NI 43-101 TECHNICAL REPORT
MACMILLAN PASS PROJECT
YUKON TERRITORY, CANADA

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NOTICE

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

Fireweed Zinc Ltd. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

Date and Signature Page

This report entitled NI 43-101 Technical Report, Macmillan Pass Project, Yukon Territory, Canada, effective as of May 23, 2018 was prepared and signed by the following authors:

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1 Executive Summary

1.1 Introduction

The Macmillan Pass Project (also known as the Tom-Jason Project or the Project) is 100% owned by Fireweed Zinc Ltd. (Fireweed), a public company which trades on the TSX-Venture Exchange (TSX-V) under the symbol FWZ. JDS Energy & Mining Inc. (JDS) was commissioned by Fireweed to compile a Preliminary Economic Assessment (PEA) for the Macmillan Pass project. This Technical Report summarizes the results of the PEA and is prepared according to the guidelines of the Canadian Securities Administrator's National Instrument 43-101 (NI 43-101) and Form 43-101F1.

JDS managed the PEA and completed the mining, mineral processing, metallurgical testing, infrastructure, and economics sections of the report. JDS was assisted by several Fireweed-designated subcontractors to provide report information as noted below:

- CSA Global Geosciences Canada Ltd. (CSA or CSA Global): property description, geology and mineral resources; and
- Knight Piésold Ltd. (KP): mine closure, mine waste and water management.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the project presented in the PEA will be realized.

1.2 Property Description and Ownership

The Project contains a historic underground exploration adit and waste rock stockpile. The PEA plan presented in this report is to mine the deposit using conventional open pit and underground mining methods and extract zinc, lead and silver from the mineralization using a 5,000 tonnes per day (t/d) flotation mineral processing facility.

The Macmillan Pass Project is in eastern Yukon, Canada near the border with the Northwest Territories, approximately 400 kilometres (km) northeast of the city of Whitehorse. It consists of a number of contiguous blocks of claims: the Tom mining lease and the Jason claims which are 100% owned by Fireweed and the MAC, MC, MP, Jerry, BR and NS claim blocks which Fireweed has under option from third parties, for a total of 2,528 claims covering about 469 km^2, as well as a single surface lease in the Tom area comprising 120.7 hectares (ha) owned 100% by Fireweed. The Tom and Jason deposits, subject of this report, are located entirely within the Tom mining lease and the Jason claims, respectively.

Access to the site is by seasonal gravel road or by air, and there is minimal infrastructure available in the region. The nearest population centre is at Ross River located about 200 km to the southwest.

1.3 History, Exploration and Drilling

There has been a significant amount of historical exploration on the Tom and Jason claims. Commencing with the discovery of the Tom West Zone in 1951 and including the 2017 drilling by Fireweed, total drilling on the Jason property is 39,191 meters (m) in 135 drillholes and on the Tom property is 34,431 m in 219 drillholes. In addition, an adit with approximately 3,423 m of underground development and a spiral decline was excavated in stages into the Tom West deposit to assist exploration and for bulk sampling between 1969 and 1982. Exploration was effectively suspended on the properties after 1992.

Hudbay Minerals Inc. (Hudbay), the former owner of the Tom and Jason claims, commissioned a Mineral Resource Estimate (MRE) in 2007 that is historical, not to current NI 43-101 standards. Results are summarized in Section 14.12.2.

Exploration recommenced briefly in 2011 with the drilling of 11 new diamond holes for a total of 1,823 m. These holes were drilled for metallurgical testing and infill purposes in the Tom West Zone. Five of the holes were twin holes that verify historical intersections. In 2017, Fireweed carried out a program of drilling, mapping, sampling, LiDAR topographic mapping and airborne geophysics on the property. Drilling totaled 936 m in seven holes on the Tom deposit and 1,266 m in seven holes on the Jason deposit. Results of the drilling and other 2017 work are described in Sections 9 and 10.

1.4 Geology and Mineralization

The Tom and Jason zinc-lead-silver (Zn-Pb-Ag) deposits are proximal, stratiform, sediment-hosted (SEDEX) deposits formed during Devonian era rifting activity in the Selwyn Basin. They were subsequently folded during the transition of the Pacific margin of North America from a passive to convergent plate margin.

1.5 Metallurgical Testing and Mineral Processing

The most recent metallurgical test programs on the Tom and Jason deposits were completed in 2012 and 2018. The 2012 test program was performed by G&T Metallurgical Services in Kamloops, BC (Project No. KM3180) and focused solely on the Tom deposit. The 2018 test program, completed in conjunction with this study, was carried out at Base Metallurgical Laboratories Ltd. (“Base Met”) in Kamloops, BC (Project No. BL0236) and evaluated both the Tom and Jason deposits.

The 2012 program was developed using the results from a historical test program conducted by Michigan Tech in 1986. Tom mineralized material was used to evaluate mineralogy, grinding specific energy and sequential Pb, Zn flotation. A locked cycle test (LCT) was completed at a primary grind size of 80% passing (P_{80}) 72 µm with Pb and Zn regrind sizes of 12 µm and 24 µm respectively. The Pb concentrate recovered 82% of the Pb at a grade of 70.9% Pb, while the Zn concentrate recovered 79.5% of the Zn at a grade of 58.8% Zn.

In 2018, Base Met completed a metallurgical test program to evaluate both the Tom and Jason deposits. The program included mineralogy, comminution, dense media separation (DMS), settling, and rougher/cleaner Pb, Zn sequential flotation. Five variability composite samples, representing the Tom and Jason zones, were tested to develop a preliminary recovery flowsheet and associated flotation conditions. Three global composites (Tom, Jason and a Blend) were then created and locked cycle testing was completed.

Comminution testing found that both the Tom and Jason deposits can be classified as soft to moderately hard with Bond ball mill work indices ranging between 8.8 and 14 kWh/t and Axb parameters ranging between 55.8 and 80.8. The material was found to be moderately abrasive with Bond abrasion indices ranging between 0.225 and 0.445 g.

Based on the results from locked cycle testing, saleable Pb and Zn concentrates can be produced using Pb and Zn sequential flotation at a primary P_{80} grind size of 50 µm and Pb and Zn regrind sizes of 15 µm and 25 µm respectively. DMS pre-concentration was found not to benefit the process, with the low mass rejection and high fines production not justifying the corresponding metal losses.

Five LCTs were completed to predict concentrate grades and recoveries. To reflect the anticipated mine plan, a ratio of 65% Tom Composite and 35% Jason Composite was used to create the Blend Global Composite. A summary of the LCT results are shown in Table 1-1.

Table 1-1: Locked Cycle Testing Results

Composite ID	Test Number	Pb Flotation		Zn Flotation	
		Concentrate Grade (%)	Recovery (%)	Concentrate Grade (%)	Recovery (%)
Tom Composite	43	69.1	74.4	60.1	85.5
Jason Composite	44	69.9	55.7	63.2	88.4
Blend Global Composite (Zn regrind 25 µm)	45	61.5	75.4	58.4	88.9
Blend Global Composite (Zn regrind 20 µm)	49	69.1	77.5	61.4	84.1
Composite 1 (Tom West)	46	67.4	59.8	55.5	91.0

Source: Base Met (2018)

Preliminary test work results on the Blend Global Composite (LCT-45) indicate that the MacMillan Pass deposits can be treated using conventional sequential flotation techniques. The results from this test were used to predict the estimated Pb and Zn concentrate grades and recoveries for the economic model.

1.6 Mineral Resource Estimate

CSA Global was commissioned to prepare an independent estimate of Mineral Resources for the project compiled using technical data up to a cut-off date of 31 December 2017. Table 1-2 is a summary of the Base Case Mineral Resources for the Tom and Jason deposits stated as at 10 January 2018 (see Section 14 for underlying parameters used, other details and additional tables).

Table 1-2: Base Case Mineral Resource Estimate (at NSR Cut-off Grade of C$65)

Category	Tonnes (Mt)	ZnEq %	Zn %	Pb %	Ag g/t	B lbs Zn	B lbs Pb	Moz Ag
Indicated	11.21	9.61	6.59	2.48	21.33	1.63	0.61	7.69
Inferred	39.47	10.00	5.84	3.14	38.15	5.08	2.73	48.41

Source: CSA Global (2018)

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and therefore do not have demonstrated economic viability.

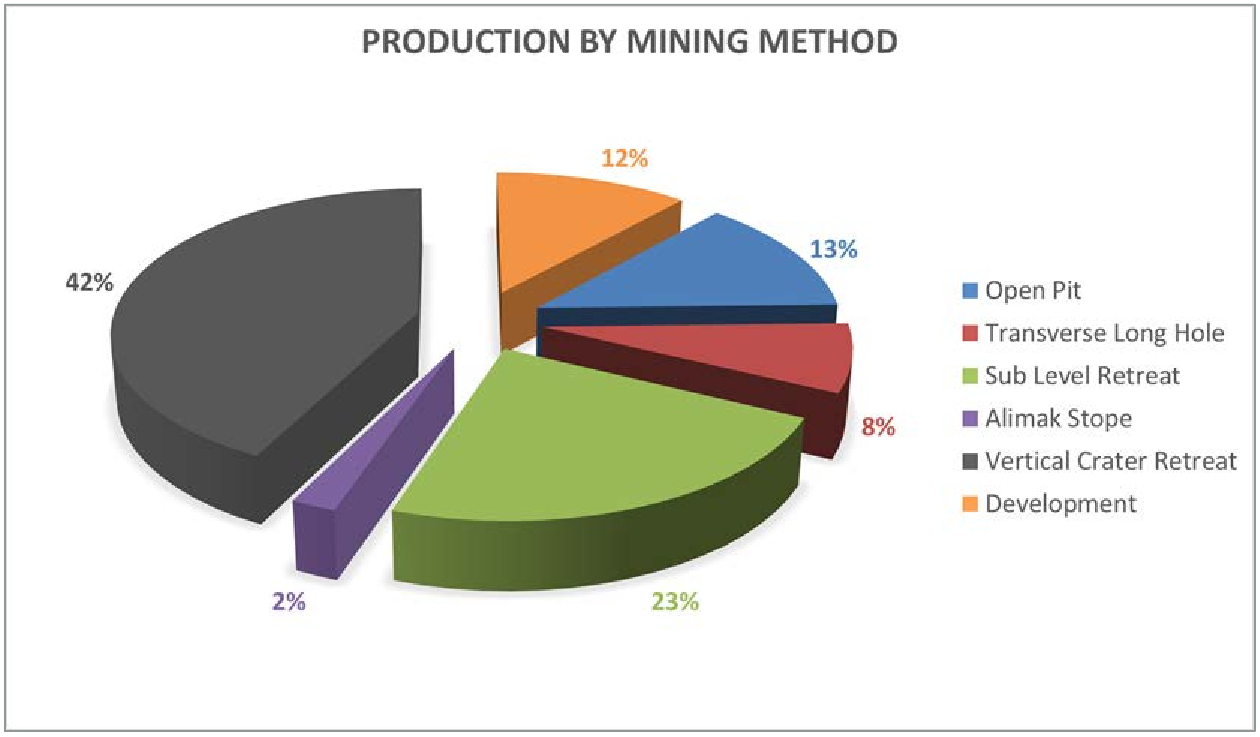
1.7 Mining Methods

The Fireweed mineable resource will be extracted from both Tom and Jason deposits using a combination of mining methods, including:

- Open Pit Mining (OPM);
- Longhole stoping (LHS);
- Vertical Crater Retreat (VCR);
- Sub-level retreat (SLR);
- Alimak Stoping (ALS); and
- Development and cross cuts (XCO).

Mining methods are selected based on geometry and the grade of the mineralized zones Figure 1-1 below outlines a summary of mining methods proposed at Macmillan Pass.

Figure 1-1: Production by Mining Method



Source: JDS (2018)

The mineral deposits will be mined by open pit methods as underground development progresses. As the open pit resources are depleted the underground operation will ramp up to sustain a nominal throughput of 5,000 tonnes per day (t/d) to the mill.

Cemented and un-cemented rock fill from open pit and underground mining will be used underground as backfill to maximize mining recovery. The underground resources will initially be accessed from the existing portal and exploration ramp with one additional portal at Tom for material transport and one new portal at Jason for underground access. Access ramps will be driven at maximum grade of 15% at a 5.5 m wide by 5.0 m high profile to accommodate 50-tonne haul trucks.

Level spacing is variable up to a maximum of 40 m. Mineralized zone development will be driven using a 5.0 m x 5.0 m profile.

The mine design was based on basic assumptions to generate lower limits for cut-off grades (COG) for each of the planned mining zones. A value between 100 to 130 $/t Net Smelter Return (NSR) was determined as the COG depending on the geotechnical criteria, mine method, and anticipated sustaining development required for each zone. These COG's were used to design initial mining shapes.

The PEA mine plan focuses on accessing and mining higher margin material early in the mine life, including Tom East and higher grade portions of Tom West and Jason Main. As the mine is developed, other high grade areas in Jason South are accessed and placed into production.

Mining recovery and dilution factors were applied to each mining shape based on the mining method used. The production plan for the MacMillan Pass project is summarized in Table 1-3.

Table 1-3: Mine Production Schedule

Year	Tonnage (kt)	Silver Grade (g/t)	Lead Grade (%)	Zinc Grade (%)	Lateral Development (km)	Vertical Development (km)	Cemented Rock Fill (kt)	Waste Fill (kt)
-2	-	-	-	-	-	-	-	-
-1	-	-	-	-	-	-	-	-
1	1,825	24.7	2.8	6.2	7.6	0.3	28	42
2	1,825	40.0	3.8	6.3	7.8	1.1	-	-
3	1,826	51.5	4.5	6.1	7.7	0.4	369	458
4	1,825	65.0	5.4	7.1	7.6	0.3	420	622
5	1,825	56.9	4.8	6.8	7.6	0.3	457	609
6	1,825	48.4	3.5	6.2	7.7	0.5	388	692
7	1,830	58.1	4.6	5.7	7.6	0.6	482	630
8	1,830	59.2	4.3	4.8	7.6	0.1	668	438
9	1,830	77.9	5.1	5.0	7.6	0.7	674	391
10	1,830	88.9	6.2	4.7	7.6	0.3	638	392
11	1,830	65.0	4.8	4.7	2.5	-	660	392
12	1,830	21.0	1.9	5.2	3.1	-	450	690
13	1,830	26.6	2.9	4.5	2.9	-	437	671
14	1,830	23.5	2.2	4.7	3.5	-	431	677
15	1,830	20.8	1.8	4.7	4.2	0.4	451	656
16	1,830	11.4	1.4	4.1	6.0	0.9	417	642
17	1,830	14.8	1.4	3.8	0.6	-	275	461
18	1,574	25.3	2.4	4.8	0.6	-	98	711

Source: JDS (2018)

1.8 Recovery Methods

A blend of the Tom and Jason deposits will feed the crushing and process plants at a rate of 5,000 t/d, producing saleable Pb and Zn concentrates. Two crushing plants, one underground for Tom and one near the crushed material stockpile for Jason, will operate on average 18 hours per day. The process plant will operate 24 hours per day, 365 days per year at an availability of 92%.

The process design criteria and recovery flowsheet have been developed based on the metallurgical test work results from the 2018 Base Met test program. The process plant will use a semi-autogenous grinding (SAG) mill / ball mill grinding circuit to achieve a P_{80} grind size of 50 µm. The material will then be fed to sequential Pb and Zn rougher/cleaner flotation circuits. The Pb and Zn regrind circuits, which will further liberate the rougher concentrates, will be designed to produce P_{80} grind sizes of 15 µm and 25 µm respectively.

The processing facilities will consist of the following unit operations:

- Two crushing circuits - single-stage jaw crushers;
- Primary grinding – SAG and ball mills;

- Pb flotation using conventional and column cells;
- Pb concentrate regrind using stirred mills;
- Zn flotation using conventional and column cells;
- Zn concentrate regrind using stirred mills;
- Pb concentrate dewatering, filtration, bagging and truck load-out facility;
- Zn concentrate dewatering, filtration and truck load-out facility; and
- Tailings disposal.

1.9 Project Infrastructure

The project envisions the upgrading and/or construction of the following key infrastructure items:

- Upgrading 230 km of the Canol Road, to an all-seasonal access road from Ross River to the project site location;
- Process facilities;
- Natural gas power plant and liquefied natural gas (LNG) receiving and storage facility;
- Tailings management facility (TMF) and Waste Rock Storage Facilities (WRSF);
- Permanent camp (established for the construction stage);
- Truck shop and warehouse;
- Mine dry and office complex;
- 300,000 L of on-site fuel oil storage and distribution;
- Airstrip extension;
- Industrial waste management facilities such as the incinerator;
- Site sewage treatment facilities;
- Site storm water management facilities; and
- Telecommunication facilities.

1.10 Environment and Permitting

In 2001, Hudbay, the former property owner, initiated the collection of baseline environmental data and, later, heritage information. The collection of environmental baseline information, which is ongoing by Fireweed, includes the disciplines of wildlife, water, hydrology (surface and groundwater), climate and aquatic life. The data collected to date reflects undisturbed areas found in and around the Project area.

The Project will be subject to an environmental and socio-economic assessment (ESA) under the Yukon Environmental and Socio-economic Assessment Act (YESAA), administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB). The ESA documents the potential environmental and socio-economic effects of the Project by evaluating baseline information, the proposed mine plan and by consulting widely with governments, First Nations, communities, stakeholders, experts and the public.

The ESA will also outline mitigation measures and management plans to be employed to minimize or eliminate possible negative effects resulting from the Project, and conduct research to address environmental priorities.

Once the adequacy review is completed, Fireweed intends to submit the application for a Type A Water Use Licence from the Yukon Water Board, a Quartz Mining Licence under the Quartz Mining Land Use Regulation, and other authorizations.

1.11 Capital and Operating Costs

1.11.1 Capital Costs

The capital cost estimate was compiled using a combination of quotations, database costs and factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study.

The capital cost (CAPEX) estimate includes the costs required to develop, sustain, and close the operation for a planned 18-year mine life. The construction schedule is based on a 20-month build period, with major construction at site taking place over 24 months. The intended accuracy of this estimate is +/- 35%.

The high-level CAPEX estimate is shown in Table 1-4. The sustaining capital is carried over operating Years 1 through 18, and closure costs are projected in Year 19.

Table 1-4: Summary of Life of Mine Capital Costs

Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
Mining	30.3	378.4	-	408.6
Site Development	12.0	1.1	-	13.1
Mineral Processing	70.6	5.5	-	76.1
Tailings Management	32.7	113.9	-	146.6
Infrastructure	129.7	21.4	-	151.1
Indirect Costs Incl. EPCM	63.5	-	-	63.5
Owners Costs	7.0	-	-	7.0
Closure Costs	-	-	56.7	56.7
Subtotal Pre-Contingency	**345.8**	**520.3**	**56.7**	**922.7**
Contingency	58.6	54.2	18.1	130.9
Total Capital Costs	**404.3**	**574.5**	**74.7**	**1,053.6**

Note:
- Numbers may not add due to rounding.

Source: JDS (2018)

1.11.2 Operating Costs

The operating cost estimate (OPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a PEA study.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies.

Total LOM operating costs amount to C$2,677.6 M or an average unit cost of C$82.00/tonne milled. The LOM costs are summarized in Table 1-5. OP mining costs average C$4.45 per OP tonne moved while UG mining costs average C$52.02 per UG tonne mined.

Table 1-5: LOM Total Operating Cost Estimate

Description	Total Estimate (C$ M)	Average Unit Cost (C$/t Processed)
OP Mining	111.9	3.43
UG Mining	1,478.7	45.28
Processing	748.5	22.92
G&A	338.6	10.37
Total Operating Costs	**2,677.6**	**82.00**

Source: JDS (2018)

1.12 Economic Analysis

This PEA is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and, as such, there is no certainty that the PEA results

will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations to represent an indicative value of the after-tax cash flows of the project. Base case metal prices used are calculated by averaging London Metal Exchange (LME) values for each of the prior three years with projected LME contract futures for the coming two years. The summary of results is presented for both base case metal prices and spot price metal prices at 30 April 2018 are shown in Table 1-8.

1.12.1 Main Assumptions

Table 1-6 outlines the metal prices and exchange rate used in the economic analysis.

Table 1-6: Metal Prices and F/X Rate

Parameter	Unit	Base Price Value	Spot Price Value
Lead Price	US$/lb	0.98	1.05
Zinc Price	US$/lb	1.21	1.42
Silver Price	US$/oz	16.80	16.38
Exchange Rate	US$:C$	0.77	0.78

Source: JDS (2018)

No preliminary market studies were completed on the potential sale of lead concentrate and zinc concentrate from the Macmillan Pass Project. The terms selected are in-line with current market conditions and were reviewed and found to be acceptable by QP Michael Makarenko, P.Eng.

No contractual arrangements for shipping, port usage, or refining exist at this time.

Table 1-7 outlines the terms used in the economic analysis.

Table 1-7: Net Smelter Return Assumptions

Assumptions & Inputs	Unit	Value
Lead Concentrate		
Metal Recovery to Concentrate	% Pb	75.4
	% Zn	4.8
	% Ag	59.4
Pb Concentrate Grade Produced	% Pb	61.5
Minimum Deduction	% Pb/t	3.0
	g/t Ag	50.0
Metal Payable	% Pb	95.0
	% Ag	95.0
Pb Treatment Charge	US$/dmt conc.	170
Ag Refining Charge	US$/oz	1.50
Moisture Content	%	8.0
Pb Concentrate Transportation Cost	C$/wmt	211.85
Zinc Concentrate		
Metal Recovery to Concentrate	% Pb	7.5
	% Zn	88.9
	% Ag	22.2
Zn Concentrate Grade Produced	% Zn	58.4
Minimum Deduction	% Pb/t	0.0
	%Zn/t	8.0
	g/t Ag	93.31
Metal Payable	% Pb	0.0
	% Zn	85.0
	% Ag	70.0
Zn Treatment Charge	US$/dmt conc.	190
Ag Refining Charge	US$/oz	1.50
Moisture Content	%	8.0
Zn Concentrate Transportation Cost	C$/wmt	211.85
Hg Content	%	0.0155
Base Hg Content	%	0.01
Penalty per 0.01% Hg	0.01%	1.75
Hg Content Penalty	US$/dmt conc.	0.96
SiO_2 Penalty	US$/dmt conc.	2.00

Source: JDS (2018)

1.12.2 Results

The economic results for the Project based on the assumptions outlined in Section 1.13.1 are shown in Table 1-8.

Table 1-8: Economic Results

Parameter	Unit	Base Price Value	Spot Price Value
Cash Flows			
Working Capital	C$M	22.4	22.4
Pre-Tax Cash Flow	LOM C$M	1,734.8	2,674.7
	C$M/a	96	148
Taxes	LOM C$M	615.7	945.1
After-Tax Cash Flow	LOM C$M	1,119.1	1,729.6
	C$M/a	62	96
Economic Results			
Pre-Tax $NPV_{8\%}$	**C$M**	**779**	**1,214**
Pre-Tax IRR	**%**	**31.9**	**43.5**
Pre-Tax Payback	**Years**	**3.2**	**2.4**
After-Tax $NPV_{8\%}$	**C$M**	**448**	**729**
After-Tax IRR	**%**	**23.5**	**32.1**
After-Tax Payback	**Years**	**4.0**	**3.1**

Source: JDS (2018)

1.12.3 Sensitivities

A simplistic sensitivity analysis was performed to determine which factors most affect the project economics and is discussed in Section 23. Each variable evaluated was tested using the same sensitivity values, although some may be more likely to experience significantly more fluctuation in value over the LOM (i.e. CAPEX versus metal prices).

Sensitivity analyses were performed on metal prices, exchange rate, mill feed grade, capital costs, and operating costs as variables. The value of each variable was changed plus and minus 5% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1-9.

Table 1-9: Sensitivity Results (Pre-Tax $NPV_{8\%}$)

Parameter	-15%	-10%	-5%	Base	+5%	+10%	+15%
Metal Price	204	396	588	779	971	1,163	1,354
C$:US$ FX	1,085	983	881	779	677	576	474
Mill Feed Grade	356	497	638	779	920	1,061	1,203
OPEX	973	908	844	779	715	650	586
CAPEX	891	854	817	779	742	705	667

Source: JDS (2018)

The analysis revealed that the project is most sensitive to metal price, followed by mill feed grade, exchange rate, and operating costs. The Project showed the least sensitivity to capital costs.

1.13 Conclusions and Recommendations

1.13.1 Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used in the PEA.

The Macmillan Pass project contains a substantial zinc, lead and silver resource that can be mined by open pit and underground methods and recovered with conventional flotation processing.

Based on the assumptions used for this preliminary evaluation, the project is economic and should proceed to the pre-feasibility stage.

There is also a likelihood of improving the project economics by identifying additional mineral resources within the development area that may justify increased mine production or extend the mine life as well as improvement opportunities that may be found with further study and work on technical and economic parameters.

The most significant potential risks associated with the Project are uncontrolled dilution, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

To date, the QPs are not aware of any fatal flaws for the Project.

1.13.2 Recommendations

It is recommended that the project proceed to the pre-feasibility study (PFS) stage in line with Fireweed's desire to advance the project. It is also recommended that environmental work and permitting continue as needed to support Fireweed's project development plans and the work programs defined in Section 27.

It is estimated that a pre-feasibility study and supporting field work would cost approximately $10.3 million. A breakdown of the key components of the next study phase is as follows in Table 1-10.

Table 1-10: Cost Estimate to Advance to Pre-feasibility Study Phase

Component	Estimated Cost ($C M)	Comment
Resource Drilling	5.0	Conversion of inferred to indicated resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.5	Comminution, flotation optimization, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Geochemistry	0.5	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	0.8	Site investigation drilling, sampling and lab testing.
Geotechnical, Hydrology & Hydrogeology	1.0	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	1.5	PFS-level mine, infrastructure and process design, cost estimation, scheduling & economic analysis.
Environment	1.0	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology.
Total	**10.3**	Excludes corporate overheads and future permitting activities.

Source: JDS (2018)

2 Introduction

2.1 Basis of Technical Report

This PEA Technical Report was prepared for Fireweed by JDS, CSA Global and KP; collectively referred to as Consultants.

This document has been prepared following the guidelines of the Canadian Securities Administrator's NI 43-101 and Form 43-101F1.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Consultant's services, based on:

- Information available at the time of preparation;
- Data supplied by outside sources; and
- The assumptions, conditions, and qualifications set forth in this report.

Given the nature of the mining business, these conditions can change significantly over relatively short periods. Consequently, actual results may vary significantly. The user of this document should ensure that this is the most recent Technical Report for the property as it may not be valid if a new Technical Report has been issued.

2.2 Scope of Work

This report summarizes the work carried out by the Consultants, all of which are independent of Fireweed. The scope of work for each company is listed below and when combined, makes up the total Project scope.

JDS scope of work included:

- Compile the Technical Report that also includes the data and information provided by other consulting companies;
- Mine planning, optimal pit and underground design, and production schedule;
- Mining equipment selection and cost estimation;
- Determine mine geotechnical criteria and establish pit slope angles and stope sizes;
- Provide recommendations on the execution and development of the metallurgical test work program;
- Interpret the past and current test work results and develop the Project process design criteria;
- Develop an appropriate process flowsheet and preliminary mass and water balance;
- Preparation of layouts, drawings, lists, and other deliverables to support the plant design basis;
- Prepare an operating cost estimate for the process plant;
- Design required plant infrastructure, estimate power requirements, and identify proper sites, plant facilities, and other ancillary facilities;

- Estimate OPEX and CAPEX for the Project;
- Prepare a financial model and conduct an economic evaluation including sensitivity and Project risk analysis; and
- Interpret the results and make conclusions that lead to recommendations to improve value, reduce risks, and move toward a pre-feasibility level study.

CSA Global scope of work included:

- Establish a Mineral Resource estimate for the Project following NI 43-101 guidelines; and
- Summarize geology, mineralization and drilling information.

KP scope of work included:

- Assess tailings management alternatives;
- Design the tailings a management facility (TMF) and determine which methodology would be feasible;
- Develop the mine rock management plan;
- Determine the Project water balances and establish water management plans; and
- Summarize waste disposal operating and post closure requirements and plans.

2.3 Qualified Person Responsibilities and Site Inspections

The Qualified Persons (QPs) preparing this Technical Report are specialists in the fields of geology, exploration, Mineral Resource estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in Fireweed. The QPs are not insiders, associates, or affiliates of Fireweed. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Fireweed and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience, and professional association, are considered QPs as defined in the NI 43-101 standard for this report, and are members in good standing of appropriate professional institutions. The QPs are responsible for specific sections as shown in Table 2-1.

Table 2-1: QP Responsibilities

Qualified Persons	Company	Report Section(s)
Michael Makarenko, P. Eng.	JDS Energy & Mining Inc.	1.1, 1.2, 1.7, 1.9 to 1.13, 2, 3, 15, 16 (except 16.4), 18 (except 18.6, 18.7, 18.8, 18.9), 19 20, 21, 22 (except 22.3), 23, 24, 25, 26, 27, 28, 29
Kelly McLeod, P. Eng.	JDS Energy & Mining Inc.	1.5, 1.8, 13, 17
Mike Levy, P.E., P.G.	JDS Energy & Mining Inc.	16.4
Dennis Arne, P. Geo.	CSA Global Geosciences Ltd.	1.3, 4, 5, 6, 7, 8, 9, 11, 12
Leon McGarry, P. Geo.	CSA Global Geosciences Ltd.	1.4, 1.6, 10, 14
Kenneth Embree, P. Eng.	Knight Piesold Ltd.	18.6, 18.7, 18.8, 18.9

Source: JDS (2018)

QP site visits were conducted as follows:

- Michael Makarenko, P. Eng., completed a site visit on 5 September 2017;
- Mike Levy, P.E., P.G., completed a site visit on 5 September 2017;
- Kenneth Embree, P. Eng., completed a site visit on 5 September 2017;
- Dennis Arne, P. Geo., completed site visits 31 August to 2 September 2011, 24 to 28 July 2017, and 25 to 29 June 2018; and
- Kelly McLeod, P. Eng. and Leon McGarry, P. Geo. did not visit the site and relied upon the observations of QPs Makarenko and Arne respectively.

2.4 Sources of Information

The sources of information include data and reports supplied by Fireweed personnel as well as documents cited throughout the report and referenced in Section 29. In particular, background Project information was directly taken from the technical report titled "*NI 43-101 Technical Report on the Macmillan Pass Zinc-Lead-Silver Project, Watson Lake and Mayo Mining Districts Yukon Territory, Canada*" with an effective date of 10 January 2018 produced by CSA Global (2018) and summarized in Fireweed's 10 January 2018 news release.

2.5 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals).

All dollar figures quoted in this report refer to Canadian dollars (C$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 28.This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3 Reliance on Other Experts

The QPs opinions contained herein are in large part based on information provided to the consultants by Fireweed throughout the course of the investigations. JDS has relied upon the work of other consultants in Project areas in support of this Technical Report.

The QPs used their experience to confirm the information supplied by Fireweed and from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending.

Neither JDS nor the authors of this Technical Report are qualified to provide extensive comment on legal issues associated with the ownership or control of the Macmillan Pass property. As such, portions of Section 4 dealing with the types and numbers of mineral tenures and licences, the nature and extent of Fireweed's title and interest in the Macmillan Pass property, the terms of any royalties, back-in rights, payments, or other agreements and encumbrances to which the property is subject, are descriptive in nature and are provided exclusive of a legal opinion.

4 Property Description and Location

4.1 Property Location

The Macmillan Pass Project is located in eastern Yukon, Canada near the border with the Northwest Territories (Figure 4-1). It is located approximately at latitude 63°10'N and longitude 130°09'W on NTS map sheet 105O-01, approximately 400 km northeast of Whitehorse, a regional capital city, and 200 km northeast of the community of Ross River, which is the nearest settlement.

