	mg/L	0.002	0.004	0.072	0.04	0.65	423
Loading Rates Assuming 7 L/s	max	kg/week	0.0423	0.0423	3.81	0.694	10.9	2370
	median	kg/week	0.0254	0.0318	1.28	0.224	9.84	1980
	min	kg/week	0.00847	0.0169	0.305	0.169	2.75	1790
Area Normalized Loading Assuming wall rock area of 25,600 m2	max	mg/m^2/wk	1.65	1.65	149	27.1	424	92600
	median	mg/m^2/wk	0.992	1.24	50	8.76	384	77400
	min	mg/m^2/wk	0.331	0.662	11.9	6.62	107	70000

Note: Concentrations are converted into loading rates by multiplying by the max flow rate during this time period (7 L/s). The loading rates are then normalized by assuming a wall rock area of 25,600 m^2.

Table 3-43:
Range of concentrations observed once flow rates had stabilized during the 1999 dewatering event (December 4th, 1999 to February 7th, 2000)

Description	statistic (n = 10)	unit	D-Cd	D-Cu	D-Fe	D-Pb	D-Zn	SO4
Observed Concentrations	max	mg/L	0.020	0.02	1.26	0.013	2.02	-
	median	mg/L	0.0049	<0.001	<0.03	<0.001	1.12	-
	min	mg/L	0.0023	<0.001	<0.03	<0.001	0.933	-
Loading Rates Assuming 13 L/s	max	kg/week	0.157	0.157	9.91	0.102	15.9	-
	median	kg/week	0.0385	0.00786	0.236	0.00786	8.81	-
	min	kg/week	0.0181	0.00786	0.236	0.00786	7.34	-
Area Normalized Loading Assuming wall rock area of 37,900 m2	max	mg/m^2/wk	4.15	4.15	262	2.7	419	-
	median	mg/m^2/wk	1.02	0.21	6.23	0.21	233	-
	min	mg/m^2/wk	0.48	0.21	6.23	0.21	194	-

Note: Concentrations are converted into loading rates by multiplying by the max flow rate during this time period (13 L/s). The loading rates are then normalized by assuming a wall rock area of 37,900 m^2

The median area normalized loading rates presented in Table 3-42 and Table 3-43 are averaged to produce the base case wall rock source term, while the maximum area normalized loading rates are averaged to produce the upper case wall rock source term. The source term applied to the wall rock is presented in Table 3-44. Note that number of parameters were either not measured were below relatively high detection limits during time periods used to calibrate the wall rock source term. For these parameters, loading rates are estimated based on the ratio between the parameter of interest and SO_4 concentrations observed in recent adit drainage chemistry (2010 to current). Note that since 2010, a full suite of metals has been monitored with improved detection limits. The observed ratio is then multiplied by the predicted SO_4 concentration to produce an estimated loading rate for the parameter in question. For instance, As was not measured during the 1984 dewatering event and was below the relatively high detection limit of 0.2 mg/L in all samples collected during the 1999 dewatering event. Since 2010, the average D-As concentration of 0.002 mg/L has been observed while the average SO_4 concentration is 311 mg/L. This ratio of As/SO_4 is multiplied by the SO_4 loading rates predicted by calibration of loading rates to produce an estimate for As. This is done for all parameters which were either not monitored or below detection limits during the 1984 and 1999.

Table 3-44:
Wall rock loading rates used to predict underground source term

	unit	Base Case	Upper Case
SO_4	$mg/m^2/week$	77400	92600
As	$mg/m^2/week$	0.531	0.635
Ca	$mg/m^2/week$	52800	57700
Cd	$mg/m^2/week$	1.0	2.9
Cu	$mg/m^2/week$	0.724	2.9
Mg	$mg/m^2/week$	8790	10300
Ni	$mg/m^2/week$	6.75	8.08
Pb	$mg/m^2/week$	4.49	14.9
Sb	$mg/m^2/week$	2.2	2.64
Se	$mg/m^2/week$	0.249	0.297
U	$mg/m^2/week$	0.916	1.1
Zn	$mg/m^2/week$	309	422

During mine life wall rock is expected to remain at a neutral pH. However, upon mine closure Earn group wall rock exposures above the water table have the potential to become acid generating. In order to estimate the metal leaching rate of acidic Earn group wall rock, the calibrated loading rates in Table 3-44 are scaled by NAG test results. That is, the rate of wall rock loading under neutral conditions is scaled by the ratio of NAG extractable metals in acidic vs. neutral NAG tests. The average concentration produced by NAG extractions which produced neutral and acidic leachate is provided in Table 3-45. The ratio of acidic to neutral NAG tests and the scaled acidic wall rock loading rate are also provided in this Table. Note that the scaled acidic wall rock loading rates are only applied to the upper case-closure scenario.

**Table 3-45:
Scaled wall rock loading to predict acidic wall rock loading rate aplied to unsaturated Earn group wall rock in the upper case-closure scenario.**

	Neutral-NAG mg/L	Acidic-NAG mg/L	Acidic:Neutral Ratio	Upper Case-Closure Acidic Wall Rock mg/m^2/week
SO_4	117	533	4.57	354000
As	0.0208	0.0134	0.645	0.342
Ca	39.5	73.7	1.87	98700
Cd	0.00042	0.0145	34.6	34.7
Cu	0.0126	0.21	16.6	12.1
Mg	9.69	29.5	3.05	26800
Ni	0.0986	0.301	3.05	20.6
Pb	0.000143	0.0191	133	598
Sb	0.00607	0.00483	0.797	1.76
Se	0.00749	0.0133	1.77	0.441
U	0.000653	0.00288	4.41	4.04
Zn	0.281	3.67	13.1	4030

3.3.7.2 Saturated Underground Source Term

Upon mine closure much of the underground mine workings will become flooded and fully saturated. Saturated conditions will inhibit metal leaching and acid generation from sulphide oxidation, however, some leaching of water soluble phases and reducible oxides may occur. Currently, a majority of underground mine workings are saturated, with only a small volume of unsaturated void space (approx. 332 m^3 in McDame limestone). Hence, recent water quality can provide an indication of metal leaching from saturated underground mine workings.

Dissolved concentrations measured at WQ9 have been relatively constant since 2010, while flow rates have varied from 0.7 L/s to 5.4 L/s with no apparent relationship to drainage chemistry (Table 3-40 and Figure 3-5). The wide range in flow rates, and relatively constant drainage chemistry suggest that dissolved concentrations are not affected by flow rate. This indicates that metal release is equilibrium controlled, and is not related to kinetic loading rates. For the saturated source term it is assumed that a similar concentration range will be imparted in infiltrating ground water when mine workings flood at the end of operations. The median concentration observed at WQ9 between 2010 and 2013 is assigned to the base case-closure saturated source term, while the upper 10th percentile concentrations observed at WQ9 during this time period are assigned to the upper case-closure scenario. These concentrations are provided in Table 3-46. Note that this source term is a constant concentration and independent of flow rate. The closure scenarios also consider metal leaching from unsaturated mine workings and backfill above the saturated mine workings.

Table 3-46:
Saturated mine workings soruce term for closure scenario.

	unit	Base Case	Upper Case
SO_4	mg/L	311	329
As	mg/L	0.00537	0.00647
Ca	mg/L	156	167
Cd	mg/L	0.0122	0.0156
Cu	mg/L	0.000905	0.001
Mg	mg/L	26.4	27
Ni	mg/L	0.0281	0.0348
Pb	mg/L	0.000485	0.000778
Sb	mg/L	0.0093	0.0099
Se	mg/L	0.001	0.00161
U	mg/L	0.00388	0.00413
Zn	mg/L	2.54	3.18

3.3.7.3 Backfill Loading Rates

A majority of the underground void space produced over mine life will be filled in with backfill. This is in part a requirement of the Drift-and-Fill mining method, which requires the placement of backfill for full extraction of the ore zones. Primary stopes will be filled with cemented tailings backfill requiring a cement content of approximately 5% to provide stability of the backfilled walls, which are exposed when mining the neighboring secondary stopes. The secondary stopes can be backfilled with less cement, since a bottom-up mining sequence will be applied. The secondary stope backfill (secondary backfill) requires a cement content of 1%. However, due to lack of geochemical characterization of secondary backfill a cement content of 5% will be used until the characterization is complete.

During the first few months of mining, tailings will not be available for backfilling stopes. During this time period approximately 7,000 tonnes of cemented rock fill (CRF) will be required. Earn group waste rock will used for CRF production as this rock type requires underground storage to prevent the development of ARD. To produce CRF, the Earn group waste rock will be crushed before being mixed with 3% cement slurry. At the end of mine life, any remaining Earn group waste rock in the PAG stockpile will backfilled into the underground mine workings that will flood at closure to mitigate the stockpiles ARD potential.

Tailings Backfill

Tailings backfill will be composed of desulphurized (DeS) tailings and pyrite concentrate at a ratio of approximately 2:3 when backfilling ore zones below the portal elevation. Acid base accounting results indicate that backfill is PAG, however, the lag time until ARD develops exceeds the operational mine life (Appendix C). When backfilling zones above the portal elevation, only DeS tailings are used. No testwork has been carried out on cemented backfill composed entirely of DeS. As a conservative assumption it is assumed to have the same metal leaching potential as backfill containing pyrite concentrate.

Humidity cell testwork is available for two paste backfill samples produced from metallurgical tailings. The samples were produced using a 3:2 ratio of pyrite concentrate to DeS tailings with 6 wt.% cement (BF-HC 4 and BF-HC5). The 90th percentile loading rates and the median loading rates of the final 5 cycles of data for BF-HC4 and BF-HC5 which are used to calculate metal leaching are presented in Table 3-47. The 90th percentile loading rates are applied to the operations source terms, while the final loading rate is applied to the closure source term.

Table 3-47:
Backfill tailings humidity cell loading rates

		90th Percentile	**Final Loading Rate**
SO_4	mg/kg/wk	143	69.5
As	mg/kg/wk	0.00449	0.00302
Ca	mg/kg/wk	73.1	32.7
Cd	mg/kg/wk	0.00447	0.00172
Cu	mg/kg/wk	0.0196	0.00113
Mg	mg/kg/wk	1.28	0.783
Ni	mg/kg/wk	0.000707	0.000466
Pb	mg/kg/wk	0.101	0.0281
Sb	mg/kg/wk	0.0249	0.0212
Se	mg/kg/wk	0.00187	0.000985
U	mg/kg/wk	0.0000893	0.0000634
Zn	mg/kg/wk	0.184	0.061

A series of physical scaling factors are applied to adjust loading rates to account for differences between lab-based kinetic tests and field conditions. These include but are not limited to hydrogeological pathways, water/rock ratio, temperature, grain size distribution, and reaction mechanisms. A summary of scaling factors used to scale humidity cell results are presented in Table 3-48.

Table 3-48:
Scaling factors for cemented tailings backfill

		Operations		**Closure**	
Scaling Factors	*Unit*	*Base Case*	*Upper Case*	*Base Case*	*Upper Case*
Temperature Correction	-	0.3	0.3	0.3	0.3
Contact Water Factor	-	0.5	1.0	0.5	1.0
Grain Size Correction Factor	-	0.01	0.05	0.01	0.05

The cemented backfill will be relatively impermeable, with flow dominated by flow along the backfill surfaces and fractures. Note that the backfill humidity cells have a relatively fine grain size (Klohn, 2014), with a p50 of <0.06 mm. Therefore a relatively small grain size correction factor is applied, it is assumed that in the cemented backfill will only have 1% to 5% of the available surface area that is present in the humidity cell sample. The percentage of the surface area experiencing flow can be expected to be relatively high as most of the available surface area is limited to cement surfaces. The contact water correction factor is conservatively estimated to be between 50% and 100%. A temperature correction factor of 0.3 is applied to account for reduced sulphide oxidation rates at an expected underground temperature of approximately 7.5°C (Appendix C). This range in estimates of grain size and contact water correction factors are used to differentiate between upper case and base case source terms (Table 3-48). Scaled loading rates are provided in Table 3-49.

Table 3-49:
Scaled Loading Rates for Backfill Tailings

		Operations		**Closure**	
		Base Case	**Upper Case**	**Base Case**	**Upper Case**
SO_4	mg/kg/wk	11.2	112	5.42	54.2
As	mg/kg/wk	0.00035	0.0035	0.000235	0.00235
Ca	mg/kg/wk	5.7	57	2.55	25.5
Cd	mg/kg/wk	0.000349	0.00349	0.000134	0.00134
Cu	mg/kg/wk	0.00153	0.0153	0.0000882	0.000882
Mg	mg/kg/wk	0.0997	0.997	0.0611	0.611
Ni	mg/kg/wk	0.0000551	0.000551	0.0000363	0.000363
Pb	mg/kg/wk	0.00789	0.0789	0.00219	0.0219
Sb	mg/kg/wk	0.00195	0.0195	0.00165	0.0165
Se	mg/kg/wk	0.000146	0.00146	0.0000768	0.000768
U	mg/kg/wk	6.96E-06	6.96E-05	0.00000494	4.94E-05
Zn	mg/kg/wk	0.0144	0.144	0.00476	0.0476

Waste Rock Backfill

As a conservative assumption, it is assumed that CRF has the same metal leaching potential as Rock fill. Note that the presence of cement will provide additional neutralization capacity and reduce the available surface area for oxidation of mine waste. Thus, the assumption that CRF has the same metal leaching potential as rock fill is considered conservative. The same loading rates and scaling factor applied to the PAG stockpile are used to predict metal leaching rates for Rock Fill. These scaled loading rates are presented in Table 3-22.

3.3.7.4 Predicted Concentrations and Solubility Controls

In order to apply solubility controls the scaled loading rates described in the previous sections must be converted into a concentration, and an estimate of pH and alkalinity must be derived. Concentrations are estimated for the end of operations scenario when the maximum backfill volumes and wall rock exposures will occur, and for closure scenario the underground mine workings below the portal elevation flood.

Loading Rates

Total mass of tailings backfill, PAG rock fill, CRF that will be stored in underground mine workings are presented in Table 3-50. The total mass of backfill is assumed to contribute to metal loading at the end of operations when the mine workings are unsaturated. The closure source term only considers loading rates from backfill and wall rock above the water table. Note that a saturated source term is also developed for the closure scenario.

Table 3-50:
Masses of backfill and volume of excavation used to calculate loading rates for the end of operations and closure source terms. Note that the closure scenario only includes loading from unsaturated backfill and mine workings.

Description	unit	End of Operations	Closure*
Tailings backfill	tonnes	914,000	67,100
PAG Rock Fill	tonnes	30,400	-
CRF	tonnes	7,700	-
Earn Group Excavation	m^3	14,700	4,590
McDame Group Excavation	m^3	213,000	1,880
Ore Excavation	m^3	305,000	35,300

*Only includes backfill and wall excavations above portal elevation.

Approximately 38,000 tonnes of PAG rock fill and CRF will be stored underground. This mass of rock fill and CRF is assigned the source term developed for the PAG stockpile presented in Table 3-25. Note that all PAG and CRF will be flooded upon mine closure, hence, this source term is not included in the closure scenario. For the operations scenario it is assumed that 100% of backfill present underground is oxidizing at the loading rate presented in Table 3-49. For the closure scenario only the backfill mass that will be stored above the portal elevation is assigned the loading rate presented in Table 3-49.

Excavation volumes of Earn group, McDame group and ore are shown in Table 3-50. In order to convert the volume of Earn group and McDame group waste rock into a surface area, it is assumed that these non-mineralized units are mined as 4 m x 4 m tunnels. Volumes (m^3) can be converted to surface area (m^2) at a 1:1 ratio by making this assumption. The excavated ore volume will be completely backfilled with primary or secondary cement. Therefore, ore wall rock loading is not incorporated into source term development.

Earn group and McDame group wall rock exposures are assigned loading rates presented in Table 3-44 for the base case-operations, upper case-operations and base case-closure scenarios. In the upper case-closure scenario, it is assumed that Earn group wall rock becomes acid generating and the scaled acidic loading rate presented in Table 3-45 is applied, while the upper case loading rate for the McDame limestone is unchanged between closure and operations scenarios.

Flow Rate

In order to convert the cumulative loading rates discussed in the previous section into concentrations an estimate of flow rate is required. During operations, flow rates are estimated to range from 8 L/s to 15 L/s (Klohn, 2014). For the purpose of calculating an end of operations concentration, a flow rate of 10 L/s is assumed. Upon mine closure the flow rate is expected to range from 4 L/s to 6 L/s. For the purpose of calculating a closure source term, a flow rate of 5 L/s is assumed.

Alkalinity and pH

During mine operations, the pH is assumed to be similar to pH values measured during previous dewatering events. No field pH or alkalinity samples were collected during these dewatering events. Note that the difference between laboratory and field pH values are expected to be less significant while mine workings are dewatered, as described in section 3.3.7.1. The base case and upper case operations are assigned the median and 10th percentile lab-pH values observed during the 1984 and 1999 dewatering events (pH 7.7 and pH 7.5, respectively). Alkalinity is estimated by assuming a pCO_2 of -3.0.

The pH of portal drainage is expected to decline slightly upon closure when the mine workings flood due to the reduced equilibrium pH of carbonate minerals under closed conditions (cutoff from atmosphere). Note that some Earn group waste rock is assumed to become acid generating in the upper case-closure scenario. Considering that the exposure of Earn Group wall rock is not directly connected to the portal which is in McDame group limestone, any acidic seepage will be neutralized by carbonate minerals prior to discharge. Therefore the pH and alkalinity is expected to be similar to what is currently produced from the mine adit. For the drainage pH is assigned the median and 10th percentile alkalinity and field pH values measured while the mine workings were flooded for the upper case and base case scenarios, respectively.