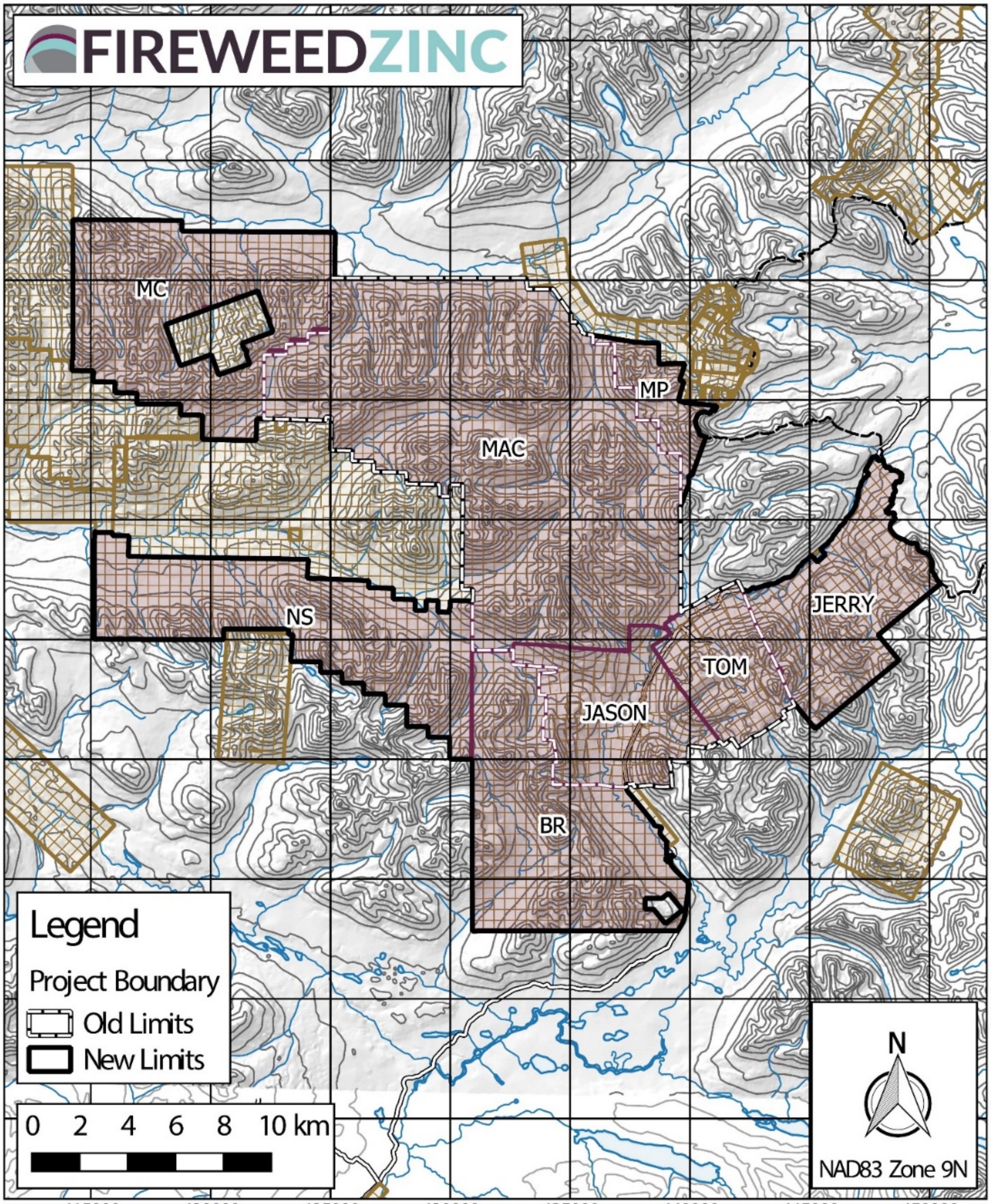
Figure 4-1: Location of the Macmillan Pass Project

Source: CSA Global (2018)

4.2 Property Description and Mineral Tenure

The Macmillan Pass Project consists of a number of historically distinct but contiguous claim groups located in the Watson Lake and Mayo Mining Districts of Yukon, Canada: the Tom mining lease claims and the Jason, MAC, MC, MP, Jerry, BR and NS “quartz claim” blocks (Figure 4-2). The Tom mining lease and the Jason claims are 100% owned by Fireweed and the MAC, MC, MP, Jerry, BR and NS claim blocks are held under option by Fireweed from third parties, for a total of 2,528 claims covering about 469 km^2, as well as a single surface lease in the Tom area comprising 120.7 hectares (ha) owned 100% by Fireweed. The Tom and Jason deposits, subject of this report, are located entirely within The Tom mining lease and the Jason claims, respectively.

Figure 4-2: Macmillan Pass Project Claim Groups



Source: Fireweed News Release dated 27 March 2018

The Tom property / mining lease consists of 144 claims covering an area 2,295 ha with an anniversary date of 12 October 2018 which Fireweed reports will be extended. The group also includes a surface lease comprising 120.68 ha over the Tom West deposit which expires on 28 February 2022 but will be extended. The Jason property/claim group consists of 283 “quartz claims” covering an area of 3,528 ha with a current renewal date of 31 December 2023. The MAC property/claim group consists of 820 “quartz claims” covering an area of 16,780 ha with a current renewal dates of 31 December 2022. The MC claim block consists of 333 “quartz claims” with a current renewal dates varying between 4 April 2019 and 4 April 2022. The MP claim block consists of 74 “quartz claims” with a current renewal dates varying between 4 April 2020 and 4 April 2022. The Jerry claim block consists of 217 “quartz claims” with a current renewal dates varying between 7 April 2019 and 7 April 2022. The BR claim block consists of 326 “quartz claims” with a current renewal date on all claims of 4 January 2020. The NS claim block consists of 333 “quartz claims” with a current renewal date on all claims of 13 January 2020. All claim renewal dates can be extended by carrying out additional work on the claims.

Continued tenure to mineral rights on a lode mineral claim (termed a “quartz claim” in the Yukon) is dependent upon work performed on the claim or a group of claims. When work has been done on a claim and is being used for the renewal of that claim, a full report of the work done must be submitted to the Mining Recorder Office. A renewal certificate will not be issued until the report and/or survey has been approved for the value required. The Yukon Quartz Mining Act (QMA) does not specify work to be performed, except in dollar terms. Renewal of a quartz claim requires that C$100 of work be done per claim per year, based on the Schedule of Representation Work outlined in the QMA. Where work is not performed, the claimant may make a payment in lieu of work. The fee for payment in lieu is C$100 per claim per year plus C$5 for the certificate of work per claim per year. Work must be performed on every claim unless groupings are filed. An application can be made to group adjoining claims; the maximum number of claims per grouping is 750. Grouping allows work to be performed on one or more claims and can be distributed to any or all other claims in the group. As such, annual work requirements for the 283 Jason claims total C$28,300 per year and the other claim blocks similarly require $100 per claim of work per year to extend renewal dates. The Tom claims are a mining lease and are only subject to annual permit fees totaling $28,960 per year. In recent years, these work requirements and fees have been waived by the Yukon government due to the staking withdrawal in the region (described below under First Nations Consultations). The annual fee for the 120.68 ha surface lease on the Tom property is $2,311 per year.

4.3 Royalties, Agreements, and Encumbrances

The following information has been provided to CSA Global by Fireweed as described in Section 3.

4.3.1 Tom and Jason Claims

Fireweed signed a Definitive Option Agreement with Hudbay on 14 December 2016 to acquire the Tom mining lease and Tom surface lease, the Jason quartz claims and associated permits, licences and hard assets. On 7 February 2018, Fireweed exercised the option and is now 100% registered owner of these assets.

The Jason quartz claims were purchased by Hudbay on 3 August 2006 from a consortium of companies operating as MacPass Resources Limited. As per a royalty agreement dated 3 August 2006, the Jason property is subject to a 3% NSR which Fireweed has the right to purchase, at any time as to 1.5% of the NSR for C$1.25 million and the remaining 1.5% of the NSR for C$4.0 million.

There is no NSR encumbrance on the Tom mining lease.

4.3.2 MAC Claims

Fireweed signed an option agreement with Newmont Canada Holdings, ULC (Newmont) on 24 July 2017 to acquire the MAC claims (Figure 4-2). On 29 May 2018, Maverix Metals Inc. (Maverix) (TSXV:MMX), a royalty company, announced the acquisition of a portfolio of royalties and other assets from Newmont which included assignment of the MAC claims option agreement to Maverix which transfer Fireweed acknowledged making Maverix the new optionor of the MAC claims.

Fireweed must pay a total of C$450,000 to acquire 100% interest in the MAC claims as follows:

a. C$50,000 on signing of the option agreement (paid);
b. C$80,000 on or before 24 July 2018;
c. C$95,000 on or before 24 July 2019;
d. C$110,000 on or before 24 July 2020; and
e. C$115,000 on or before 24 July 2021.

Fireweed must also carry out sufficient work to maintain the property in good standing during the term of the option (work completed).

Upon completion of the payment schedule, Maverix will be entitled to receive NSR royalties on future production as follows: 0.25% NSR on base metals, 1% NSR on silver and 3% NSR on gold. Maverix will also have an exclusive but limited 30-day right of first offer on any future proposed sale, transfer or disposition by Fireweed of its interest in the MAC claims. This right of first offer shall not apply to (i) any internal corporate reorganization of Fireweed or (ii) to any transfer of control of Fireweed itself to a third party if the book value of Fireweed's interest in the MAC claims (based on Fireweed financial statements) does not exceed 50% of the combined book value of all the assets of Fireweed.

4.3.3 MC, MP and Jerry Claims

On 27 March 2018, Fireweed announced signing of an option agreement with joint venture partners Constantine Metal Resources Ltd. ("Constantine") and Carlin Gold Corporation ("Carlin") for the 624 MC, MP and Jerry "quartz claims" covering an area of 11,700 hectares (Figure 4-2).

Fireweed can exercise the Option and acquire 100% interest in the claims by making payments totaling C$500,000 and 300,000 Fireweed shares over three years to Constantine and Carlin as follows:

a. C$75,000 and 50,000 shares upon TSX Venture Exchange approval of the Option (paid);
b. On or before 9 May 2019, C$125,000 and 50,000 shares;
c. On or before 9 May 2020, C$150,000 and 100,000 shares; and
d. On or before 9 May 201, C$150,000 and 100,000 shares.

Fireweed may prepay any of the Option Payments and/or prepay the entire Purchase Price at any time.

Although not part of the consideration payable to exercise the Option, Fireweed will pay an additional C$750,000 or equivalent in shares at Fireweed's option, upon receiving a resource calculation of at least 2.0 million tonnes of indicated (or better) resource on any part of the MC, MP and Jerry claims.

Constantine-Carlin will retain the right to receive an NSR on any future mine production from the MC, MP and Jerry claims as follows: on base metals and silver 0.5% NSR, and on all other metals including gold 2% NSR. Fireweed maintains a right of first refusal on the sale of any NSR royalty from these claims by Constantine and/or Carlin.

4.3.4 BR and NS Claims

On 27 March 2018, Fireweed announced signing of an option agreement with Golden Ridge Resources Ltd. ("GLDN") for the 659 BR and NS "quartz claims" covering an area of 12,700 hectares (Figure 4-2).

Fireweed can exercise the Option and acquire 100% interest in the claims by making payments totaling C$500,000 and 450,000 Fireweed shares over three years to GLDN as follows:

a. C$75,000 and 75,000 shares upon TSX Venture Exchange approval of the Option (paid);
b. On or before 9 May 2019, C$75,000 and 75,000 shares;
c. On or before 9 May 2020, C$150,000 and 100,000 shares; and
d. On or before 9 May 2021, C$200,000 and 200,000 shares.

Fireweed may prepay any of the Option Payments and/or prepay the entire Purchase Price at any time.

Although not part of the consideration payable to exercise the Option, Fireweed will pay an additional C$750,000 or equivalent in shares at Fireweed's option to GLDN, upon receiving a resource calculation of at least 2.0 million tonnes of indicated (or better) resource on any part of the BR and NS claims.

GLDN will retain the right to receive a NSR on any future mine production from the BR and NS claims as follows: on base metals and silver 0.5% NSR, and on all other metals including gold 2% NSR. There is also a third party 3% NSR on any future cobalt production from the BR and NS claims. Fireweed will have the right to purchase one-half of these NSR royalties (excluding the cobalt royalty) for $2.0 million at any time prior to the commencement of commercial production. Fireweed maintains a right of first refusal on the sale of any NSR royalty from the BR and NS claims by GLDN.

4.4 Environmental Liabilities

The lower adit on the Tom property was partially plugged in 2010 to flood the mine workings and reduce the flow of acid mine drainage (AMD) from oxidation of sulphides in the mine workings. A waste pile from underground development at Tom West has also been covered with an impermeable barrier to reduce AMD from the site. The lower adit continues to make water as designed and metal contents and other parameters of the discharge water are monitored and have been within standards set in the current Type B water use licence (see Permitting Considerations below) (G. Gorzynski, personal communication, February 2018).

A preliminary environmental investigation of the Jason property in 2006 by Gartner Lee Limited noted that several exploration boreholes below an elevation of 1,250 m were discharging water. Water samples from one of these boreholes and four samples of surface water exceeded the Canadian Council of Ministers of the Environment (CCME) Aquatic Life guidelines for several metals, including Cd and Zn. Elevated metal concentrations and lowered pH levels reflect natural groundwater discharge from the site, as the Earn Group sediments are regionally elevated with respect to several metals, including Zn, Cd, Pb and Ag (Mackie et al., 2015). In 2015, a number of drill pads and collars at the Jason property were rehabilitated and holes plugged with cement when ground conditions allowed it. Water still flows from some holes where proper cementing has not yet been completed (G. Gorzynski, personal communication, 2018).

4.5 Permitting Considerations

Exploration work is subject to the Mining Land Use Regulations of the Yukon QMA and to the Yukon Environmental and Socio-Economic Assessment Act (YESAA). A land use permit must be obtained and YESAA Board approval issued before large-scale exploration is conducted.

Since the exercise of the property option on 7 February 2018, all title and project permits have been transferred into Fireweed's name. Fireweed currently holds a Class 3 land use permit for exploration activities on the Tom and Jason properties (LQ00325) under the QMA and Quartz Mining Land Use Regulations with a renewal date of 21 September 2021. However, Fireweed has applied for a new upgraded land use permit to allow for larger exploration programs on the project and in the meantime continues to operate under the old permit. A waste management permit issued in 2011 (81-029) has been extended to 31 December 2021.

Currently water use and discharge of water from the Tom adit are governed by a Type B water use licence (QZ15-060-01) granted on 24 July 2015 and extended until 31 December 2020. The discharge from the lower Tom adit has naturally elevated metals levels and has been the subject of water quality monitoring and water sampling a minimum of six times per year and reporting since 2001. Continued efforts will be required to monitor compliance with the water licence.

Any potential future development of the Tom and Jason deposits will require an environmental assessment under YESAA and a Yukon Mining Licence and Lease issued by the Yukon Government. A preliminary environmental investigation was undertaken on the Jason deposit by Gartner Lee Limited (Pearson, 2006). Additional permits will be required from the territorial and federal governments to further develop the deposits. For example, development of mining activities in the Yukon requires the issuance of a Type A water licence by the Yukon Water Board.

4.6 First Nations Consultations

The Macmillan Pass Project lies within an area of overlapping territorial claims by the Kaska First Nations and Na-Cho Nyak Dun First Nation. This area has been withdrawn from staking (Ross River Area OIC 2013/224 and OIC 2013/60) pending settlement of land claims. The First Nations have not reached a land claim settlement with the Yukon government, and so the terms of any future development of the Tom and Jason deposits remain uncertain and will require First Nations consultations. Also, to obtain and renew permits for the project Fireweed is required, under Yukon permitting procedures, to consult with the affected First Nations. However, the current staking moratorium does not prevent exploration or development work to be carried out on existing claims and Fireweed reports good relations with local First Nations during the 2017 and 2018 work programs in which they hired local First Nations workers and purchased supplies and fuel from local First Nations businesses.

4.7 Other Significant Factors and Risks

As of the effective date of this report, JDS is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Macmillan Pass Project.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

The Macmillan Pass Project is in the Hess Mountain region of the Selwyn Mountains, part of the western North American Cordillera. Elevations in the Project area vary between approximately 1,125 m and 1,200 m in the flat, wide valley bottom of Macmillan Pass to approximately 2,100 m at mountain peaks on the Tom Property (Figure 4-2, Figure 5-1 and Figure 5-2). The tree line occurs at approximately 1,350 m, and mountain tops are covered by alpine vegetation. Vegetation below 1,350 m is dominated by mixed deciduous and conifer (mainly black spruce) forest. Limited mapping and pitting on the Property including 40 test pits in the area of the proposed plant site (Figure 18-1) indicates discontinuous permafrost patches at lower elevations (Thompson, 1984). Fireweed reports plans for surficial mapping including permafrost distribution in 2018.

Figure 5-1: View of the Tom Camp Taken in 2017 Looking North Down Sekie Creek #2

Source: CSA Global (2018)

Figure 5-2: Aerial View of the Jason Property in the Middle Distance Looking Northwest (Access Tracks Through Forested Areas are Visible)

Source: CSA Global (2018)

5.2 Access to Property

Access to the property is via the sealed Robert Campbell Highway from Whitehorse, the capital of Yukon with an international airport, to the town of Ross River, a distance of approximately 400 km (Figure 4-1). The North Canol Road (Yukon Highway 6) continues to the Project area at Macmillan Pass from Ross River approximately 200 km (Figure 4-1). The road is maintained by the Territorial government and supports large truck traffic. This road can only be accessed by a ferry / barge across the Pelly River near the town of Ross River during the summer months (Figure 5-3) or an ice bridge crossing in the winter.

The Tom Property can be accessed directly from the North Canol Road. A wooden bridge across the South Macmillan River previously provided access to the Jason Property, but this bridge is now derelict. It is possible to ford the South Macmillan River during low water in the summer. Numerous tracks provide access to various areas of both projects (Figure 5-2).

A seasonal government-maintained gravel airstrip, approximately 740 m long is located on the property and supports exploration activities in the region (Figure 5-4).

Figure 5-3: Pelly River Barge Near Ross River

Source: CSA Global (2018)

Figure 5-4: Gravel Airstrip at MacMillan Pass (circled) in Middle Distance Looking North

Source: CSA Global (2018)

5.3 Climate

The climate of the region is sub-arctic. Weather data collected by a government weather station at the Macmillan Pass airstrip averages about -16°C in the winter and +17°C in the summer.

Precipitation data are not available for Macmillan Pass airstrip but between 1974 and 1982 at the Mactung Project located 14 km to the north, the average recorded annual precipitation for this period was 490 millimetres (mm), with an average annual snowfall of 294 cm (Rennie, 2007).

The effective summer season for field exploration operations in the Project area runs from June through early October, and road access is dependent upon when the Pelly River ferry commences and ceases operations for the season. Mine operations with supporting infrastructure, can operate year-round in the region.

5.4 Infrastructure

There are no services available at the project site. Electricity must be generated locally by diesel or liquid natural gas generators.

A 20-person trailer camp was installed at the Tom property in 2011 (Figure 5-1), including a septic system. Fireweed is currently expanding the camp to accommodate 50 persons with the addition of trailers and tents. The majority of historical drill core from both the Tom and Jason deposits, including holes drilled in 2011 and 2017, are stored just upstream from the Tom camp in and around a shed (Figure 5-5).

Hudbay excavated an adit and underground workings in stages between 1969 and 1982 to access the Tom West Zone for bulk sampling and underground drilling, for a total of 3,423 m of underground workings. The adit was subsequently partially plugged on 26 August 2010 to flood existing workings and reduce the flow of AMD from the opening. An upper level decline into the deposit was developed in 1982, also for exploration purposes, but was subsequently backfilled.

Figure 5-5: Core Storage Area Near the Tom Camp Viewed in 2017

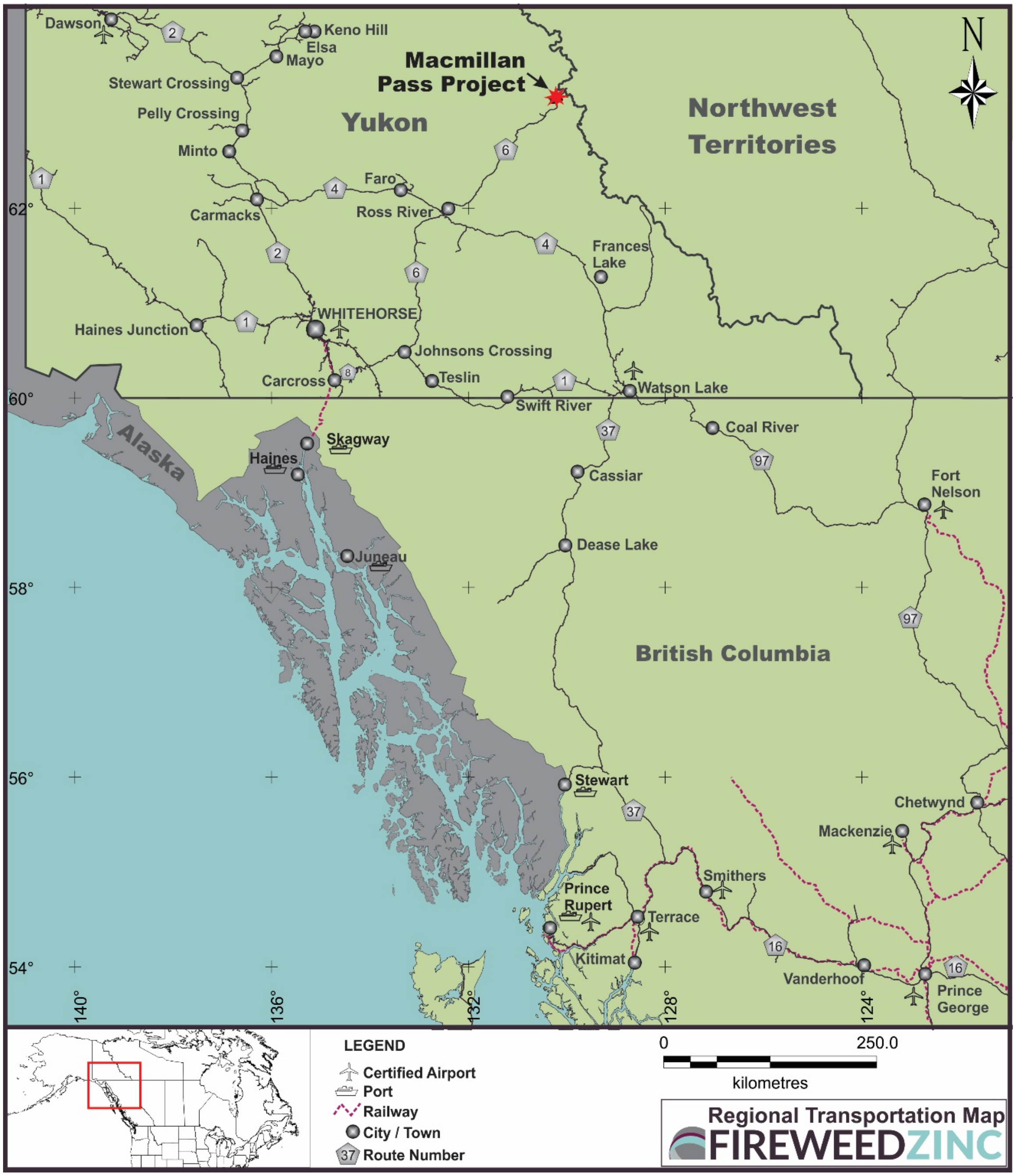
Note: Cross-piled core adjacent to the core shed is covered with fitted and ventilated nylon tarps.
Source: CSA Global (2018)

Project infrastructure needs in the event of potential development of the Tom and Jason deposits to production stage have not been assessed in detail, but CSA Global is of the opinion that existing property surface rights along with likely available government land are sufficient for potential mining operations, processing plant sites and waste and tailings storage areas, provided necessary permits are obtained and a satisfactory settlement is reached with the Kaska First Nations. Water is readily available, provided necessary permits can be obtained from the Yukon Water Board. The North Canol Road would require upgrading and the construction of a bridge across the South Macmillan River to the Jason area. Power needs would probably require installation of diesel or liquid natural gas generators at the site. The nearest year-round ice-free port facilities are in Skagway, Alaska and Stewart, British Columbia (Figure 5-6).

The city of Whitehorse, 600 km via road from the Project, is the major center of supplies and communications in the Yukon and is a source of skilled labor for exploration diamond drilling, construction and mining operations. There is daily jet airplane service from Whitehorse to Vancouver, British Columbia and other points south. The closest population centres to the Project via road (Figure 5-6) from which local supplies may be obtained are:

- Ross River (population 350; 200 km);
- Faro (population 400; 275 km);
- Carmacks (population 500; 435 km); and
- Watson Lake (population 1,200; 570 km).

Figure 5-6: Regional Transportation Map for the Macmillan Pass Project



Source: CSA Global (2018)

6 History

6.1 Ownership

6.1.1 Tom Property

The Tom property was held continuously by Hudbay through various subsidiaries since its discovery and staking in 1951, although it was temporarily optioned to Cominco Ltd between 1988 and 1992. On 14 December 2016, Hudbay signed a Definitive Option Agreement for the Tom and Jason properties with Fireweed as described in Section 4, and on 7 February 2018 Fireweed exercised the option and acquired 100% interest in the Tom and Jason claims.

6.1.2 Jason Property

The following history of ownership of the Jason property is taken largely from Rennie (2007). The Jason claims were first staked in 1971 by the Ogilvie Joint Venture. In 1975 the discovery drill hole on the Jason Main deposit was drilled. An interest in the property was obtained by Pan Ocean Oil Ltd in 1979 before being acquired by Aberford in 1981. Aberford's interest in the property was transferred to Abermin Corporation (Abermin) in 1985, and thence to CSA Gold Corporation (no connection to CSA Global). All parties transferred their interest to MacPass Resources Ltd and the property was then purchased by Hudbay in 2007 subject to a purchasable 3% NSR (see Section 4.3 – Property Agreements and Encumbrances, for details). On 14 December 2016, Hudbay signed a Definitive Option Agreement for the Tom and Jason assets with Fireweed as described in Section 4 and on 7 February 2018 Fireweed exercised the option and acquired 100% interest in the Tom and Jason claims and assets.

6.1.3 MAC Property

The MAC property was staked by Newmont in 2011 who carried out exploration for gold in 2011, 2012 and 2013. In August 2017, Fireweed signed an Option and Exploration Agreement for the property with Newmont which Agreement was recently transferred to Maverix (see Section 4.3 – Royalties, Agreements and Encumbrances, for details).

6.1.4 MC, MP and Jerry Claims

The MC and MP claims were staked by Carlin in March 2011, and the Jerry claims were staked by Constantine in March 2011. On 27 March 2018, Fireweed announced signing of an option agreement for the properties (see Section 4.3 – Royalties, Agreements and Encumbrances, for details).

6.1.5 BR and NS Claims

The BR and NS claims were staked by Golden Ridge Resources Ltd. ("GLDN") in March 2011 who carried out airborne geophysics and ground exploration work. On 27 March 2018, Fireweed announced signing of an option agreement for the properties (see Section 4.3 – Royalties, Agreements and Encumbrances, for details).

6.2 Project Results – Previous Owners

6.2.1 Tom Property

A brief history of exploration activity presented below is taken from Wells (2012). Key events include:

- Discovery of the Tom West Zone in 1951 with commencement of drilling in 1952;
- Discovery of the Tom East Zone in 1953;
- Commencement of Tom adit development in 1969 (lower adit) with 1,703 m of lateral development in 1970;
- Discovery of an extension to the Tom West Zone in 1979;
- Completion of a spiral decline in 1982 (upper adit);
- Optioning of the property to Cominco Ltd between 1988 and 1992;
- Partial plugging of the lower adit and covering of waste rock pile between 2007 and 2010;
- 201 drillholes totalling 31,672 m completed between 1952 and 2007; details of this drilling are provided by Rennie (2007);
- 11 additional diamond drillholes totalling 1,823 m were drilled for metallurgical and infill drilling at the Tom property in 2011, followed by metallurgical testing; and
- Orientation surface geochemical soil sampling surveys on the Tom and Jason properties in 2011.

The next material exploration work carried out on the Tom property was by Fireweed in 2017 which is described in Sections 9 and 10. Fireweed announced start of the 2018 field exploration program on 5 June 2018

6.2.2 Jason Property

The following summary of exploration is taken from Rennie (2007) and includes:

- Drilling of 87 holes, including 45 diamond and 33 rotary overburden holes, between 1974 and 1978;
- Drilling of 42 diamond drillholes between 1980 and 1982 for a total of 128 historical diamond and rotary holes totalling 37,924 m. Details of this drilling are provided by Rennie (2007). No drilling occurred on the property since 1991 until Fireweed's 2017 drill program;
- An option of the property to Phelps Dodge Corporation of Canada between 1990 and 1992; and
- Purchase of the Jason claims by Hudbay in 2006.

CSA Global are unaware of any material exploration on the Jason property undertaken since 1992 until the work carried out by Fireweed in 2017.

A majority of the historical exploration work carried out at Tom and Jason was drilling with the goal of defining economic resources.

6.2.3 MAC Property

Newmont carried out reconnaissance exploration for gold in 2011 (stream sediment BLEG – bulk leach extractable gold), 2012 (a small ridge and spur soil sampling program) and 2013 (ridge and spur soil sampling, mapping and prospecting). This work outlined several gold anomalous areas as well as zinc, lead and silver anomalies. Smits (2014) is an unfinished report which included maps of results but no interpretations or conclusions.

6.2.4 MC, MP and Jerry Claims

The following information was compiled from Yukon government assessment files which indicates that the MC, MP and Jerry claims have seen limited exploration work in the past.

The MC claims saw limited intermittent surface exploration between 1973 and 1998 by various groups. This work identified the Walt (Yukon Minfile No. 105O 021) and Tryala (Yukon Minfile No. 105O 022) barite occurrences and a low grade zinc horizon.

At the MP claims, at total of 131 soil samples, 7 stream silt samples and 14 rock samples were collected in 2011.

At the Jerry claims, 51 soil samples were collected by Cominco Limited in 1991 with low results. In 2011 an additional 317 soil samples, 46 stream silt samples and 57 rock samples were collected with some anomalous base metal values that Fireweed indicated it intends to investigate in 2018.

6.2.5 BR and NS Claims

The following information is summarized from Cathro (2012):

The BR and NS claims cover four Yukon Minfile showings: Bremner copper-lead-zinc prospect (Yukon Minfile No. 105O 025); the Sim tungsten skarn showing (Yukon Minfile No. 105O 043); and two barite-zinc occurrences named Bailes (Yukon Minfile No. 105O 052) and Hasten (Yukon Minfile No. 105O 043). The Bremner showing has seen 324m of drilling in 1978 alongside a limited program of geochemistry and ground geophysics. The Sim showing saw limited geochemical sampling and mapping in 1981-82. In 1982-83, the Bailes barite-zinc occurrence was explored with 1011 soil, 4 stream silt and 41 rock samples plus 43.9 km of VLF-EM geophysics, 43.7 km of magnetics, and limited hand trenching. Eighteen VLF-EM anomalies were outlined, some coincident with soil anomalies. The Bailes prospect has not been drilled. The Hasten prospect was explored by Cominco in 1983 with one 302m drillhole. In 2011, GLDN carried out airborne geophysics over the BR and NS claims identifying a number of anomalies. Fireweed indicated that they intend to further evaluate these areas in 2018.

6.3 Historical Mineral Resource Estimates

In 2007, Scott Wilson of Roscoe Postle Associates Inc. (RPA) completed a MRE on the Tom and Jason deposits for Hudbay in accordance with NI 43-101 of that time (Rennie, 2007). After those estimates were made, there were 11 diamond holes drilled at Tom in 2011 by Hudbay, seven holes drilled at Tom in 2017 by Fireweed and seven holes drilled at Jason in 2017 by Fireweed. These new diamond holes along with other work carried out in 2017 by Fireweed, led to the current MRE described in this report (see Section 14 – Mineral Resource Estimates).

The 2007 MRE is not in compliance with current NI 43-101 standards. Results are presented in Section 14.12.2.

Drill cores, historic and recent, from both the Tom and Jason deposits are stored just upstream from the Tom camp (Figure 5-5). Much of the Tom deposit core is stored in a metal shed and the Jason deposit core was transported and cross-piled beside the shed and protected by thick vinyl covers in 2015. Hudbay carried out an inventory of the core stored in the building and reported that 79 holes are stored there, comprising some 4,000 boxes with 11,500 m of core. Some core was donated to the Yukon Geological Survey H.S. Bostock core library in Whitehorse where it is accessible for viewing and, with permission, sampling. This includes core from 70 drillholes from the Tom deposit, mainly from underground, and core from 20 Jason drillholes. Some core drilled from surface prior to 1975 was dumped in with mine waste and covered during rehabilitation of the site in 2010. One of the Authors (Dennis Arne) viewed the core stored at the Tom site during his 2017 visit and confirms it is generally in good condition given its age.

6.4 Production History

There is no known production from the property. An exploration adit and decline were excavated for underground bulk sampling and exploration purposes at the Tom West deposit in stages between 1969 and 1982.

7 Geological Setting and Mineralization

7.1 Regional Geology

The regional geology of the Tom and Jason properties has previously been described by Rennie (2007), Goodfellow (2007) and Wells (2012). A summary is presented here from those sources.

7.1.1 Stratigraphy

The Macmillan Pass Project lies within the Selwyn Basin (Figure 7-1), a deep water marine basin that was initiated off the ancestral coast of North America during the late Proterozoic era with deposition continuing through the early to middle Paleozoic era. The Selwyn Basin consists of a package of sedimentary rocks beginning with continentally-derived sediments of the late Proterozoic to Cambrian Windermere Supergroup. These units were overlain in the late Cambrian to Ordovician by carbonate rocks of the Rabbitkettle Formation, and then by deep water cherts and shales of the Ordovician to early Devonian Road River Group. The Road River Group is in turn overlain by chert, black shales and turbidite sediments of the Devonian to Mississippian Earn Group, the host to the Tom and Jason deposits, as well as other zinc-lead-silver and barite mineralization in the Macmillan Pass region (Figure 7-2).

The stratigraphy of the Selwyn Basin and the adjacent Mackenzie carbonate platform that existed to the north and east of the basin (Figure 7-1) is given in Figure 7-3. A detailed stratigraphic description of the Macmillan Pass area is available in Abbott and Turner (1991).