Solubility Controls

In order to identify potential solubility controls, the predicted solution chemistry is speciated in PHREEQC using the llnl.dat database. A number of potential mineral solubility controls were found to be supersaturated. Select mineral phases found to be supersaturated are set to equilibrium as outlined in Table 3-51. The PHREEQC input files are provided in Appendix D.

Table 3-51:
Secondary mineral controls applied to portal drainage source term

		Operations		**Closure**	
Mineral Phase	**Chemical Formula**	**Base Case**	**Upper Case**	**Base Case**	**Upper Case**
Gypsum	$CaSO_4:2H_2O$	x	x		
$ZnCO_3:H_2O$	$ZnCO_3:H_2O$	x	x		
Cerusite	$PbCO_3$	x	x	x	x
Siderite	$FeCO_3$				x
Otavite	$CdCO_3$				

Predicted Concentration

Predicted concentrations for the portal at the end of operations and closure mine scenarios are presented in Table 3-52. The operations source terms are calculated using the maximum extent of underground mine workings and backfill mass that will be reached at the end of mine life. Upon mine closure the water table will rebound to the portal elevation flooding much of the underground workings. This leads to a considerable reduction in total metal loading, however, concentrations for a number of parameters show little or no change due to the coinciding decline in flow rate. That is, the flow rate is assumed to be 10L/s in the operations scenario, versus 5L/s in the closure scenario. Note that estimates of nitrogen loadings from the mine portal are described in Appendix B.

Table 3-52:
Predicted portal drainage chemistry

Parameter	Unit	Operations		Closure	
		Base Case	Upper Case	Base Case	Upper Case
pH	s.u.	7.7	7.5	7.1	6.5
T-Alkalinity	$mgCaCO_3/L$	73.4	52.7	168	161
$N-NO_3$	mg/L	1.0	3.3	0.0864	1.31
$N-NO_2$	mg/L	0.4	0.88	0.0054	0.0818
$N-NH_3$	mg/L	3.2	7.7	0.0148	0.224
SO_4	mg/L	1920	2060	902	1470
F	mg/L	0.0134	0.134	1.44	2.34
Cl	mg/L	5.41	12.1	1.53	1.62
Ag	mg/L	0.00182	0.0165	0.000296	0.000245
Al	mg/L	0.0818	0.279	0.0194	0.256
As	mg/L	0.0214	0.0355	0.00953	0.0111
Ba	mg/L	0.0889	0.402	0.0302	0.147
Ca	mg/L	589	576	343	612
Cd	mg/L	0.0466	0.151	0.0199	0.167
Co	mg/L	0.0805	0.0877	0.0178	0.0301
Cr	mg/L	0.00964	0.013	0.0029	0.00323
Cu	mg/L	0.0328	0.158	0.00704	0.0832
Fe	mg/L	1.07	7.79	0.955	42.1
Hg	mg/L	0.00243	0.00864	0.000144	0.000163
K	mg/L	17.1	27.2	5.29	8.13
Mg	mg/L	170	204	57.1	121
Mn	mg/L	3.77	5.97	1.06	2.72
Mo	mg/L	0.0133	0.0418	0.00186	0.00203
Na	mg/L	2.5	6.02	2.93	3.03
Ni	mg/L	0.259	0.323	0.0792	0.118
Pb	mg/L	0.0187	0.0249	0.0198	0.0389
Sb	mg/L	0.0904	0.162	0.0268	0.0299
Se	mg/L	0.0104	0.0177	0.00294	0.0042
Si	mg/L	23.7	30.0	7.79	9.59
Sr	mg/L	2.92	4.16	0.458	0.78
Tl	mg/L	0.00219	0.00493	0.000617	0.00149
U	mg/L	0.035	0.0432	0.0108	0.0174
Zn	mg/L	5.15	13.1	4.86	13.1

3.3.8 Mill Process Water

Mill process water will be produced from dewatering of tailings and concentrate. Process water will carry a dissolved load owing to reagents used in ore processing (*i.e.* lime and cyanide), and release of water soluble metals associated with the ore. A majority of the process water discharged from the plant will be produced from dewatering of tailings waste (pyrite concentrate and DeS tailings) with lesser amounts of water discharged from the Zn concentrate thickener overflow circuit. Discharge of mill process water is directed to the MCP where it is either used as makeup water by the mill or discharged to the environment via the HDS lime treatment plant.

Metallurgical test programs were conducted in 2011, 2013 and 2014 to develop and assess metal recovery from the Silvertip ore. During each program, tailings supernatant was collected for environmental testwork. The most recent metallurgical testwork (2014) used ore from the oxidized ore stockpile present on site. This ore was originally mined in the 1980's and represents ore at a relatively advanced weathering state compared to what is expected for ROM produced during mining operations. Therefore the supernatant chemistry produced during this testwork is not considered an appropriate estimate of supernatant chemistry that will occur at site. The testwork conducted in 2013 and 2011 used fresh ore collected from drill core. The 2013 testwork produced supernatant with a relatively high pH (pH 9.5) due to higher lime addition compared to 2011 and 2014 testwork, which produced rougher tailings ranging at a more neutral pH (pH 7.4 to pH 7.5, respectively). For these reasons, the supernatant chemistry from pyrite rougher tailings produced during lock cycle testwork conducted in 2011 is used to estimate dissolved metal concentrations in mill process water chemistry (Table 3-53). Note that some modifications are made to this chemistry as described below.

Sulphate concentrations can be expected to accumulate in process water until gypsum saturation is reached. Therefore the SO_4 is set at the approximate concentration of gypsum saturation of 1600 mg/L (compared to 544 mg/L measured in supernatant). Nitrogen and cyanide species are below detection limit in the 2011 pyrite rougher tailings. Therefore concentrations of nitrogen and cyanide species from the Wolverine mine in southeastern Yukon. The Wolverine mine is an underground Zn-Ag-Cu-Pb-Au mine that uses minor amounts of CN similar to the process planned at Silvertip. The maximum concentration observed in process water at the Wolverine mine during the first two years of operations is assigned to the process water source term for Silvertip. For the purposes of the water balance model, the same process water source term is used for the upper case and base case scenarios (Table 3-53).

Table 3-53:
Mill Process Water Source Term

Parameter	Unit	Base case and Upper Case
pH	s.u.	7.41
T-Alkalinity	$mgCaCO_3/L$	1600
$N\text{-}NO_2$	mg/L	0.8
$N\text{-}NO_3$	mg/L	5.8
$N\text{-}NH_3$	mg/L	3.0
T-CN	mg/L	1.5
WAD-CN	mg/L	0.065
SO_4	mg/L	1600
Cl	mg/L	4.9
F	mg/L	0.23
Al	mg/L	0.0044
Sb	mg/L	0.0274
As	mg/L	0.0027
Cd	mg/L	0.00626
Ca	mg/L	220
Cr	mg/L	0.0005
Co	mg/L	0.00995
Cu	mg/L	0.0296
Fe	mg/L	0.029
Pb	mg/L	0.0702
Mg	mg/L	3.80
Mn	mg/L	0.529
Hg	mg/L	0.000001
Mo	mg/L	0.0229
Ni	mg/L	0.0125
K	mg/L	2.89
Se	mg/L	0.00568
Ag	mg/L	0.00506
Na	mg/L	16.3
Tl	mg/L	0.00118
U	mg/L	0.000084
V	mg/L	0.00004
Zn	mg/L	0.437

3.4 Water Treatment Plant Effluent

3.4.1 Overview of Approach

All contact water produced from the Silvertip Mine project will be collected in the Main Collection Pond and be treated through the mine water treatment plant (MWTP). As such, effluent from the MWTP is a key source term in the water quality modeling for the project. Preliminary effluent quality standards (EQS) for the MWTP have been developed to inform the design and desired performance of the system. The methodology employed to develop defensible EQS for the Silvertip project that are protective of aquatic life in the receiving environment of Silvertip Creek at station WQ8 followed a stepwise progression that duly considered:

- BC water quality guidelines for the protection of freshwater aquatic life (BCWQG) in Silvertip Creek at station WQ8 for those parameters that are not elevated in the background above BCWQG;
- Non-degradation water quality benchmarks for those parameters currently elevated in WQ8 background (minus current portal discharge loadings and presented in Table 3-9), namely Cd and Zn. The maximum observed mean monthly value plus 10% was used to establish the non-degradation water quality benchmark for these parameters. The benchmarks derived this way were less than the 95th percentile of WQ8 background data.
- Background monthly water quality (*i.e.* minus current portal loadings) and flow conditions at WQ8;
- Predicted seasonally-variable discharge volumes of mine contact water from the project from the MWTP during operations and early closure;
- Published acute toxicity values (*e.g.* LC50) for a suite of parameters of environmental interest for salmonids to ensure that proposed discharge limits were reasonable and unlikely to induce acute toxicity; and
- Metal Mining Effluent Regulations (MMER) concentrations as a minimum performance objective.

3.4.2 Water Quality Benchmarks and Non-Degradation Benchmarks

Water quality benchmarks for Silvertip Creek have been developed to be protective of aquatic life as providing a basis for the development of effluent quality standards (EQS) for the project for all mine contact waters exiting the MWTP. For Silvertip Creek, water quality benchmarks have been developed for station WQ8, which is located downstream of all potentially affected mine contact waters and thus serves as the overall project compliance point.

In establishing water quality benchmarks for the project, existing baseline water quality conditions for Silvertip Creek have been considered as well as generic water quality guidelines currently published by British Columbia Ministry of Environment (BCMOE). For most parameters, generic water quality guidelines as published in BCMOE have been adopted. The objectives selected are based on chronic guidelines (30-day average concentrations) not maximum allowable concentrations. A number of parameters have hardness dependent criteria and include SO_4, Cd, Cu, Pb, Mn, Ni, Ag and Zn. As such, characterization of the hardness of the receiving stream was performed prior to calculation of the appropriate water quality objective. As previously introduced, for Cd and Zn a non-degradation approach was employed to establishing water quality benchmarks for these parameters. The proposed water quality benchmarks for Silvertip Creek at WQ8 are summarized in Table 3-53.

Table 3-54:
Water Quality Benchmarks used in the development of EQS for the MWTP

Parameter	Benchmark	Rationale
pH	6.5 - 9.0	
Nitrate (as N)	3	BC long-term WQG
Nitrite (as N)	0.02	BC long-term WQG for water with chloride <2 mg/L.
Ammonia (as N)	0.87	BC long-term WQG, based on an average pH of 8.1 and summer temp estimate of 15 C
WAD-CN	0.005	BC 30-day average
Sulfate (SO4)	309	BC long-term WQG based on hardness ranging from
Fluoride (F)	1.55	BC long-term WQG for annual mean hardness of 170 mg/L.
Chloride (Cl)	0.15	BC 30-day average
Silver (Ag)-Total	0.0015	BC long-term WQG for annual mean hardness of 170 mg/L.
Aluminum (Al)-Dissolved	0.05	BC long-term WQG for water with pH>6.5.
Arsenic (As)-Total	0.005	BC short-term WQG. BC only has a max guideline for As, no 30-d avg WQG is proposed.
Cadmium (Cd)-Total	0.0013	SBEB calculated as maximum monthly mean plus 10%
Cobalt (Co)-Total	0.004	BC 30-day average
Copper (Cu)-Total	0.0068	BC long-term WQG for annual mean hardness of 170 mg/L.
Iron (Fe)-Dissolved	0.35	BC short-term WQG. No long-term Fe guideline has been developed
Iron (Fe)-Total	1	Equal to BC short-term WQG. No long-term Fe guideline has been developed
Mercury (Hg)-Total	0.000020	Equal to BC long-term WQG assuming T-Hg is composed of 0.5% MeHg
Manganese (Mn)-Total	1.3	BC 30-day average assuming hardness of 170 mg/L
Molybdenum (Mo)-Total	1	BC 30-day average
Nickel (Ni)-Total	0.110	BC short-term WQG for annual mean hardness of 170 mg/L
Lead (Pb)-Total	0.0096	Equal to BC long-term WQG for annual mean hardness of 170 mg/L
Antimony (Sb)-Total	0.02	Draft working BC guideline
Selenium (Se)-Total	0.002	Equal to BC long-term WQG
Thallium (Tl)-Total	0.0017	Water plus organism
Zinc (Zn)-Total	0.332	SBEB calculated as maximum monthly mean plus 10%

3.4.3 Determination of Acceptable Threshold Discharge Concentrations

Water treatment requirements for the project for the MWTP were a key focus of water quality modeling. Geochemical source terms for mine contact water and temporary stockpiles have been developed for each month of each mine-year and incorporated into the water balance model. Similarly, source terms for non-contact flows, including undisturbed runoff and background water chemistry in Silvertip Creek have been incorporated into the model. The combined water balance/geochemical Goldsim® water quality model has been run under average case flow and climate conditions to determine water treatment requirements for the project and to establish target treatment requirements for those parameters anticipated to be elevated in mine contact water. As previously introduced, the parameters that have been modeled and their respective proposed water quality benchmarks at WQ8 in Silvertip Creek are provided in Table 3-53.

For the purposes of developing EQS, the water quality model was initially used to determine the threshold (*i.e.*, maximum allowable) concentrations in the MWTP discharge that would still maintain water quality benchmarks at WQ8 for all parameters modeled. In order to determine the threshold concentrations, the MWTP discharge was the independent variable and WQ8 water quality was set at the water quality benchmark for each parameter. The model was then operated to "back-calculate" the maximum allowable concentration in the MWTP discharge. The lowest observed "back-calculated" threshold concentration was then selected as the MWTP benchmark EQS.

3.4.3.1 MWTP EQS

As described above, the water quality model was used to hindcast the threshold (*i.e.*, maximum allowable) concentrations in the MWTP discharge that would still meet water quality benchmarks at WQ8 in Silvertip Creek under all flow conditions. The results of this exercise were used to establish effluent quality standards/benchmarks (EQS) for the MWTP that were ultimately protective of aquatic life in the receiving environment and did not cause degradation to existing water quality for those parameters naturally elevated in the background. Table 3-54 summarizes the proposed effluent quality standards for the MWTP for the entire period in which it is in operation and Table 3-55 provides the detailed monthly threshold values calculated and used to derive the EQS for Table 3-54. Certain parameters, namely TSS, Ni and Zn had benchmark EQS that were controlled by MMER limits. For example, Ni and Zn had minimum hindcast threshold values of 0.53 mg/L and 0.60 mg/L, respectively. However, these values are in excess of MMER limits and therefore the proposed benchmark EQS for these parameters is shown as 0.5 mg/L.

For sulphate, the proposed EQS was set at a conservative sulphate concentration assuming some degree of gypsum solubility control.

Table 3-55:
MWTP EQS Benchmarks

Parameter	*Proposed Effluent Quality Standards (mg/L)*
pH	**6.5 to 8**
TSS	**15**
SO_4	**1800**
Nitrate-N	**15**
Nitrite-N	**0.09**
NH_3-N	**4.3**
CN_{WAD}	**0.024**
Al (diss)	**0.23**
Sb	**0.1**
As	**0.024**
Cd	**0.003**
Co	**0.02**
Cu	**0.03**
Fe	**4**
Pb	**0.045**
Hg	**0.00006**
Mn	**6**
Mo	**5**
Ni	**0.5**
Se	**0.006**
Ag	**0.007**
Zn	**0.5**

Table 3-56:
MWTP monthly threshold benchmarks used to propose EQS

Parameter	**WTP EQS (mg/L)**											
	Jan	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**
Nitrate (as N)	17.6	14.7	18.4	28.9	51.5	66.5	47.3	43.4	23.6	30.8	23.1	21.2
Nitrite (as N)	0.114	0.095	0.118	0.187	0.334	0.426	0.303	0.224	0.123	0.198	0.149	0.139
Ammonia (as N)	5.2	4.3	5.3	8.4	15.2	19.4	13.7	12.6	6.8	9.0	6.7	6.3
WAD-CN	0.0295	0.0244	0.0306	0.0483	0.0869	0.1107	0.0784	0.0721	0.0392	0.0512	0.0384	0.0359
Sulfate (SO4)	1506	1254	1488	2296	4809	5990	4162	3767	2074	2683	1985	1819
Silver (Ag)-Total	0.00889	0.00735	0.00919	0.01455	0.02421	0.03323	0.02360	0.02172	0.01180	0.01542	0.01156	0.01079
Aluminum (Al)-Dissolved	0.281	0.232	0.286	0.448	🟥	0.525	0.648	0.613	0.308	0.468	0.364	0.343
Arsenic (As)-Total	0.029	0.024	0.029	0.045	0.027	0.103	0.073	0.068	0.037	0.050	0.037	0.035
Cadmium (Cd)-Total	0.00426	0.00384	0.00310	0.00355	0.00708	0.01578	0.01223	0.00983	0.00578	0.00810	0.00605	0.00472
Cadmium (Cd)- Dissolved	0.00329	0.00315	0.00278	0.00222	0.01164	0.01386	0.01080	0.00842	0.00548	0.00759	0.00540	0.00335
Cobalt (Co)-Total	0.0233	0.0193	0.0241	0.0382	0.0546	0.0817	0.0619	0.0571	0.0307	0.0404	0.0303	0.0283
Copper (Cu)-Total	0.0382	0.0317	0.0386	0.0609	🟥	0.1108	0.0953	0.0903	0.0448	0.0649	0.0494	0.0463
Iron (Fe)-Dissolved	1.9	1.6	2.0	3.2	5.1	7.2	5.2	4.8	2.6	3.3	2.5	2.3
Iron (Fe)-Total	5.8	4.8	6.0	9.5	🟥	20.2	15.0	14.2	6.3	10.1	7.3	7.0
Mercury (Hg)-Total	0.00007	0.00006	0.00007	0.00011	0.00019	0.00023	0.00017	0.00016	0.00009	0.00011	0.00009	0.00008
Manganese (Mn)-Total	8.04	6.65	8.31	13.15	22.75	29.89	21.34	19.61	10.58	13.96	10.46	9.76
Molybdenum (Mo)-Total	5.95	4.93	6.16	9.75	17.53	22.36	15.83	14.55	7.90	10.33	7.75	7.23
Nickel (Ni)-Total	0.64	0.53	0.66	1.04	1.83	2.39	1.71	1.57	0.85	1.12	0.84	0.78
Lead (Pb)-Total	0.055	0.046	0.056	0.090	🟥	0.161	0.138	0.134	0.059	0.095	0.070	0.066
Antimony (Sb)-Total	0.118	0.097	0.121	0.192	0.339	0.442	0.312	0.287	0.156	0.204	0.153	0.143
Selenium (Se)-Total	0.007	0.006	0.009	0.013	0.019	0.024	0.018	0.017	0.009	0.013	0.010	0.008
Zinc (Zn)-Total	0.84	0.71	0.72	0.60	2.54	4.69	3.09	2.40	1.49	1.96	1.23	1.01
	🟥	represent elevated background concentrations due to high TSS that are above WQ8 water quality benchmark										

3.4.3.2 MWTP Expected Performance

The proposed MWTP EQS benchmarks are to be used to inform water treatment plant design. The proposed high density sludge MWTP is expected to produce water quality on a routine basis that is better than the proposed EQS benchmarks for a number of parameters. Table 3-56 summarizes the expected performance of the MWTP in comparison to proposed EQS benchmarks.

Table 3-57:
MWTP expected performance compared to proposed EQS

Parameter	*Proposed EQS (mg/L)*	*Expected Performance (mg/L)*
pH	**6.5 to 8**	**6.5 to 8**
TSS	**15**	**<5**
SO_4	**1800**	**1800**
Nitrate-N	**15**	**2.9**
Nitrite-N	**0.09**	**0.4**
NH_3-N	**4.3**	**6.9**
CN_{WAD}	**0.024**	**0.024**
Al (diss)	**0.23**	**0.2**
Sb	**0.1**	**0.05**
As	**0.024**	**0.01**
Cd	**0.003**	**0.001**
Co	**0.02**	**0.02**
Cu	**0.03**	**0.01**
Fe	**4**	**0.1**
Pb	**0.045**	**0.02**
Hg	**0.00006**	**0.000067**
Mn	**6**	**0.1**
Mo	**5**	**0.05**
Ni	**0.5**	**0.01**
Se	**0.006**	**0.008**
Ag	**0.007**	**0.001**
Zn	**0.5**	**0.1**

4. Model Results

4. Model Results

4.1 Model Validation

Calibration of the model was completed through a trial-and-error approach by comparing the model predictions to the collected baseline data (including flows and water quality concentrations) in the receiving environment until a reasonably accurate model performance was achieved; particularly for the water quality predictions. For the purpose of calibration and validation, the rectified monthly averages from baseline flow data presented in Table 3-1 and the baseline concentrations presented in Section 3.2 were used. In the model, the monthly runoff and discharges was simulated assuming the same monthly runoff distribution as the Rancheria watershed because Silvertip Creek is located in the Rancheria watershed. The runoff coefficients for the background stream (*i.e.*, the Silvertip Creek and Tootsee River) catchments were also assumed to be approximately equal, which would be adjusted through model calibration.

Figure 4-1 and Figure 4-2 show the comparisons of the average monthly flows at the modeling location WQ8 on the Silvertip Creek and the background location WQ4 on the Tootsee River during the pre-mine phase (reflecting current average conditions), demonstrating a good match between the rectified baseline data and model estimates and verifying the average flow scenarios has been properly incorporated in the model (with a runoff coefficient of 0.37).

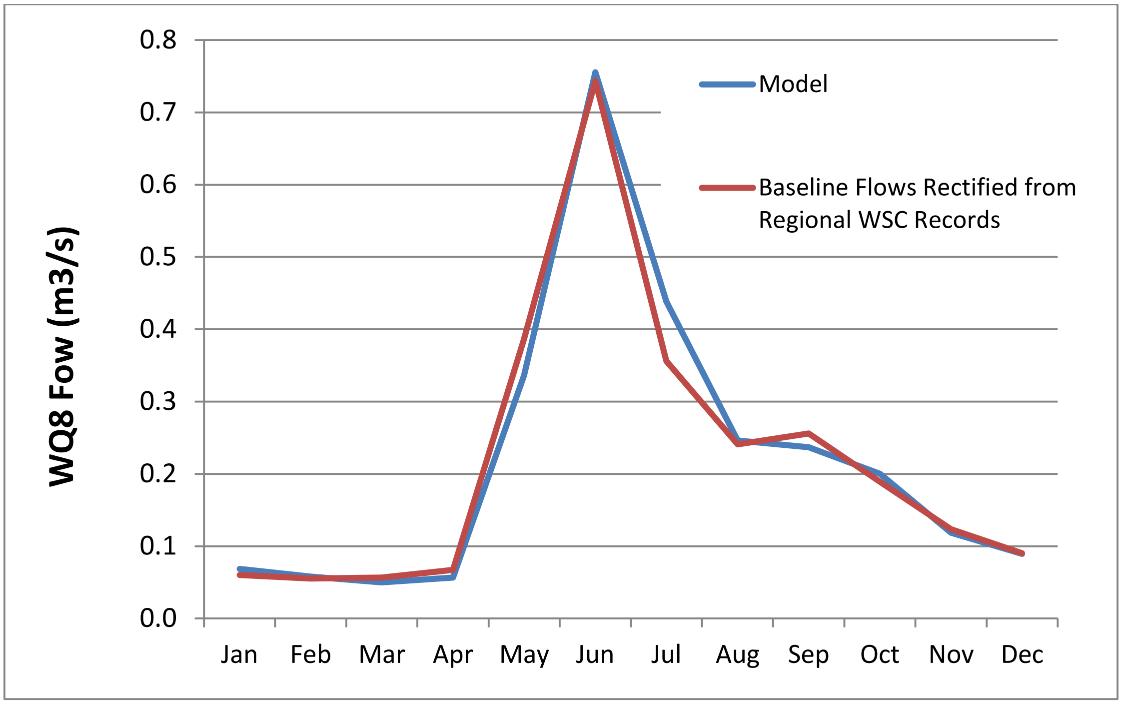


Figure 4-1: Comparison of model estimated average monthly flows and the baseline observations at the Silvertip Creek WQ8

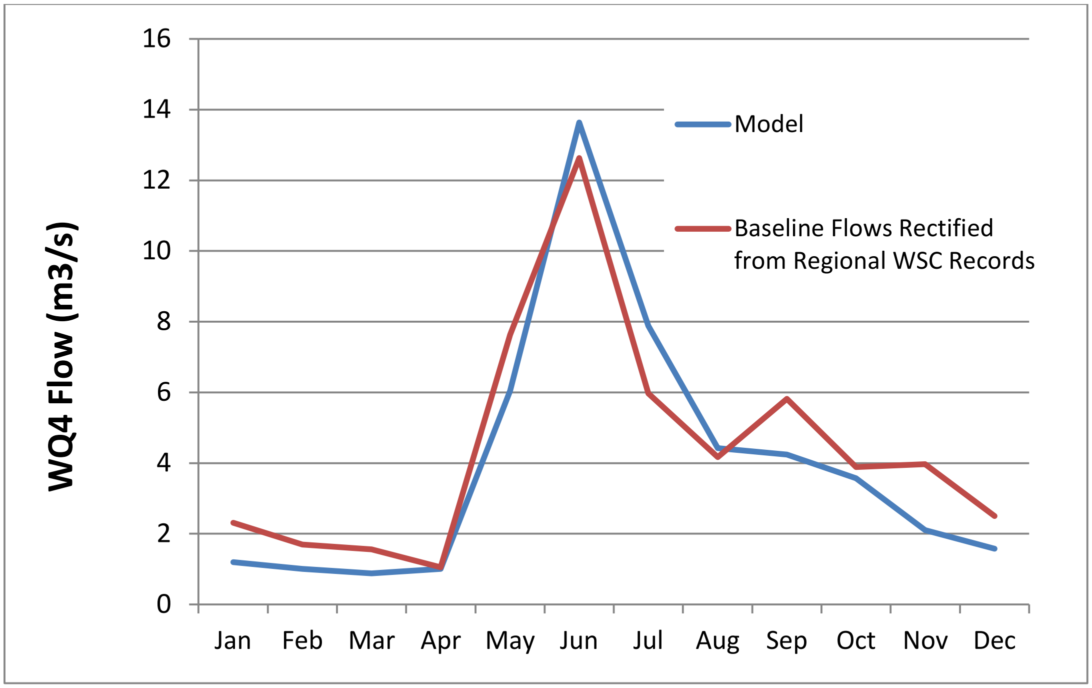


Figure 4-2: **Comparison of model estimated average monthly flows and the baseline observations at the Silvertip Creek WQ4**

Validation of the model performance on the concentration predictions were evaluated through correlation analyses of the monthly average baseline data and the model results at WQ8 on the Silvertip Creek. As presented in Table 4-1, the correlation coefficients for most water quality parameters are greater than 0.95, indicating the model has been well calibrated for the water quality predictions at WQ8. The exception to this is for antimony (Sb) with a correlation coefficient of R^2 = 0.46. This was investigated through comparing the model prediction to baseline Sb concentrations, as plotted on Figure 4-3.

Table 4-1:
Correlation analyses on the observed and model predicted monthly concentrations at the Silvertip Creek WQ8 and WQ25 stations.

Water Quality Parameters	R^2 (WQ8)	R^2 (WQ25)
Alkalinity, Total (as $CaCO_3$)	0.99	0.95
Nitrate (as N)	0.99	0.35
Nitrite (as N)	0.99	0.99
Ammonia (as N)	0.99	0.30
Sulfate (SO_4)	0.99	0.98
Fluoride (F)	0.99	0.84
Chloride (Cl)	0.99	0.97
Silver (Ag)-Total	0.99	0.82
Aluminum (Al)-Dissolved	0.99	0.97
Arsenic (As)-Total	0.99	0.78
Cadmium (Cd)-Total	0.99	0.25
Cobalt (Co)-Total	0.96	0.74
Copper (Cu)-Total	0.97	0.79
Iron (Fe)-Total	0.99	0.76
Potassium (K)-Total	0.99	0.93
Magnesium (Mg)-Total	0.99	0.97
Manganese (Mn)-Total	0.99	0.59
Molybdenum (Mo)-Total	0.99	0.37
Sodium (Na)-Total	0.97	0.90
Nickel (Ni)-Total	0.99	0.52
Lead (Pb)-Total	0.97	0.82
Antimony (Sb)-Total	0.46	0.27
Selenium (Se)-Total	0.97	0.81
Zinc (Zn)-Total	0.97	0.50

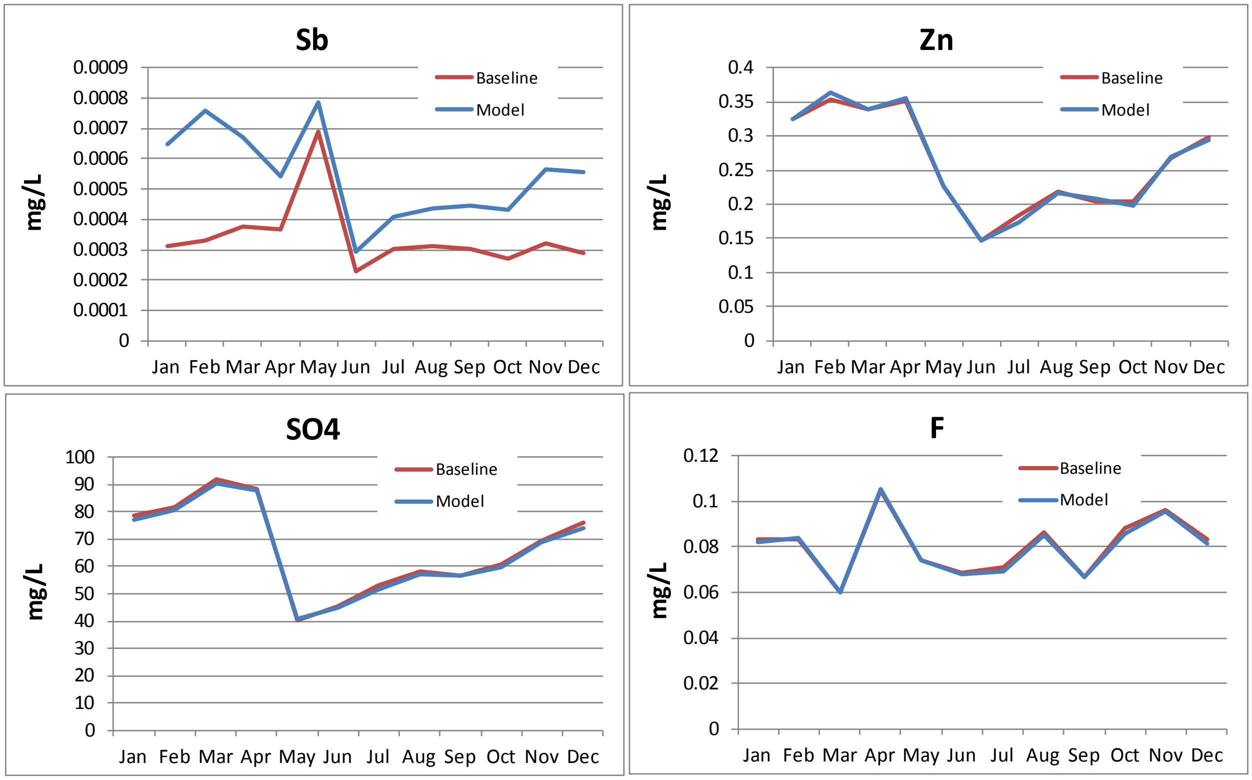


Figure 4-3: Comparison of observed and model predicted concentrations of sulphate, fluoride, antimony, and zinc at WQ8 on Silvertip Creek

It can be seen that the model tends to overestimate the monthly Sb concentration and therefor provides conservative predictions. For the comparison purpose, also presented in Figure 4-3 are the modeled and baseline concentrations at WQ8 for other three water quality parameters (*i.e.*, sulphate, fluoride, and zinc). Of note, sulphate and fluoride are chemically conservative tracers of contaminant loadings to the receiving environment, and zinc is the primary contaminant of concern for the project.

The model performance on the water quality prediction was further examined by comparing the model results to the baseline concentrations at the downstream station WQ25 on Silvertip Creek (upstream of the confluence to Tootsee River) (Figure 4-4). Similar to WQ8, Figure 4-4 shows an acceptable or conservative model prediction on the WQ25 water quality concentrations. The correlation analyses on the model predictions and baseline data at WQ25 (as also presented in Table 4-1) demonstrate a relatively poor model performance at WQ25 as compared to WQ8, but most of correlation coefficients are still high (*e.g.*, greater than or close to 0.80).

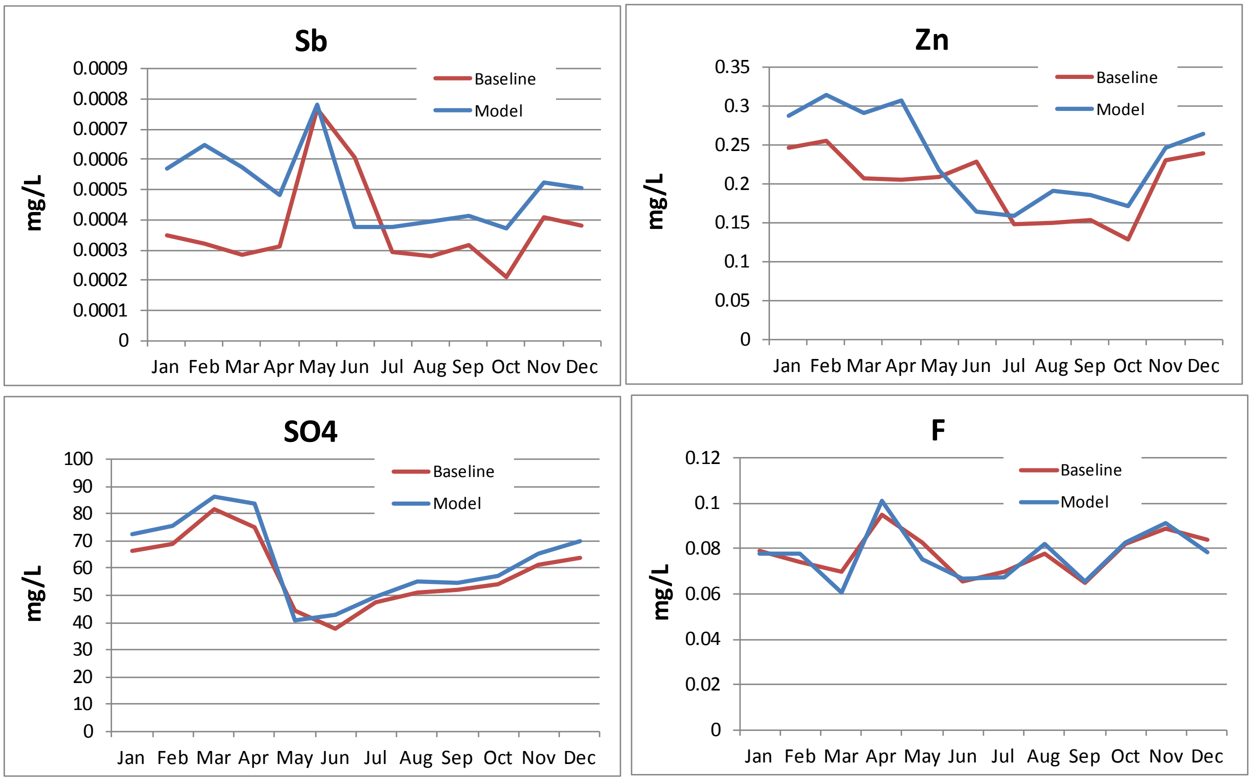


Figure 4-4: Comparison of observed and model predicted concentrations of sulphate, fluoride, antimony, and zinc at WQ25 on Silvertip Creek

In summary, a runoff coefficient of approximately 0.37 for the Silvertip Creek catchment was achieved and the model validation exercise has demonstrated that the developed water quality model is sufficiently accurate or conservative for Silvertip Creek water quality predictions.