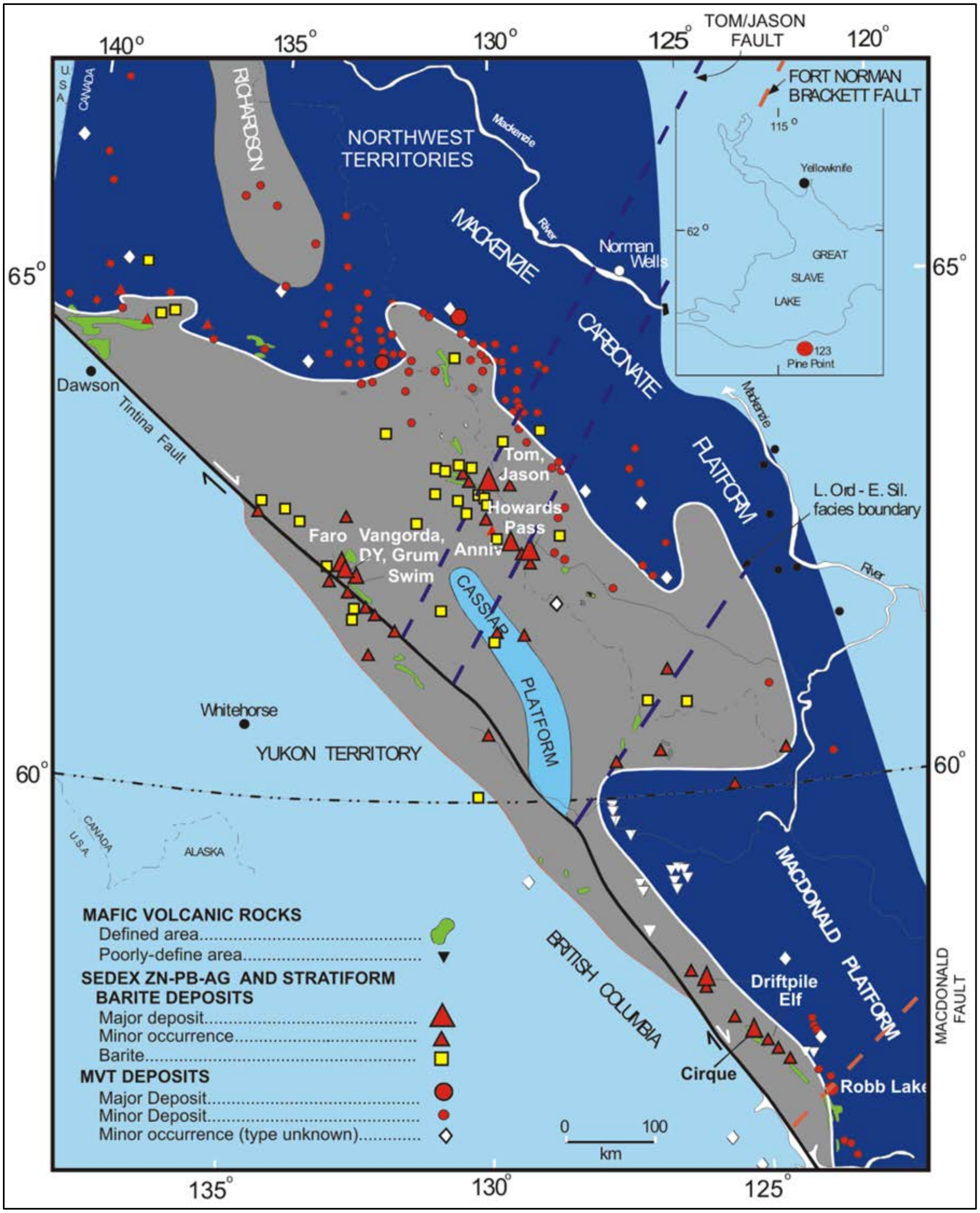
7.1.2 Magmatism

Locally, mafic volcanic rocks were erupted during deposition of both the Road River and Earn Groups and coincide regionally with the formation of zinc-lead-silver and barite deposits in the Selwyn Basin. The region was intruded by quartz monzonite plutons during the waning stages of the Jurassic to Cretaceous periods.

7.1.3 Regional Tectonics and Structure

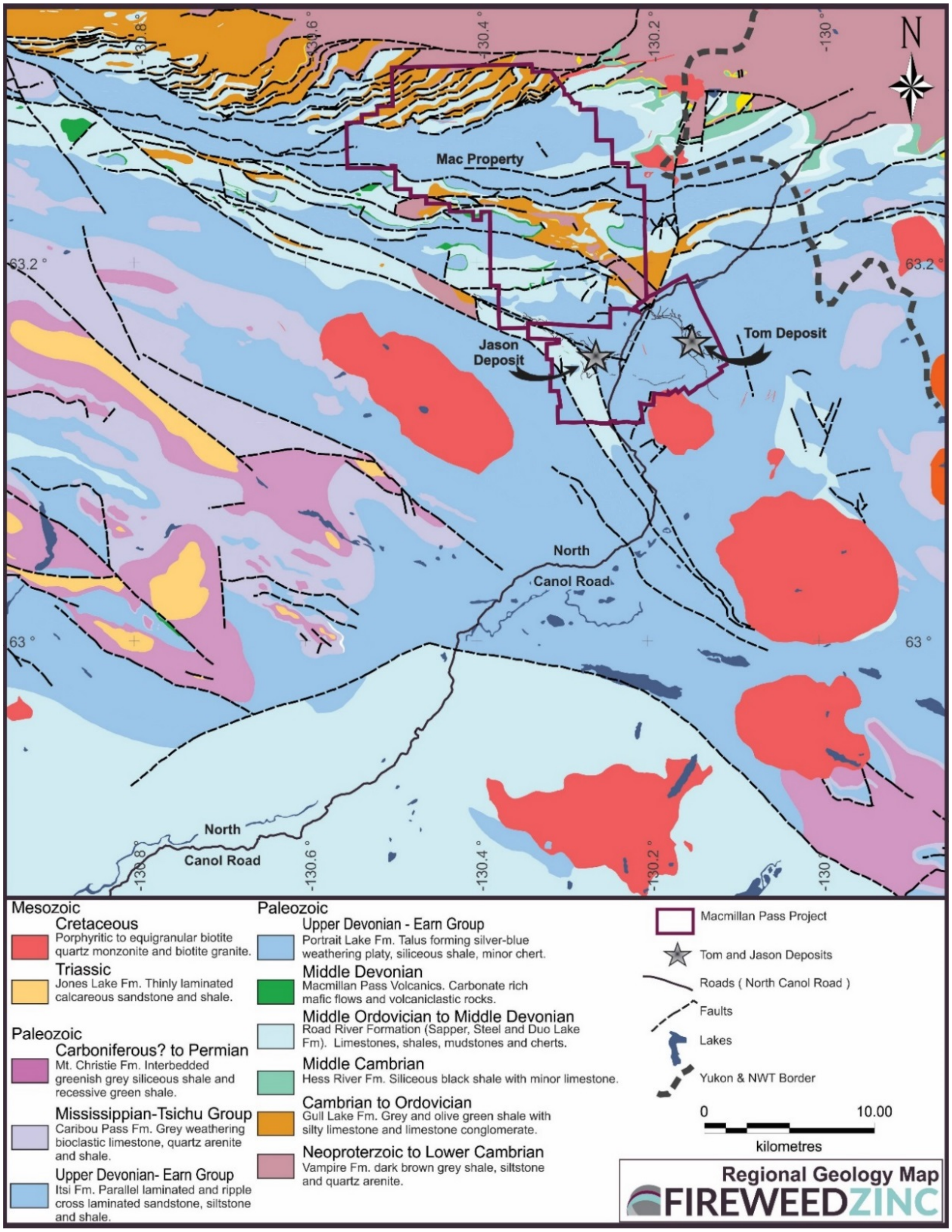
The Selwyn Basin formed in a passive margin ocean setting following a major phase of rifting in the late Proterozoic to Cambrian. Gradual subsidence continued through the Paleozoic until the Antler Orogeny in the Devonian, at which time intracontinental rifting was initiated in a back-arc graben setting in the Macmillan Pass region. Extension faults controlling the circulation of hydrothermal fluids were active at this time and are characterized by significant thickness variations in stratigraphic units across the structures, consistent with growth faulting, and the presence of sedimentary breccias, mass flow deposits (diamictites) and conglomerates indicative of syn-sedimentary faulting. The region was subject to compression during regional east-west shortening during the Jurassic to Cretaceous, resulting in likely re-activation of normal faults, folding and thrust faulting. The Macmillan Pass region occurs in the Central Block of the Macmillan Fold Belt where south-verging thrust faults and folds may be truncated by strike-slip re-activation of Devonian normal faults (Abbott et al., 1991).

Figure 7-1: Regional Geological Setting and Zinc-Lead-Silver Deposits of the Selwyn Basin, including the Tom and Jason Deposits (from Goodfellow, 2007)



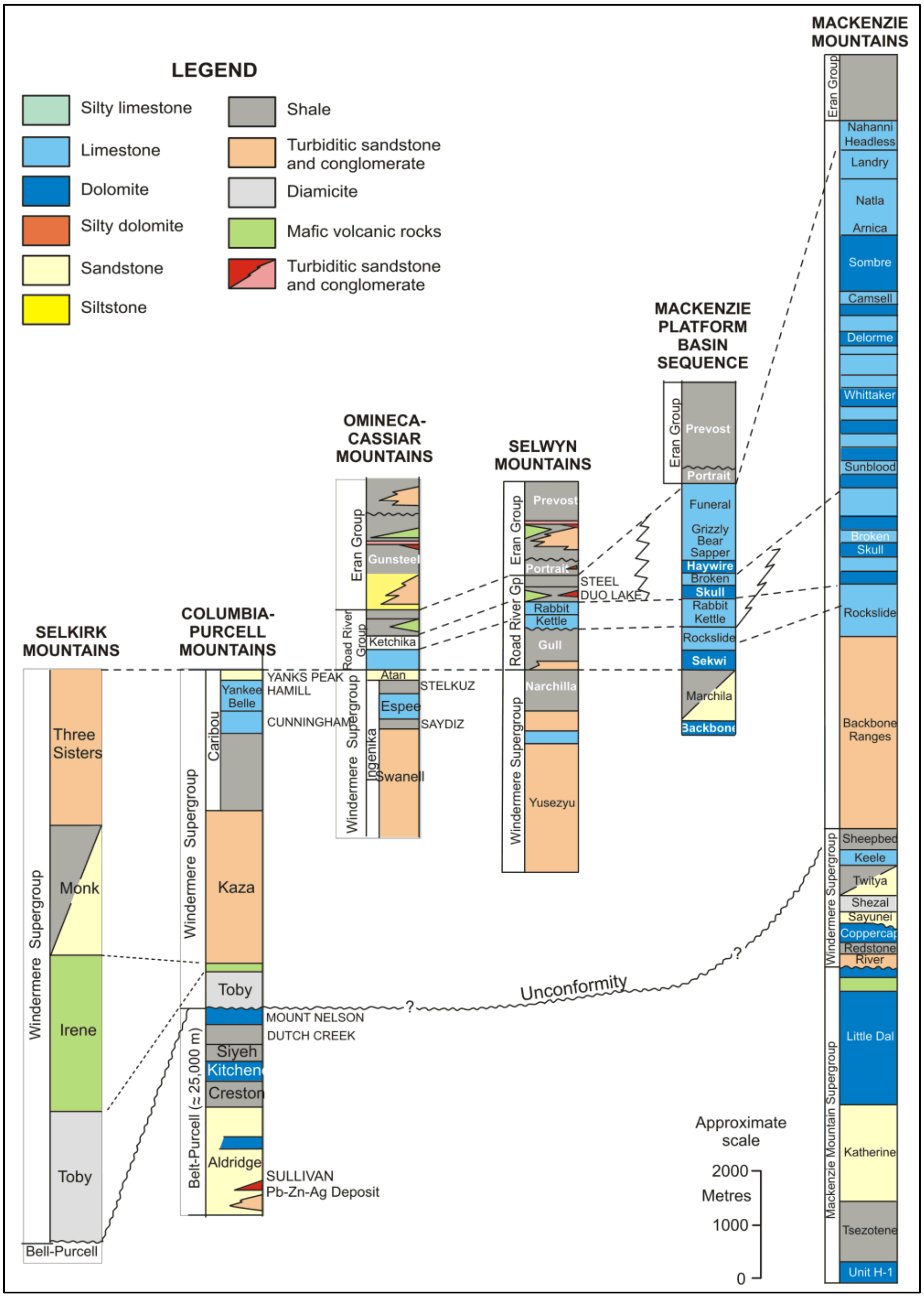
Source: CSA Global (2018)

Figure 7-2: Geology of the Macmillan Pass Region (modified from Abbott and Turner, 1991)



Source: CSA Global (2018)

Figure 7-3: Stratigraphy of the Selwyn Basin and Belt Purcell Group (from Goodfellow, 2007)



Source: CSA Global (2018)

7.2 Local Geology

The local geology of the Project area is presented in Figure 7-4 and the local stratigraphy is summarized in Figure 7-5. Detailed descriptions are provided by Turner (1991) for the Jason deposit and Goodfellow (1991) for the Tom deposit. Summary descriptions of both deposits are provided in Rennie (2007) and Goodfellow (2007). The following descriptions are taken from those sources.

7.2.1 Tom Deposit

The Tom deposit is hosted by the Portrait Lake Formation of the Devonian Earn Group. Specifically, sulphide mineralization occurs within an informal unit called the Tom Sequence (Goodfellow, 1991). The Tom Sequence is characterized by abrupt changes in sedimentary facies and unit thickness, demonstrating the influence of syn-sedimentary faulting. It consists of well banded carbonaceous and radiolarian chert, with occasional sandier intervals, barite nodules and pyrite laminae. It overlies sandy to silty laminated shales and siltstones of the MacMillan Pass Member which are interpreted to have been deposited by deep water turbidites (Goodfellow, 1991). The shales and siltstones are interbedded with occasional detrital chert layers and chert pebble conglomerates, and with mixed clast diamictite, both indicative of submarine slumping near syn-sedimentary faulting. The Tom Sequence is unconformably overlain by fine grained clastic rocks of the informal Itsi Member. The sequence has been folded about a steeply south to southeast plunging upright anticline (Figure 7-6). The Tom Sequence is well exposed near the Tom deposit, although it is locally displaced along scree slopes and disrupted by frost heave in the alpine areas.

7.2.2 Jason Deposit

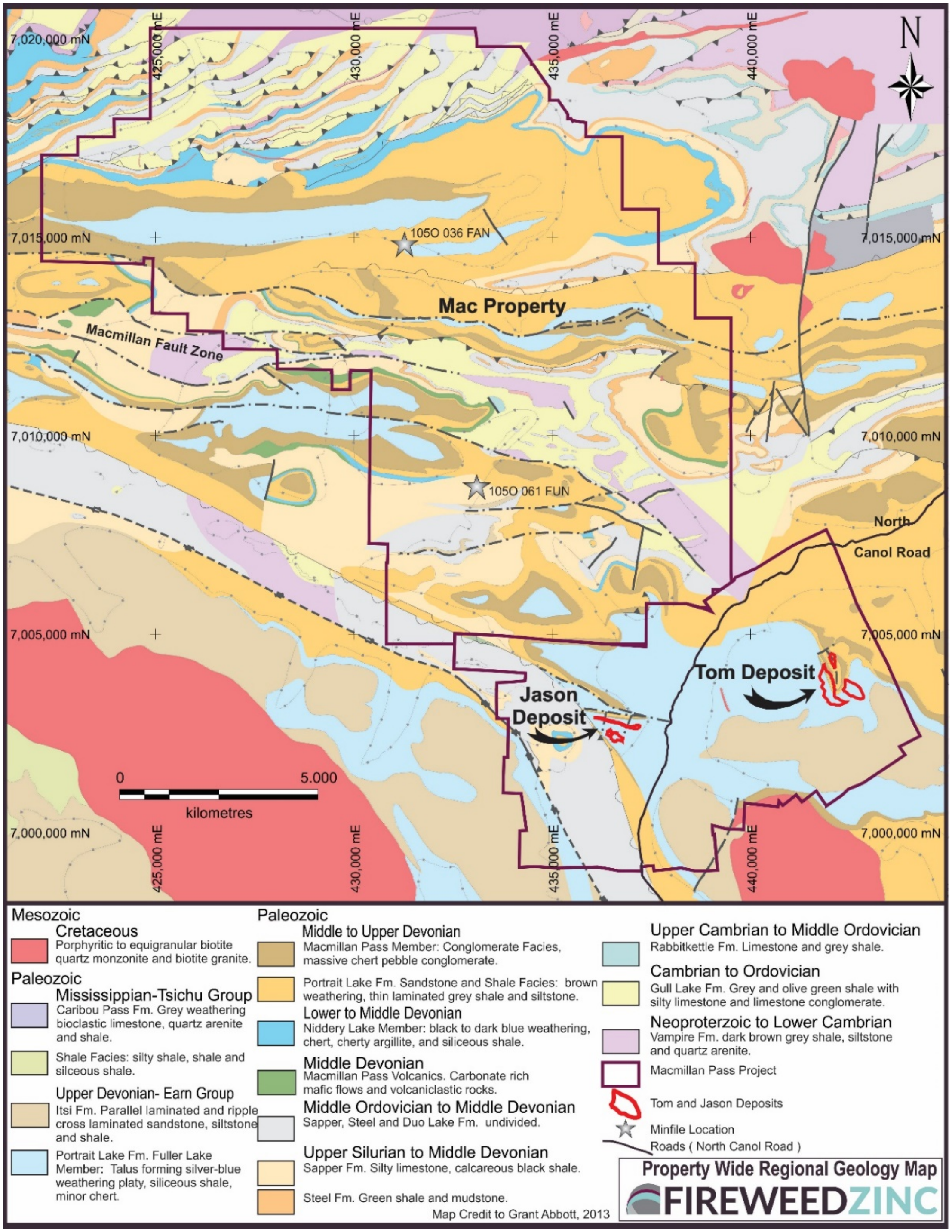
The Jason deposit is hosted by a Devonian sequence disrupted by the Hess Fault and folded into a series of “upright tight west-trending, shallowly east-plunging folds” (Turner, 1991) (Figure 7-7). The position of the Jason deposit is controlled by the location of the Jason Fault, a syn-sedimentary growth fault that brings older rocks of the Road River Group and lower Portrait Lake Formation of the Earn Group into contact with the Macmillan Pass Member and a stratigraphic package considered to be the lateral equivalent of the Tom Sequence (Goodfellow, 1991). The latter contains well developed sedimentary breccias, conglomerates and mass flow deposits (diamictites) that thicken towards the position of the Jason Fault, consistent with syn-sedimentary fault movement. Bedrock exposure is good within the alpine areas, but the valley bottoms and walls at lower elevations are concealed by a blanket of till that has inhibited exploration.

7.3 Regional Mineralization

The following information on regional SEDEX zinc-lead-silver mineralization is taken from Goodfellow and Lydon (2007) and Goodfellow (2007).

The Selwyn Basin is one of the most prolific basins for SEDEX zinc-lead-silver deposits in the world. The basin hosts 12 large deposits including the Tom and Jason deposits, the subject of this report (Figure 7-1). Past producers were Faro (aka Anvil), Grum and Vangorda. The Howards Pass deposit (aka Selwyn) is currently one of the world’s largest undeveloped zinc deposits. SEDEX mineralization of the Selwyn Basin occurs in four main districts of different ages: Anvil/Faro (Cambrian), Howards Pass/Selwyn (Silurian), Gataga / Cirque (Late Devonian) and Macmillan Pass/Tom-Jason (Late Devonian). Synchronous and genetically related Mississippi Valley Type zinc-lead mineralization occurs in the carbonate platforms along the east side of the Selwyn Basin (Figure 7-1).

Figure 7-4: Geology of the Macmillan Pass Project Area (from Abbott, 2015)



Note: Two base metal occurrences on the Mac claim group (Fan and Fun) are also shown.

Source: CSA Global (2018)

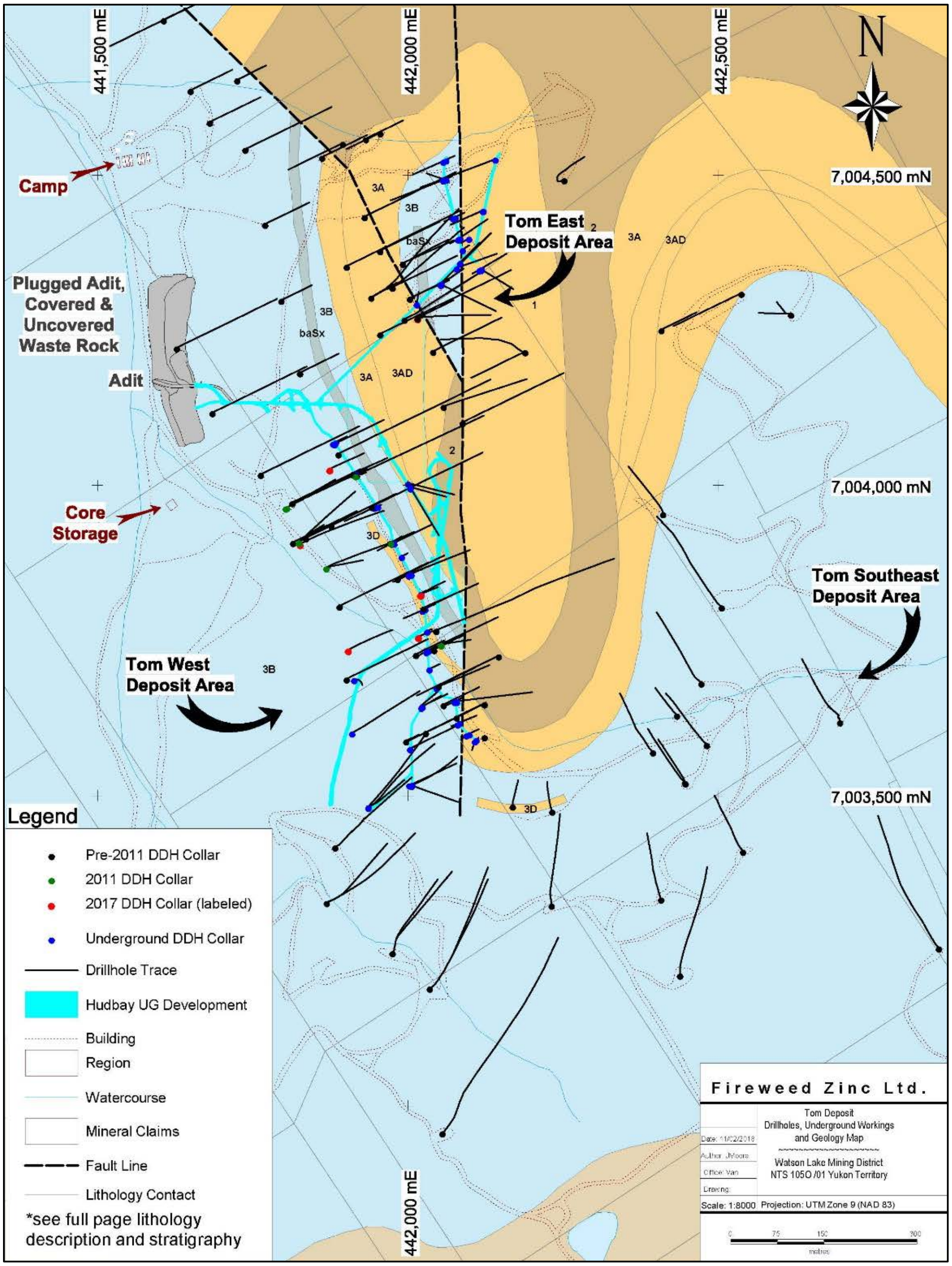
Figure 7-5: Stratigraphic Column for the Macmillan Pass Project Area

Period			Unit	Description	Hydrothermal Alteration & Hydrothermal Deposits
Cretaceous			5	Quartz monzonite Quartz feldspar porphyry dykes	
Mississippian	Earn Group Portrait Lake Fm	Itsi Member	4	Argillite, siltstone, sandstone, massive to thickly bedded, parallel and cross-laminated, rusty.	Pyritic hornfels adjacent to quartz monzonite
Devonian		Tom Sequence	3B	Black, carbonaceous, siliceous mudstone,local spotted bright horizons.Laminated sulphide, sulphate deposits (Tom, Jason) near base.	Tom-Jason Horizon: Laminated barite, galena, sphalerite, pyrite, chert interbedded with 3B or 3D. BaSx.
			3D	Diamictite Unit - Local fault scarp breccias,homolithic argillite breccias, polylithic breccias, interbedded silt-banded argillites.	
		Macmillan Pass Member	3A 3D	3A-Silt and sand banded argillite, local intraformational breccias are at base, local-3AD	Iron carbonate flooding of permeable layers, cross cutting iron carbonate veining and quartz veining, silicification.
			2	2-Dominantly chert pebble conglomerate, with chert grit and sandstone, argillite and silt banded argillite	
			1	Silt and sand banded argillite Local massive argillite	
Ordovician Silurian Devonian	Road River Group		RR	Mudstone, chert, local calcareous siltstone, limestone; fossiliferous; rare volcanics	Iron carbonate and quartz veining.

Geological interpretation by R.Cameron, Fox Geological Consultants Ltd.

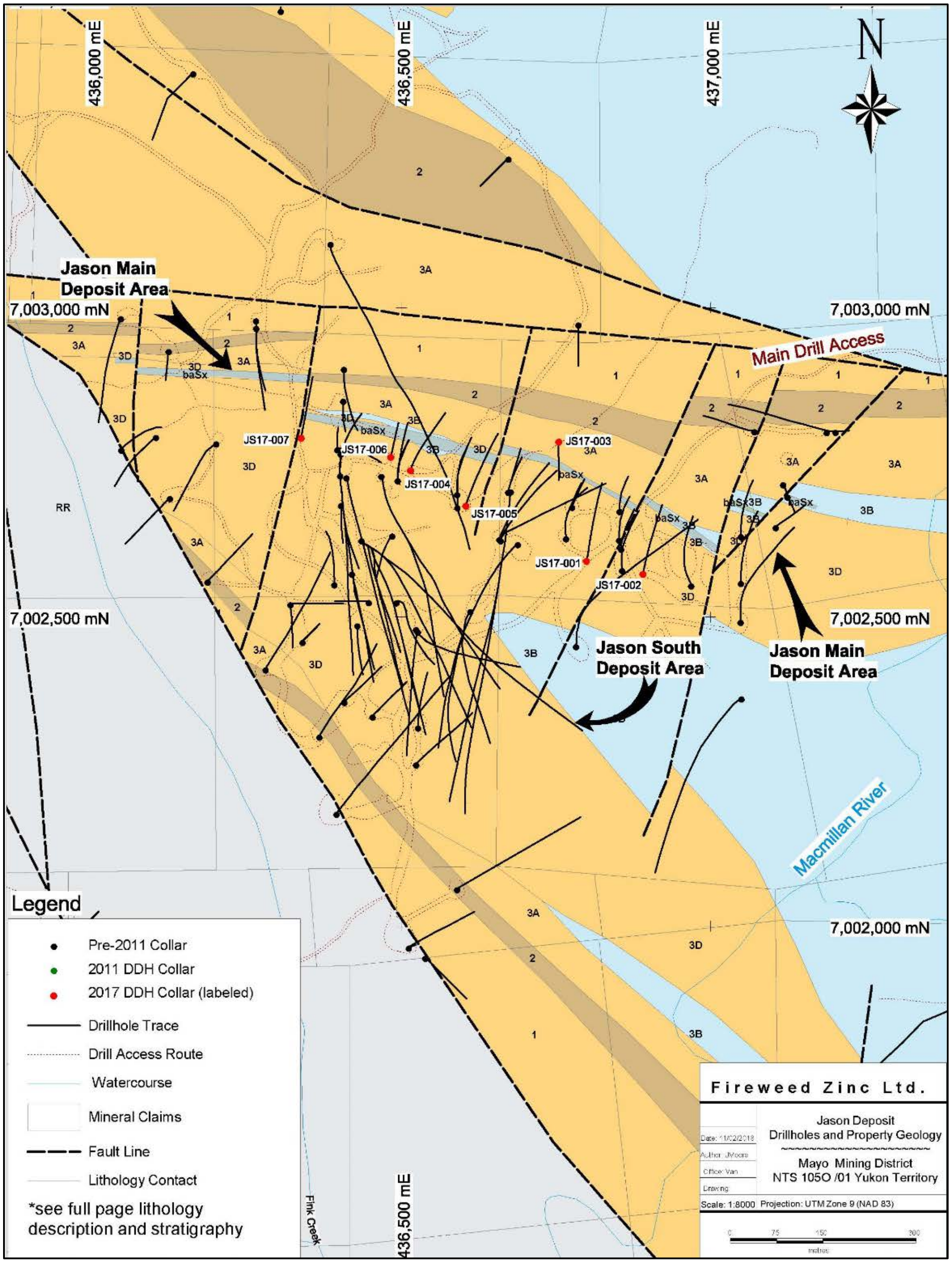
Source: CSA Global (2018)

Figure 7-6: Geology of the Tom Deposit (Historical and Recent Drillhole Collar Locations are also shown)



Source: CSA Global (2018)

Figure 7-7: Geology of the Jason Deposit (Historical and Recent Drillhole Collar Locations and Traces are also shown)



Source: CSA Global (2018)

7.4 Property Mineralization

Detailed descriptions of the Jason and Tom deposits are provided by Turner (1991) and Goodfellow (1991), respectively. The following descriptions of the Tom and Jason deposits have been taken from summaries by Goodfellow (2007) and Rennie (2007).

7.4.1 Tom Deposit

Zinc-lead-silver-barite mineralization at the Tom deposit varies from well laminated and stratiform (parallel to sedimentary layering) to a brecciated stockwork zone adjacent to the Tom normal fault (Figure 7-8). The Tom West and Tom East zones, both of which are exposed at surface (Figure 7-9), are interpreted to have formed one continuous strata-bound controlled lens prior to folding and faulting of the Tom Sequence, whereas the Southeast Zone is interpreted to have formed in a separate sub-basin to the main graben structure hosting the Tom West and Tom East zones (Goodfellow, 1991). All three zones have been affected by folding (Figure 7-6), with evidence for the possible development of a crenulation cleavage (Figure 7-10) as opposed to the chaotic folding of laminae due to soft-sediment deformation (Figure 7-11). Ferroan carbonate alteration and quartz veining are common in footwall conglomerates near vent facies at Tom West.

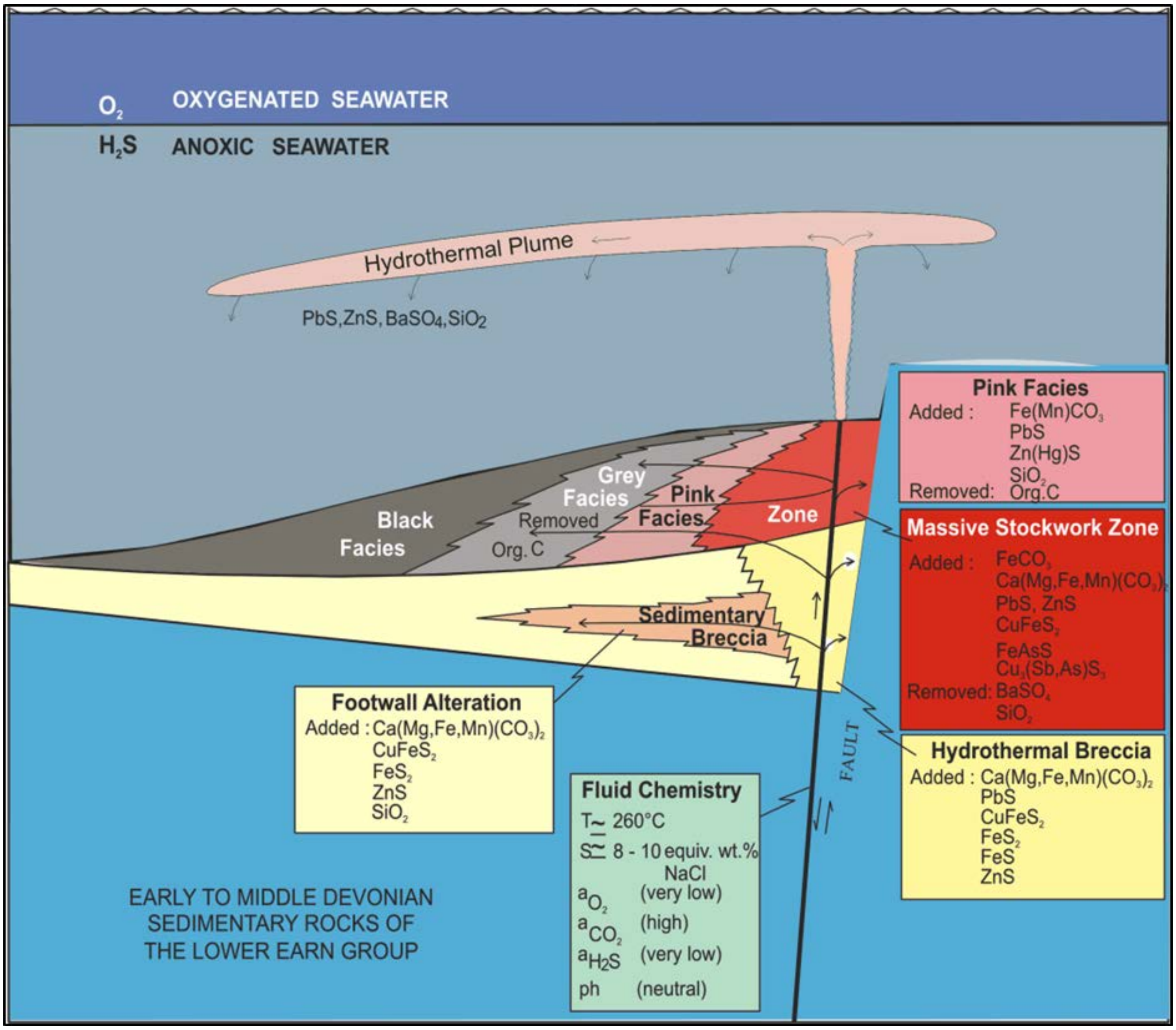
The Tom West Zone dips 60° to the southwest, has a strike extent of approximately 1 km and extends up to 400 m down dip. It is about 40 m thick at its widest point and breaks into two discrete layers in the centre at depth. Contacts vary from transition over <1 m (Figure 7-12) or are faulted and abrupt. The highest-grade portion of the Tom West Zone occurs along the southern and near surface portion of the zone where Pb+Zn grades exceed 10% with elevated silver. The Tom West Zone hosts the bulk of the resource at the Tom deposit.