4.2 Model Predictions for Operations and Closure

All contact water from the Silvertip Mine project will report to the MWTP and includes:

- TMF seepage and contact runoff;
- RST seepage and contact runoff;
- Mine site runoff from the processing plant area;
- Paste and PAG stockpiles;
- Process water (when mill is operating); and
- Portal water (when mill is shut down or early closure has commenced).

The combined water balance and mass loading Goldsim® water quality model results for the base case and upper case loading scenarios and are presented for key parameters of interest below and the full model output is presented in Appendix E. The model results are presented for conditions assuming:

- no water treatment;
- treatment only to the EQS; and
- expected performance from the MWTP.

The model assumes that the cessation of operations occurs in December 2033 and that the MWTP continues to operate during the early stages of reclamation and closure treating contact water from the TMF, RSF, mine site and portal discharge. Treatment using the MWTP is assumed in the model to continue for another 10 years until December 2043. Following decommissioning of the MWTP and completion of closure activities on the TMF, RSF, mine site and complete flooding of the underground workings, the Main Collection Pond and Sedimentation Pond will be converted into wetland treatment ponds. These facilities will be filled with organics from lower Silvertip Creek area and placed in the ponds to provide passive treatment of Zn and Cd from primarily the portal discharge but also a small amount of seepage flow from the TMF.

Provided below are detailed discussions of model results for key parameters of interest namely sulphate, nitrite, cadmium and zinc. As previously mentioned, full model output for all scenarios and all parameters is provided in Appendix E.

4.2.1 Sulphate

Figure 4-5 and Figure 4-6 present the predicted sulphate concentrations at WQ8 assuming base case and upper case source loading assumptions, respectively. In the absence of any gypsum control treatment of sulphate in the MWTP during operations, sulphate concentrations would be expected to approach values of approximately 430 mg/L in Silvertip Creek at WQ8 under base case loading assumptions. Peak values assuming upper case loading values are on the order of 450 mg/L in the absence of any gypsum control in the treatment system. Assuming sulphate concentrations are controlled in the treatment system discharge at approximately 1800 mg/L results in predicted sulphate concentrations in Silvertip Creek that are below BC MOE guidelines for sulphate considering background hardness values (Figure 4-7).

During the early closure and closure periods, sulphate concentrations in Silvertip Creek at WQ8 are not predicted to exceed 120 mg/L for the base case. For the upper case loading assumptions, sulphate concentrations at WQ8 are not anticipated to exceed 140 mg/L. Thus under both source term loading assumptions, sulphate concentrations at WQ8 are not predicted to exceed the BC MOE guideline for hardness based sulphate at any time during the operations or closure.

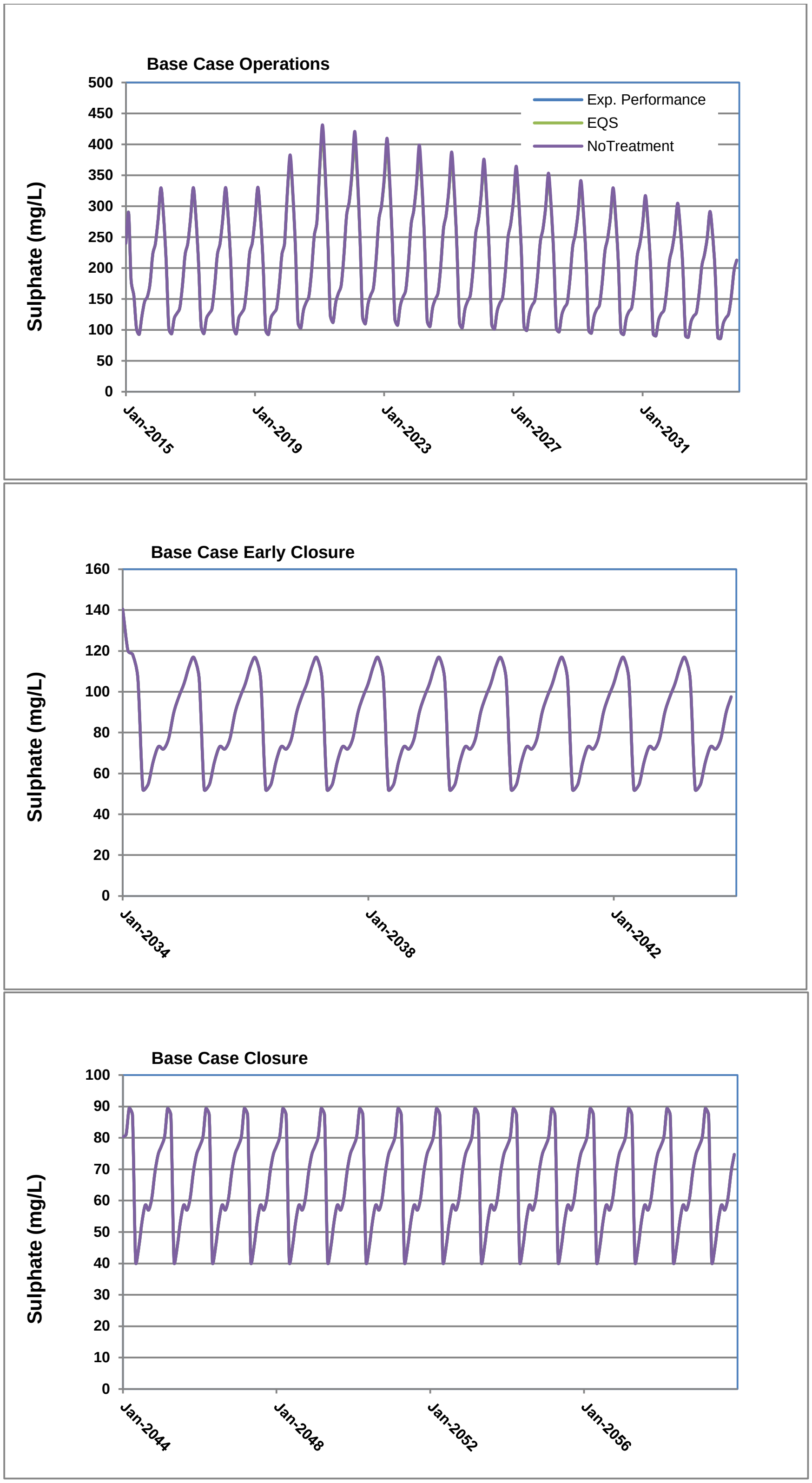


Figure 4-5: **Model predicted sulphate concentrations at WQ8 for operations, early closure and closure – Base Case source loading assumptions**

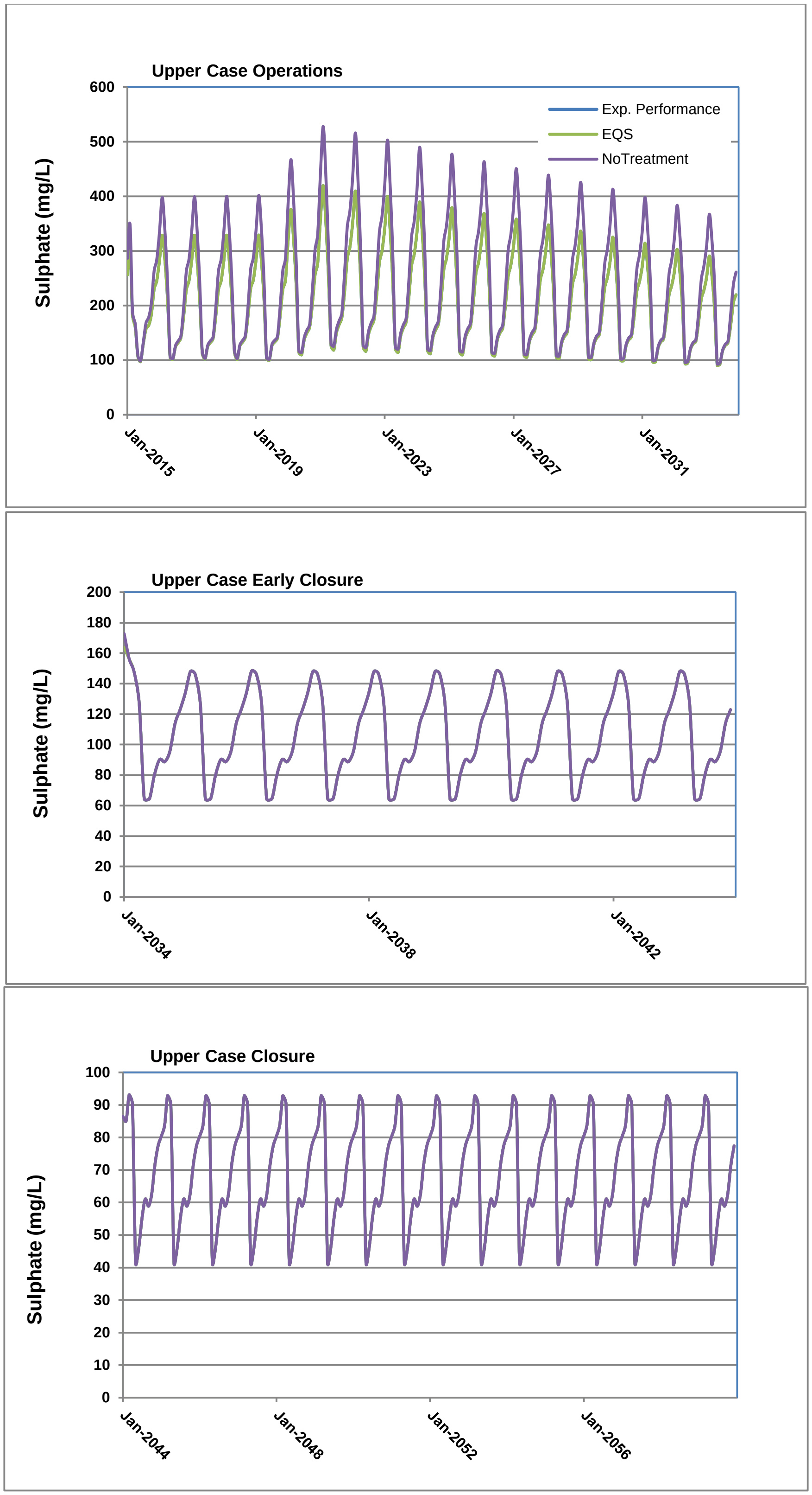


Figure 4-6: **Model predicted sulphate concentrations at WQ8 for operations, early closure and closure – Upper Case source loading assumptions**

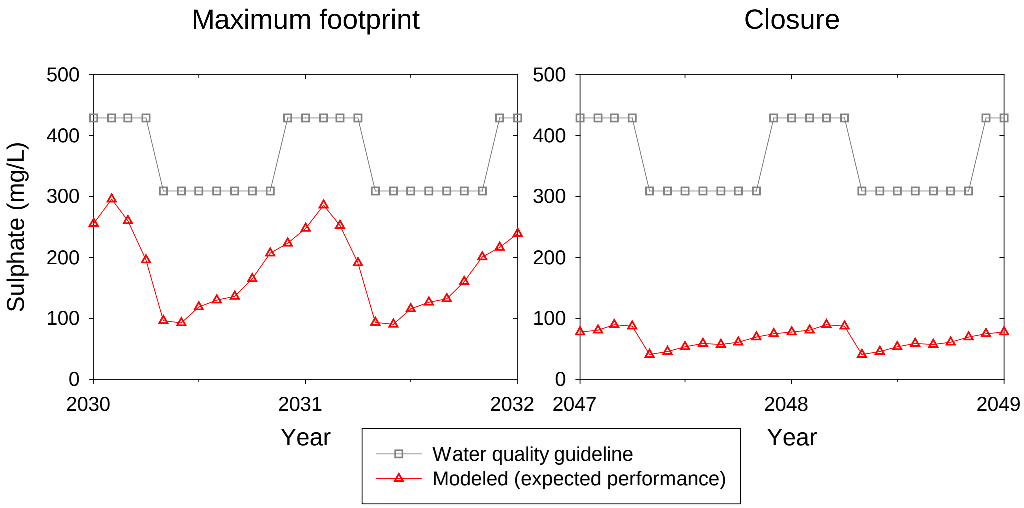


Figure 4-7: Detail of modeled sulphate concentration in Silvertip Creek at WQ8 for expected treatment efficiency, compared to the BC water quality guideline based on background hardness at WQ8

4.2.2 Nitrite

Nitrite is the only nitrogen species modeled that is predicted to exceed the BC MOE benchmark of 0.02 mg/L NO_2-N. Figure 4-8 and Figure 4-9 present the predicted results for nitrite-N for operations and closure for the base case and upper case source loading conditions, respectively. As described in Appendix C, nitrogen loading terms for the portal discharge, which represents the most significant source loading term at Silvertip, were based on analogue data from underground sump water quality data from the Wolverine Mine, Yukon. Data from Wolverine is representative of higher mining rates (approximately 1000 to 1400 tpd) and lower groundwater inflow rates than anticipated at Silvertip. Control of nitrogen species at mining operations is primarily an explosives management issue rather than a geological/geochemical issue associated with ore and waste extraction. These considerations notwithstanding, the results indicate that careful attention will need to be afforded to explosives management at Silvertip. Presently, the MWTP does not have a specific nitrogen removal component, other than the limited potential of nitrite oxidation to nitrate. However, residual nitrite-N concentrations following oxidation are not anticipated to be below 0.4 mg/L and a higher than the EQS benchmark of 0.09 mg/L. Conversely, nitrate and ammonia-N concentrations are not predicted to exceed benchmark concentrations for these parameters in Silvertip Creek at WQ8 (Appendix E).

After operations cease, nitrite concentrations at WQ8 at closure for the base case and upper case loading conditions are predicted to be well below the nitrite-N benchmark of 0.02 mg/L.

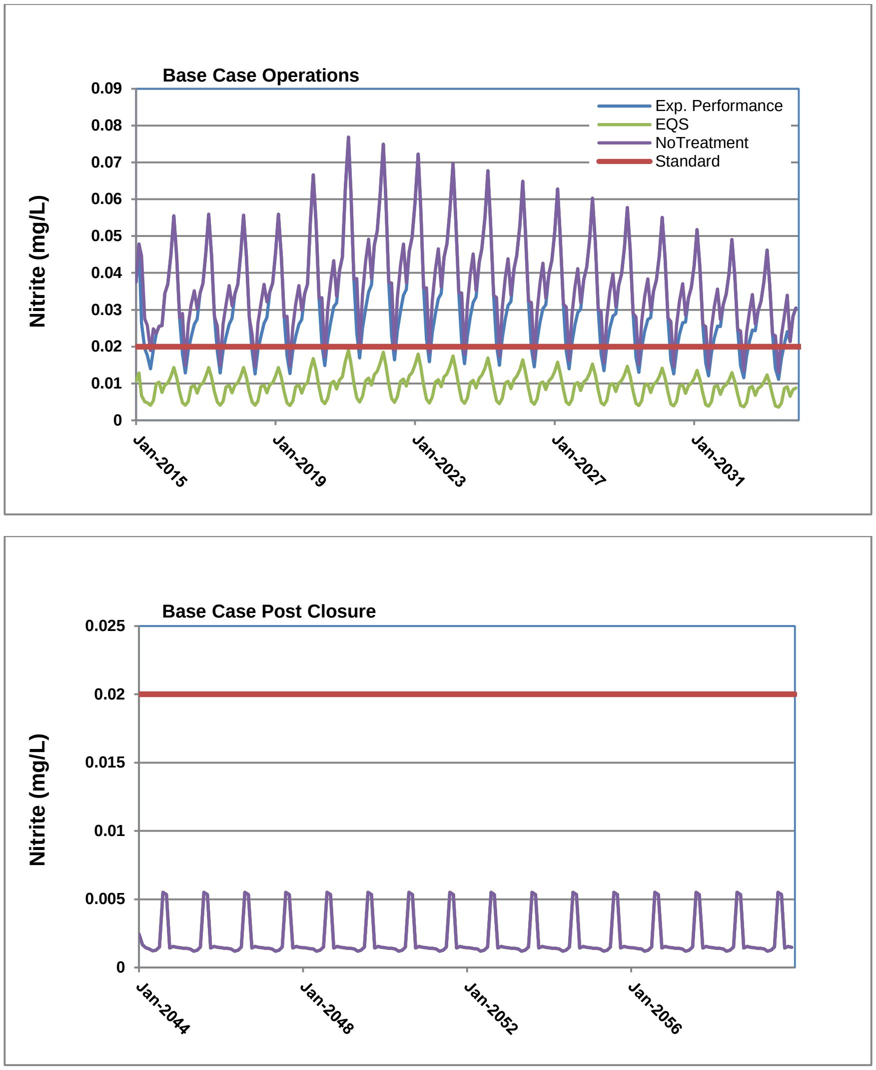


Figure 4-8: Model predicted nitrite concentrations at WQ8 for operations and closure – Base Case source loading assumptions