The Tom West Zone can be divided into a series of mineralization facies (after Goodfellow, 1991; 2007) consisting of:

- Vent facies – Stockwork of pyrite, pyrrhotite, galena, sphalerite, with minor chalcopyrite, arsenopyrite and tetrahedrite with a gangue of ferroan carbonates, quartz and barite subdivided into five types, including an upper high-grade zone with 15–30% Pb+Zn, Ag between 150 g/t and 200 g/t and a low Zn/(Zn+Pb) ratio;
- Pink facies – Interbedded barite, chert, cream-coloured sphalerite, fine grained pyrite and black Ba-carbonate, overprinted by pink and yellow sphalerite resulting in locally high grades in the range of 10–30% combined Pb and Zn;
- Gray facies – Interbedded pink sphalerite, fine grained galena and pyrite, white to pale gray barite, pale grey chert and grey to white Ba-carbonate / Ba-feldspar, typically with grades in the range of 4–5% Pb+Zn with negligible Ag;
- Black facies – Black mudstone and chert interbedded with barite, witherite (Ba-carbonate) and fine-grained sphalerite, galena and pyrite, typically with grades in the 4–10% Pb+Zn range and a high Zn/(Pb+Zn) ratio;
- The Tom East Zone occurs near the hinge of the anticline that has folded the originally planar deposit, and which plunges northward in this area. It consists of interbedded high-grade sphalerite, galena, barite and chert thought to have formed within the same stratigraphic interval as Tom West (McClay and Bidwell, 1986); and

- The Tom Southeast Zone is not exposed at surface, and consists of a tabular, stratiform body 0.5 m to 6 m thick with a strike length of approximately 400 m and a down-dip extension of at least 350 m dipping 60–70° to the east. It is located near the nose of the southeast-plunging Tom anticline on its eastern limb. Mineralization consists of finely laminated sphalerite, galena, pyrite and black cherty mudstone (Goodfellow, 1991).

Figure 7-8: Schematic Stratigraphic Reconstruction of the Mineralization Facies (zones) at the Tom Deposit (from Goodfellow, 2007)



Source: CSA Global (2018)

Figure 7-9: View of the Tom West Zone (defined by black lines) Exposed at Surface

Note: Bulldozer for scale below the mineralized band.
Source: CSA Global (2018)

Figure 7-10: Minor Folds of Likely Tectonic Origin in Gray Facies Mineralization from Tom West

Note: NQ diameter drill core from TYK-010 drilled in 2011 (depth 103.5 m).
Source: CSA Global (2018)

Figure 7-11: Chaotic Folds Related to Soft-Sediment Deformation in Gray Facies Mineralization from Tom West

Note: NQ diameter drill core from TYK-010 drilled in 2011 (depth 104.7 m).
Source: CSA Global (2018)

Figure 7-12: Hangingwall and Footwall Contacts for Tom West in Hole TYK-006 Drilled in 2011

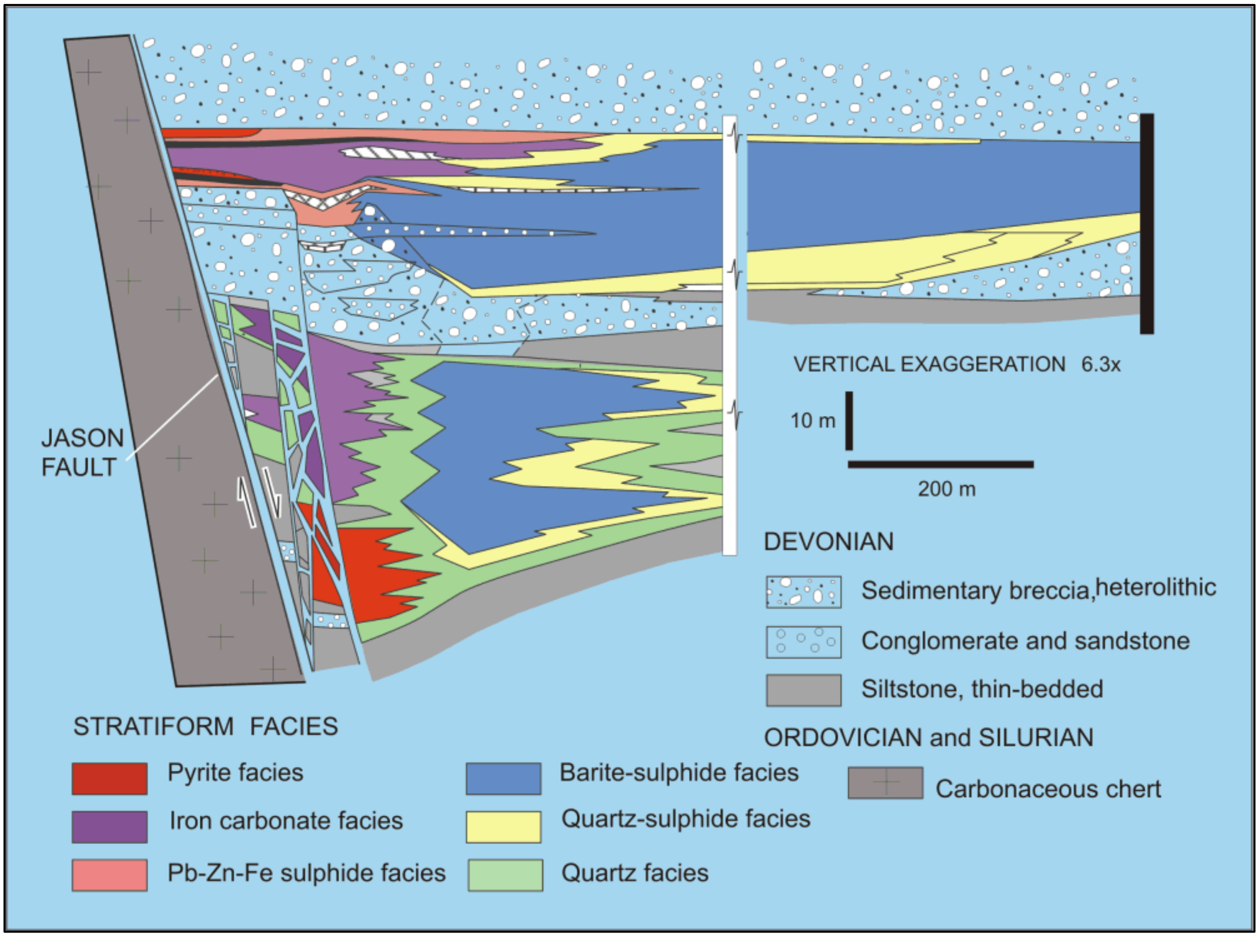
Note: core boxes from the main zone are missing from this image.
Source: CSA Global (2018)

7.4.2 Jason Deposit

A stratigraphic reconstruction of the Jason deposit at the time of mineralization is presented in Figure 7-13. The Jason Main Zone is located on the northern limb of the east-plunging Jason syncline, while the Jason South Zone occurs on the southern limb (Figure 7-7). The South Zone consists of two separate horizons whereas the Main zone is defined by a single horizon. These two separate zones are likely connected through the hinge of a syncline, but this has yet to be demonstrated through drilling. These horizons can be divided into several distinct mineralization facies (zones), including (after Turner, 1991):

- Pb-Zn-Fe sulphide facies – Massive, banded sphalerite-galena and galena-pyrite overlain by debris flow deposits containing clasts of earlier deposited massive sulphides;
- Barite-sulphide facies – Interbedded fine-grained sphalerite, galena, barite, chert and ferroan carbonate forming the bulk of the mineralization at Jason;
- Quartz-sulphide facies – Interbedded sphalerite, pyrite, quartz and carbonaceous chert with quartz-celsian (barium feldspar) bands in the lower lens;
- Massive pyrite facies – Massive pyrite beds interbedded with sphalerite, galena, chalcopyrite, pyrrhotite and quartz located near the Jason Fault; and
- Ferroan carbonate facies – Massive beds of siderite and ankerite up to several metres across with irregularly distributed galena, sphalerite, pyrrhotite, pyrite, quartz, muscovite and pyrobitumen; spatially associated with a breccia pipe.

Figure 7-13: Stratigraphic Reconstruction of the Mineralization Facies (Zones) at the Jason Deposit (from Goodfellow, 2007)



Source: CSA Global (2018)

8 Deposit Types

The Tom and Jason deposits are examples of stratiform, strata-bound sediment-hosted, exhalative ("SEDEX") zinc-lead-silver-barite deposits (Figure 8-1; Goodfellow et al., 1993; Leach et al., 2005; Goodfellow et al., 2007; Goodfellow, 2007). Historically the term SEDEX was first used in a report describing the zinc-lead-silver deposits of the Selwyn Basin by Carne and Cathro (1982) and since then the term has been used to describe these deposits worldwide. SEDEX deposits (also known as clastic-dominated or CD deposits) formed in rift basins primarily in the late Paleoproterozoic and in the early Phanerozoic, with typical grades of 10% combined Pb+Zn in producing mines.

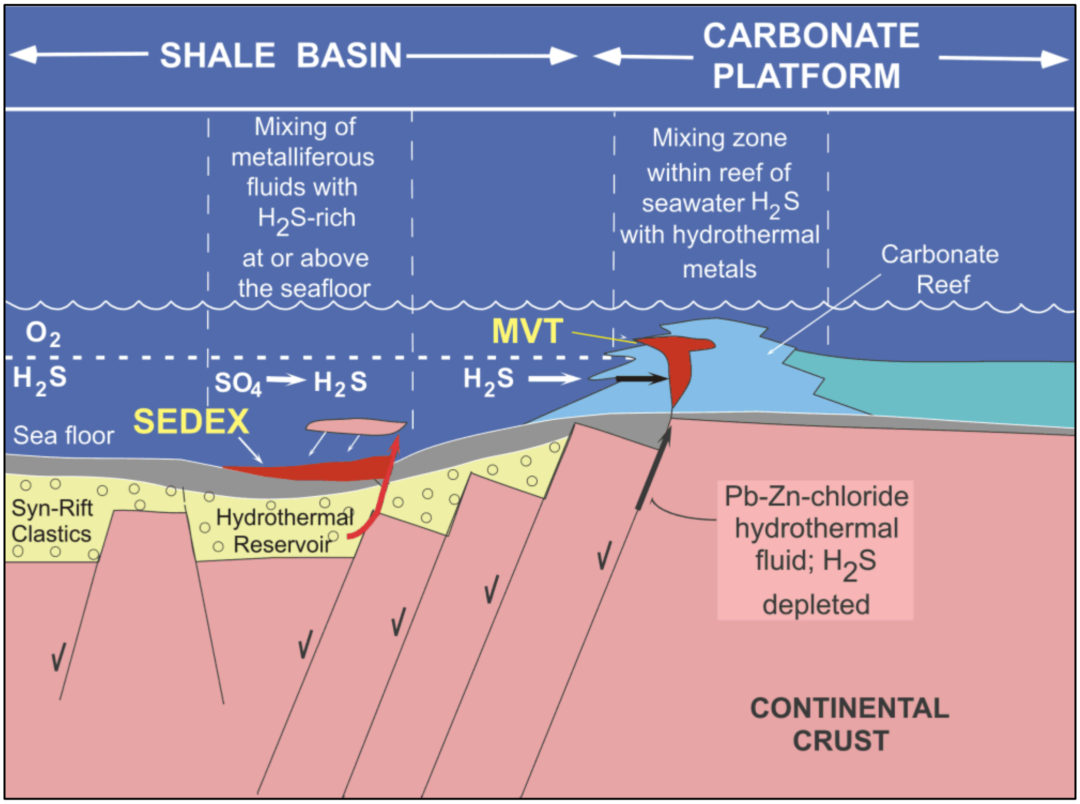
Mineralization is interpreted to have formed at or close to the seawater-sediment interface either proximal or distal to syn-sedimentary faults that controlled the subseafloor movement of mineralizing fluids and possible related submarine exhalative vents (Figure 8-2). Euxinic conditions may have been present during deposition of sulphides, but these may not have been necessary (e.g. Magnall *et al.,* 2015). The more distal deposits are therefore largely stratiform in nature in that the mineralized zones are concordant with sedimentary layering, whereas proximal deposits show more complex metal zonation and replacement textures. Proximal deposits are more closely linked spatially with syn-sedimentary feeder faults. A clear understanding of structural geology and stratigraphy are therefore important aspects of exploration for SEDEX mineralization. Metal ratios, such as Ag/Pb, Pb/(Pb+Zn), Cu/((Zn+Pb), Zn/Fe and Zn/Ba typically increase towards the feeder faults and vents providing a vector towards the central and potentially higher-grade parts of the hydrothermal system. Both the Tom and Jason deposits are proximal SEDEX deposits (Goodfellow, 2007).

Other important guides to exploration for SEDEX mineralization include (after Goodfellow, 2007):

- The presence of footwall feeder zones involving silicification of the footwall sedimentary package, brecciation, veining and trace element enrichments (Cu, Co, Ni, Mo, As, Sb, Zn, Cd, Pb and Hg);
- Laterally extensive stratigraphic horizons equivalent to the main deposit lens with elevated Zn, Cd, As and Hg;
- Hangingwall alteration characterized by elevated Ba, Zn and pyrite enriched in Co, Ni and Cu;
- The presence of pyrite and/or pyrrhotite in vent complexes that may be detectable by electrical and/or electromagnetic geophysical exploration methods; and
- Positive gravity anomalies that may be directly indicative of massive sulphide concentrations at depth.

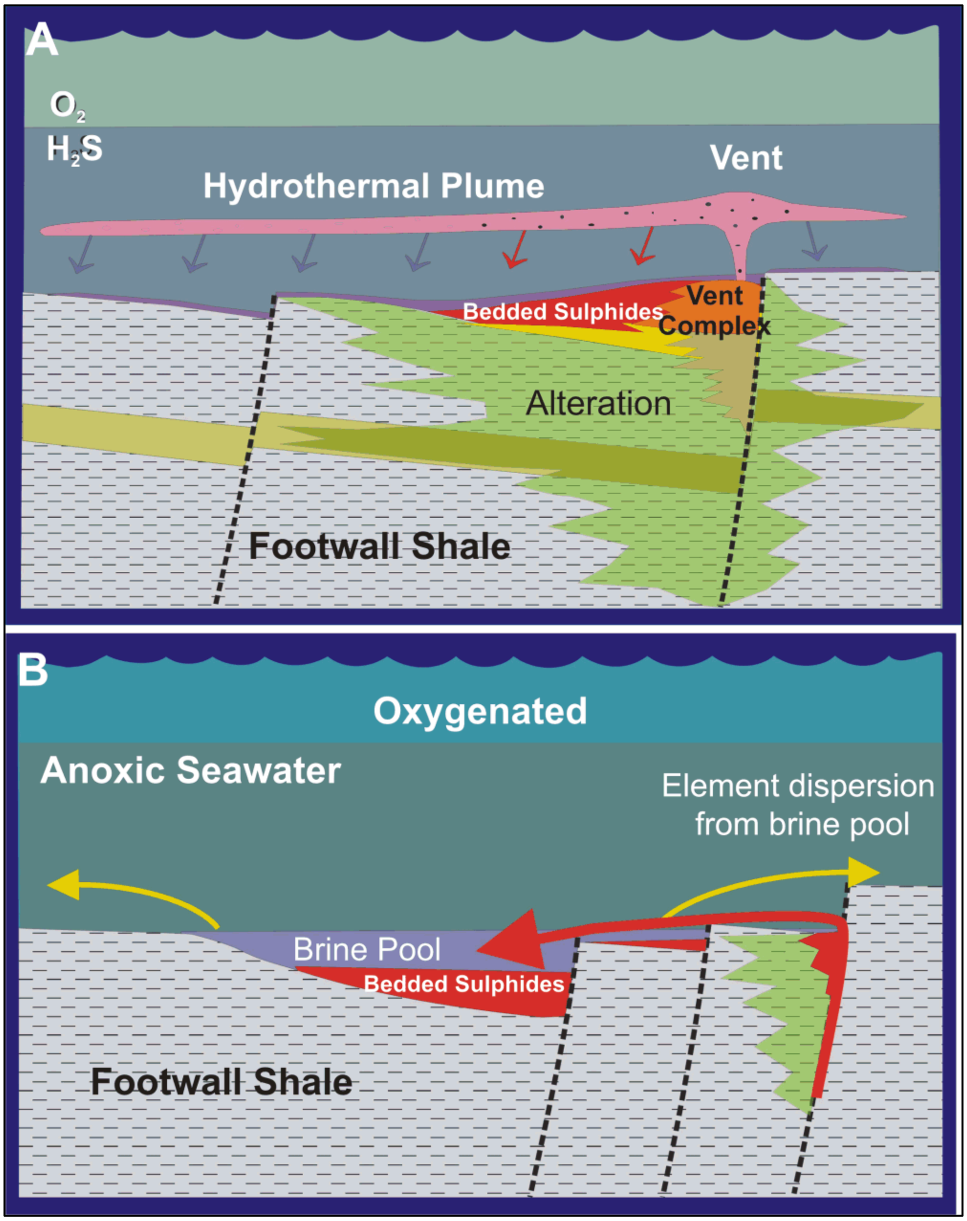
Many of the exploration guides described in this section were developed through extensive research into the Tom and Jason deposits, as well as into other SEDEX deposits found within the Selwyn Basin. Much of this research was carried out by the Geological Survey of Canada (GSC) prior to 1991. There has been little in the way of meaningful exploration work carried out on the Tom and Jason properties since this research was completed and many of the concepts developed by the GSC have not yet been tested by modern exploration.

Figure 8-1: Conceptual Models for SEDEX and MVT (Mississippi Valley-type) Pb-Zn Deposits (from Goodfellow, 2007)



Source: CSA Global (2018)

Figure 8-2: Conceptual Models for Proximal (A) and Distal (B) SEDEX Deposits (from Goodfellow, 2007)



Source: CSA Global (2018)

9 Exploration

In 2017, Fireweed carried out a program of drilling, mapping, sampling, LiDAR topographic mapping and airborne geophysics on the property. Drilling totalled 936 m in seven holes on the Tom deposits and 1,266 m in seven holes on the Jason deposits. The following summary is taken from Fireweed's news release dated 27 December 2017. Drilling results from 2017 are described in Section 10 (Drilling).

9.1 Airborne Geophysics

The airborne geophysics program was designed to rapidly cover the entire area of the Tom and Jason claims as well as the southern portion of the adjacent MAC claims with the objectives of helping to map critical subsurface geology and identify drill targets for new discoveries and extensions of known mineralization. The geophysics work employed a state-of-the-art helicopter-borne Versatile Time-Domain Electromagnetic (VTEM) system and a high sensitivity magnetometer. Parallel lines were flown at 100 m spacing on a north-northeast bearing for a total of about 1,000 line kilometres. Preliminary results have been received and were analyzed and interpreted to define areas for exploration and potential new discoveries in 2018.

9.2 Airborne LiDAR Topographic Mapping

A program of airborne LiDAR (Light Detection and Ranging) surveying was carried out and the Jason portion of the property completed before it was suspended late in the season due to poor weather. The LiDAR work over the Tom and other areas will be completed in 2018. The purpose of the LiDAR survey is to produce a very accurate topographic map of the property for engineering and mapping work as well as aid in the mapping of geological features. High definition aerial photography was also carried out during the survey which will aid in geological and engineering work.

9.3 Field Work

Exploration field work carried out in 2017 included surface geological mapping and geochemical sampling in the search for new discoveries in the Tom and Jason deposit areas. The mapping has resulted in a better understanding of the geology and setting of the mineralization. Geochemical results were interpreted with follow up plans for 2018. A more extensive 2018 program of mapping and geochemistry has now begun.

10 Drilling

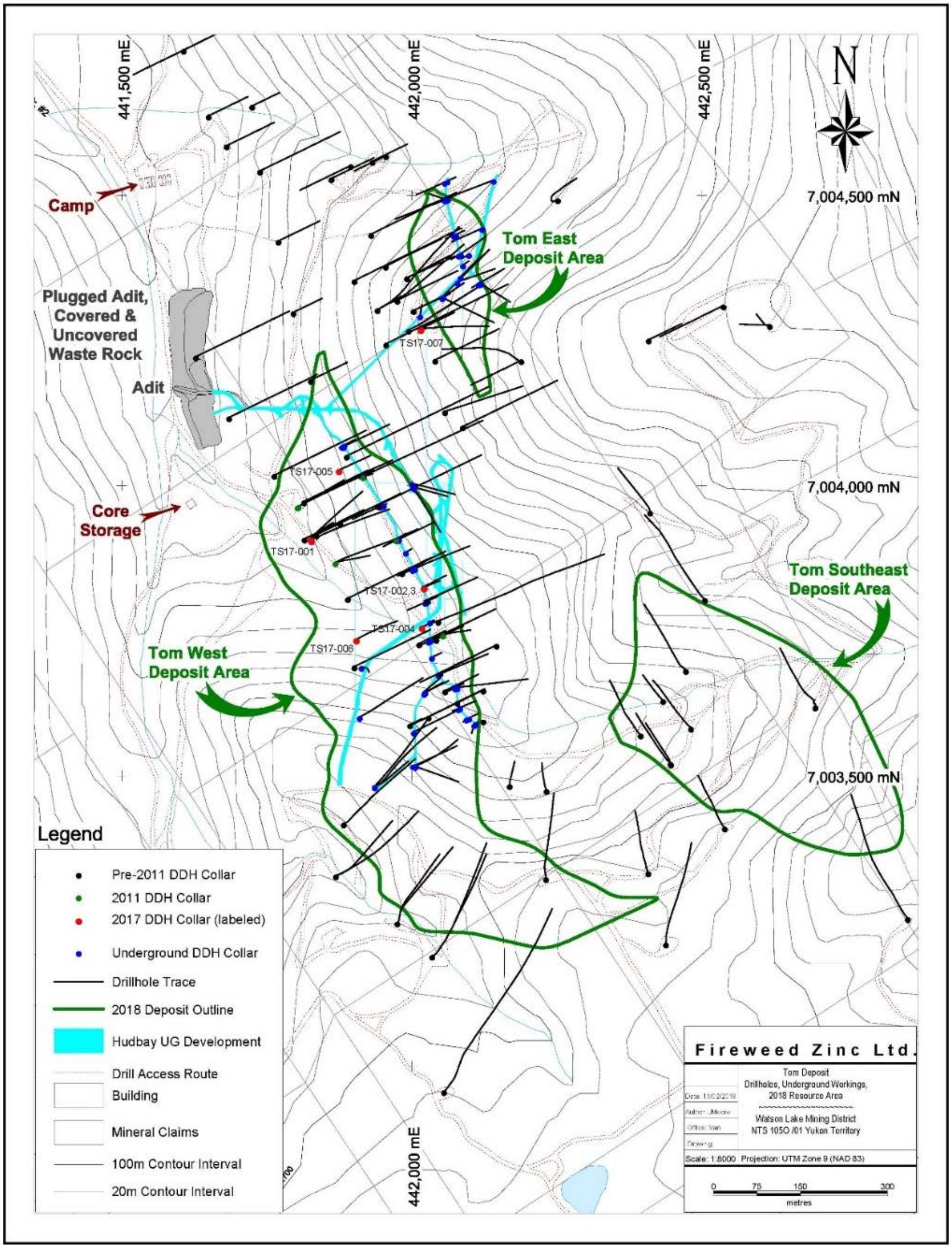
Fireweed carried out drilling on both the Tom and Jason areas in 2017 (Figure 10-1 and Figure 10-2). The following description is based on Fireweed's news release dated 27 December 2017.

The objectives of the 2017 drill program were to:

- Complete sufficient new drilling and resampling of old drill core to verify historic drill results for use in a NI 43-101-compliant mineral resource report;
- Step-out drill holes on the known zones of mineralization to expand on historic drill results; and
- Collect fresh rock core samples from the new drilling for metallurgical test work.

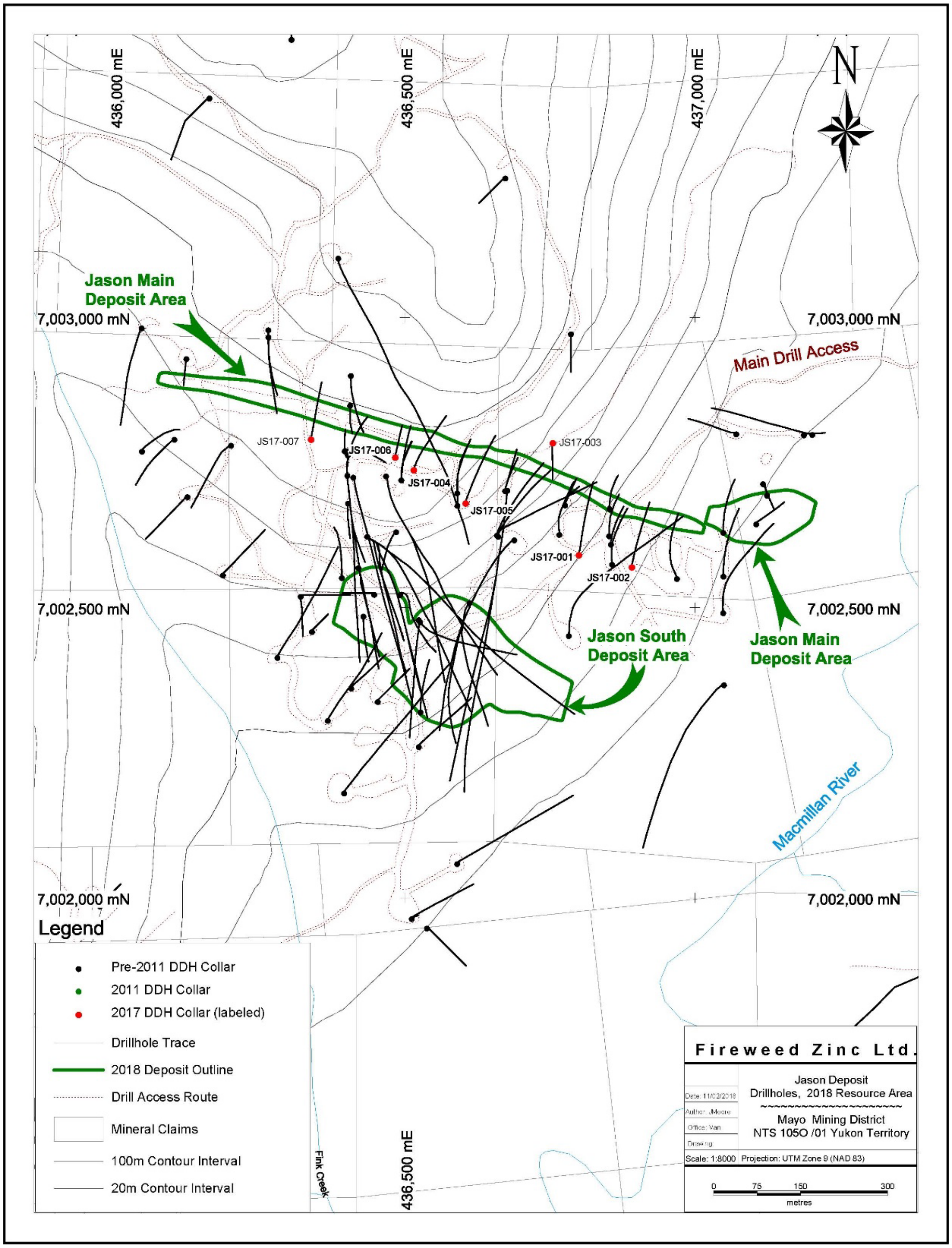
All these objectives were met and locally exceeded with drill results that were, in places, higher grade and/or wider than historic drill data indicated. Significant drill results from the 2017 program are summarized in Table 10-1.

Figure 10-1: Drillhole Collars and Traces from the Tom Deposit



Source: CSA Global (2018)

Figure 10-2: Drillhole Collars and Traces from the Jason Deposit



Source: CSA Global (2018)

Table 10-1: Significant Drilling Intercepts from the 2017 Drill Program

Hole No.	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Zn (%)	Pb (%)	Ag (g/t)
Tom West Zone Drill Results							
TS17-01	98.25	157.00	58.75	50.9	5.05	1.22	0.6
TS17-02	17.25	40.00	22.75	21.6	11.26	7.88	136.7
Including:	30.00	40.00	10.00	9.5	15.88	12.04	290.4
Including:	35.40	40.00	4.60	4.4	21.57	19.24	491.8
TS17-03	16.70	42.47	25.77	24.4	10.20	6.30	87.7
Including:	34.00	42.47	8.47	8.0	14.66	9.82	234.1
Including:	38.65	42.47	3.82	3.6	19.20	13.95	379.8
TS17-04	40.50	69.65	29.15	21.5	6.53	2.93	18.2
Including:	55.40	66.50	11.10	8.2	7.23	4.65	38.8
TS17-05	57.55	94.20	36.65	27.7	6.35	3.15	34.2
Including:	79.90	93.80	13.90	10.5	7.55	5.99	87.0
Including:	83.50	86.00	2.50	1.9	14.99	2.36	54.4
Including:	90.00	93.80	3.80	2.9	10.33	7.15	166.7
TS17-06	196.85	239.00	42.15	28.5	5.27	0.70	0.4
Including:	198.60	206.00	7.40	5.0	8.45	0.41	1.5
Tom East Zone Drill Results							
TS17-07	61.00	154.20	93.20	38.0	8.73	7.62	129.7
Including:	88.55	150.45	61.90	25.2	10.62	10.32	178.0
Including:	121.00	150.45	29.45	12.0	11.76	11.80	228.5
Including:	121.60	124.65	3.05	1.2	15.55	23.41	389.4
Including:	132.00	142.18	10.18	4.2	17.66	12.95	277.0
Including:	138.38	142.18	3.80	1.5	23.84	17.70	392.2
Jason Main Zone Drill Results							
JS17-01	172.30	183.26	10.96	7.0	12.16	3.13	1.6
JS17-02	155.18	172.76	17.58	10.5	7.82	1.39	1.3
Including:	165.00	172.76	7.76	4.6	11.19	1.94	1.2
JS17-03	Drillhole abandoned before reaching main zone due to drilling and survey problems						
JS17-04	154.19	179.00	24.81	11.2	9.07	1.60	0.7
Including:	170.70	179.00	8.30	3.7	14.03	1.29	1.1
JS17-05	177.98	206.72	28.74	15.7	10.22	1.95	0.5
Including:	184.60	193.22	8.62	4.7	15.02	3.05	0.3
Including:	187.16	191.17	4.01	2.2	19.53	3.97	0.6
Including:	203.50	206.00	2.50	1.4	18.75	1.12	1.8
JS17-06	57.50	83.83	26.33	13.1	13.24	3.38	1.4
Including:	57.50	61.30	3.80	1.9	12.93	4.29	3.0

Hole No.	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Zn (%)	Pb (%)	Ag (g/t)
Including:	64.70	68.40	3.70	1.8	25.06	5.00	3.4
Including:	77.20	83.83	6.63	3.3	20.66	3.95	0.8
JS17-07	61.00	85.05	24.05	16.9	5.25	1.24	2.0
Including:	79.95	85.05	5.10	3.6	8.91	1.58	0.4

Notes to Table 10-1:

- Maps and sections of the Tom and Jason drillholes are provided elsewhere in this report and are available on Fireweed's website at www.FireweedZinc.com .
- True width estimates are based on Fireweed's understanding of the orientation of the mineralized bodies in the area of the drill intersections at the time of the news release (27 December 2017).
- Details on the drilling procedures, sampling and assay methods are in Section 11.

Source: CSA Global (2018)

11 Sample Preparation, Analyses and Security

Fireweed conducted exploration activities on the property in 2017. All other data are historical in nature. Sampling, analyses and quality control are discussed for three distinct phases of drilling on the Property:

1. Historical drilling prior to 2011 for which the records are incomplete;
2. Drilling carried out by Hudbay in 2011; and
3. Drilling carried out by Fireweed in 2017.

11.1 Pre-2011 Sample Preparation, Analyses and Security

Due to its historic nature, CSA Global has been unable to confirm the sampling protocols, core-handling procedures, or site security utilized on diamond drill programs prior to 2011. As previously described in Section 6, some of the archived pre-2011 core is stored in a metal-clad building at the site or cross-stacked in the surrounding area. This building is not presently locked but is nailed shut when no one is on site and could be made secure easily. There is no evidence that any vandalism has taken place. Core from 70 Tom holes is stored in Whitehorse, in a secure government warehouse (Rennie, 2007) at the H.S. Bostock (Yukon government) Core Library. The Jason core was moved from a shed on the east bank of the South Macmillan River by Hudbay to the Tom core storage site between 2011 and 2015. The Jason core has been cross-piled and covered with breathable canvas covers (Figure 5-5). The core, for the most part, is secure although it is in a remote site that is accessible by road and so is vulnerable in some degree to tampering. Core from 20 Jason holes is also stored at the H.S. Bostock Core Library.

Pre-2011 core samples were collected using a diamond saw or a blade splitter. Core samples from both Tom and Jason were sent to a number of labs including Bondar Clegg and Company Ltd, Chemex Labs Ltd and Hudson Bay Mining and Smelting Co. Limited (Rennie, 2007). CSA Global notes that the analytical work carried out on samples from the Tom deposit at the Hudson Bay Mining and Smelting lab was not independent as it was performed by employees of the same company which was undertaking the exploration drilling at the time.

Assay certificates for some historical analyses are available for the Tom deposit from the 1980s. Only random spot checks of digital copies of historical assay certificates have been undertaken. Original Ag values were either reported as oz/ton or gram/tonne and conversions to ppm appear to have been done correctly. Some assays are recorded only on drill core strip logs, in which case low values were often recorded as “tr”, for trace amounts. These values have been recorded as “0” values in the historical database, and an effort has been made to replace these false “0” values with a nominal detection limit amount so that it is clear that a sample was assayed and that no significant metal value was returned.

No quality assurance and quality control (QAQC) data are available for the pre-2011 historic analyses beyond check assays aside from a number of samples that appear to have been analyzed at a different laboratory. For this reason, a re-assay program of historical drill core was undertaken in 2017 to verify historical data. The results of this program are discussed in Section 12.

Despite the incomplete documentation for historical assays, it is CSA Global’s opinion that the historic sample preparation and analyses would have been carried out using industry standard procedures for that time by reputable laboratories. There is no reason to suspect that analytical results contained in the Tom and Jason historic drill database are not representative of in situ mineralization and CSA Global considers the data adequate for the purposes of this report.

11.2 2011 Hudbay Drill Core Sample Preparation and Security

Sample preparation, analyses and security methods and protocols for the 2011 drilling program carried out by Revelation on behalf of Hudbay (Wells, 2012) are described in this section.

Drill core was halved for sampling using a diamond saw installed at the new Tom camp. Quarter core was sampled for assay where the half core was required for metallurgical testing. Samples for analysis were collected into polypropylene bags. Security of samples prior to dispatch to the analytical laboratory was maintained by limiting access of unauthorized persons to the site. Samples were stored in a secure storage area at the base camp on the Property. Detailed records of sample numbers and sample descriptions provided integrity to the sampling process. Labelled samples bags were packed in polypropylene rice bags and sealed for shipping. Samples remained under the supervision of Revelation personnel while onsite at the Project and during delivery to the ACMELabs (ACME) preparation facility in Whitehorse, Yukon. ACME completed sample preparation at their Whitehorse facility, and employed bar coding and scanning technologies that provided complete chain of custody records for every sample. Master pulps were then shipped by ACME to their Vancouver laboratory for analysis.

The ACME Whitehorse preparation facility is certified to standards within ISO 9001:2008. The Vancouver analytical facility was certified to standards within ISO 9001:2008 and, at the time of the 2011 program, was in the process of accreditation to ISO/IEC 17025:2005 from the Standards Council of Canada (SCC). ISO/IEC 17025:2005 accreditation conforming to requirements of CAN-P-1579 and CAN-P-4E was received in October 2011 for methods including the determination of Ag, Cu, Pb and Zn by multi-acid digestion with an atomic absorption spectrometry (AAS) finish. ACME sample preparation procedures and analytical methods are routine and follow industry best practices and procedures. CSA Global notes however that ACME's ISO/IEC 17025:2005 accredited analytical methods do not include those utilized for the analysis of the 2011 drill core samples.

ACME and its employees were independent from CSA Global, Fireweed, Hudbay and its consultant Revelation. Hudbay and Revelation personnel, consultants and contractors were not involved in the 2011 sample preparation and analysis.

11.3 2011 Drill Core Sample Analytical Method

Drill core samples from the Tom Zn-Pb -Ba-Ag deposit were analyzed by ACME following crushing and pulverization of the samples to >85% less than 75 microns. The pulps were analyzed for a suite of 24 elements using inductively-couple plasma optical emission spectroscopy (ICP-OES), including base metals, following a hot modified aqua regia digestion consisting of a 1:1:1 ratio of $HCl:HNO_3:H_2O$ (ACME group 7AR). Samples with greater than 4% Pb or 20% Zn were re-digested using a dilution to obtain data within range for the ICP-OES. Two samples with greater than 300 ppm Ag were also re-analyzed by fire assay. Barium was determined by fused disc X-ray fluorescence (XRF) (ACME group 8X – Ba). Gold was determined by aqua regia digestion of a 15 g charge (ACME group 3A01) as a preliminary check of Au levels, there being few previous analyses. It was not intended to provide rigorous Au assay data.

11.4 2011 Drill Core Sample QAQC

11.4.1 Overview

Several in-house certified reference materials (CRMs) manufactured from Flin Flon, Manitoba area base metal material and supplied by Hudbay were included with the core sample submissions. These were A5 (seven samples), B5 (seven samples), E5 (seven samples) and the base metal blank F6 (42 samples). Because these samples are not matrix-matched to the sediment-hosted base metal mineralization at Tom, two additional Pb-Zn-Ag CRMs manufactured from base metal material from the Mount Isa district in Australia were purchased from Ore Research & Exploration and included in the sample submission – Oreas 133a (six samples) and 134a (nine samples). In addition, data for two ACME internal CRMs, Oreas 131b (27 analyses) and Geostats GBM997-6 (19 analyses) were also assessed. Oreas 131b is a low-grade Pb-Zn-Ag CRM made from the same material as Oreas 133a and 134a, and GBM997-6 is a high-grade Pb-Zn CRM.

11.4.2 Analysis of 2011 QAQC Data

A summary of CRM performance is provided in Table 11-1. Samples with a bias and no failures lie mainly within two standard deviations of the calculated mean for the CRMs (i.e. the expected value). A failure is taken to be any analysis that lies more than three standard deviations away from the expected value, or two consecutive analyses with the same bias (i.e. positive or negative) more than two standard deviations from the expected value.

Table 11-1: Summary of CRM Performance for 2011 Assays

CRM	No.	Pb	Zn	Cu	Ba	Ag
HBMS A5	7	NA	Positive bias	Negative bias	NA	NA
HBMS B5	7	NA	Positive bias	Negative bias	NA	NA
HBMS E5	7	Acceptable	Positive bias	Excellent	NA	Positive bias
HBMS F6	42	No failures	1 failure	No failures	n/a	No failures
Oreas 133a	6	Negative bias	Acceptable	Negative bias	2 failures	6 failures; positive bias
Oreas 134a	9	1 failure	3 failures	2 failures; positive bias	3 failures	1 failure; positive bias
Oreas 131b	27	6 failures; negative bias	Acceptable but with drift	Not assessed	Not assessed	9 failures; positive bias
GBM997-6	19	1 failure; negative bias	Negative bias	NA	NA	NA

NA = Not applicable
Source: CSA Global (2018)

The Hudbay CRM F6 is not an ideal blank material because the material is already pulverized and thus does not pass through the crushing and pulverizing stream at the laboratory. Therefore, the blank tests only for laboratory contamination during digestion and analysis. Aside from a single instance of probable Zn cross-contamination, the results are acceptable when the data are filtered to remove all data within an order of magnitude of the lower limit of detection.

Laboratory precision has been assessed through an assessment of pulp duplicate analyses provided by ACME. This estimate of laboratory precision does not include any variance introduced during the sample preparation stages and assesses only the combined effects of subsampling the final pulp, sample digestion and instrumental uncertainties. The analysis used the square root of the average relative variances for individual duplicate pairs (relative standard deviation = RSD; RMS method of Stanley and Lawie, 2007). The data were filtered to remove any values within an order of magnitude of the lower limit of detection, as these data are inherently imprecise. The results of this analysis for the main commodity elements are summarized in Table 11-2. There were insufficient Ag data for pulp duplicates greater than an order of magnitude above the detection limit to allow an assessment of laboratory precision for Ag. The results for Pb, Zn and Ba are all less than 5% and considered to be best practice for base metals assays (Abzalov, 2008). In general, the relative standard deviation for pulp duplicate pairs decreases with increasing grade.

Pulp splits from 38 samples processed by ACME were obtained and submitted to ALS Minerals of North Vancouver with Oreas 133a and 134a for check assays. The ALS North Vancouver analytical facility is individually certified to standards within ISO 9001:2008 and has received accreditation to ISO/IEC 17025:2005 from the SCC for methods including: fire assay Au by AAS; fire assay Au and Ag by gravimetric finish; aqua regia Ag, Cu, Pb, Zn and Mo by AA; and aqua regia multi-element analysis by ICP-OES and ICP-MS. ALS sample preparation procedures and analytical methods are routine and follow industry best practices and procedures.

ALS and its employees were independent from CSA Global, Fireweed, Hudbay and its consultant, Revelation. Hudbay and Revelation personnel, consultants and contractors were not involved in the 2011 sample preparation and analysis.

The analytical methods used by ALS were similar to those used by ACME Labs: Pb, Zn, Ag, S and Fe were analyzed by ICP-OES following an aqua regia digestion (ALS method ME-OG46); Ba was analyzed by fused disc XRF (ALS method Ba-XRF15c); Au was analyzed by 30g fire assay to check the validity of the aqua regia Au data from ACME (ALS method Au-ICP21). The data for the two CRMs submitted with the check assays are acceptable. While Au values by fire assay are systematically higher than those obtained by aqua regia, the values are all typically only an order of magnitude above background levels and are not considered to be economically significant.

Aside from Ba, the other main commodity elements show a negative bias in the check assay results compared to the original assays (Table 11-2), indicating that the original ACME data are slightly higher, on average, relative to the check assays from ALS Minerals. In the case of Zn, this bias occurs at all grades and is consistent with the positive bias shown by some of the CRMs submitted to ACME (Table 11-2). By contrast, the negative bias is strongest at lower grades in the case of Pb and may even give way to a positive bias at higher grades, consistent with the bias observed from the CRMs (Table 11-2). The negative bias in the Ag check assays is also supported by a positive bias in the ACME Ag data for the CRMs (Table 11-2). These biases appear to account for most of the variation between the two datasets.

Table 11-2: Summary of Laboratory Precision and Bias from Check Assays for 2011

Element	Pb	Zn	Ba	Ag
Precision (average % RSD)	3.9	4.9	2.4	n/a
Bias (average % relative difference)	-10	-5	1	-6

Source: CSA Global (2018)

11.5 2017 EEC Drill Core Sample Preparation and Security

The 2017 drill program at the Project was managed by Equity Exploration Consultants Ltd (EEC), an independent mineral exploration consultancy, contracted by Fireweed. The drill core was received in a purpose-built trailer and re-aligned in the core trays, prior to collection of structural information, photographing and determination of rock quality designation (RQD). Half samples of oriented HQ3 (spilt tube) core (61 mm diameter) were cut with a diamond saw on site at the Tom camp. The other half of the core was returned to the core box and is stored on site for future reference. Highly weathered, soft intervals of core from the Jason deposit were sampled with a putty knife. Some of this 2017 core was later quarter cut to provide samples for metallurgical testing (see Section 13).

Entire core intervals sampled for assay were also measured for dry bulk density using the water immersion method before the samples were shipped from camp. Samples were enclosed in individual plastic bags and then placed into rice bags for shipment from site. The rice bags were sealed with security tags. Drill core samples were either flown directly to Whitehorse from Tom camp by charter aircraft using Tintina Air or transported via road by Tu-lidlini Petroleum truck to Ross River where they were stored in a secure compound. Samples were then transported by truck from either Ross River or the Tintina Air hangar by Small's Expediting Services Ltd directly to the Bureau Veritas (formerly ACME; "BV") sample preparation facility in Whitehorse. All rice bags were received intact by BV.

11.6 2017 Drill Core Sample Analytical Method

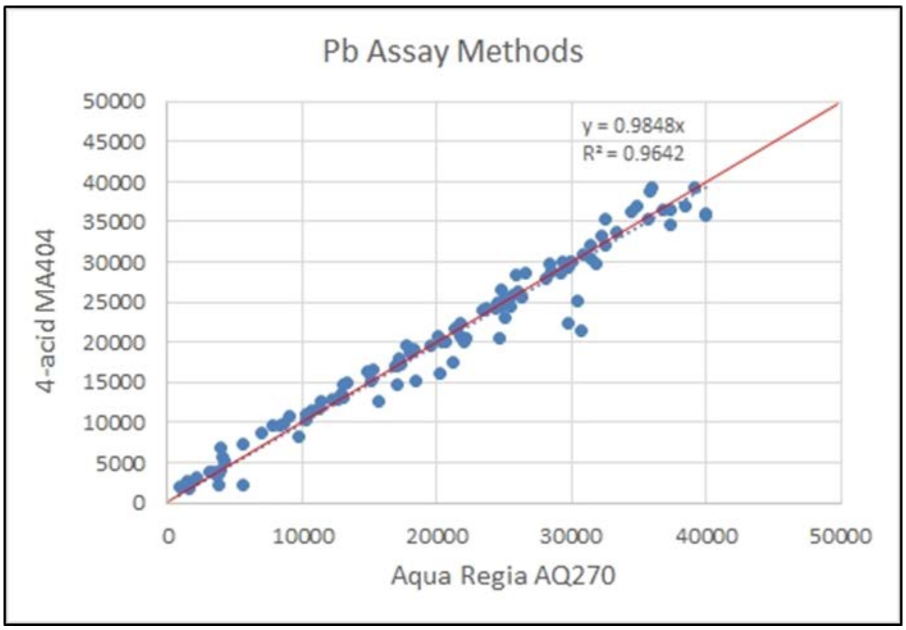
Sample preparation and analytical methods were selected to conform as closely as possible to those used previously. The details of historical analyses are not known, but samples from the 2011 drill program were analyzed at ACME using a hot modified aqua regia digestion. ACME was purchased by Bureau Veritas in 2012, and the previous preparation and method codes were renamed. The equivalent method codes at BV to those used during 2011 are as follows: preparation – PRP70-500 and hot modified aqua regia digestion – AQ370. The latter method was used for the re-sampling and assay of historical drill core. Due to lower limits of detection for some potentially deleterious elements and additional elements included in the package, the decision was made in late July to analyze all new drill core samples from 2017 using AQ270, noting that AQ270 and AQ370 use the same modified hot aqua regia digestion. The over-range method code for base metals by the aqua regia digestions with an atomic absorption finish was MA404 for samples exceeding the upper limit of detection of 4% Pb and 20% Zn (reduced for this program to 8% Zn to trigger over-range analyses by MA404), followed by a classical titration method for any further over range Pb assays exceeding 20% (BV code GC817) or Zn assays over 30% (BV code GC816). The Vancouver BV laboratory is accredited by ISO/IEC 17025:2005 for AQ370 and MA404 methods.

Barium by fused disk XRF–XF700 was not available as a single element method at BV. Instead, a 250 g pulp split was sent from the BV preparation laboratory in Whitehorse (code SPTPL) to MS Analytical in Langley, British Columbia for Ba analysis by WRA-3Ba. This method involves a total fusion of the sample using a lithium metaborate fusion, digestion in acid and then analysis by ICP-ES. MS Analytical is not accredited for this method but was selected because the quoted upper limit using a similar method at BV is 5% Ba and much higher levels than this were anticipated.

The use of a four-acid digestion for over-range samples greater than 4% Pb and 8% Zn raised issues as to whether the results would be comparable to those obtained using a modified, hot aqua regia digestion. There are a sufficient number of samples that overlap in the range 8% to 20% Zn that were analyzed by both digestions and which allow a direct comparison (Figure 11-1). For the most part, there is good agreement between assays obtained using either acid digestion method. However, there are a few samples, generally containing very high Ba contents >20%, where the four-acid digestion under-reports Pb due to precipitation of Ba and Pb sulphates during digestion (J. Sader, personal communication, 2017). On average, the aqua regia Pb data are 6% higher than the four-acid Pb data. This discrepancy for those few samples was resolved through the use of a reverse aqua regia (3:1 HNO_3:HCL) digestion (J. Sader, personal communication, 2017), although this was not implemented on a routine basis for the 2017 assay results. Note also that the Oreas CRM used to monitor data accuracy do not contain significant Ba (i.e. <1,000 ppm) and are not likely to be affected by this phenomenon.

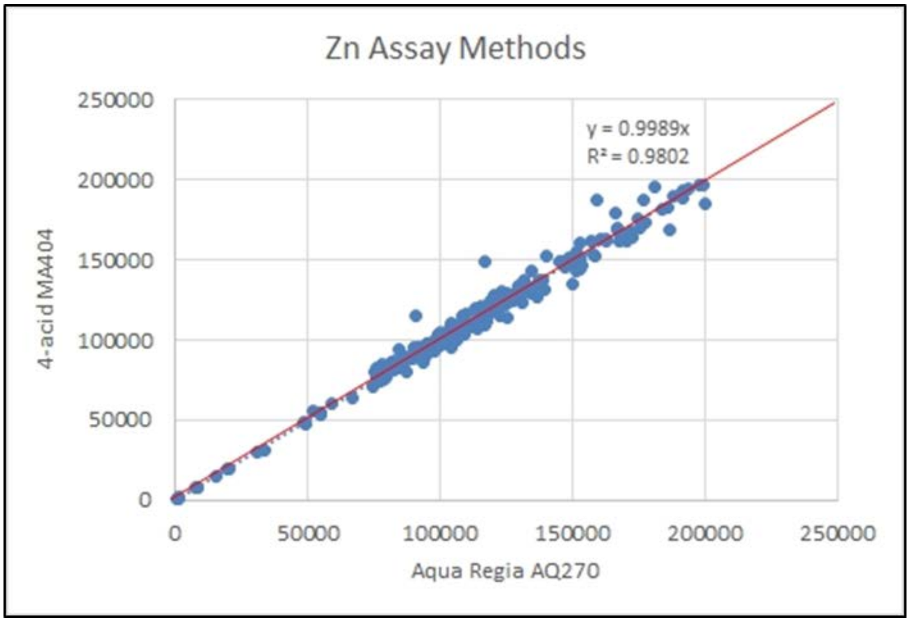
Recognition that some Zn assays might also be under-reported in the modified, hot aqua regia digestion resulted in a lowering of the trigger value for the use of the four-acid method to 8% from 20%. This resulted in a large dataset for which data were obtained by both acid digestions (Figure 5-1). As in the case of the Pb data, there is general agreement between Zn data obtained using both digestions, but the occasional sample for which the difference is significant. However, overall, there is no statistically significant difference between the two datasets and the use of a four-acid digestion for the over-range analyses for Zn introduces no bias to the data overall.

Figure 11-1: Comparison of Modified Hot Aqua Regia and Four-acid Digestion for Pb Assays



Source: CSA Global (2018)

Figure 11-2: Comparison of Modified Dilute Aqua Regia and Four-acid Digestion for Zn



Source: CSA Global (2018)

11.7 2017 Drill Core Sample QAQC

11.7.1 Overview

A comprehensive QAQC program accompanied the 2017 drilling program on the Property. It involved the use of CRM, quarter-core field duplicates, assessment of coarse crush and pulp duplicate data, and the measurement of bulk density field standards and duplicates. Minimal check assays were conducted on samples from the 2017 drill program. The results of this program are summarized in the following section.

11.8 Analysis of 2017 QAQC Data

The CRM used for the 2017 drill program are shown in Table 11-3 along with estimated mean biases. While mean biases are useful for summarizing overall laboratory performance, they can disguise significant individual CRM failures. The performance of individual CRM has been assessed using Z-score plots (Figure 11-3, Figure 11-4 and Figure 11-5), and these charts give an indication of systematic biases that may exist in the data. CRM Z-scores are calculated as follows:

CRM Z-score = (Observed CRM value – Certified CRM value)/Certified standard deviation

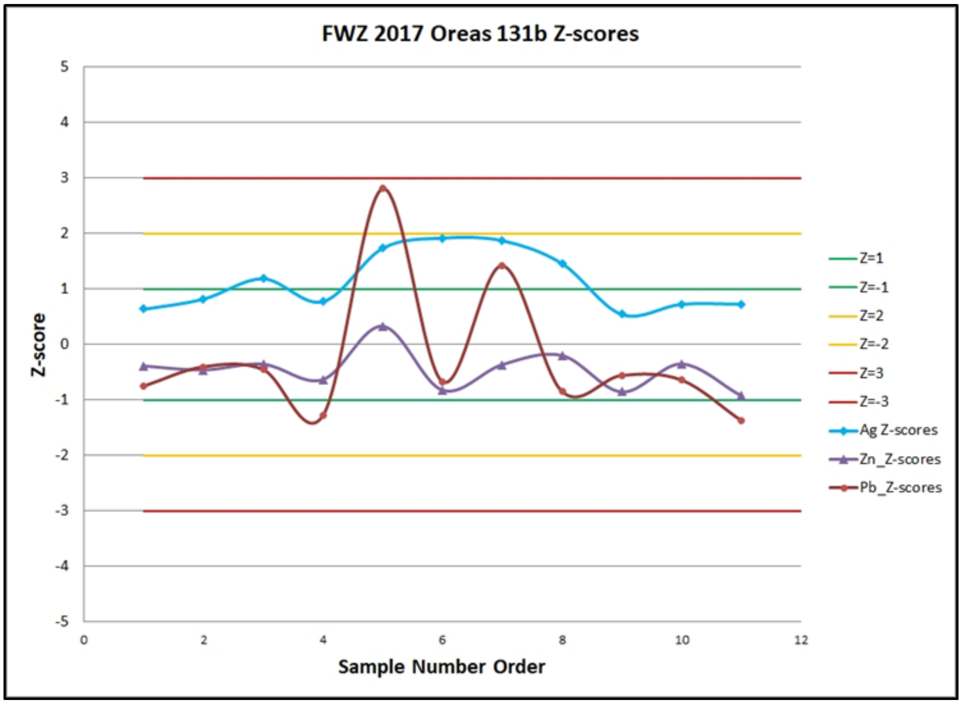
A Z-score >3 or <-3 would constitute a failure at 3 standard deviations, and two out of three consecutive CRM with a Z-score >2 or <-2 of the same polarity would also constitute a quality control failure. No failures of CRM inserted by Fireweed occurred during 2017, although there is a significant positive bias to the Ag data, similar to that observed in 2011 at ACME. Mean biases observed for Pb and Zn are generally within an acceptable range of +/-2%.

Table 11-3: Summary of CRM Values and Mean Biases for 2017

CRM	No.	Certified Pb (%)	Mean Pb Bias (%)	Certified Zn (%)	Mean Zn Bias (%)	Certified Ag (ppm)	Mean Ag Bias (%)
Oreas 131b	11	1.86	-1.2	3.03	-2.4	32.1	+7.7
Oreas 132a	11	3.6	-0.8	4.86	-0.7	55.6	+2.4
Oreas 133a	11	4.9*	-0.2	10.97*	-0.8	96.9	+3

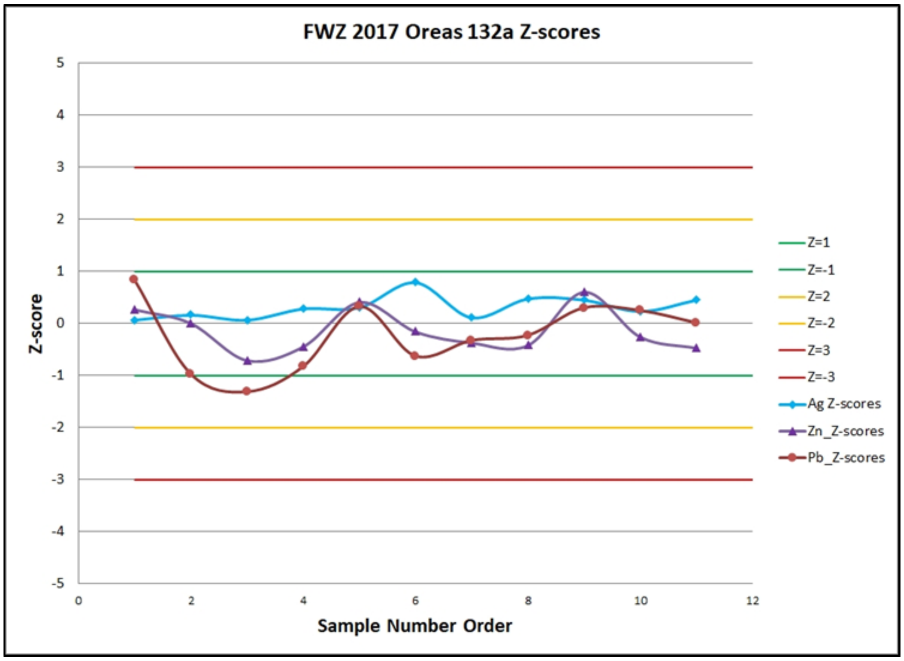
*Certified values for a four-acid digestion. All others are for an aqua regia digestion.
Source: CSA Global (2018)

Figure 11-3: Z-score Chart for Pb, Zn and Ag for CRM Oreas 131b from the 2017 Drilling Program



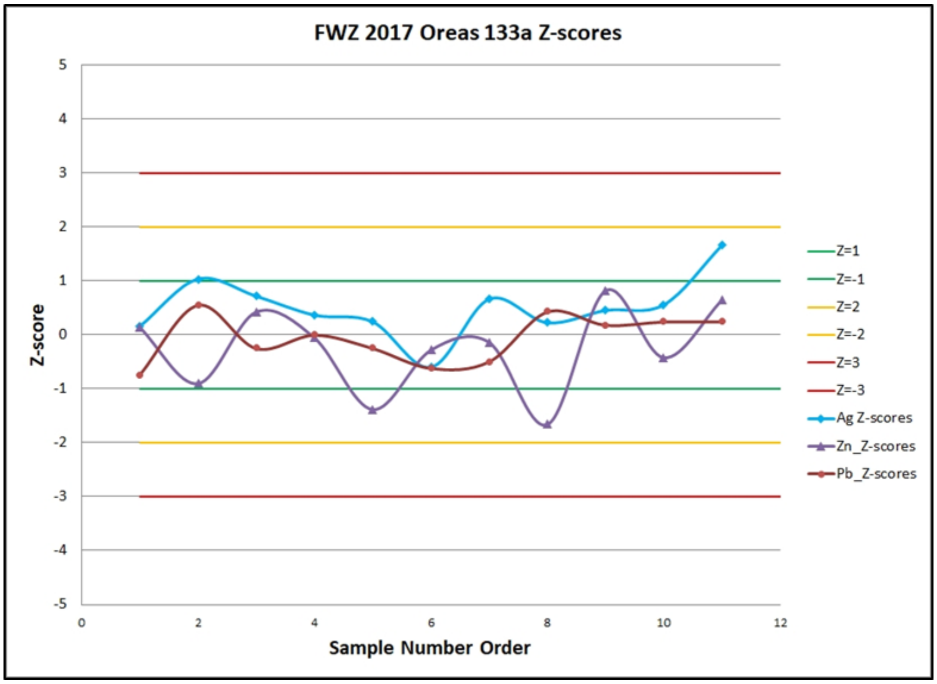
Source: CSA Global (2018)

Figure 11-4: Z-score Chart for Pb, Zn and Ag for CRM Oreas 132a from the 2017 Drilling Program



Source: CSA Global (2018)

Figure 11-5: Z-score Chart for Pb, Zn and Ag for CRM Oreas 133a from the 2017 Drilling Program



Source: CSA Global (2018)

A total of 61 coarse blank samples were submitted with the core samples in 2017. This material consisted of approximately 0.5 kg of 20 mm crushed granite from a quarry in the lower mainland area near Vancouver. The samples all returned Ag values less than five times the detection limit of 0.5 ppm Ag. However, the Pb and Zn values are typically more than 10 times the lower limit of detection (LLD) for the analytical method used. Possible cross contamination of samples is measured in terms of a blank Z-score calculated as:

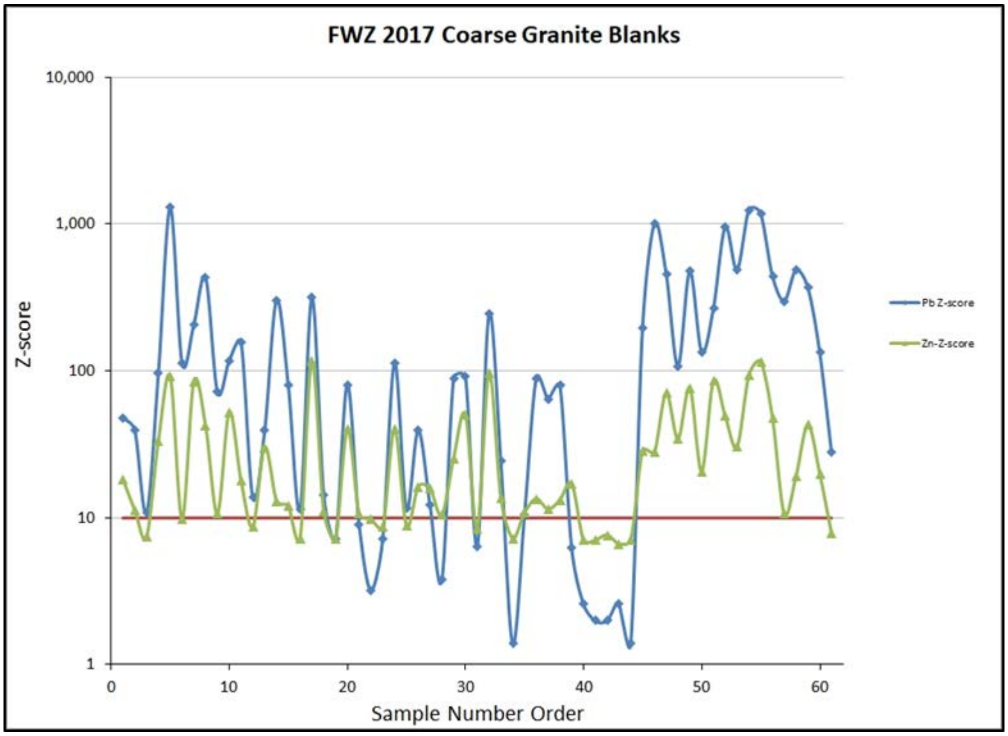
Blank Z-score = (Observed value – lower limit of detection)/Lower limit of detection

Some coarse blanks contain up to 1,000 times the LLD for Pb, which is 0.5 ppm, and up to 100 times the LLD for Zn, which is 5 ppm. These values are well above what would be expected in a granite and indicate the potential level of cross-contamination, or carryover, between samples. This carryover is likely a small percentage of the base metals contained in the sample preceding the coarse blank and is not considered to be significant (i.e. carryover <1%).

Precision has been assessed in a similar fashion to 2011, through the use of quarter duplicate, coarse crush duplicate and pulp duplicate data assessed using average relative standard deviations calculated following the RMS method of Stanley and Lawie (2007). The data were filtered to remove any values within an order of magnitude of the lower limit of detection for the analytical method, as these data are inherently imprecise. A summary of data is provided in Table 11-4 for Zn, Pb, Ag and Ba.

In general, the samples show excellent repeatability. The coarse crush and pulp duplicates give average RSD values that are within best practice guidelines from Abzalov (2008; 2011) for magmatic base metal and skarn deposits, and for VMS Cu from Arne and Cobb (2017). The field duplicate RSD values, which have been mass corrected for the use of quarter samples to estimate the precision of routine half core samples, are also good, and lie within the range of acceptable practice for most coarse crush duplicates.