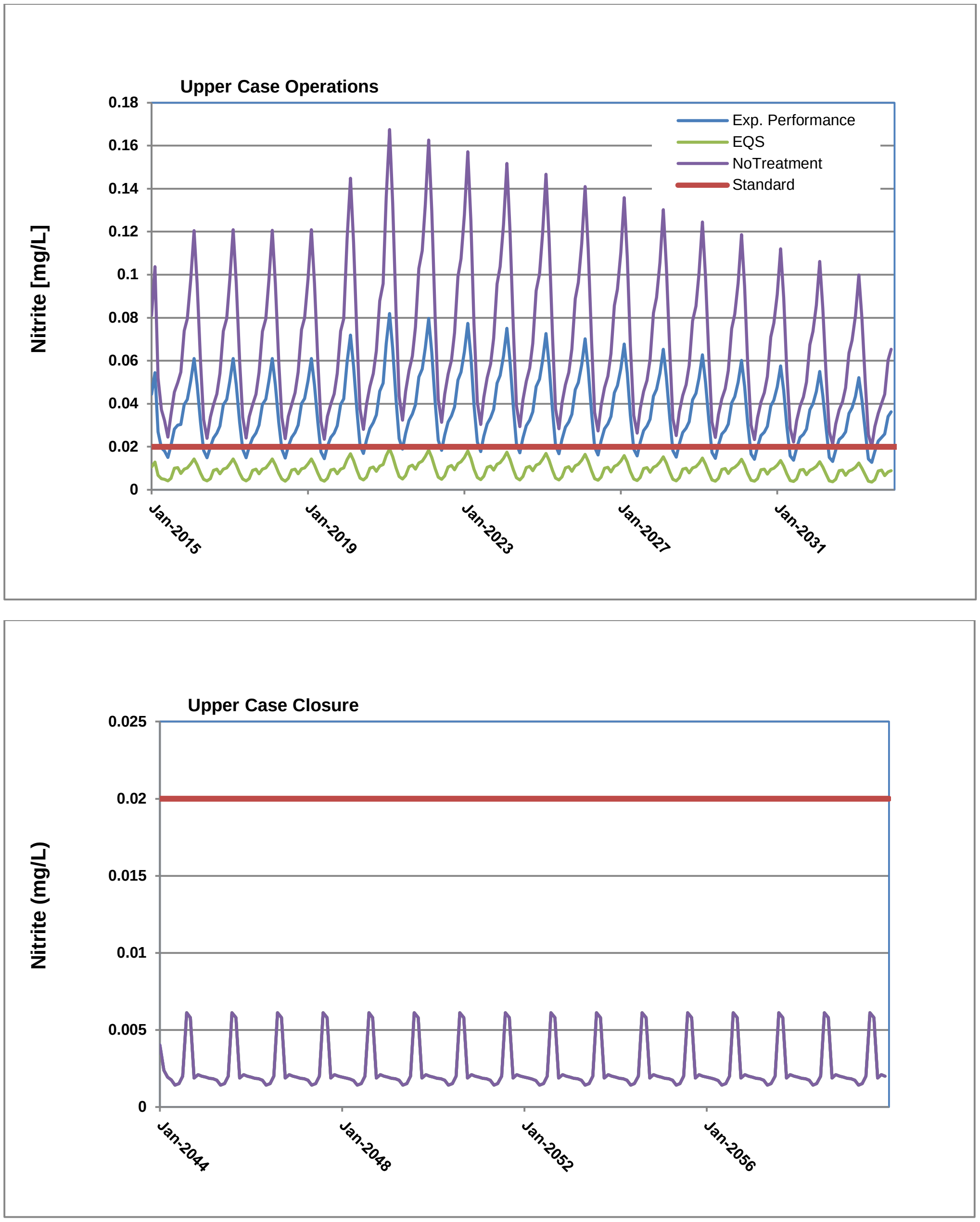


Figure 4-9: **Model predicted nitrite concentrations at WQ8 for operations and closure – Upper Case source loading assumptions**

4.2.3 Cadmium

Cadmium is a contaminant of concern in Silvertip Creek owing to background concentrations in Silvertip Creek at WQ8 exceeding the current working BC MOE hardness based Cd guideline. For Cd, a non-degradation approach was therefore adopted. As described in Section 3.4.2, a benchmark Cd concentration of 0.0013 mg/L was developed that represents the maximum monthly mean Cd concentration plus 10%. Figure 4-10 and Figure 4-11 illustrate the predicted

Cd concentrations at WQ8 for the base case and upper case source loading condition, respectively.

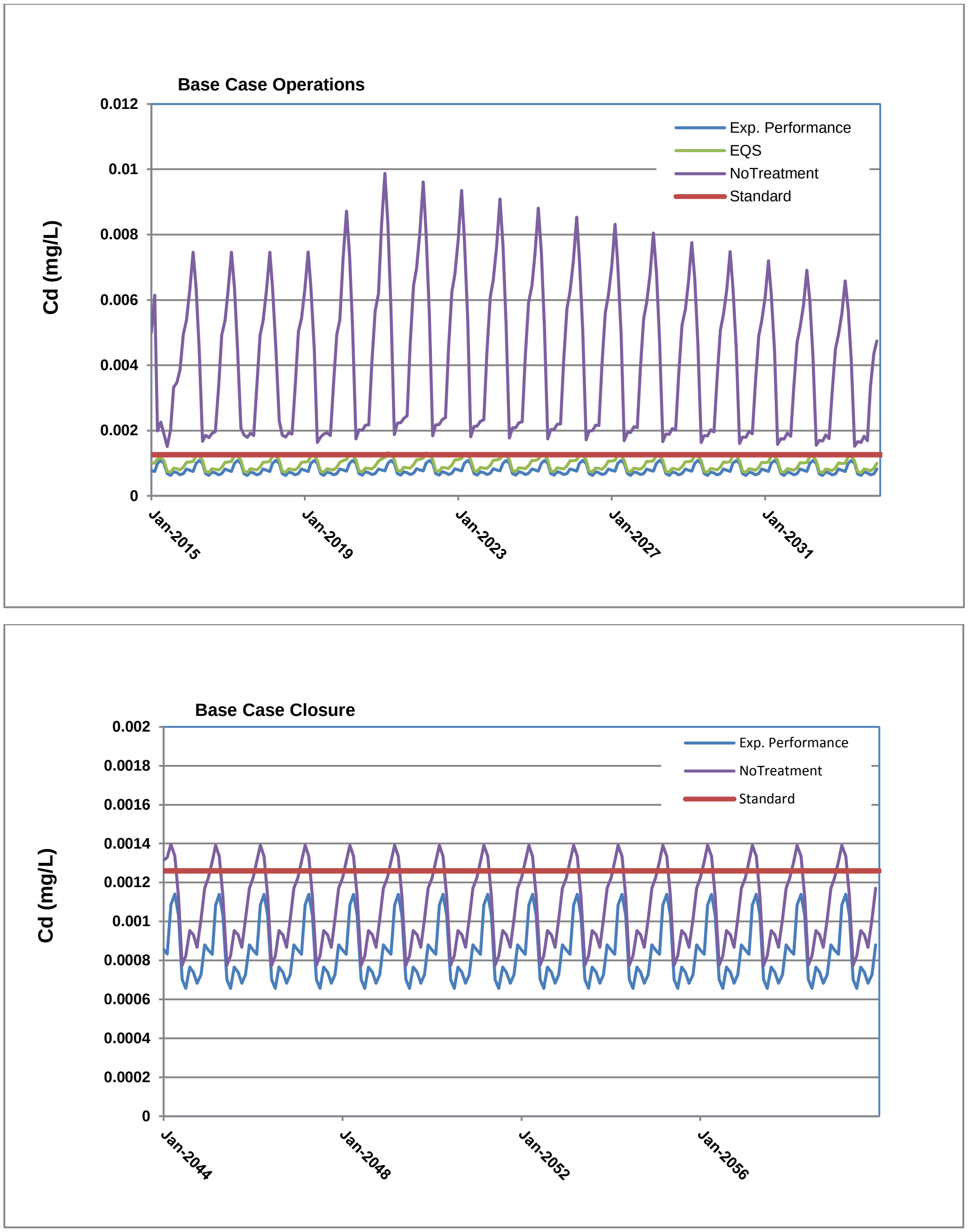
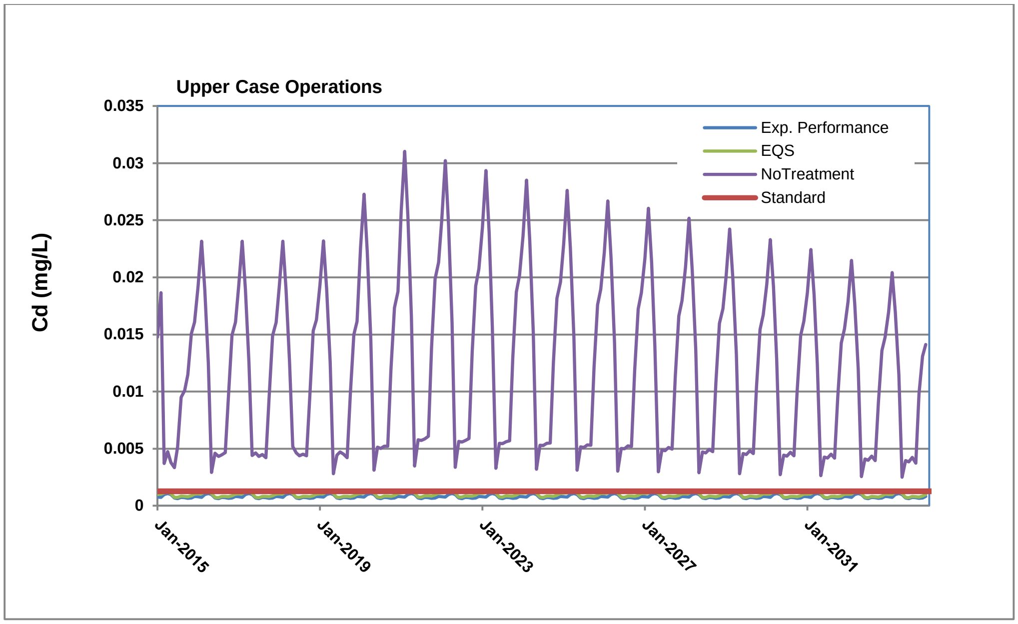


Figure 4-10: Model predicted cadmium concentrations at WQ8 for operations and closure – Base Case source loading assumptions



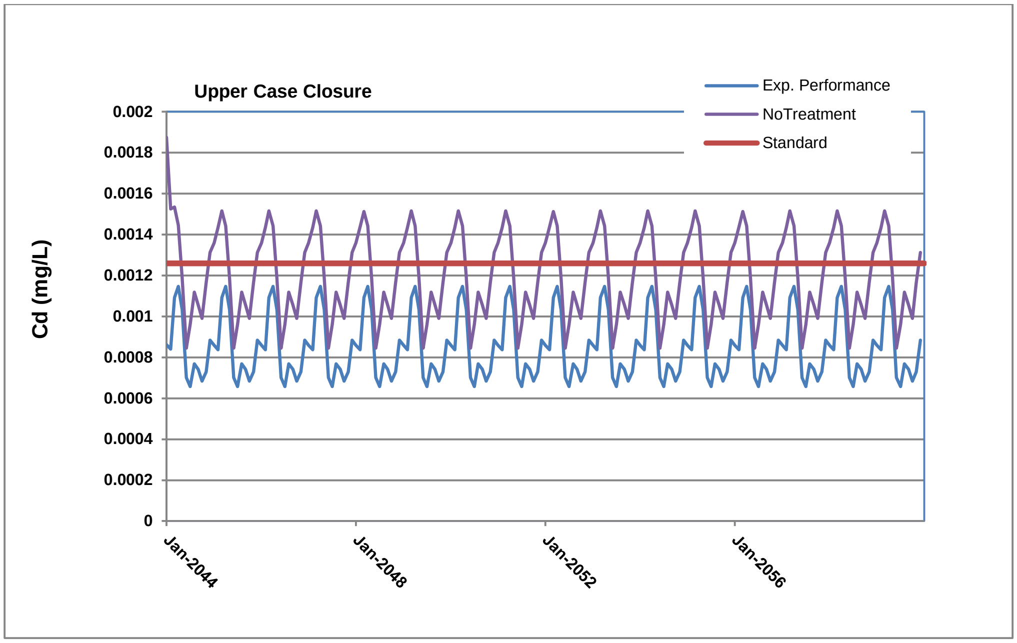


Figure 4-11: Model predicted cadmium concentrations at WQ8 for operations and closure – Upper Case source loading assumptions

In the absence of treatment for Cd, predicted concentrations for the base case condition approach 0.01 mg/L and greatly exceed the benchmark background concentration of 0.0013 mg/L. However, the MWTP is focused on treating Cd and predicted concentrations following treatment are predicted to remain well below the proposed benchmark value (Figure 4-10 and Figure 4-11). For the closure condition, passive Cd treatment is assumed to occur in the wetlands constructed within the Main Collection Pond and Main Settling Pond. Assuming 30% porosity within the organic backfilled ponds, residence times for passive treatment range from roughly 2 days during freshet to upwards of 17 days during winter low flows (Table 4-2).

Table 4-2:
Residence time and required Cd reductions through the Main Collection Pond and Main Settling Pond wetland closure treatment

Month	Water Balance					Cd Water Quality					Required Cd Reduction
	MineSite (m3/day)	Portal (m3/day)	TMF (m3/day)	Total (m3/day)	Residence Time (days) 0.3 porosity	MineSite (mg/L)	Portal (mg/L)	TMF (mg/L)	Total (mg/L)	Discharge Limit (mg/L)	
Jan	11	216	41	269	9	0.0001	0.0128	0.0064	0.0113	0.003	73%
Feb	9	234	34	278	8	0.0001	0.0130	0.0061	0.0117	0.003	74%
Mar	8	152	30	190	12	0.0001	0.0115	0.0059	0.0101	0.003	70%
Apr	9	96	33	138	17	0.0001	0.0126	0.0052	0.0100	0.003	70%
May	57	329	180	567	4	0.0002	0.0118	0.0033	0.0079	0.003	62%
Jun	128	472	416	1017	2	0.0002	0.0133	0.0039	0.0078	0.003	61%
Jul	74	415	272	761	3	0.0002	0.0168	0.0065	0.0115	0.003	74%
Aug	42	279	160	481	5	0.0002	0.0154	0.0074	0.0114	0.003	74%
Sep	40	310	143	493	5	0.0002	0.0149	0.0060	0.0111	0.003	73%
Oct	34	322	120	475	5	0.0002	0.0123	0.0060	0.0098	0.003	70%
Nov	20	271	75	366	6	0.0002	0.0130	0.0071	0.0111	0.003	73%
Dec	15	220	55	290	8	0.0002	0.0126	0.0066	0.0108	0.003	72%

Based on the water balance and expected Cd concentrations entering the wetland system and the discharge limit of 0.003 mg/L, required reduction factors of between 60% and 74% would be needed in order to achieve discharge concentrations of 0.003 mg/L or below.

Figure 4-12 and Figure 4-13 provide an important illustration of the Cd predicted concentrations at WQ8 during operations and closure relative to current concentrations at WQ8 (measured concentrations that include current loadings from the portal) and existing background conditions at WQ8 (calculated concentration minus existing portal loadings). Closure predictions assume that Cd concentrations in the discharge to Silvertip Creek do not exceed 0.003 mg/L. As shown, the operations and closure model predictions track very closely to existing background concentrations for Cd at WQ8 and indicate that the proposed mining operation does not degrade existing water quality with respect to Cd in Silvertip Creek.