Figure 11-6: Coarse Field Blank Z-score Chart for Pb and Zn from the 2017 Drilling Program



Source: CSA Global (2018)

Table 11-4: Summary of Precision Estimates for Duplicates Analyzed in 2017

Duplicate Type	Number	Zn % RSD	Pb % RSD	Ag % RSD	Ba % RSD
Quarter-core field*	55	12.7	11.9	11.6	8.4
Coarse crush	37	2.6	2.8	3.3	n/a
Pulp	8-65	3.6	4.5	2.2	n/a

* Mass corrected using the a **priori** method of Stanley (2014); n/a = not available.
Source: CSA Global (2018)

Dry bulk densities were determined in the field using the water immersion (buoyancy) method. Four core samples were selected to represent a range of bulk densities for repeated measurement. The results of repeated measurements of these four core samples are presented in Table 11-5. Data from 45 core samples were analyzed in duplicate for dry bulk density are also summarized in Table 11-5. The repeat analyses of the bulk density measurements are all within +/-5% and so reflect good reproducibility. As the dry bulk densities of the core samples used as standard reference materials are not known independently from the field determinations, nothing definitive can be said about accuracy of the data, although the balance was checked before every use using a known 1 kg weight. The temperature of the weighing room within the core shed was kept relatively constant compared to outside temperatures, and the water was changed on a regular basis when it became dirty in order to maintain a constant density.

Table 11-5: Summary of Repeated Analysis of Dry Bulk Density Reference Materials

Reference	No. of Measurements	Rock Type	Mean Bulk Density (g/cm^3)	Standard Deviation (g/cm^3)	Relative Standard Deviation (%)
1	9	Wall rock	2.802	0.034	1.2
2	10	Mineralized	3.630	0.069	1.9
3	10	Mineralized	3.395	0.106	3.1
4	7	Barite-bearing	4.049	0.127	3.1
Duplicates	45	Various	NA	NA	2.6

NA = not applicable
Source: CSA Global (2018)

11.9 QAQC Analysis Summary

The quality control data for the 2017 drilling program are generally excellent, with no failures of CRM inserted by Fireweed and demonstrated good repeatability of field, coarse crush and pulp duplicates. Duplicate data lie within best practice guidelines when compared to published studies from similar deposit types. Coarse blank material inserted into the sample stream show evidence of some cross-contamination of base metals, particularly when inserted with high-grade samples, but the carryover from mineralized samples is estimated to be <1% and so within acceptable limits.

In general, the 2011 data for the in-house Hudbay CRM were acceptable for Zn and Cu. The Pb and the Ag levels are too close to the lower limit of detection available for the Tom deposit assays in many of the Hudbay CRM for a precise assessment of accuracy. There is a clear bias toward lower Au values from the aqua regia digestion of the Hudbay CRM that probably reflects incomplete digestion compared to the certified fire assay results. The poor performance of the Au assays at Tom is not considered to be relevant.

Of concern are the strong positive biases displayed by the aqua regia Ag data in the Oreas CRM, including the ACME internal CRM Oreas 131b, and the Hudbay CRM E5. Clearly, the Ag data for the samples are over-estimated by these assays, probably on the order of 5% to 10%. This positive bias is confirmed by the results of analyses of Oreas CRM during the 2017 drilling program, with mean positive biases ranging from 2.4% to 7.7%.

In general, the Zn assays for the Oreas CRMs in 2011 are acceptable, but there appears to have been a problem with the initial dilutions for the over-range samples. The original Zn analyses for Oreas 134a were generally acceptable, except for one failure outside of three standard deviations below the expected value and a clear negative bias. However, the over-range re-assays show erratic data for several early analyses, before steadying at quite good results. The Zn data for Oreas 131b show a distinct drift through the sample sequence from a negative to positive bias. The Pb data for Oreas 131b in 2011 also show a negative bias, with numerous analyses greater than three standard deviations below the expected value. Although a slight negative bias for Pb is apparent in the 2017 analyses from BV for similar Oreas CRM, it does not appear to be as significant as that observed in 2011.

The 2011 Ba data for both Oreas 133a and 134a are erratic, with both positive and negative failures. Both CRMs have low Ba contents and the values are only an order of magnitude or so above the lower limit of detection. Imprecise data are expected at these levels. The accuracy of the Ba data generated in 2017 at MS Analytical was monitored by the laboratory using commercially available CRM.

Given the poor performance of the CRMs for the higher-grade material in 2011, re-assays of two batches were requested at ACME. Re-assays of over-range samples using method 7AR and a dilution method showed a slight positive bias for both Pb and Zn compared to the original analytical results, and this is reflected in Zn data from CRMs. Despite these slight biases, the re-assay data are generally within a 20% relative difference from the original data in the case of Pb, and within 10 % in the case of Zn. Given the absence of significant differences between the original and re-assay data, as well as evidence of positive bias in the ACME data relative to check assays performed at ALS Minerals, retention of the original data in the database was recommended.

Uncertainties associated with incomplete recoveries of Pb from acid digestions in the 2017 dataset have been assessed and, for the most part, data obtained using acid digestions are similar regardless of whether a modified hot aqua regia or four-acid digestion was used. A re-assessment of assay methods in future drilling campaigns on the Property was carried out and for 2018 analyses by fused disk XRF for base metals will be used.

11.10 CSA Global’s Opinion on 2011 and 2017 Sample Preparation, Security and Analytical Procedures

It is CSA Global’s opinion that sample security, collection, preparation and analysis undertaken on the Macmillan Pass Project during 2011 by Hudbay and in 2017 by Fireweed were appropriate for the sample media and mineralization type and conform to industry standards. The Pb, Zn and Ag data from both ACME in 2011 and BV in 2017 show evidence of minor systematic biases, but these are generally <5%, and are acceptable for the estimation of a MRE. The precision of the 2011 and 2017 data is industry best practice.

CSA Global recommends that approximately 5% of samples from the 2017 drill program and future drilling programs be submitted for check assaying at an accredited laboratory using similar assay procedures.

12 Data Verification

12.1 Geology, Drilling and Assaying

12.1.1 Drill Collar Locations

Rennie (2007) recommended that the drill collars on the Jason property be re-surveyed. Wells (2012) noted that verification checks of a limited number of historical drillhole collars at the Jason deposit using a Trimble GeoExplorer 6000 GeoXH model DGPS receiver indicated that there was a locational error in the positions of these collars in the Hudbay database; database collars were located approximately 53 m northwest of their actual locations in NAD83 UTM-Z9. All Jason drill collars that could be identified in the field were re-surveyed using a Trimble R10 DGPS during the 2017 field season and this error rectified.

A similar exercise was conducted in 2017 on the historical Tom drill collar locations. These revised locations were cross-referenced with survey plans of the historical drillhole locations to correct all remaining drill collar locations that could not be measured directly in the field. While there is always potential to locate further historical drillhole collars, the collar locations are now considered to have been located as accurately as can be expected given the passage of time since they were drilled.

12.1.2 Database

The historical drilling information up to and including the 2011 data was reviewed by Arne (2017). Many of the issues raised in that review, including uncertainties in precise drill collar locations, have been rectified for the present report.

Digital assay certificates are not available for all historical drillholes, and so random spot checks of pdf copies of historical assay reports and core logs with assay data transcribed onto them have been compared to the digital database supplied to Fireweed by Hudbay. The 2011 data, which contained omissions in the data provided by Hudbay, have been updated from records retained from the 2011 drilling program by CSA Global. Random spot checks of digital assay certificates from the 2011 and 2017 drilling programs have been undertaken.

Verification of the complete database has been complicated by the fragmental nature of the data residing in a number of spreadsheets and Microsoft Access databases. A historical compilation provided by Fireweed and the 2017 digital assay certificates provided by the laboratory directly to CSA Global were loaded into Maxwell Geoservices DataShed™ SQL database to allow verification of the data and to correct various data entry errors that existed in the historical compilations (e.g. overlapping intervals, data entries extending beyond bottom of hole, inconsistent units). Issues encountered included conversion of trace amounts of metal from assays recorded on historical drill logs to “0” values in some versions of the historical data compilation. In some instances, “0” values have replaced with below detection limit values and the value “-1” used to designate that no sample was taken. These entries have been cross-referenced in the database used for this report using various historical files and “0” values replaced with values either at (trace) or below historical detection limits. Logged intervals not sampled have been converted to null in the database to avoid confusion. The identification of historical core intervals not sampled versus those sampled but relatively barren of metal is significant for modelling of the deposits.

CSA Global has taken what it considers to be reasonable steps to validate and correct the data compilation provided to it by Fireweed. CSA Global strongly recommends the adoption of an auditable SQL database for the storage of the existing data and for the addition of new data for future programs. Digital assay certificates from the laboratory and logging data from site should be loaded directly into the database and standard verification rules applied to the data on a routine basis. Data entry procedures should be modified to include the collection of quality control data for the measurement of dry bulk density data in the field.

12.1.3 Re-sampling of Historical Drill Core

Arne (2017) recommended a program of re-sampling and assaying of historical drill core given the lack of assay certificates for some drillholes and the absence of historical quality control data. Intervals for re-sampling were selected to provide a representative sampling from various historical drilling campaigns by different operators, and to obtain material from various zones within both the Tom and Jason deposits. An effort was made to sample identical sample intervals where these could be identified in the core trays. A total of 111 samples were collected from historical drill core from the Jason deposit and 108 samples were collected from historical drill core from the Tom deposit. A listing of intervals resampled is included in Table 12-1.

Table 12-1: Historical Core Intervals Re-sampled in 2017

Deposit	Zone	Hole No.	No. of Samples	From (m)	To (m)
Jason	South – middle	JS82-087	7	635.56	644.06
	South – upper	JS82-087	10	583.22	593.52
	South – middle	JS82-088	5	357.05	364.57
	South – lower	JS82-088	5	344.1	349.2
	South – middle	JS81-070	8	797.27	803.91
	South – middle	JS81-068D (W4)	9	679.96	687.54
	South – upper	JS81-070	15	740	754.78
	Main	JS77-025	16	218.39	232.56
	Hangingwall	JS77-026	13	247	260.6
	Main	JS81-075	10	485.84	499
	Hangingwall	JS81-071	7	81.12	99.28
	Hangingwall	JS81-081	6	92.52	106.38
Tom	East	TU001	13	21.03	37.16
		TU024	4	3.78	9.88
		TS091	14	66.10	87.50
	West	TS085	9	35.99	51.24
		TU053	15	3.05	21.24
		TU015	10	0.00	11.98
		TU017	10	1.71	15.85
		TS086	2	60.30	78.33
		TS087	5	33.70	44.20
		TS89-007W1	17	568.00	579.00
		TS88-004	9	546.50	555.50

Source: CSA Global (2018)

12.1.4 Analysis of Data from the 2017 Re-sampling Program

The re-sampled data are effectively half core where all remaining material was sampled (majority) or quarter core (minority) duplicate samples of historical drill core. As such they can be assessed in a similar fashion to the quarter core duplicates discussed in Section 11, although they have not been mass corrected for the use of some quarter core samples as the majority of samples were of half core. Therefore, the RSD estimates provided in Table 12-2 should be treated as maximum estimates where quarter core was sampled.

The estimates of RSD for the re-sampled assays for Zn is within acceptable levels, although higher than the variability obtained for field duplicates during the 2017 drilling program. More importantly, they show very little bias. Most of the variability in the data occurs at low to background values where the historical data will be imprecise (Figure 12-1, Figure 12-2 and Figure 12-3). The relative bias for the re-sampled assays is well within acceptable limits.

The Pb data show a positive bias in the re-sampled core, particularly at the higher grades (6.1%), and this may reflect historical difficulties experienced in assaying for Pb in samples with high Ba contents. Historical Pb is therefore likely under-reporting compared to the more recent assays from 2011 and 2017.

Reproducibility of the Ag data is excellent, with minimal relative bias (0.6%) and a RSD of 13.9%, which would fall within best practice for field duplicates for a precious metal.

12.1.5 Geology and Resource QP's Opinion on the Project Data

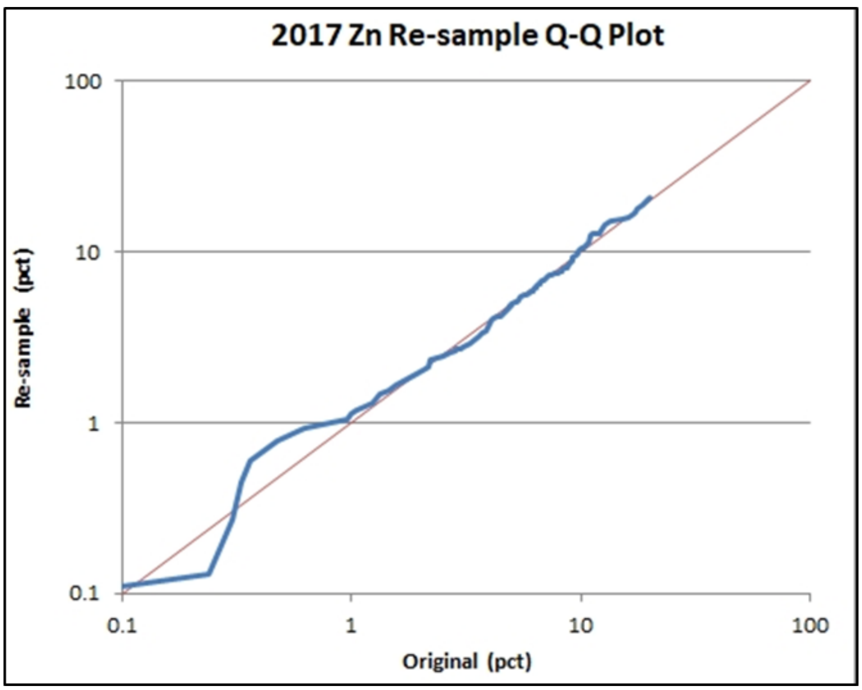
The re-sampling program of historical drill core completed in 2017 indicates that the historical Zn and Ag data show no appreciable bias compared to modern assays. However, historical Pb assays are likely under-reporting Pb by an average of 6%. The data compilation provided to CSA Global has been verified and is adequate to support a MRE.

Table 12-2: Summary Comparison of Historical Assays for Drill Core Re-sampled in 2017

Element	Relative Bias (%)	RSD (%)
Zn	0.5	22.0
Pb	6.1	25.7
Ag	0.6	13.9

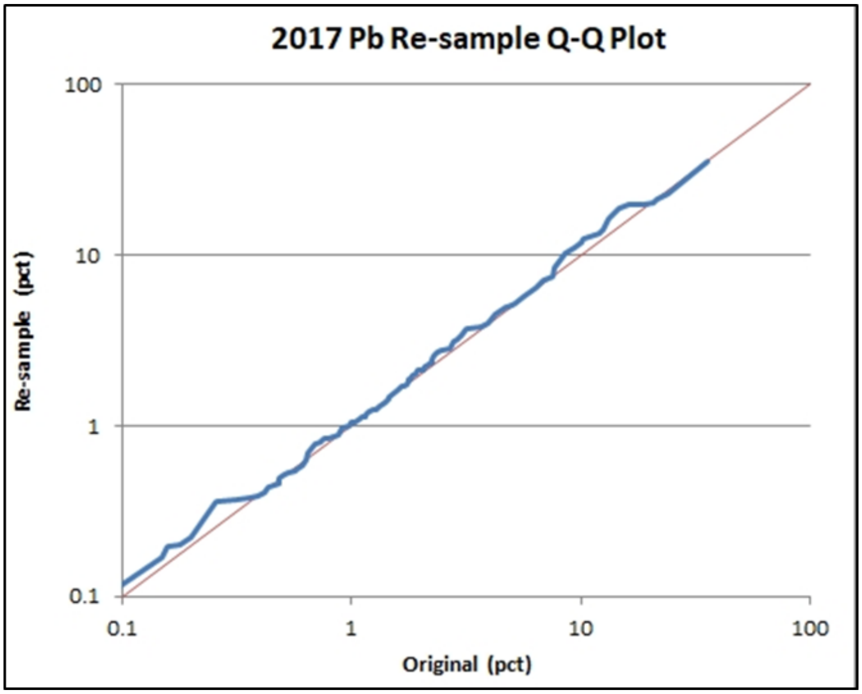
Source: CSA Global (2018)

Figure 12-1: Quantile-Quantile Plot of Historical Assays and Re-sampled 2017 Assays for Zn



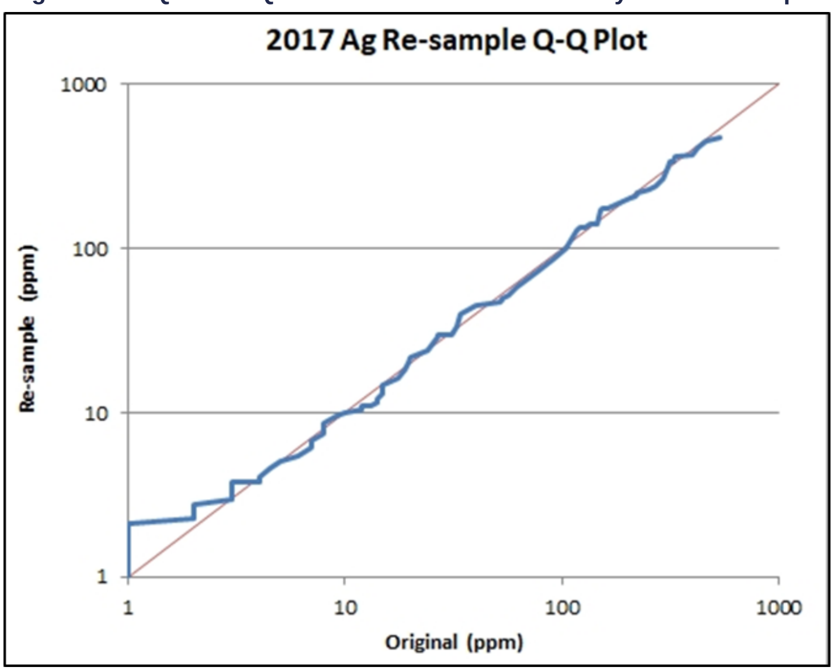
Source: CSA Global (2018)

Figure 12-2: Quantile-Quantile Plot of Historical Assays and Re-sampled 2017 Assays for Pb



Source: CSA Global (2018)

Figure 12-3: Quantile-Quantile Plot of Historical Assays and Re-sampled 2017 Assays for Ag



Source: CSA Global (2018)

12.2 Metallurgy

Metallurgical test data was verified through a review of previous studies and testwork reports and an analysis of the new results from the 2017 metallurgical testwork program. Any studies and reports referred to were thoroughly reviewed and align with the PEA metallurgical design and analysis in this report. All metallurgical data was verified and is adequate for this Preliminary Economic Assessment Technical Report as required by NI 43-101 guidelines.

12.3 Mining

Mining design data was verified through review of studies and reports. Any studies and reports referred to were thoroughly reviewed and summarized in this report and align with the PEA mine design and mine plan in this report. All mining data was verified and is adequate for this Preliminary Economic Assessment Technical Report as required by NI 43-101 guidelines.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

In 2012, a preliminary metallurgical test program was carried out on the Tom deposit by G&T Metallurgical Services of Kamloops, BC. The test program was developed based on a historical test program conducted by Michigan Tech in 1986. A blended composite comprised of 83% mineralized material and 17% waste was tested to evaluate mineralogy, grinding specific energy and sequential Pb, Zn flotation. The results were positive. The composite was found to be soft with respect to grinding specific energy, measuring a Bond ball mill work index of 11.5 kWh/t. After optimizing flotation conditions through rougher and cleaner flotation testing, a single locked cycle test was conducted. A primary grind size of 80% passing (P_{80}) 72 µm was chosen with Pb and Zn regrind sizes of 12 µm and 24 µm respectively. The Pb concentrate recovered 82% of the Pb at a grade of 70.9% Pb, while the Zn concentrate recovered 79.5% of the Zn at a grade of 58.8% Zn.

In December 2017, another metallurgical test program was commenced at Base Metallurgical Laboratories Ltd. ("Base Met") in Kamloops, BC to evaluate both the Tom and Jason deposits using quarter core samples from 2017 drilling. Testwork included mineralogy, comminution, dense media separation (DMS), settling, and rougher/cleaner Pb, Zn sequential flotation. Five composite samples, representing the Tom and Jason zones, were tested to develop a preliminary recovery flowsheet for producing saleable Pb and Zn concentrates. Once the flowsheet was developed, global composites were created and locked cycle testing was carried out to project recoveries for economic analysis. The results from this test program were used for the process design discussed in Section 17.

A full breakdown of the results for both test programs can be found in G&T (2012) and Base Met (2018).

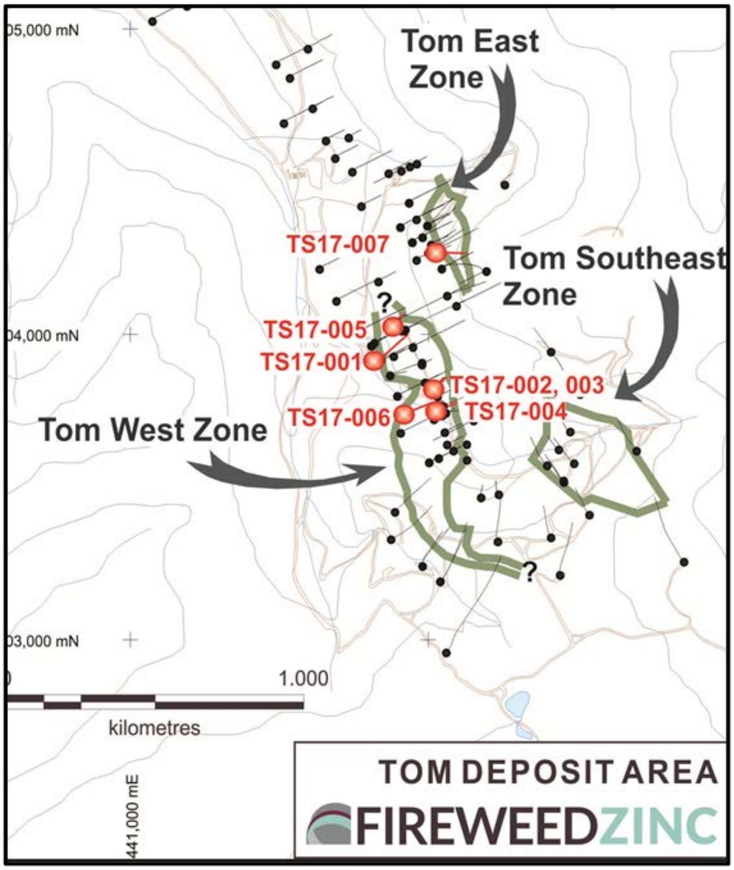
Based on the results from Base Met (2018), saleable Pb and Zn concentrates can be produced using Pb and Zn sequential flotation with a primary P_{80} grind size of 50 µm. For the Tom Composite, locked cycle test results achieved recoveries of 74.4 % Pb and 85.5% Zn at concentrate grades of 69.1% Pb and 60.1% Zn. For the Jason Composite, locked cycle testing achieved recoveries of 55.7% Pb and 88.4% Zn at concentrate grades of 69.9% Pb and 63.2% Zn.

A blended global composite was also generated at a ratio of 65% Tom Composite and 35% Jason Composite to reflect the anticipated mine plan. Preliminary testwork results indicate that at a P_{80} grind size of 49 µm, Pb and Zn sequential flotation can achieve recoveries of 75.4% Pb and 88.9% Zn at concentrate grades of 61.5% Pb and 58.4% Zn.

13.2 Base Met (2018) Sample Selection

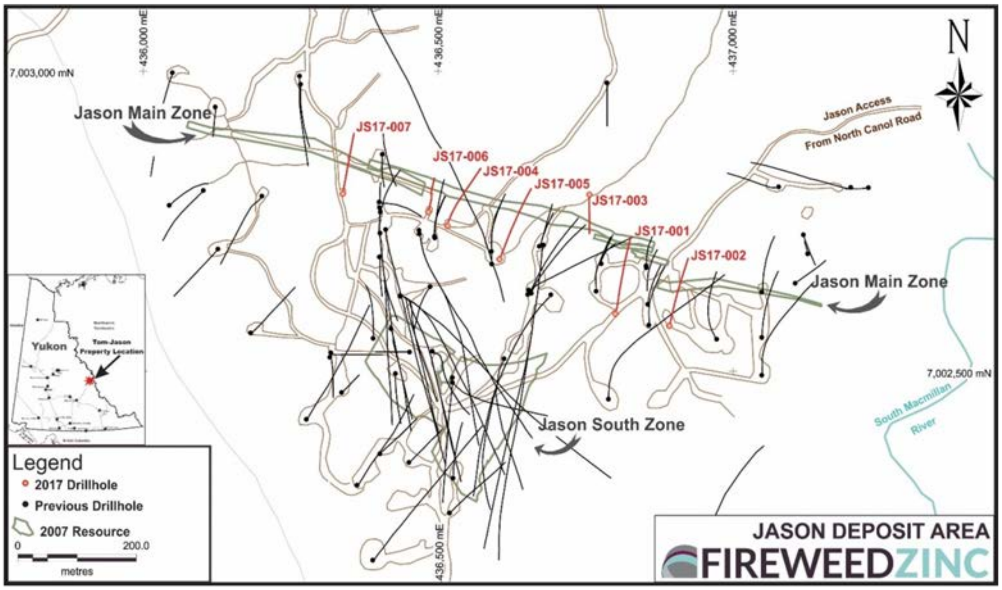
Five composites representing Tom West, Tom East and Jason Main were generated covering sections of the zones and containing representative samples based on mineralogy, grade and location in the deposit. Hanging Wall and Footwall dilution was included with each composite to represent actual mined material. A high grade sub sample of Tom East that contained mercury, designated Composite 3B, was also created and blended with Composite 3A. The location of drill holes for Tom and Jason are presented in Figure 13-1 and Figure 13-2 respectively.

Figure 13-1: Drill Hole Locations for the Tom Zone



Source: Fireweed (2018)

Figure 13-2: Drill Hole Locations for the Jason Zone



Source: Fireweed (2018)

Head assays for the five composites are summarized in Table 13-1. Total Organic Carbon (TOC) assays were performed, indicating a significant portion of the carbon in the sample was present as organic carbon, measuring between 0.3% and 0.9%. Organic carbon is naturally hydrophobic and can contaminate concentrates if it is not adequately controlled. High ratios of organic carbon to Pb in the feed may indicate samples or zones which require control of organic carbon.

Table 13-1: Composite Sample Head Assays

Composite ID	Drill Hole ID	Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	S (%)	C (%)	TOC (%)
Composite 1 TOM WEST	TS17-001	1.21	4.71	1.1	1	8.27	1.22	0.56
Composite 2 TOM WEST	TS17-002 TS17-003	5.39	8.50	2.2	91	11.8	0.63	0.43
Composite 3A TOM EAST	TS17-007	7.60	7.90	3.8	118	9.94	1.38	0.87
Composite 3B TOM EAST	TS17-007	21.0	24.6	2.0	327	16.9	1.31	0.39
Composite 4 JASON MAIN	JS17-001 JS17-002	1.47	6.50	6.2	2	10.4	1.07	0.75
Composite 5 JASON MAIN	JS17-004 JS17-006	2.03	8.70	12.9	1	23.5	0.42	0.30

Source: Base Met (2018)

13.3 Base Met (2018) Mineralogy Results

Feed samples from each composite were submitted for Bulk Mineral Analysis (BMA) using QEMSCAN to determine mineral composition. Table 13-2 summarizes the deportment of minerals in each sample. The Pb mineralization consists mainly of galena; while sphalerite is the main Zn mineral of interest.

Composite 1 was relatively low in sulphide minerals, at about 10% content, while Composite 5 was high in sulphide minerals at about 49%. There was significant pyrite in composites 4 and 5, but very limited pyrite in Composites 1, 2 and 3A. Galena was quite variable, ranging from 1.2% to 8.9%, while sphalerite ranged from 7.4% to 14.6%.

Table 13-2: Composite Sample Mineral Content

Mineral	Comp #1 Tom West	Comp #2 Tom West	Comp #3A Tom East	Comp #4 Jason Main	Comp #5 Jason Main
Galena (%)	1.2	6.1	8.8	1.3	2.0
Sphalerite (%)	7.4	14.6	13.7	10.9	14.0
Pyrite (%)	1.6	4.7	5.8	13.1	32.8
Iron Oxides (%)	0.1	1.4	2.4	0.3	0.1
Quartz (%)	35.0	36.7	42.4	61.4	42.3
Barite (%)	37.2	28.8	9.9	0.9	0.6
Hyalophane/Celsian (%)	6.8	3.1	9.0	2.2	1.0
Edingtonite (%)	1.8	0.3	1.3	0.5	0.2
Witherite BaCO3 (%)	4.3	-	0.4	-	-
'Kaolinite' (clay) (%)	0.7	1.8	1.6	4.9	4.8
Calcite/Dolomite (%)	0.8	0.1	1.0	1.0	0.3
Other Minerals (%)	3.1	2.5	3.8	3.5	1.9

Source: Base Met (2018)

Mineral liberation analysis was also carried out on each composite. The results are shown in Table 13-3 and Table 13-4. At a P_{80} grind size of 66 - 76 µm. Sphalerite has a higher degree of liberation than galena, but both have adequate liberation from gangue material. It should be noted that Composite 2 shows a high degree of galena – sphalerite interlocking. This could pose a problem in achieving a clean separation of lead and zinc for this mineralized material.

Table 13-3: Galena Liberation Analysis

Description	Comp #1 Tom West	Comp #2 Tom West	Comp #3A Tom East	Comp #4 Jason Main	Comp #5 Jason Main
P_{80} Grind Size	66 µm	74 µm	76 µm	69 µm	76 µm
Liberated Galena	39.7%	50.4%	52.9%	41.2%	39.3%
Galena – Chalcopyrite binary	0.1%	0.4%	0.3%	0.2%	0.1%
Galena – Sphalerite binary	3.5%	17.2%	13.3%	13.1%	9.7%
Galena – Pyrite binary	1.5%	3.4%	4.6%	10.8%	14.4%
Galena – Barite binary	7.9%	3.9%	2.2%	0.2%	1.1%
Galena – Gangue binary	21.6%	9.5%	12.8%	16.3%	15.2%
Galena containing multiphase	25.8%	15.2%	13.9%	18.1%	20.2%

Source: Base Met (2018)

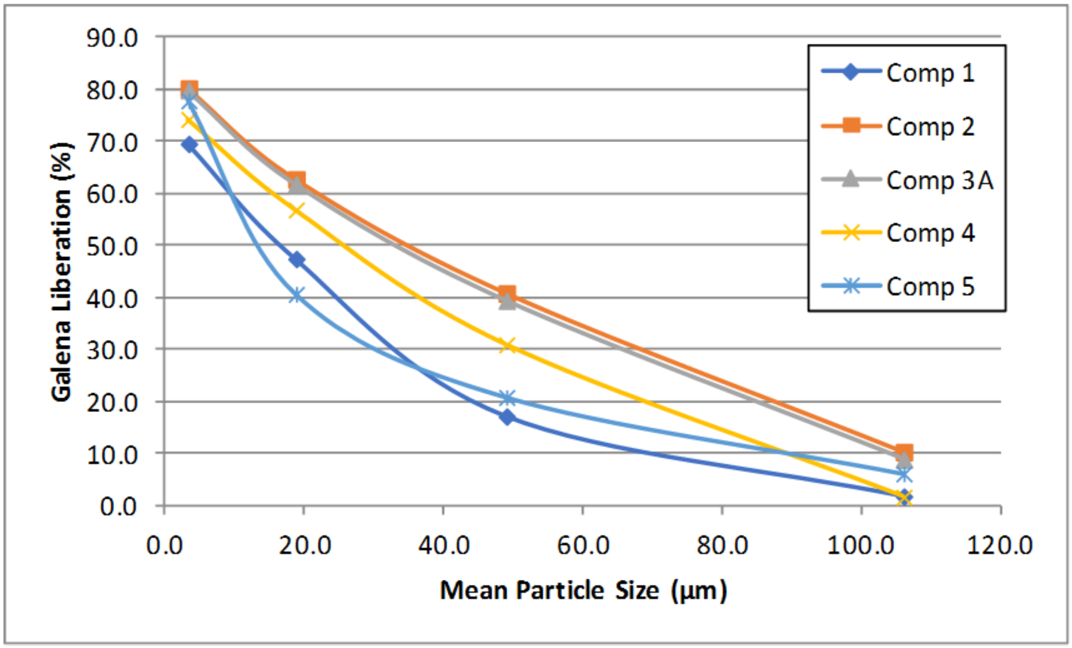
Table 13-4: Sphalerite Liberation Analysis

Description	Comp #1 Tom West	Comp #2 Tom West	Comp #3A Tom East	Comp #4 Jason Main	Comp #5 Jason Main
P_{80} Grind Size	66 µm	74 µm	76 µm	69 µm	76 µm
Liberated Sphalerite	52.4%	54.4%	55.1%	59.4%	66.1%
Sphalerite – Chalcopyrite binary	-	1.8%	0.1%	0.1%	0.7%
Sphalerite – Galena binary	0.7%	6.7%	9.1%	1.7%	1.7%
Sphalerite – Pyrite binary	1.4%	2.5%	2.9%	6.3%	6.6%
Sphalerite – Barite binary	5.6%	3.2%	1.4%	0.2%	0.1%
Sphalerite – Gangue binary	27.2%	22.1%	21.4%	25.2%	19.5%
Sphalerite containing multiphase	12.6%	9.3%	10.0%	7.1%	5.3%

Source: Base Met (2018)

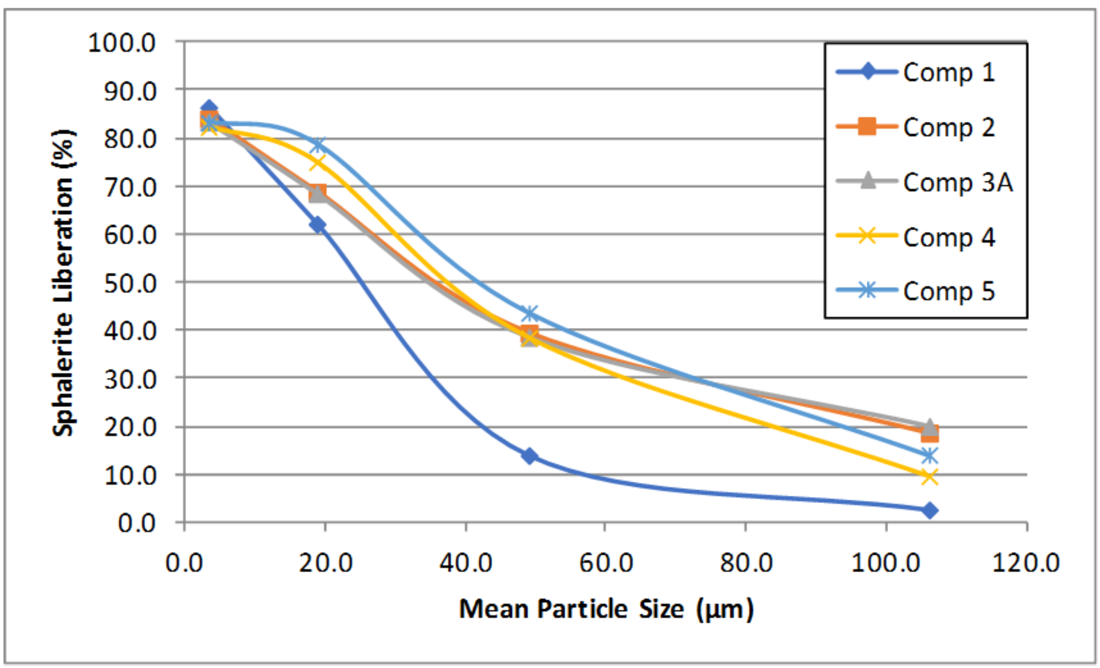
Using the mineral liberation data presented above, release curves for the five composites were generated to evaluate galena and sphalerite liberation as grind size decreased. The results are presented in Figure 13-3 and Figure 13-4 respectively. The curves show that adequate liberation is achievable at a P_{80} primary grind size of 50 µm with the exception of the finer grained material in composite 1. Mineralogy indicates a finer grind size will be required at the cleaning stage to achieve the desired grades and recoveries.