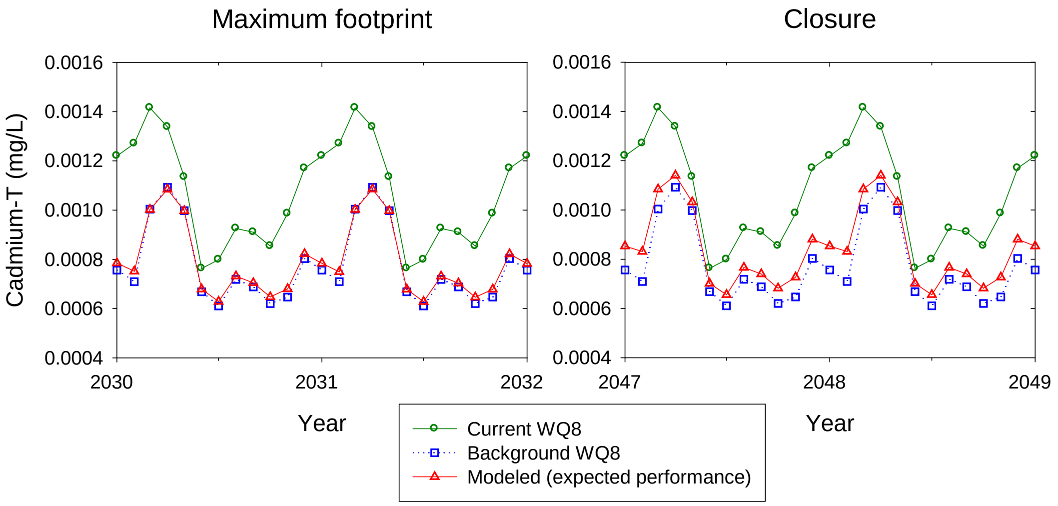


Figure 4-12: Detail of modeled total cadmium concentration in Silvertip Creek for expected treatment efficiency, compared to the current (2010-2012) monthly mean concentration at WQ8 and the background (pre-development) concentration at WQ8

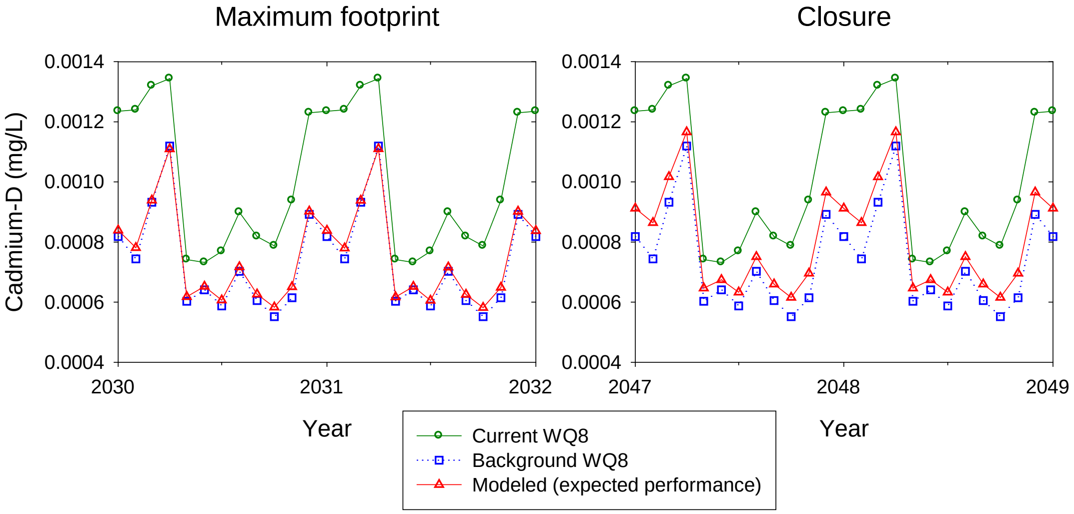


Figure 4-13: Detail of modeled dissolved cadmium concentration in Silvertip Creek for expected treatment efficiency, compared to the current (2010-2012) monthly mean concentration at WQ8 and the background (pre-development) concentration at WQ8

4.2.4 Zinc

Similar to cadmium, zinc is a contaminant of concern in Silvertip Creek owing to background concentrations in Silvertip Creek at WQ8 exceeding the current working BC MOE hardness based Zn guideline. For Zn, a non-degradation approach was therefore adopted. As described in Section 3.4.2, a benchmark Zn concentration of 0.332 mg/L was developed that represents the maximum monthly mean Zn concentration plus 10%. Figure 4-14 and Figure 4-15 illustrate the predicted Zn concentrations at WQ8 for the base case and upper case source loading condition, respectively.

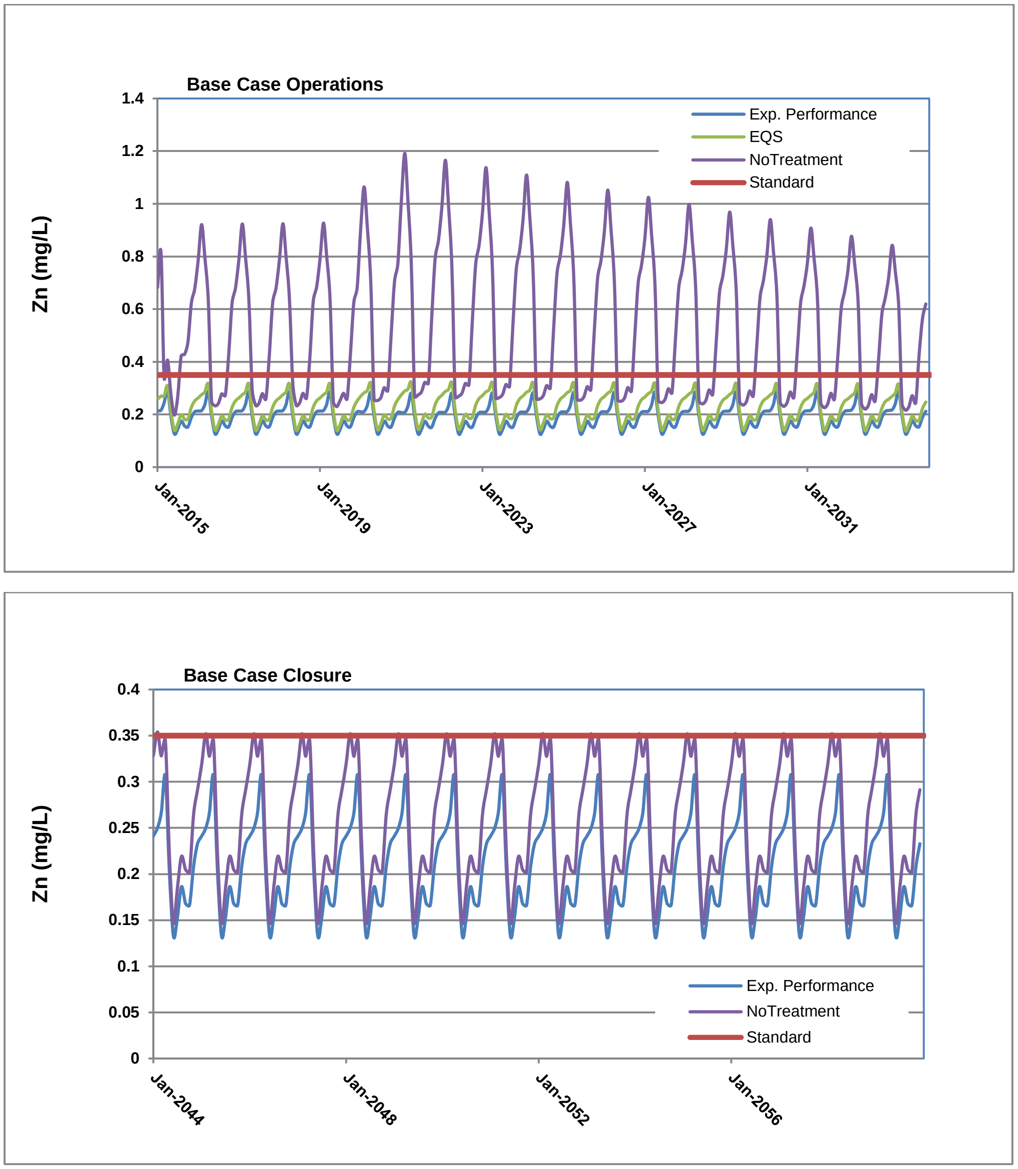
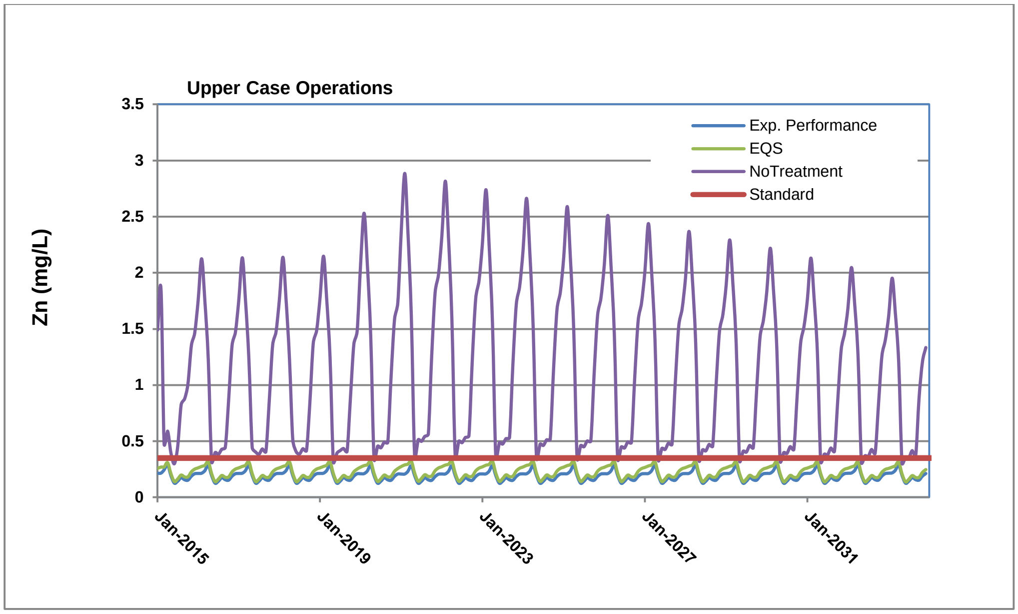


Figure 4-14: Model predicted zinc concentrations at WQ8 for operations and closure – Base Case source loading assumptions



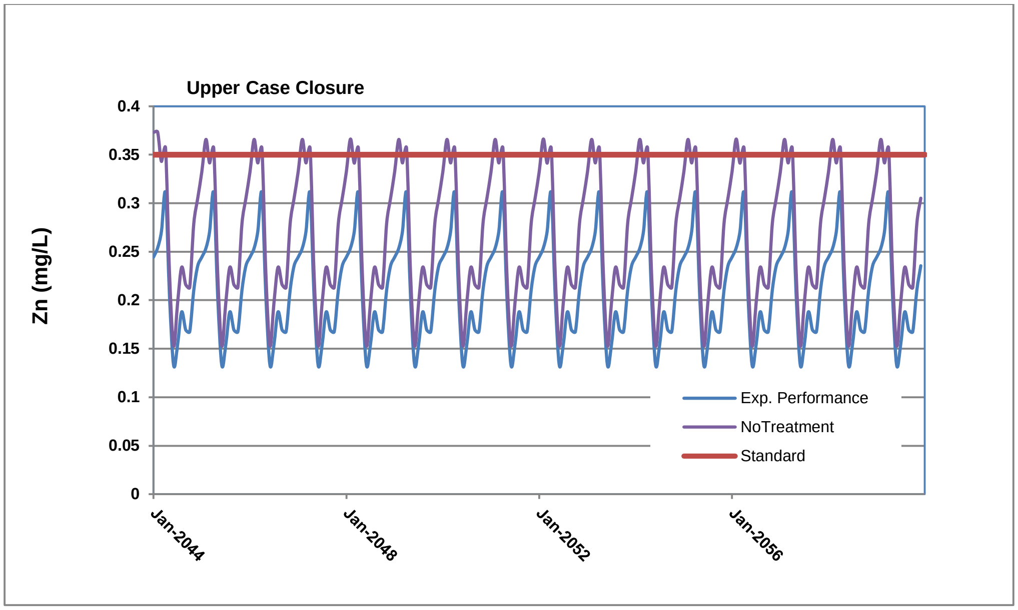


Figure 4-15: Model predicted zinc concentrations at WQ8 for operations and closure – Upper Case source loading assumptions

In each scenario following treatment, Zn concentrations at WQ8 never exceed the proposed non-degradation benchmark concentration of 0.33 mg/L Zn. As described for Cd, the Zn concentrations at closure are assumed to be attenuated in wetland cells constructed in the Main Collection Pond and Main Settling Pond. Based on the water balance and expected Zn concentrations entering the wetland system and the discharge limit of 0.5 mg/L, required reduction factors of between 65% and 79% would be needed in order to achieve discharge concentrations of 0.5 mg/L or below (Table 4-3).

Table 4-3:
Residence time and required Zn reductions through the Main Collection Pond and Main Settling Pond wetland closure treatment

Month	Water Balance					Zn Water Quality					Required Zn Reduction
	MineSite (m3/day)	Portal (m3/day)	TMF (m3/day)	Total (m3/day)	Residence Time (days) 0.3 porosity	MineSite (mg/L)	Portal (mg/L)	TMF (mg/L)	Total (mg/L)	Discharge Limit (mg/L)	
Jan	11	216	41	269	9	0.031	2.65	0.66	2.23	0.5	78%
Feb	9	234	34	278	8	0.031	2.72	0.63	2.37	0.5	79%
Mar	8	152	30	190	12	0.031	2.27	0.61	1.91	0.5	74%
Apr	9	96	33	138	17	0.029	2.50	0.53	1.87	0.5	73%
May	57	329	180	567	4	0.038	2.33	0.35	1.47	0.5	66%
Jun	128	472	416	1017	2	0.028	2.71	0.39	1.43	0.5	65%
Jul	74	415	272	761	3	0.019	3.44	0.65	2.11	0.5	76%
Aug	42	279	160	481	5	0.017	2.97	0.74	1.97	0.5	75%
Sep	40	310	143	493	5	0.017	2.98	0.60	2.05	0.5	76%
Oct	34	322	120	475	5	0.021	2.46	0.60	1.82	0.5	72%
Nov	20	271	75	366	6	0.026	2.64	0.72	2.10	0.5	76%
Dec	15	220	55	290	8	0.028	2.57	0.67	2.08	0.5	76%

Figure 4-16 provides an important comparison of the Zn predicted concentrations at WQ8 during operations and closure relative to current concentrations at WQ8 (*e.g.* measured concentrations that include current loadings from the portal) and existing background conditions at WQ8 (*e.g.* calculated concentration minus existing portal loadings). Closure predictions assume that Zn concentrations in the discharge to Silvertip Creek do not exceed 0.5 mg/L As shown, the operations and closure model predictions track very closely to existing background concentrations for Zn at WQ8 and indicate that the proposed mining operation does not degrade existing water quality with respect to Zn in Silvertip Creek.

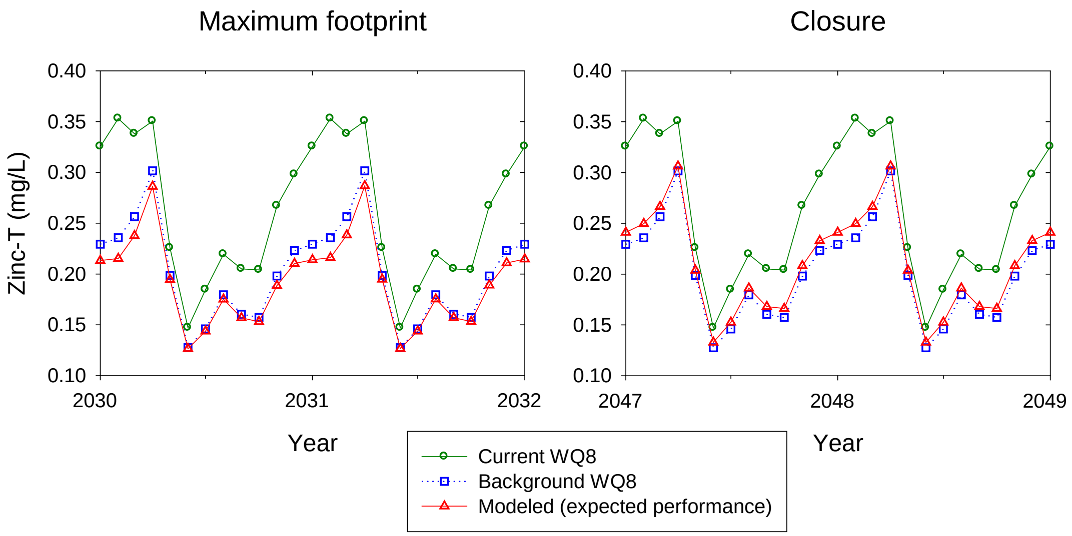


Figure 4-16: **Detail of modeled total zinc concentration in Silvertip Creek for expected treatment efficiency, compared to the current (2010-2012) monthly mean concentration at WQ8 and the background (pre-development) concentration at WQ8**

4.2.5 Other Mine Site Components at Closure

In the description of closure model results above, it has been assumed that, in addition to surface discharge from flooded underground workings, that the TMF seepage and runoff will require collection and discharge to the proposed wetland system in the Main Collection Pond and Main Settling Pond. The primary driver behind this assumption is that the long-term predicted Zn concentrations in the TMF seepage will exceed the MMER limit of 0.5 mg/L. No other parameters exceed MMER limits and therefore Zn is the controlling parameter for this facility. Model results for the TMF seepage at closure are presented in Figure 4-17 and illustrate that seasonal peak Zn concentrations on the order of 0.7 mg/L are anticipated under the mass loading assumptions. As such, all TMF seepage reports to the wetland treatment system prior to release to Silvertip Creek.

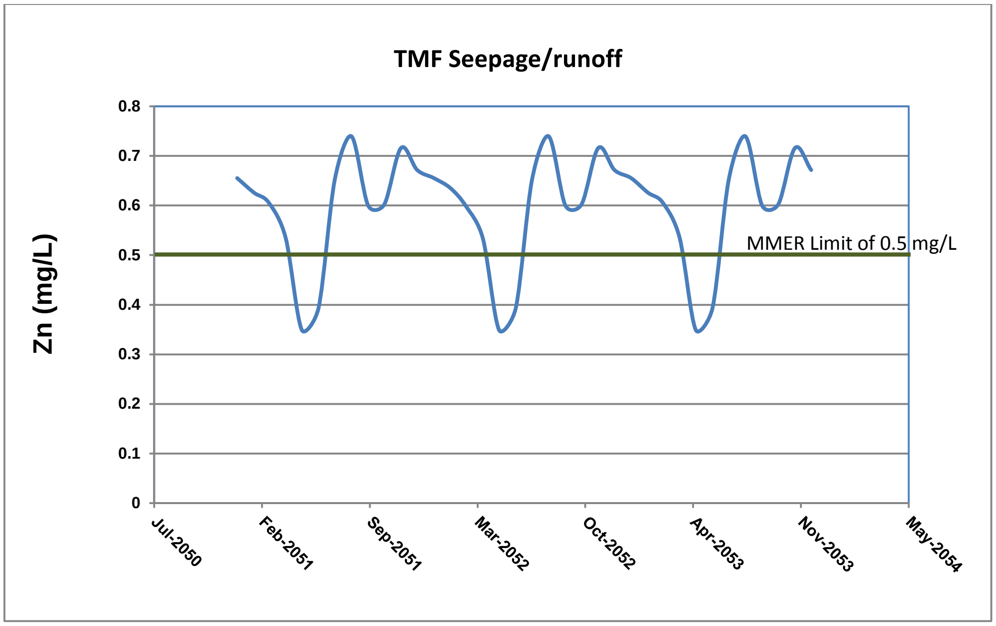


Figure 4-17: Detail of modeled total zinc concentration in TMF seepage/runoff at closure

Conversely, seepage/runoff from the RSF is assumed to be released directly to Silvertip Creek. Model results for the RSF seepage at closure are presented in Figure 4-18 and illustrate that seasonal peak Zn concentrations are not predicted to exceed 0.2 mg/L under the mass loading assumptions. As such, all RSF seepage is assumed to report directly to Silvertip Creek.