Figure 13-3: Release Curve for Galena Liberation



Source: Base Met (2018)

Figure 13-4: Release Curve for Sphalerite Liberation



Source: Base Met (2018)

13.4 Base Met (2018) Comminution Results

Comminution testwork was carried out to determine the grinding energy required to liberate Pb and Zn minerals prior to flotation. Bond ball mill work index (BWi) tests at a sieve size of 106 µm were completed on all five composites. The results are summarized in Table 13-5. The Tom East composite (Composite 3A) was found to be the hardest sample, with a BWi of 14 kWh/t. This value was used for sizing the grinding equipment referenced in Section 17. Overall, the Tom and Jason material can be ranked at a medium hardness.

Table 13-5: Bond Ball Mill Work Index Results

Composite ID	Sieve Size (µm)	Grams per Revolution (g)	F_{80} (µm)	P_{80} (µm)	Bond Ball Mill Work Index (kWh/t)
Composite 1 TOM WEST	106	1.69	2,274	73	**11.3**
Composite 2 TOM WEST	106	2.27	2,618	73	**8.8**
Composite 3A TOM EAST	106	1.36	2,367	77	**14.0**
Composite 4 JASON MAIN	106	1.45	2,366	76	**13.1**
Composite 5 JASON MAIN	106	1.85	1,916	74	**10.9**

Source: Base Met (2018)

Due to sample size suitability, SMC testing was only completed on Composites 1 and 3A. The results are shown in Table 13-6 and indicate that the samples are soft to moderately hard.

Table 13-6: SMC Data for Composites 1 and 3A

Composite ID	DWi (kWh/m^3)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	A x b	t_a
Composite 1 TOM WEST	4.2	10.7	7.2	3.7	**80.8**	**0.61**
Composite 3A TOM EAST	5.7	14.5	10.4	5.4	**55.8**	**0.46**

Source: Base Met (2018)

Bond abrasion tests were also conducted on Composites 1 and 3A to determine potential wear rates for the crushing and grinding equipment. The results are summarized in Table 13-7. At an average Bond abrasion index of 0.335, the samples are considered moderately abrasive. A weighted average of 0.27 was used for predicting wear rates and estimating annual operating costs.

Table 13-7: Bond Abrasion Results for Composites 1 and 3A

Composite ID	Bond Abrasion Index (g)
Composite 1 TOM WEST	**0.225**
Composite 3A TOM EAST	**0.445**

Source: Base Met (2018)

13.5 Base Met (2018) DMS Results

DMS testing was carried out on Composites 1 and 3A to evaluate the potential to pre-concentrate the sulphide minerals prior to flotation. Twenty kilogram samples were crushed and screened at three size fractions, ¾", ½" and ¼". The -¼" fines were put aside and the three coarse samples were subjected to heavy liquid separation at specific gravities (SG) of 2.85 and 3.00.

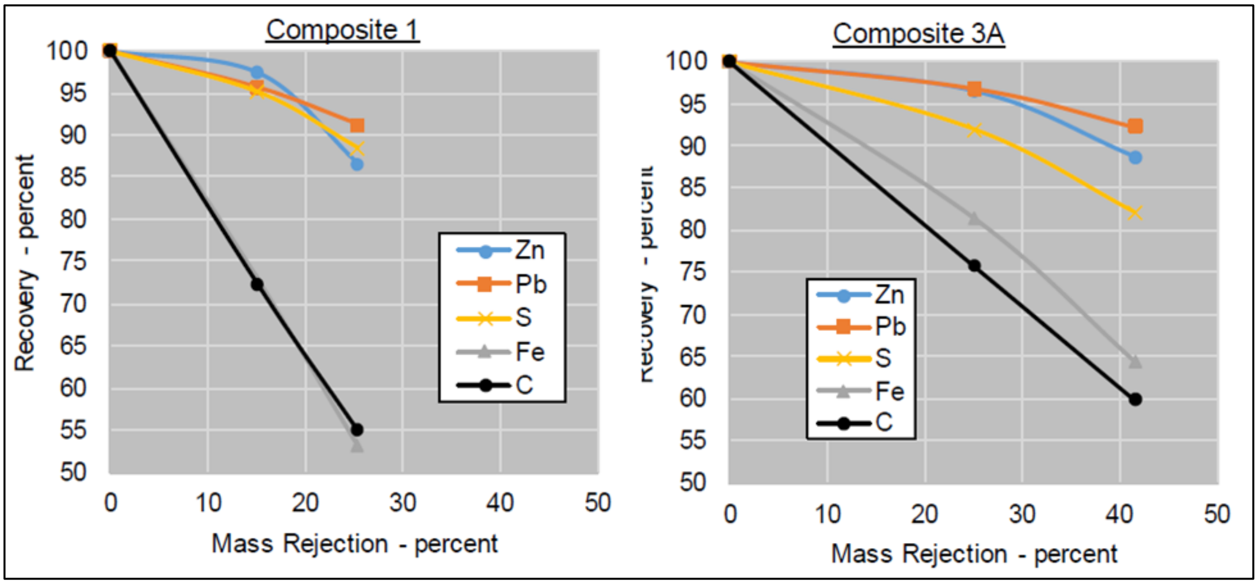
The DMS results are summarized in Table 13-8 and the mass rejection versus recovery curves are shown in Figure 13-5. At a separation SG of 2.85, DMS would be able to reject 15 to 25% of the material while losing approximately 4% Pb and 3% Zn. The mass rejection was quite limited, likely a function of barite content in the samples, which would be concentrated to the sinks along with the sulphide minerals. Since the low mass rejection did not justify the corresponding metal losses, DMS was not considered in the remaining test program

Table 13-8: DMS Results for Composites 1 and 3A

Parameter	Units	Composite 1	Composite 3A
Feed Grade			
Pb Feed Grade	%	1.21	4.93
Zn Feed Grade	%	4.56	7.34
Feed Size Distribution			
+ ¾"	kg	6.2	6.8
- ¾", + ½"	kg	5.1	4.7
- ½", + ¼"	kg	4.0	3.8
- ¼"	kg	4.6	4.9
Product (2.85 Sinks + Fines)			
Mass Pull	%	84.9	74.9
Pb Concentrate Grade	%	1.37	6.38
Zn Concentrate Grade	%	5.24	9.47
Pb Recovery	%	**95.7**	**96.9**
Zn Recovery	%	**97.4**	**96.6**

Source: Base Met (2018)

Figure 13-5: DMS Mass Rejection vs. Recovery Curves



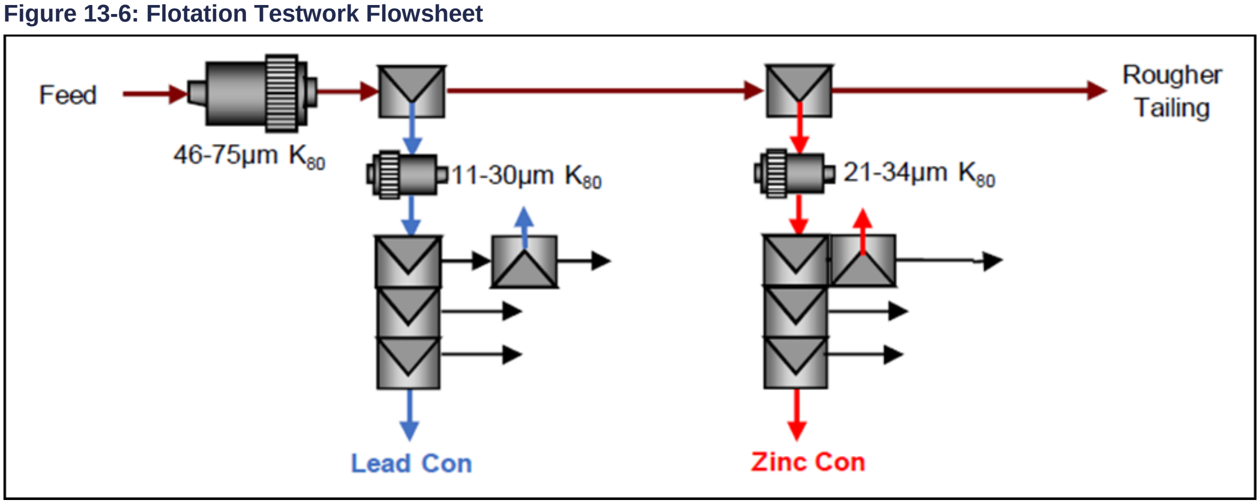
Source: Base Met (2018)

13.6 Base Met (2018) Flotation Variability Results

Sequential Pb and Zn flotation testwork was carried out using the flowsheet presented in Figure 13-6. All five composites were subjected to both rougher and cleaner flotation testing to determine the appropriate conditions required to achieve saleable Pb and Zn concentrates. The key parameters investigated in rougher flotation were primary grind, lead selectivity over Zn (Composite 2), and carbon depression with CarboxyMethyl Cellulose (CMC). The CMC used was marketed under the product name PE26.

After completing flowsheet development, global composites were created for locked cycle testing. These results were used for the recovery projections summarized in Section 13.10.

Figure 13-6: Flotation Testwork Flowsheet

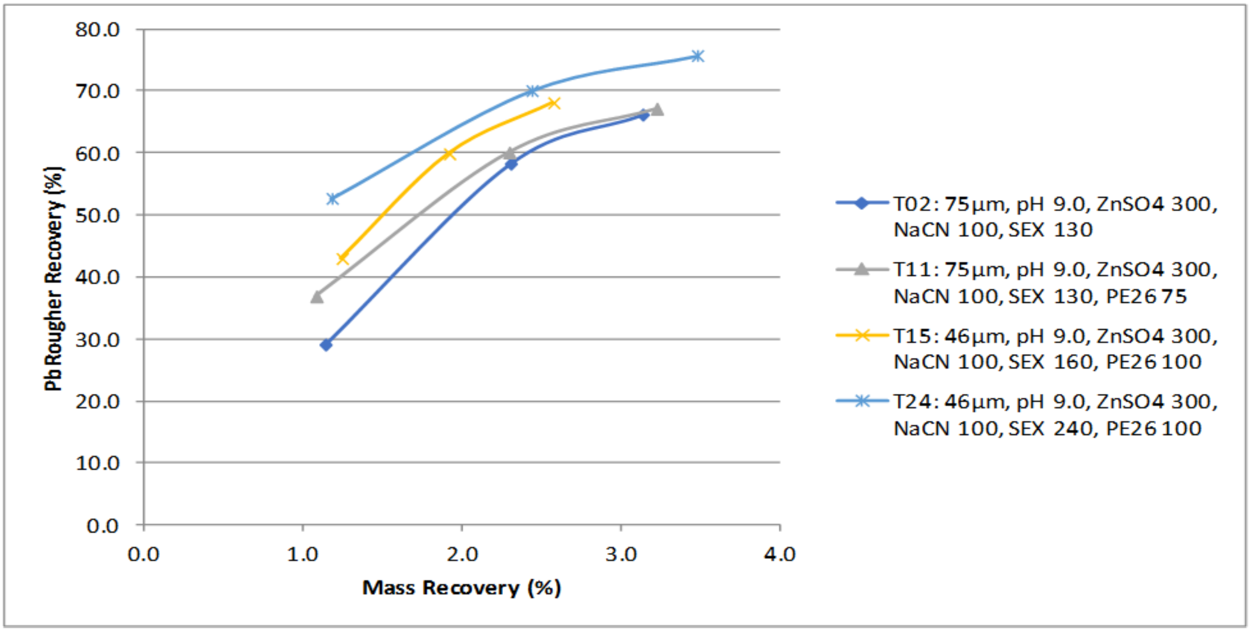


Source: Base Met (2018)

13.6.1 Composite 1 – Tom West

Four initial rougher flotation tests were completed on Composite 1 to evaluate how grind size and reagent dosage affected rougher kinetics. The selected standard reagent scheme was selected. For the lead circuit, sodium ethyl xanthate (SEX) was the collector, soda ash was used as a pH modifier, and sodium cyanide (NaCN) and zinc sulphate ($ZnSO_4$) were used as sphalerite and pyrite depressants. For the zinc circuit, sodium iso-propyl xanthate (SIPX) was the collector, lime the pH modifier and copper sulphate ($CuSO_4$) as the sphalerite activator. MIBC and Polyfroth H57 were used as the frothers in both circuits Concentrate samples were taken every two minutes for a total flotation time of 6 minutes. The results for the Pb rougher circuit are shown in Figure 13-7. A finer P_{80} grind size of 46 µm improved rougher kinetics; while an increased dosage of sodium ethyl xanthate (SEX) improved Pb recovery. PE26 was added to suppress organic carbon and reduce reagent consumption.

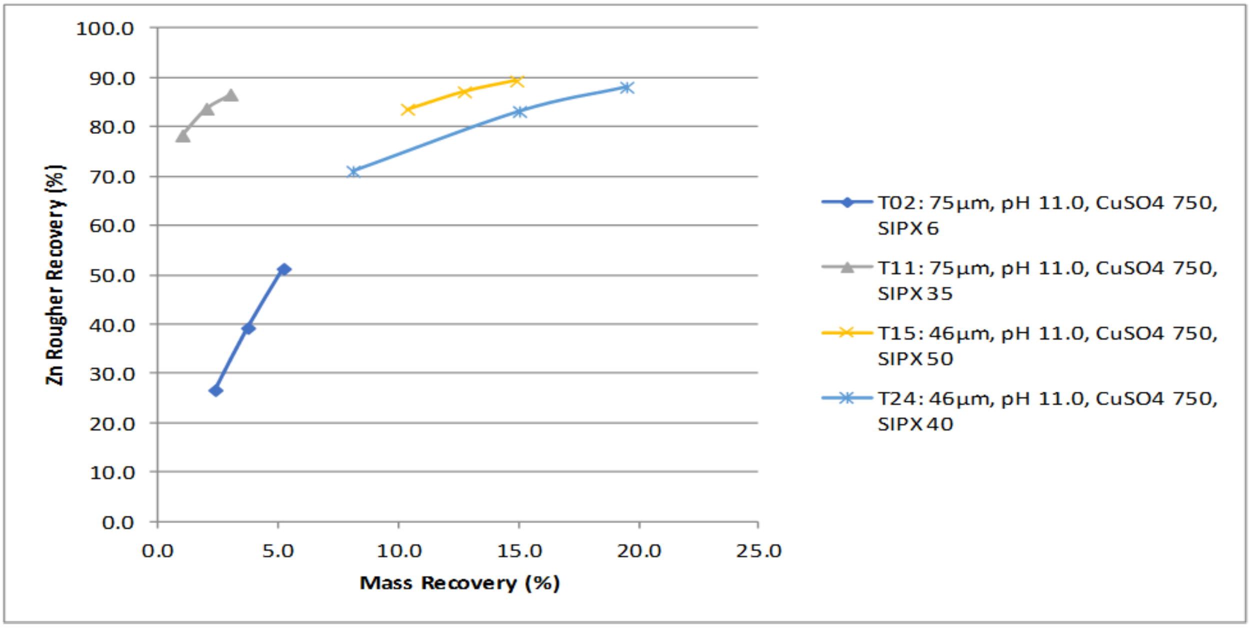
Figure 13-7: Pb Rougher Kinetic Testing for Composite 1



Source: Base Met (2018)

The Pb rougher tailings were then used as the feed for Zn flotation. The results for the Zn rougher circuit are shown in Figure 13-8. In similar findings to the Pb results, a finer P_{80} grind size of 46 µm achieved better flotation kinetics.

Figure 13-8: Zn Rougher Kinetic Testing for Composite 1

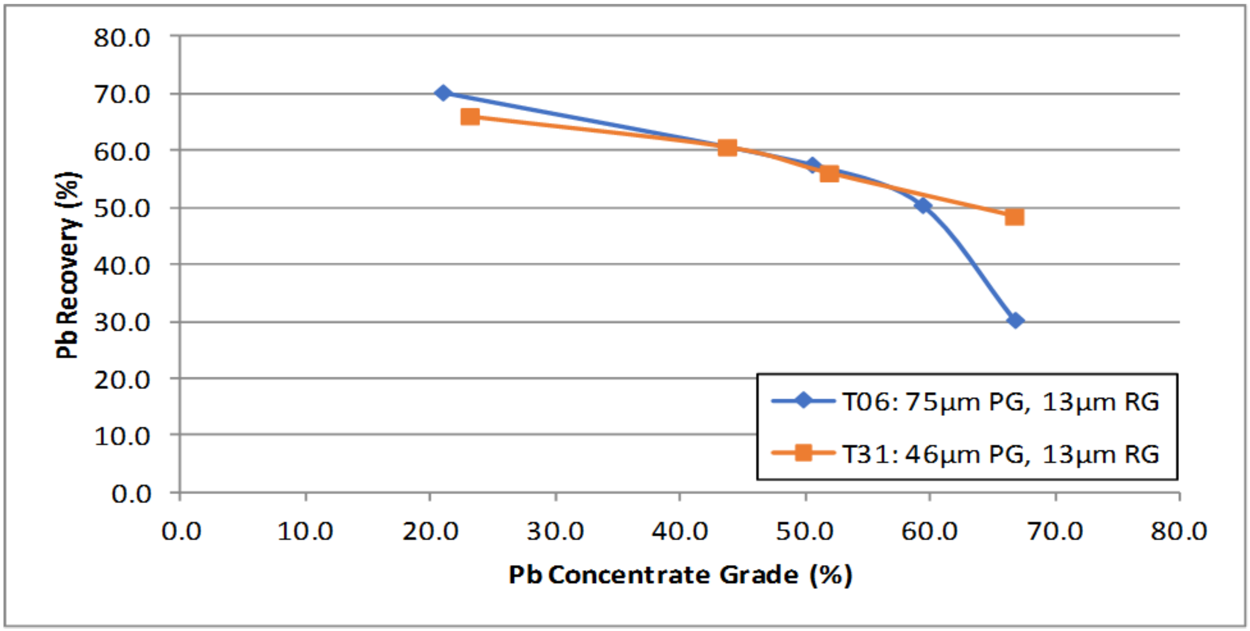


Source: Base Met (2018)

Two cleaner flotation tests were performed to determine if high concentrate grades could be achieved at high overall Pb and Zn recoveries. These tests were completed at primary P_{80} grind sizes of 75 µm and 46 µm. In the Pb circuit, rougher concentrate was reground to a P_{80} grind size of 13 µm and subjected to three stages of cleaner flotation. Pb rougher tailings were then used to feed the Zn circuit.

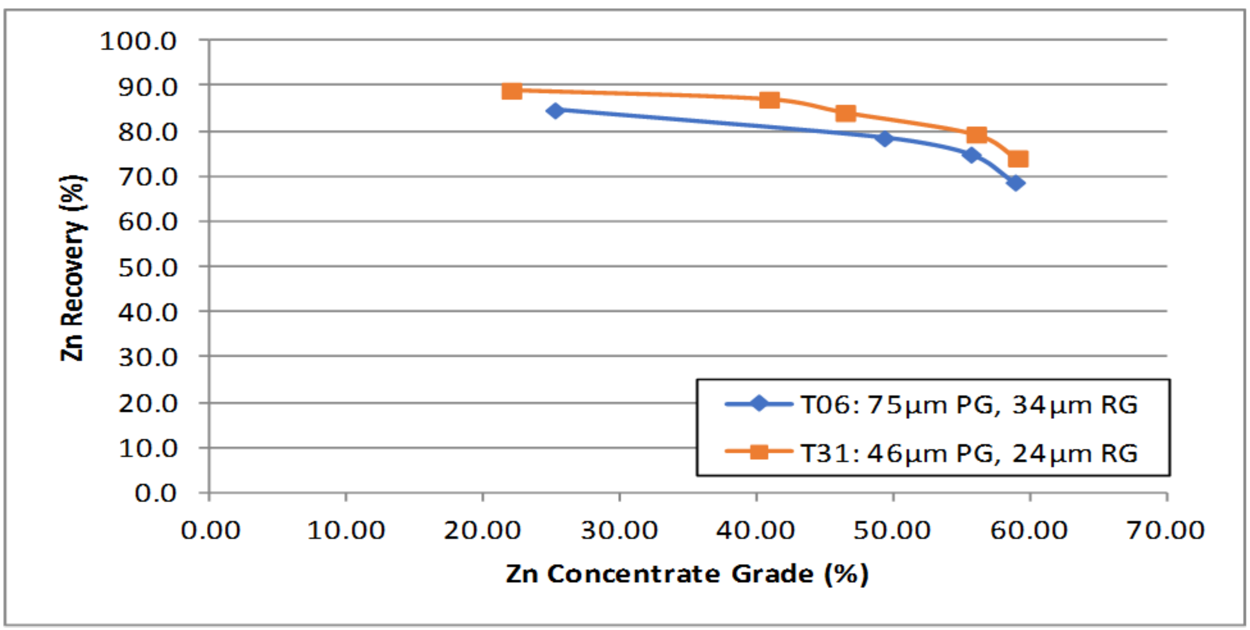
Zn rougher concentrate was reground to a P_{80} of 24-34 µm and subjected to three stages of cleaner flotation and one stage of cleaner scavenger flotation. Grade vs. recovery curves for Pb and Zn at the two selected primary grind sizes are shown in Figure 13-9 and Figure 13-10 respectively. For both circuits, a P_{80} grind size of 46 µm achieved higher recoveries at higher concentrate grades.

Figure 13-9: Pb Grade vs. Recovery Curves for Composite 1



Source: Base Met (2018)

Figure 13-10: Zn Grade vs. Recovery Curves for Composite 1



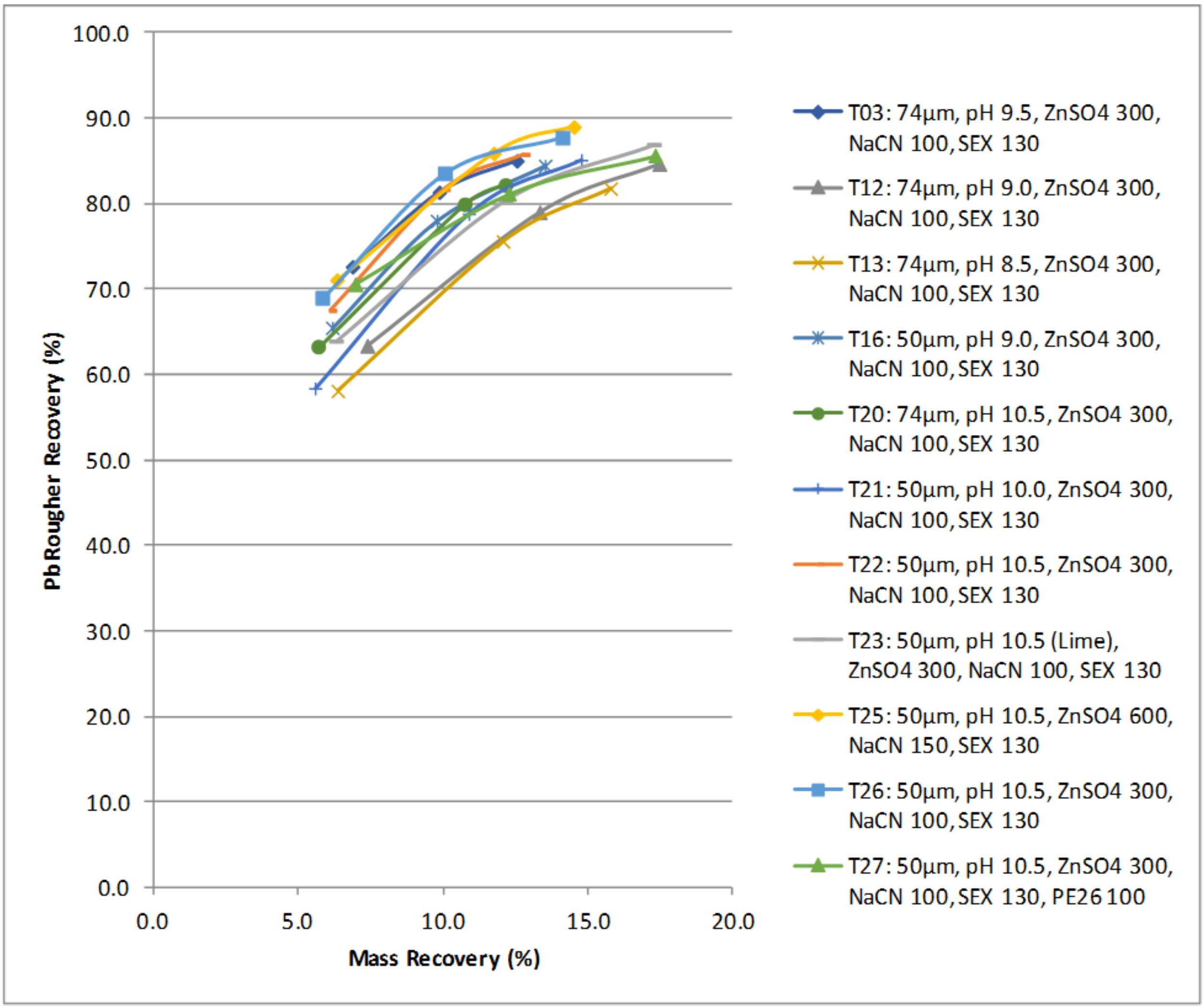
Source: Base Met (2018)

13.6.2 Composite 2 – Tom West

A significant portion of flotation testing for Composite 2 focused on improving Pb rougher flotation selectivity against Zn. Up to 69% of the Zn in the feed was recovered into the Pb rougher concentrate. This suggested possible chemical activation of sphalerite by Pb or Cu ions.

Composite 2 required eleven rougher flotation tests to evaluate how grind size, pH and reagent dosage affected rougher kinetics. Concentrate samples were taken every two minutes for a total flotation time of 6 minutes. The results for the Pb rougher circuit are shown in Figure 13-11. A finer P_{80} grind size of 50 µm at a pH of 10.5 with a SEX dosage of 130 g/t resulted in the best Pb rougher performance, reducing Zn recovery to 38.8% in the Pb rougher concentrate. The addition of PE26 did not have a positive effect on rougher recovery.

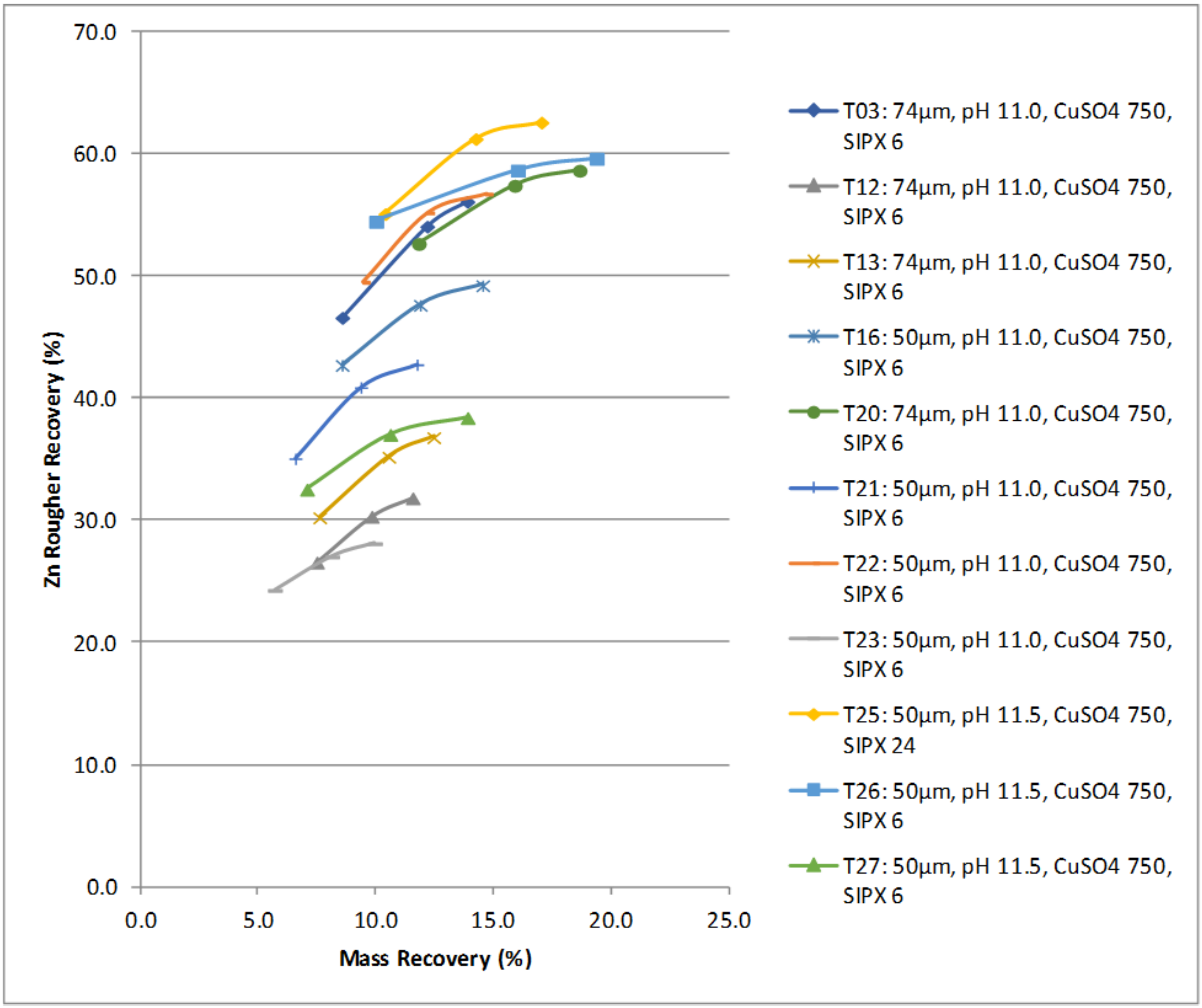
Figure 13-11: Pb Rougher Kinetic Testing for Composite 2



Source: Base Met (2018)

The Pb rougher tailings were then used as the feed for Zn flotation. The results for the Zn rougher circuit are shown in Figure 13-12. In similar findings to the Pb results, a finer P_{80} grind size of 50 µm achieved better flotation kinetics. It should be noted that a higher dosage of Zn depressants ($ZnSO_4$ / NaCN) in the lead circuit resulted in increased Zn rougher recovery, as Zn losses to the Pb circuit were reduced.