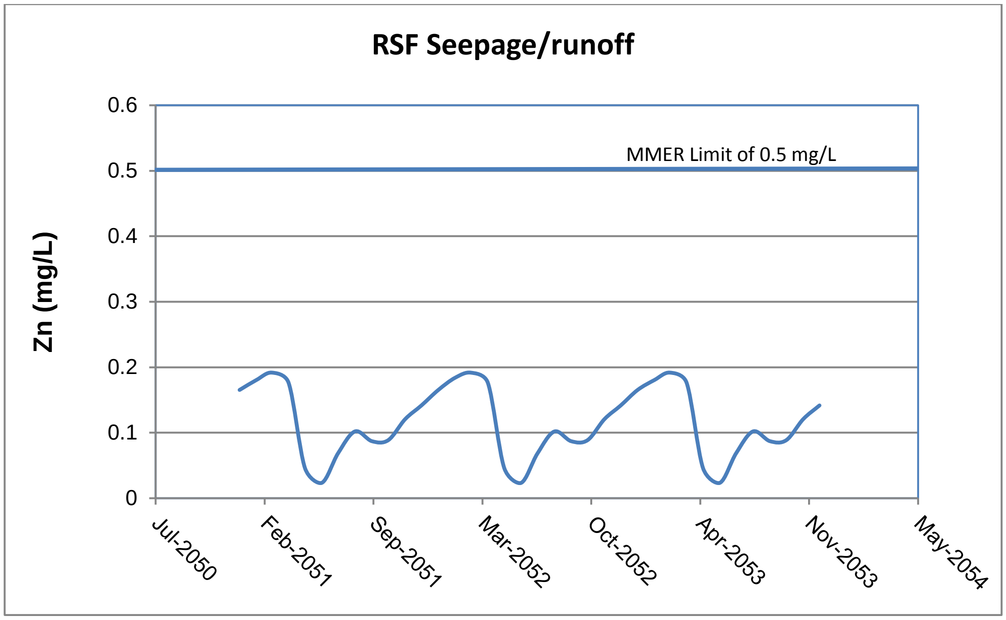


Figure 4-18: Detail of modeled total zinc concentration in RSF seepage/runoff at closure.

5. Summary and Conclusions

5. Summary and Conclusions

Lorax was retained by JDSS to update the water quality predictions in the Silvertip MAPA. Water quality predictions were generated in the MAPA using a Goldsim® mass balance model. Following a screening level review by MEM and MOE, a number of concerns were identified that were not resolvable with the MAPA modeling effort. Lorax indicated that in order to adequately address the concerns of MEM and MOE, the following tasks were required for generating water quality predictions:

- create a new Goldsim model for the Project;
- derive new source term concentrations and flows for input to the model;
- derive new background Silvertip Creek concentration and flows for input to the model.

New model input concentrations for source terms and background water quality were required largely because the material presented in the MAPA and its appendices was deemed to be inappropriate and/or the derivation of specific input flow and chemical terms were not presented in a way that could be reviewed. While the climate data presented in the MAPA was assumed correct, the source term inputs, receiving water quality and flows of background water and contact flows were revised. Accordingly, the updates to these model inputs and the model results presented in this addendum can be considered to supersede those in the MAPA.

Water quality was modeled for the following conditions:

- base case and upper estimates of source term chemistry;
- monthly mean flows;
- no treatment of mine water, treatment of mine water to levels equal to effluent quality standards, and treatment of mine water to levels expected for the proposed high density sludge MWTP; and
- operation and closure phases of the mine life.

Modeled water quality for the no-treatment cases indicated that treatment of mine contact water will be required in order to meet BC water quality guidelines for protection of freshwater aquatic life (BCWQG) in the receiving environment at monitoring site WQ8.

The Goldsim® water quality model was run in a hindcast mode to derive effluent quality standards (EQS) that, if met in the discharge from the mine waste treatment plant (MWTP), would result in water quality in the receiving environment that meets BCWQG. Cadmium and zinc are naturally present in Silvertip Creek at levels that exceed BCWQG, so even if these metals were completely removed by the MWTP the concentration at WQ8 would be above BCWQG. Therefore effluent quality standards for Cd and Zn were determined as levels that would not degrade water quality from background levels. Non-degradation benchmarks were estimated as 10% above the maximum mean monthly background concentration.

The MWTP is expected to treat most parameters to levels lower than EQS (Table 3-56). Water quality for operation phase of the Project, using the expected contaminant levels in effluent from the MWTP, was below BCWQG for all parameters except nitrite, Cd, and Zn. In BC, water quality guidelines are developed to be protective of all species of aquatic life, and all aquatic stages of their life cycle, during indefinite exposure. Therefore, water quality predictions that are below BCWQG indicate that the predicted levels of contaminants will not harm aquatic life in the receiving environment. Predicted cadmium and zinc levels were approximately equal to background concentrations of these parameters at WQ8. Background concentrations were determined as the concentration of contaminants in Silvertip Creek at station WQ8 without any loading from the existing portal discharge. Therefore the results of modeled cases using expected levels of Cd and Zn in treated effluent indicate that concentrations of these parameters will be approximately equal to their concentration prior to mine development.

The maximum modeled nitrite level during operation phase is 0.08 mg/L, which is 4 times higher than the BCWQG for nitrite. The model assumed that the MWTP has no effect on nitrite. Nitrogen parameters, including nitrite, in mine water typically come from residue from explosives. The model results indicate that mitigation of nitrogen species in mine contact water should be addressed through best management practices of explosives in the underground during operations.

The MWTP will be shut down at the closure phase of the Project. Mine contact water will be directed to a constructed wetland, which will be designed to produce a discharge that meets the EQS derived for the MWTP. The Goldsim® model was run to determine the removal efficiency necessary meet these EQS in the water discharged from the wetland. Removal of 61-74% of the estimated Cd levels in water flowing into the wetland, and 65 – 79% removal of Zn, would produce levels in water exiting the wetland that meet EQS, therefore would not degrade Silvertip water quality compared to background levels.

References

References

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MEND report, 1995. Hydrogeology of waste rock dumps, MEND report PA-1, Editors by L. Smith, D.L. Lòpez, R. Beckie, K. Morin, R. Dawson and W. Price

Appendix A
Silvertip Production Summary – Aug. 12, 2014

Appendix A1: Annual Schedule Summary

Appendix A2: Monthly Schedule Summary

JDS SILVER

Appendix A1: Annual Schedule Summary

CALENDAR	UNITS	TOTAL/AVERAGE	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
DAYS PER PERIOD	days	**365**	365	365	365	365	365	365	365	365	365	365	365	365	365	365
PRODUCTION	UNITS	TOTAL/AVERAGE														
ORE TONNES	tonnes	**1,400,000**	6,587	67,413	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000
ACCUMULATIVE TONNES			6,587	74,000	148,000	222,000	296,000	370,000	444,000	518,000	592,000	666,000	740,000	814,000	888,000	962,000
Production Rate (tpd)			212	494	503	503	516	462	500	500	500	500	500	500	500	500
PROCESS PLANT THROUGHPUT																
PROCESS PLANT OPERATING DAYS																
PROCESSED ORE (TONNES)	tonnes	**1,400,000**	0	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000
STOCKPILE (TONNES)	tonnes	**329**	6,587	0	0	0	0	0	0	0	0	0	0	0	0	0
MATERIAL TYPE	UNITS	TOTAL/AVERAGE														
SUMMARY																
ORE TONNES	tonnes	**1,400,000**	6,587	67,413	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000
MUD STONE TONNES (PAG)	tonnes	**38,190**	346	0	0	0	1,941	313	0	750	221	0	1,237	0	22,622	1,782
LIMESTONE TONNES (NAG)	tonnes	**583,418**	24,315	18,636	39,473	18,732	35,149	23,545	14,730	44,096	22,448	12,418	48,712	23,296	34,257	52,727
TOTAL TONNES		**2,021,607**	31,249	86,048	113,473	92,731	111,090	97,858	88,730	118,846	96,669	86,418	123,949	97,296	130,879	128,509
BACKFILL SCHEDULE																
BACKFILL SUMMARY																
CEMENTED ROCKFILL PLACED UNDERGROUND	cu.m	**7,730**	1810.254	5919.471	0	0	0	0	0	0	0	0	0	0	0	0
PASTE BACKFILL PLACED UNDERGROUND	cu.m		0	46502.69	55834.26	52018.63	61675.56	46250.38	44527.37	44527.37	44527.37	44527.37	45055.44	45413.64	53822.73	48128.08
PYRITE TAILS USED FOR PASTE FILL	cu.m		0	27052.76	24363.5	25967.31	29730.68	29188.99	29486.09	25990.15	28027.6	28781.55	29730.68	29730.68	29730.68	29730.68
PYRITE TAILS SURFACE STOCKPILE (CUMMULATIVE)	cu.m	**18,114**	0	3,532	6,042	4,196	1,211	3,134	0	0	0	0	0	0	0	0
DRYSTACK TAILS SURFACE STOCKPILE (CUMMULATIVE)	cu.m		0	11,024	14,391	20,319	22,332	25,653	36,503	48,292	61,820	74,335	85,946	96,727	99,098	107,164
MINE WASTE BROUGHT TO SURFACE	cu.m		8478.19	5301.31	14619.7	6937.681	13736.91	6424.828	5455.584	16609.72	8395.785	4599.409	18499.48	8628.076	21066.43	20188.66
MINE WASTE BROUGHT BACK UNDERGROUND (BACKFILL)	cu.m		0	409	409	0	0	0	0	0	0	0	0	0	0	0
MINE WASTE SURFACE STOCKPILE	cu.m		28,058	42,673	82,146	100,878	137,968	152,861	167,591	212,437	235,106	247,524	297,473	320,769	377,648	432,158
CEMENT CONSUMED REQUIRED	cu.m	**1,224**	33	612	823	639	909	855	652	667	692	512	842	839	639	761
SHOTCRETE CONSUMED	cu.m	**3,029**	256	1871	2242	2247	3029	2708	2422	2259	2140	2161	2329	2364	2651	2386
DEVELOPMENT METRES	UNITS	TOTAL/AVERAGE														
SUMMARY																
REHAB	metres	**1,925**	442	0	219	125	0	0	0	0	345	0	0	0	794	0
RAMP	metres	**3,771**	136	262	264	20	92	0	0	339	0	0	380	70	436	264
LEVEL	metres	**977**	24	41	136	9	0	0	0	68	5	0	39	46	90	57
SUB LEVEL	metres	**6,973**	204	52	267	209	200	220	191	442	399	280	586	388	249	794
RAISE	metres	**725**	13	0	18	12	0	0	0	64	0	0	96	9	232	35
PRODUCTION	metres	**24,439**	110	1,105	1,301	1,332	1,805	1,617	1,446	1,314	1,247	1,285	1,347	1,396	1,508	1,377
TOTAL METRES		**38,811**	930	1,461	2,204	1,706	2,098	1,837	1,637	2,227	1,996	1,565	2,448	1,908	3,309	2,528
EQUIPMENT REQUIREMENTS	UNITS	TOTAL/AVERAGE														
JUMBOS REQUIRED	units	**3**	2	2	3	3	3	3	3	3	3	3	3	3	3	3
SCISSOR LIFTS REQUIRED	units	**5**	4	5	5	5	5	5	5	5	5	5	5	5	5	5
TRUCKS REQUIRED	units	**3**	2	2	2	3	3	3	3	3	3	3	3	3	3	3
SCOOPS REQUIRED	units	**2**	1	2	2	2	2	2	2	2	2	2	2	2	2	2

Appendix A1: Annual Schedule Summary

CALENDAR	UNITS	2028	2029	2030	2031	2032	2033
DAYS PER PERIOD	days	365	365	365	365	365	365
PRODUCTION	**UNITS**						
ORE TONNES	tonnes	74,000	74,000	74,000	74,000	74,000	68,000
ACCUMULATIVE TONNES		1,036,000	1,110,000	1,184,000	1,258,000	1,332,000	1,400,000
Production Rate (tpd)		500	500	500	500	500	500
PROCESS PLANT THROUGHPUT							
PROCESS PLANT OPERATING DAYS							
PROCESSED ORE (TONNES)	tonnes	74,000	74,000	74,000	74,000	74,000	68,000
STOCKPILE (TONNES)	tonnes	0	0	0	0	0	0
MATERIAL TYPE	**UNITS**						
SUMMARY							
ORE TONNES	tonnes	74,000	74,000	74,000	74,000	74,000	68,000
MUD STONE TONNES (PAG)	tonnes	0	0	8,979	0	0	0
LIMESTONE TONNES (NAG)	tonnes	61,640	58,725	6,325	15,140	15,140	13,912
TOTAL TONNES		135,640	132,725	89,304	89,140	89,140	81,912
BACKFILL SCHEDULE							
BACKFILL SUMMARY							
CEMENTED ROCKFILL PLACED UNDERGROUND	cu.m	0	0	0	0	0	0
PASTE BACKFILL PLACED UNDERGROUND	cu.m	44527.3699	48942.058	49940.083	48573.5009	44527.3699	44723.3879
PYRITE TAILS USED FOR PASTE FILL	cu.m	26376.2966	29730.684	29730.684	29730.6843	29005.6609	27320.1359
PYRITE TAILS SURFACE STOCKPILE (CUMMULATIVE)	cu.m	0	0	0	0	0	0
DRYSTACK TAILS SURFACE STOCKPILE (CUMMULATIVE)	cu.m	118,831	127,753	134,007	141,628	153,295	160,570
MINE WASTE BROUGHT TO SURFACE	cu.m	22829.8089	21750.103	5668.3058	5607.25114	5607.25114	5152.60916
MINE WASTE BROUGHT BACK UNDERGROUND (BACKFILL)	cu.m	0	0	0	0	0	0
MINE WASTE SURFACE STOCKPILE	cu.m	493,798	552,523	567,828	582,967	598,107	612,019
CEMENT CONSUMED REQUIRED	cu.m	617	798	830	1224	1196	1126
SHOTCRETE CONSUMED	cu.m	2251	2263	2289	1475	1475	1026
DEVELOPMENT METRES	**UNITS**						
SUMMARY							
REHAB	metres	0	0	0	0	0	0
RAMP	metres	494	346	127	220	220	100
LEVEL	metres	149	120	53	60	60	20
SUB LEVEL	metres	590	729	113	430	430	200
RAISE	metres	63	44	20	50	50	20
PRODUCTION	metres	1,289	1,299	1,361	850	850	600
TOTAL METRES		2,585	2,538	1,674	1,610	1,610	940
EQUIPMENT REQUIREMENTS	**UNITS**						
JUMBOS REQUIRED	units	3	3	3	3	3	3
SCISSOR LIFTS REQUIRED	units	5	5	5	5	5	5
TRUCKS REQUIRED	units	3	3	3	3	3	3
SCOOPS REQUIRED	units	2	2	2	2	2	2

Appendix A2: Monthly Schedule Summary

CALENDAR	UNITS	TOTAL/AVERAGE	Jan-14	Feb-14	Mar-14	Apr-14	May-14	Jun-14	Jul-14	Aug-14	Sep-14	Oct-14	Nov-14	Dec-14
DAYS PER MONTH	days	29.2	20	21	22	23	24	25	26	27	28	29	30	31
PRODUCTION	**UNITS**	**TOTAL/AVERAGE**												
ORE TONNES	tonnes	**1,400,000**	0	0	0	0	0	0	0	0	0	0	0	6,587
ACCUMULATIVE TONNES			**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**6,587**
Production Rate (tpd)														**212**
PROCESS PLANT THROUGHPUT	tonnes/day	**1,000**												
PROCESS PLANT OPERATING DAYS	days	**1,408**												
PROCESSED ORE (TONNES)	tonnes	**1,400,000**												
STOCKPILE (TONNES)	tonnes		**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**6,587**
MATERIAL TYPE	**UNITS**	**TOTAL/AVERAGE**												
SUMMARY														
ORE TONNES	tonnes	**1,400,000**	0	0	0	0	0	0	0	0	0	0	0	6,587
MUDSTONE TONNES	tonnes	**38,190**	0	0	0	0	0	0	0	0	0	0	276	70
LIMESTONE TONNES	tonnes	**588,125**	0	0	0	0	0	0	0	0	4,708	4,705	9,006	10,604
TOTAL TONNES	**tonnes**	**2,026,315**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**4,708**	**4,705**	**9,283**	**17,261**
BACKFILL & MASS BALANCE	**UNITS**	**TOTAL/AVERAGE**												
BACKFILL SUMMARY														
CEMENTED ROCKFILL PLACED UNDERGROUND	tonnes	**7,730**	0	0	0	0	0	0	0	0	0	0	0	1,810
PASTE BACKFILL PLACED UNDERGROUND	tonnes	**914,045**	0	0	0	0	0	0	0	0	0	0	0	0
PYRITE TAILS USED FOR PASTE FILL	tonnes	**539,406**	0	0	0	0	0	0	0	0	0	0	0	0
PYRITE TAILS SURFACE STOCKPILE (CUMMULATIVE)	tonnes	**0**	0	0	0	0	0	0	0	0	0	0	0	0
DRYSTACK TAILS SURFACE STOCKPILE (CUMMULATIVE)	tonnes	**160,570**	0	0	0	0	0	0	0	0	0	0	0	0
MINE WASTE BROUGHT TO SURFACE	cu.m	**227,301**	0	0	0	0	0	0	0	0	1,744	1,742	3,438	3,298
MINE WASTE BROUGHT BACK UNDERGROUND (BACKFILL)	cu.m	**2,480**	0	0	0	0	0	0	0	0	0	0	0	0
MINE WASTE SURFACE STOCKPILE	tonnes	**612,019**	0	0	0	0	0	0	0	0	4,708	9,412	18,695	28,058
CEMENT CONSUMED REQUIRED	tonnes	**15,266**	0	0	0	0	0	0	0	0	0	0	0	33
SHOTCRETE CONSUMED	tonnes	**41,875**	0	0	0	0	0	0	0	0	32	32	23	201
DEVELOPMENT METRES	**UNITS**	**TOTAL/AVERAGE**												
SUMMARY														
REHAB	metres	**2,274**	0	0	0	0	0	0	0	0	349	348	94	0
RAMP	metres	**3,771**	0	0	0	0	0	0	0	0	0	0	56	80
LEVEL	metres	**977**	0	0	0	0	0	0	0	0	0	0	18	6
SUB LEVEL	metres	**6,973**	0	0	0	0	0	0	0	0	0	0	87	117
RAISE	metres	**725**	0	0	0	0	0	0	0	0	0	0	0	13
PRODUCTION	metres	**24,439**	0	0	0	0	0	0	0	0	0	0	0	110
TOTAL METRES		**39,159**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**349**	**348**	**255**	**326**
EQUIPMENT REQUIREMENTS	**UNITS**	**TOTAL/AVERAGE**												
JUMBOS REQUIRED	units	**3**	0	0	0	0	0	0	0	0	0	0	1	2
SCISSOR LIFTS REQUIRED	units	**6**	0	0	0	0	0	0	0	0	4	3	3	4
TRUCKS REQUIRED	units	**3**	0	0	0	0	0	0	0	0	1	1	1	2
SCOOPS REQUIRED	units	**2**	0	0	0	0	0	0	0	0	1	1	1	1

Appendix A2: Monthly Schedule Summary

Jan-15	Feb-15	Mar-15	Apr-15	May-15	Jun-15	Jul-15	Aug-15	Sep-15	Oct-15	Nov-15	Dec-15
31	28	31	30	31	30	31	31	30	31	30	31
5,350	12,469	17,822	16,610	15,162	0	0	0	0	0	0	0
11,937	**24,406**	**42,228**	**58,838**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**
173	**445**	**575**	**554**	**489**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
		700	**1,000**	**1,000**	**1,000**	**1,000**					
		25	**8**	**17**	**19**	**13**					
		17,500	**8,000**	**17,000**	**19,000**	**12,500**					
11,937	**24,406**	**24,728**	**33,338**	**31,500**	**12,500**	**0**	**0**	**0**	**0**	**0**	**0**
5,350	12,469	17,822	16,610	15,162	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
7,422	373	668	3,183	6,990	0	0	0	0	0	0	0
12,772	**12,842**	**18,490**	**19,793**	**22,152**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
254	5,666	0	0	0	0	0	0	0	0	0	0
0	0	8,931	8,388	7,573	8,800	8,800	4,011	0	0	0	0
0	0	5,963	4,282	5,056	5,876	5,876	0	0	0	0	0
0	0	1,068	0	1,774	3,532	2,678	0	0	0	0	0
0	0	3,290	2,045	5,608	9,478	11,024	9,692	11,024	11,024	11,024	11,024
1,286	0	247	1,179	2,589	0	0	0	0	0	0	0
0	409	0	0	0	0	0	0	0	0	0	0
32,555	31,832	32,500	35,683	42,673	42,673	42,673	42,673	42,673	42,673	42,673	42,673
5	103	139	202	64	50	50	0	0	0	0	0
193	378	443	466	391	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
67	0	0	61	134	0	0	0	0	0	0	0
36	0	6	0	0	0	0	0	0	0	0	0
30	0	22	0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
109	227	265	277	227	0	0	0	0	0	0	0
242	**227**	**293**	**338**	**362**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
2	2	2	2	2	0	0	0	0	0	0	0
3	3	3	4	4	0	0	0	0	0	0	0
1	2	2	2	2	2	2	0	0	0	0	0
1	2	2	2	2	2	2	0	0	0	0	0

Jan-16	Feb-16	Mar-16	Apr-16	May-16	Jun-16	Jul-16	Aug-16	Sep-16	Oct-16	Nov-16	Dec-16
31	28	31	30	31	30	31	31	30	31	30	31
0	0	0	0	19,993	16,270	12,643	13,693	11,401	0	0	0
74,000	**74,000**	**74,000**	**74,000**	**93,993**	**110,263**	**122,906**	**136,599**	**148,000**	**148,000**	**148,000**	**148,000**
0	**0**	**0**	**0**	**645**	**542**	**408**	**442**	**380**	**0**	**0**	**0**
					1,000	**1,000**	**1,000**	**1,000**	**1,000**		
					24	**5**	**20**	**15**	**10**		
					24,000	**5,000**	**20000**	**15,000**	**10,000**		
0	**0**	**0**	**0**	**19,993**	**12,263**	**19,906**	**13,599**	**10,000**	**0**	**0**	**0**
0	0	0	0	19,993	16,270	12,643	13,693	11,401	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	6,677	1,755	6,534	13,689	10,819	0	0	0
0	**0**	**0**	**0**	**26,670**	**18,024**	**19,177**	**27,382**	**22,221**	**0**	**0**	**0**
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	10,448	8,922	9,387	7,921	7,135	2,972	9,049	0
0	0	0	0	0	5,957	5,694	5,289	4,764	1,985	675	0
0	0	0	0	0	3,685	0	2,746	4,009	6,042	0	0
11,024	11,024	11,024	11,024	576	6,194	4,290	8,810	11,803	14,391	11,384	14,056
0	0	0	0	2,473	650	2,420	5,070	4,007	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
42,673	42,673	42,673	42,673	49,350	51,104	57,638	71,327	82,146	82,146	82,146	82,146
0	0	0	0	245	109	53	220	174	17	6	0
0	0	0	0	585	483	316	448	409	0	0	0
0	0	0	0	0	0	96	0	123	0	0	0
0	0	0	0	9	0	63	131	61	0	0	0
0	0	0	0	0	0	43	35	58	0	0	0
0	0	0	0	130	6	0	98	33	0	0	0
0	0	0	0	0	0	0	11	7	0	0	0
0	0	0	0	345	290	179	256	231	0	0	0
0	**0**	**0**	**0**	**483**	**296**	**382**	**530**	**514**	**0**	**0**	**0**
0	0	0	0	3	2	2	3	2	0	0	0
0	0	0	0	5	3	4	6	5	0	0	0
0	0	0	0	2	2	2	2	2	0	0	0
0	0	0	0	2	2	2	2	2	0	0	0

Jan-17	Feb-17	Mar-17	Apr-17	May-17	Jun-17	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17
31	28	31	30	31	30	31	31	30	31	30	31
0	0	0	0	12,890	16,456	16,312	18,497	9,844	0	0	0
148,000	**148,000**	**148,000**	**148,000**	**160,891**	**177,347**	**193,659**	**212,156**	**222,000**	**222,000**	**222,000**	**222,000**
0	**0**	**0**	**0**	**416**	**549**	**526**	**597**	**328**	**0**	**0**	**0**
					1,000	**1,000**	**1,000**	**1,000**	**1,000**		
					24	**5**	**20**	**15**	**10**		
					24,000	**5,000**	**20000**	**15,000**	**10,000**		
0	**0**	**0**	**0**	**12,891**	**5,347**	**16,659**	**15,156**	**10,000**	**0**	**0**	**0**
0	0	0	0	12,890	16,456	16,312	18,497	9,844	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	675	6,944	6,200	2,779	2,134	0	0	0
0	**0**	**0**	**0**	**13,565**	**23,400**	**22,512**	**21,276**	**11,978**	**0**	**0**	**0**
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	5,661	8,356	10,924	11,896	8,057	840	6,284	0
0	0	0	0	0	5,579	6,072	7,943	5,380	561	433	0
0	0	0	0	0	4,063	0	92	739	4,196	0	0
14,056	14,056	14,056	14,056	8,395	14,200	11,136	14,336	17,022	20,319	18,231	20,104
0	0	0	0	250	2,572	2,296	1,029	790	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
82,146	82,146	82,146	82,146	82,821	89,765	95,965	98,744	100,878	100,878	100,878	100,878
0	0	0	0	32	61	297	196	45	5	4	0
0	0	0	0	312	504	581	560	290	0	0	0
0	0	0	0	0	125	0	0	0	0	0	0
0	0	0	0	0	0	20	0	0	0	0	0
0	0	0	0	0	9	0	0	0	0	0	0
0	0	0	0	17	54	89	28	21	0	0	0
0	0	0	0	0	12	0	0	0	0	0	0
0	0	0	0	187	293	344	335	173	0	0	0
0	**0**	**0**	**0**	**203**	**493**	**452**	**363**	**195**	**0**	**0**	**0**
0	0	0	0	1	2	3	2	1	0	0	0
0	0	0	0	2	5	5	4	2	0	0	0
0	0	0	0	1	2	2	2	1	0	0	0
0	0	0	0	1	2	2	2	1	0	0	0

Appendix A2: Monthly Schedule Summary

Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18
31	28	31	30	31	30	31	31	30	31	30	31
0	0	0	0	3,562	20,816	14,209	13,336	17,731	4,346	0	0
222,000	**222,000**	**222,000**	**222,000**	**225,562**	**246,377**	**260,587**	**273,923**	**291,654**	**296,000**	**296,000**	**296,000**
0	**0**	**0**	**0**	**115**	**694**	**458**	**430**	**591**	**140**	**0**	**0**
					1,000	**1,000**	**1,000**	**1,000**	**1,000**		
					24	**5**	**20**	**15**	**10**		
					24,000	**5,000**	**20000**	**15,000**	**10,000**		
0	**0**	**0**	**0**	**3,562**	**377**	**9,587**	**2,923**	**5,654**	**0**	**0**	**0**
0	0	0	0	3,562	20,816	14,209	13,336	17,731	4,346	0	0
0	0	0	0	0	0	0	1,175	766	0	0	0
0	0	0	0	1,185	4,359	4,766	8,936	9,954	5,948	0	0
0	**0**	**0**	**0**	**4,747**	**25,175**	**18,976**	**23,447**	**28,452**	**10,294**	**0**	**0**
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	2,158	12,628	8,210	13,809	14,948	8,812	1,110	0
0	0	0	0	0	8,431	3,220	8,035	6,026	4,018	0	0
0	0	0	0	0	1,211	0	0	0	0	0	0
20,104	20,104	20,104	20,104	17,946	22,332	19,130	20,509	16,951	15,733	14,623	14,623
0	0	0	0	439	1,615	1,765	3,745	3,970	2,203	0	0
0	0	0	0	0	0	0	0	0	0	0	0
100,878	100,878	100,878	100,878	102,063	106,422	111,189	121,299	132,019	137,968	137,968	137,968
0	0	0	0	12	229	54	256	281	70	6	0
0	0	0	0	117	688	423	675	786	340	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	47	0	46	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	16	44	21	82	37	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	70	411	250	402	468	205	0	0
0	**0**	**0**	**0**	**86**	**455**	**318**	**484**	**550**	**205**	**0**	**0**
0	0	0	0	1	3	2	3	3	1	0	0
0	0	0	0	1	5	4	5	6	2	0	0
0	0	0	0	1	2	2	2	3	1	0	0
0	0	0	0	1	2	2	2	2	1	0	0

Appendix A2: Monthly Schedule Summary

Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	Jul-19	Aug-19	Sep-19	Oct-19	Nov-19	Dec-19
31	28	31	30	31	30	31	31	30	31	30	31
0	0	0	0	13,882	13,636	14,894	17,764	11,252	2,572	0	0
296,000	**296,000**	**296,000**	**296,000**	**309,882**	**323,517**	**338,411**	**356,175**	**367,428**	**370,000**	**370,000**	**370,000**
0	**0**	**0**	**0**	**448**	**455**	**480**	**573**	**375**	**83**	**0**	**0**
					1,000	**1,000**	**1,000**	**1,000**	**1,000**		
					24	**5**	**20**	**15**	**10**		
					24,000	**5,000**	**20000**	**15,000**	**10,000**		
0	**0**	**0**	**0**	**13,882**	**3,517**	**13,411**	**11,175**	**7,428**	**0**	**0**	**0**
0	0	0	0	13,882	13,636	14,894	17,764	11,252	2,572	0	0
0	0	0	0	0	313	0	0	0	0	0	0
0	0	0	0	3,360	4,401	3,339	1,960	7,548	2,936	0	0
0	**0**	**0**	**0**	**17,242**	**18,350**	**18,233**	**19,725**	**18,800**	**5,508**	**0**	**0**
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	201	9,748	9,224	9,260	10,277	4,270	3,270	0
0	0	0	0	0	6,509	5,142	6,183	6,862	2,851	1,642	0
0	0	0	0	0	3,134	0	1,852	1,017	2,183	0	0
14,623	14,623	14,623	14,623	14,421	19,765	17,472	21,547	23,496	25,653	24,567	24,836
0	0	0	0	0	1,746	70	726	2,796	1,087	0	0
0	0	0	0	2,071	0	0	0	0	0	0	0
137,968	137,968	137,968	137,968	134,697	139,411	140,417	142,377	149,925	152,861	152,861	152,861
0	0	0	0	6	271	159	52	290	63	14	0
0	0	0	0	403	537	548	448	571	202	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	61	38	0	40	81	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0
0	0	0	0	239	321	329	267	339	122	0	0
0	**0**	**0**	**0**	**300**	**359**	**329**	**307**	**420**	**122**	**0**	**0**
0	0	0	0	2	2	2	2	2	1	0	0
0	0	0	0	3	4	3	3	4	2	0	0
0	0	0	0	2	2	2	2	2	1	0	0
0	0	0	0	2	2	2	2	2	1	0	0

Appendix A2: Monthly Schedule Summary

2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
365	365	365	365	365	365	365	365	365	365	365	365	365	365
74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	68,000
444,000	**518,000**	**592,000**	**666,000**	**740,000**	**814,000**	**888,000**	**962,000**	**1,036,000**	**1,110,000**	**1,184,000**	**1,258,000**	**1,332,000**	**1,400,000**
500	**500**	**500**	**500**	**500**	**500**	**500**	**500**	**500**	**500**	**500**	**500**	**500**	**500**
1,000	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**	**1,000**
74	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**74**	**68**
74,000	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**74,000**	**68,000**
0	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	74,000	68,000
0	750	221	0	1,237	0	22,622	1,782	0	0	8,979	0	0	0
14,730	44,096	22,448	12,418	48,712	23,296	34,257	52,727	61,640	58,725	6,325	15,140	15,140	13,912
88,730	**118,846**	**96,669**	**86,418**	**123,949**	**97,296**	**130,879**	**128,509**	**135,640**	**132,725**	**89,304**	**89,140**	**89,140**	**81,912**
0	0	0	0	0	0	0	0	0	0	0	0	0	0
44,527	44,527	44,527	44,527	45,055	45,414	53,823	48,128	44,527	48,942	49,940	48,574	44,527	44,723
29,486	25,990	28,028	28,782	29,731	29,731	29,731	29,731	26,376	29,731	29,731	29,731	29,006	27,320
0	0	0	0	0	0	0	0	0	0	0	0	0	0
36,503	48,292	61,820	74,335	85,946	96,727	99,098	107,164	118,831	127,753	134,007	141,628	153,295	160,570
5,456	16,610	8,396	4,599	18,499	8,628	21,066	20,189	22,830	21,750	5,668	5,607	5,607	5,153
0	0	0	0	0	0	0	0	0	0	0	0	0	0
167,591	212,437	235,106	247,524	297,473	320,769	377,648	432,158	493,798	552,523	567,828	582,967	598,107	612,019
652	667	692	512	842	839	639	761	617	798	830	1,224	1,196	1,126
2,422	2,259	2,140	2,161	2,329	2,364	2,651	2,386	2,251	2,263	2,289	1,475	1,475	1,026
0	0	345	0	0	0	794	0	0	0	0	0	0	0
0	339	0	0	380	70	436	264	494	346	127	220	220	100
0	68	5	0	39	46	90	57	149	120	53	60	60	20
191	442	399	280	586	388	249	794	590	729	113	430	430	200
0	64	0	0	96	9	232	35	63	44	20	50	50	20
1,446	1,314	1,247	1,285	1,347	1,396	1,508	1,377	1,289	1,299	1,361	850	850	600
1,637	**2,227**	**1,996**	**1,565**	**2,448**	**1,908**	**3,309**	**2,528**	**2,585**	**2,538**	**1,674**	**1,610**	**1,610**	**940**
1	1	1	1	1	1	1	2	2	2	1	1	1	1
2	2	2	2	2	2	3	3	3	3	2	2	2	1
1	1	1	1	1	1	1	1	2	2	1	2	2	2
1	1	1	1	1	1	1	1	1	1	1	2	2	2