Figure 13-12: Zn Rougher Kinetic Testing for Composite 2

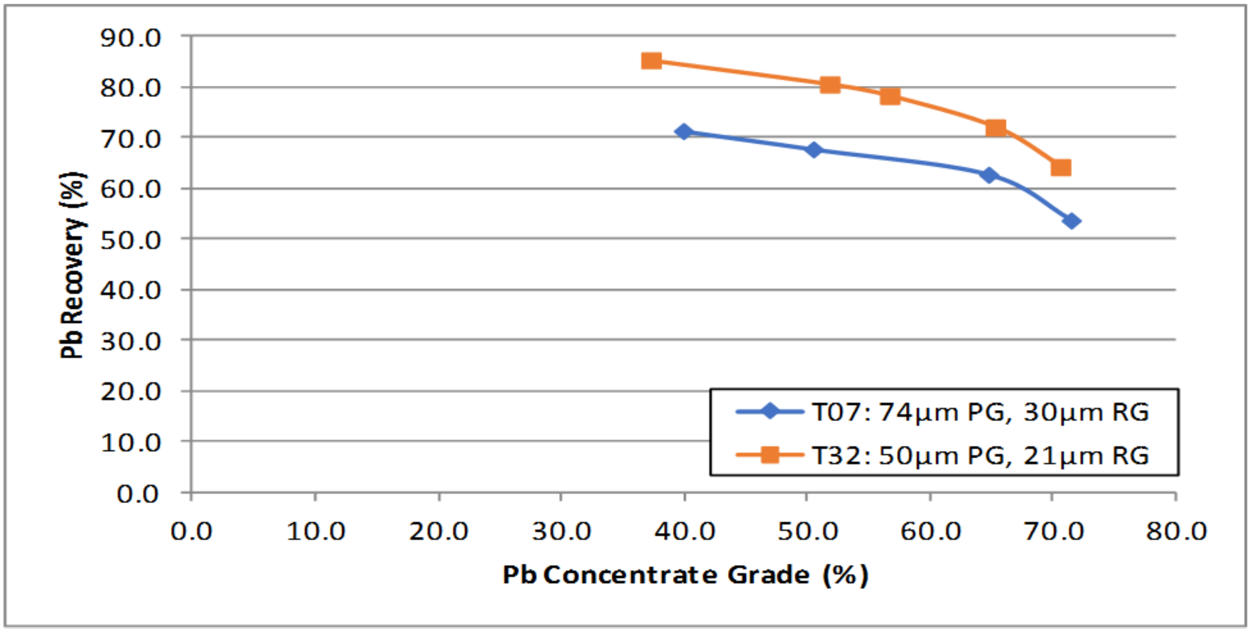


Source: Base Met (2018)

Two cleaner flotation tests were performed to determine if high concentrate grades could be achieved at high overall Pb and Zn recoveries. These tests were completed at primary P_{80} grind sizes of 74 µm and 50 µm. In the Pb circuit, rougher concentrate was reground to a P_{80} grind size of 21-30 µm and subjected to three stages of cleaner flotation. Pb rougher tailings were then used to feed the Zn circuit.

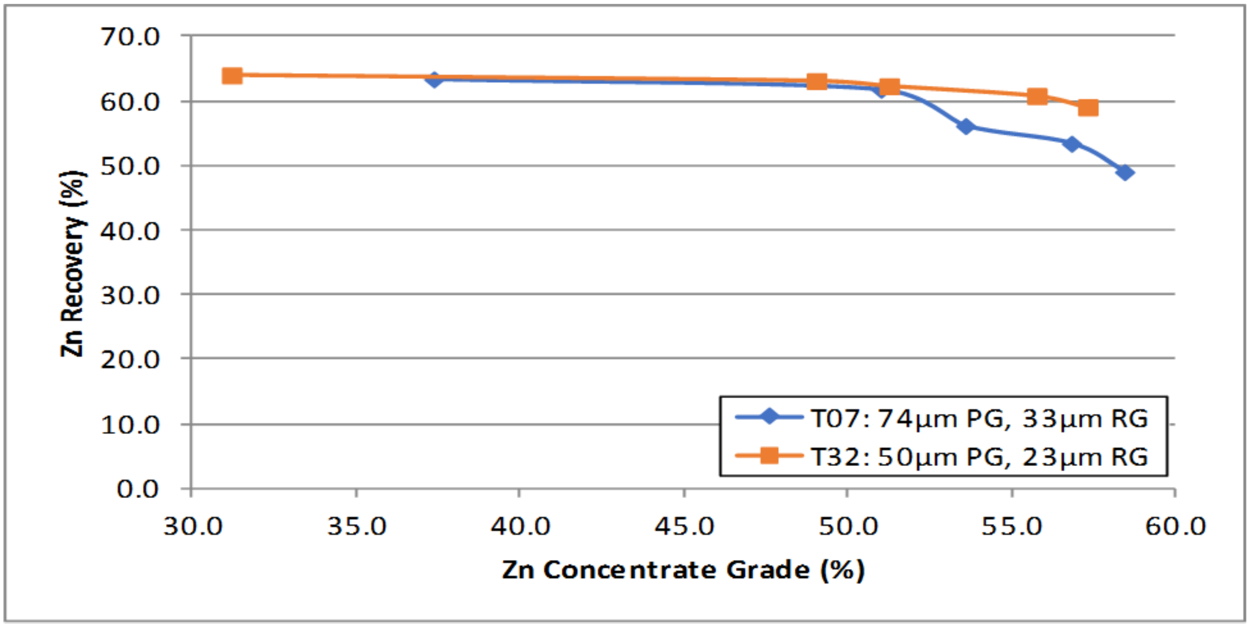
Zn rougher concentrate was reground to a P_{80} of 23-33 µm and subjected to three stages of cleaner flotation and one stage of cleaner scavenger flotation. Grade vs. recovery curves for Pb and Zn at the two primary grind sizes are shown in Figure 13-13 and Figure 13-14 respectively. For both circuits, a P_{80} grind size of 50 µm achieved higher recoveries at higher concentrate grades.

Figure 13-13: Pb Grade vs. Recovery Curves for Composite 2



Source: Base Met (2018)

Figure 13-14: Zn Grade vs. Recovery Curves for Composite 2

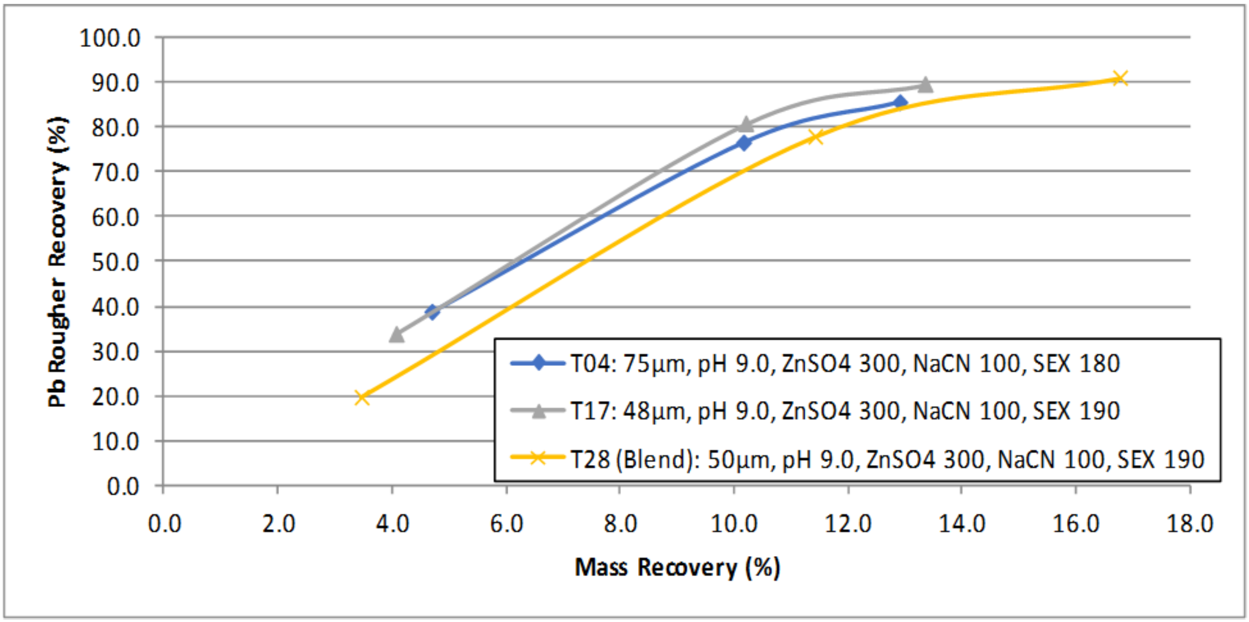


Source: Base Met (2018)

13.6.3 Composite 3A / 3B – Tom East

Two initial rougher flotation tests were completed on Composite 3A to evaluate how grind size and reagent dosage affected rougher kinetics. Concentrate samples were taken every two minutes for a total flotation time of six minutes. The results for the Pb rougher circuit are shown in Figure 13-15. A finer P_{80} grind size of 48 µm improved rougher kinetics; while an increased dosage of SEX improved Pb recovery. A 94:6 blend of Composite 3A and Composite 3B, reflecting the anticipated mine plan, was also evaluated to determine how the high grade Composite 3B affected rougher flotation (Test #28). This sample achieved the highest overall Pb recovery, but at a higher mass pull and lower concentrate grade.

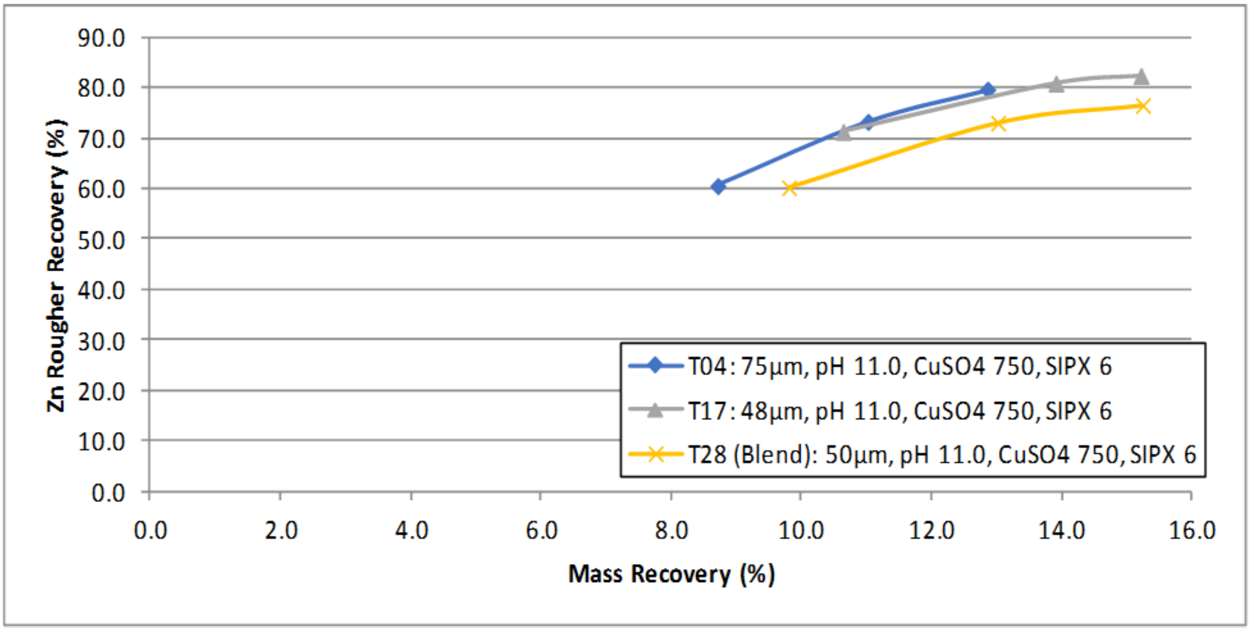
Figure 13-15: Pb Rougher Kinetic Testing for Composite 3A / 3B



Source: Base Met (2018)

The Pb rougher tailings were then used as the feed for Zn flotation. The results for the Zn rougher circuit are shown in Figure 13-16. In similar findings to the Pb results, a finer P_{80} grind size of 48 µm achieved slightly higher Zn recovery, but with a higher mass pull. The 94:6 blend did not perform as well as Composite 3A by itself. This was due to a large percentage of Zn (19.8%) reporting to the Pb rougher concentrate.

Figure 13-16: Zn Rougher Kinetic Testing for Composite 3A / 3B

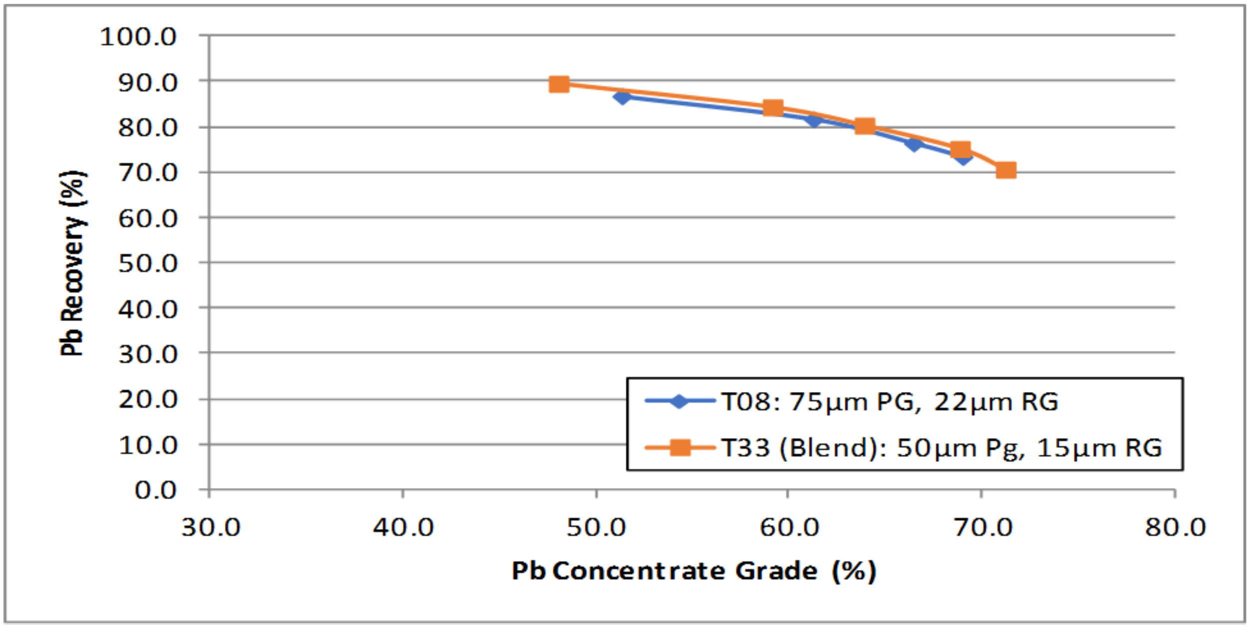


Source: Base Met (2018)

Cleaner flotation tests were performed on both the Composite 3A and the 94:6 blended samples to determine if high concentrate grades could be achieved at high overall Pb and Zn recoveries. These tests were completed at primary P_{80} grind sizes of 75 µm (Composite 3A) and 50 µm (94:6 blend). In the Pb circuit, rougher concentrate was reground to a P_{80} grind size of 15-22 µm and subjected to three stages of cleaner flotation. Pb rougher tailings were then used to feed the Zn circuit.

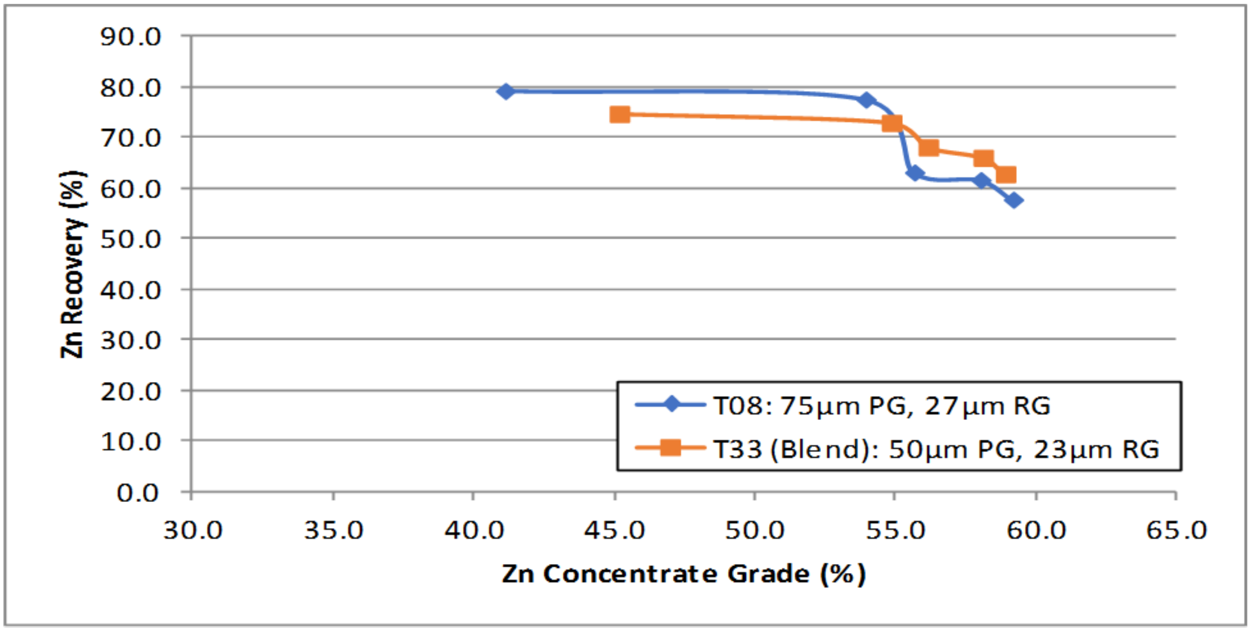
Zn rougher concentrate was reground to a P_{80} of 23-27 µm and subjected to three stages of cleaner flotation and one stage of cleaner scavenger flotation. Grade vs. recovery curves for Pb and Zn at the two primary grind sizes are shown in Figure 13-17 and Figure 13-18 respectively.

Figure 13-17: Pb Grade vs. Recovery Curves for Composite 3A / 3B



Source: Base Met (2018)

Figure 13-18: Zn Grade vs. Recovery Curves for Composite 3A / 3B

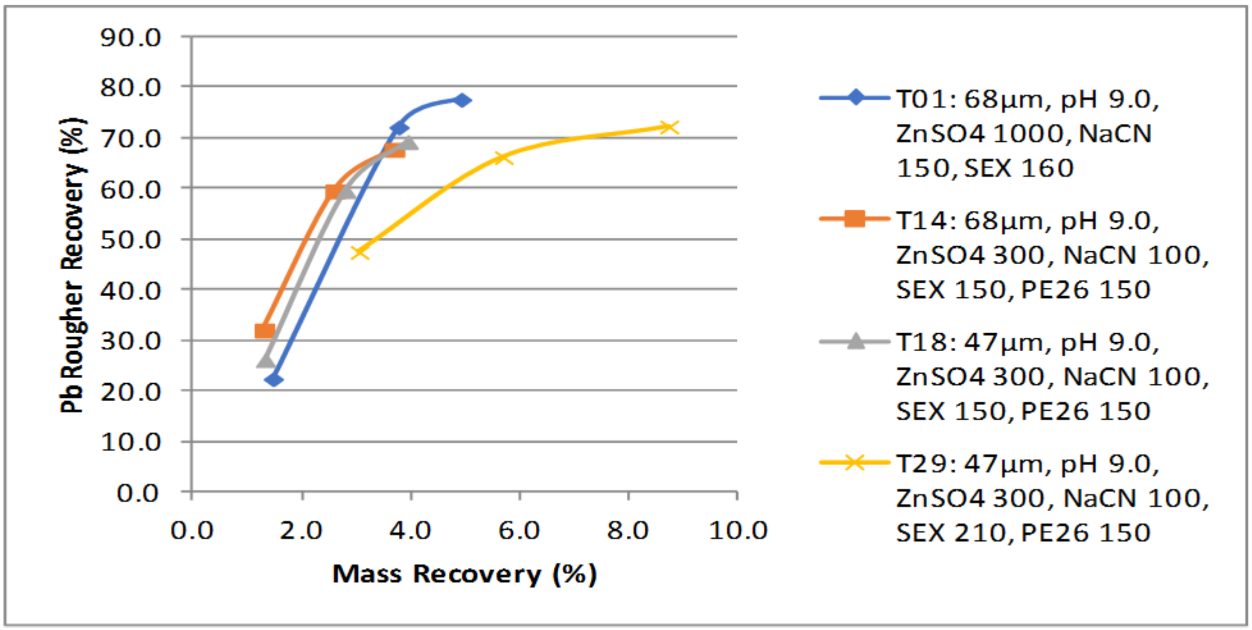


Source: Base Met (2018)

13.6.4 Composite 4 – Jason Main

Four initial rougher flotation tests were completed on Composite 4 to evaluate how grind size and reagent dosage affected rougher kinetics. Concentrate samples were taken every two minutes for a total flotation time of 6 minutes. The results for the Pb rougher circuit are shown in Figure 13-19. A finer P_{80} grind size of 47 µm had little effect on rougher kinetics; while an increased dosage of SEX improved Pb recovery, but at a much higher mass pull. PE26 was added to suppress organic carbon and reduce reagent consumption.

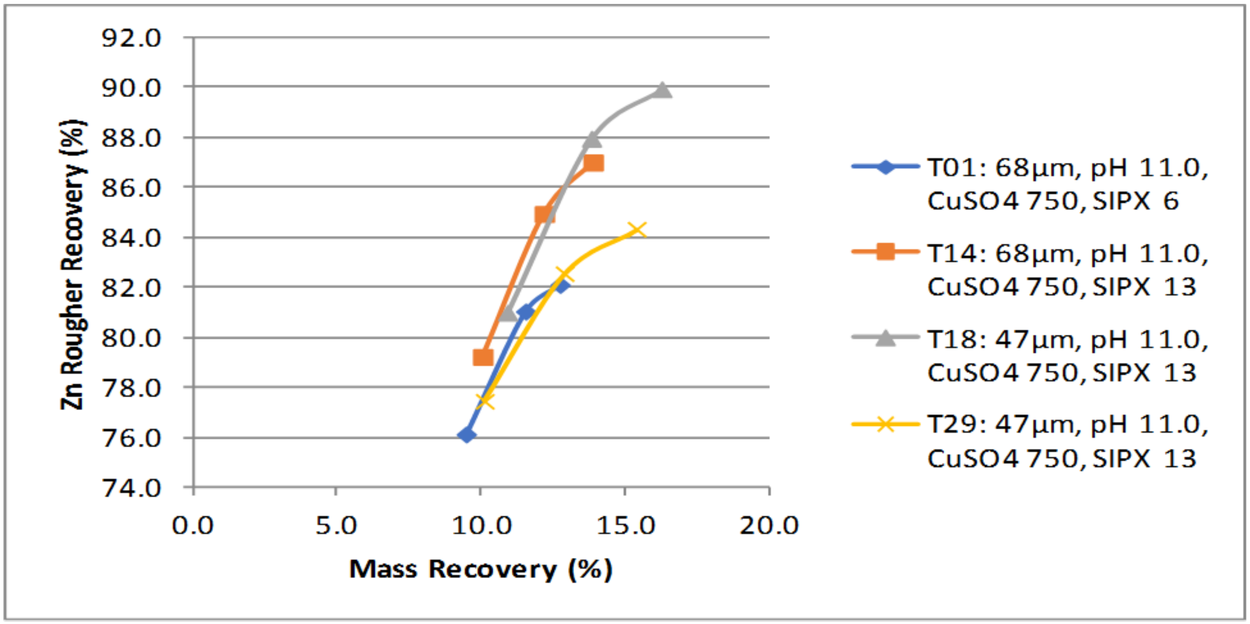
Figure 13-19: Pb Rougher Kinetic Testing for Composite 4



Source: Base Met (2018)

The Pb rougher tailings were then used as the feed for Zn flotation. The results for the Zn rougher circuit are shown in Figure 13-20. In contrast to the Pb results, a finer P_{80} grind size of 47 µm achieved slightly higher Zn recovery.

Figure 13-20: Zn Rougher Kinetic Testing for Composite 4

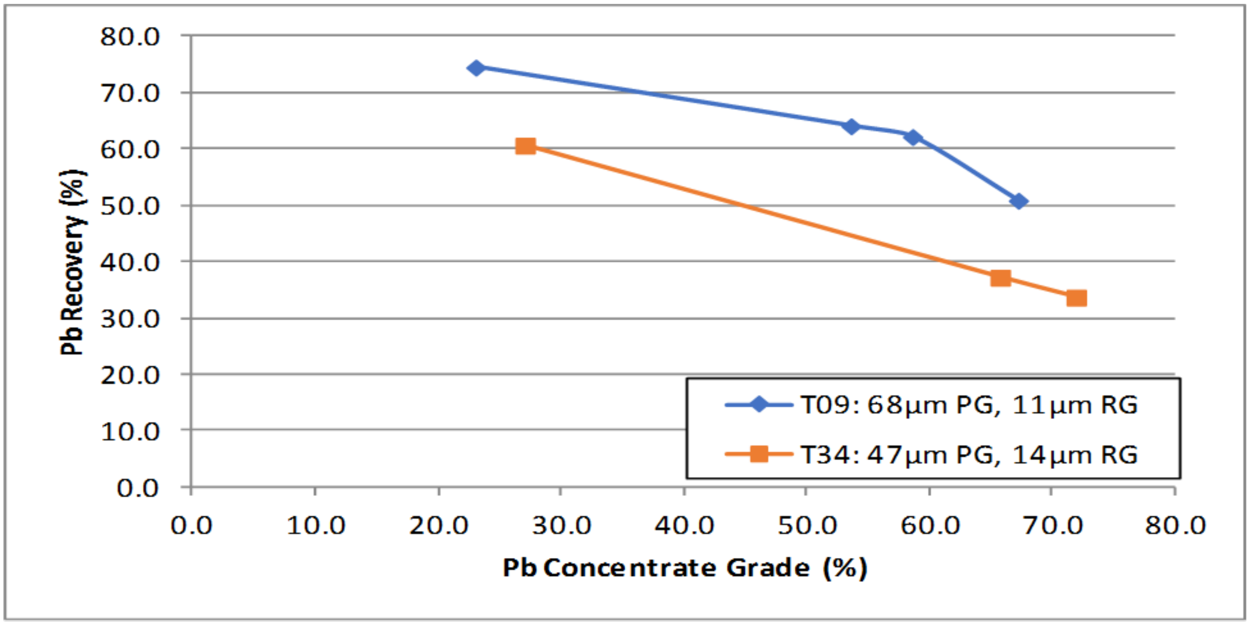


Source: Base Met (2018)

Two cleaner flotation tests were performed to determine if high concentrate grades could be achieved at high overall Pb and Zn recoveries. These tests were completed at primary P_{80} grind sizes of 68 µm and 47 µm. In the Pb circuit, rougher concentrate was reground to a P_{80} grind size of 11-14 µm and subjected to two stages (Test #34) or three stages (Test #9) of cleaner flotation. Pb rougher tailings were then used to feed the Zn circuit.

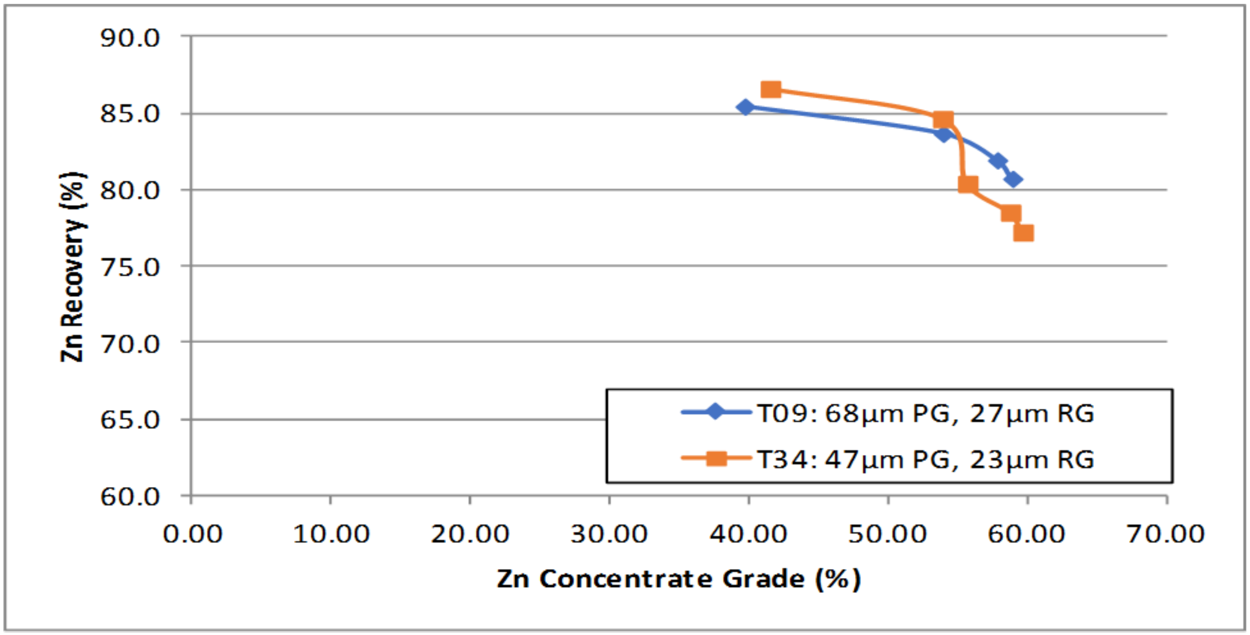
Zn rougher concentrate was reground to a P_{80} of 23-27 µm and subjected to three stages of cleaner flotation and one stage of cleaner scavenger flotation. Grade vs. recovery curves for Pb and Zn at the two primary grind sizes are shown in Figure 13-21 and Figure 13-22 respectively. For the Pb circuit, a finer P_{80} grind size of 47 µm negatively impacted recovery, although Pb concentrate grade increased. For the Zn circuit, a finer P_{80} grind size did not have a significant impact on the grade vs. recovery curve.

Figure 13-21: Pb Grade vs. Recovery Curves for Composite 4



Source: Base Met (2018)

Figure 13-22: Zn Grade vs. Recovery Curves for Composite 4

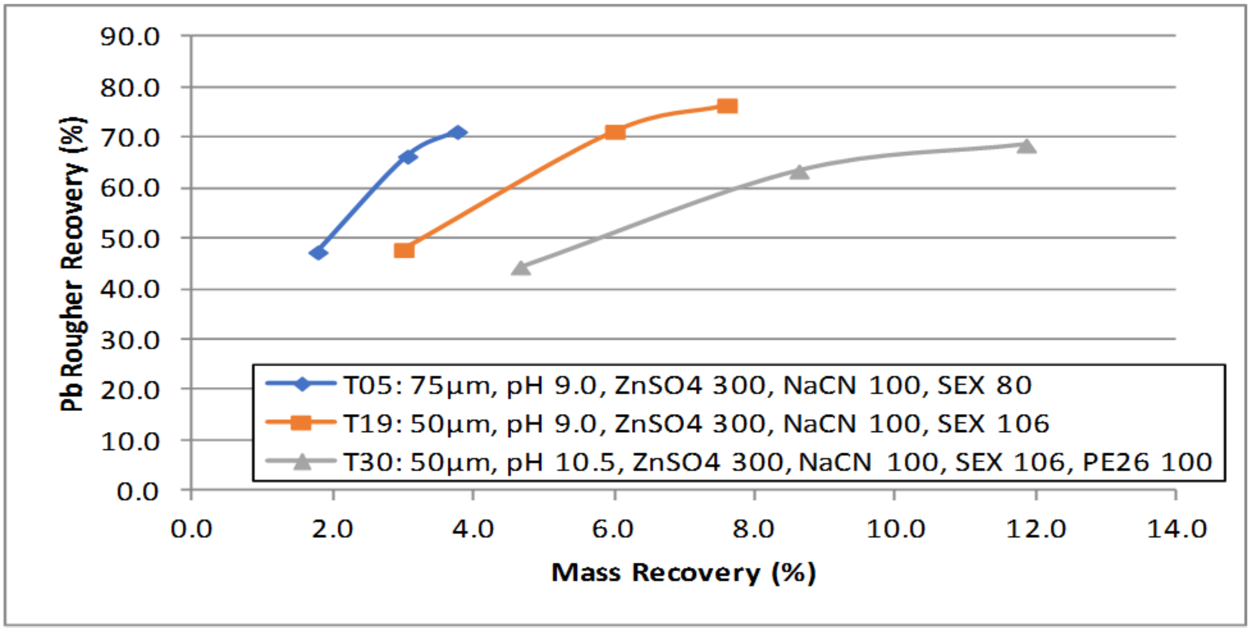


Source: Base Met (2018)

13.6.5 Composite 5 – Jason Main

Three initial rougher flotation tests were completed on Composite 5 to evaluate how grind size and reagent dosage affected rougher kinetics. Concentrate samples were taken every two minutes for a total flotation time of six minutes. The results for the Pb rougher circuit are shown in Figure 13-23. A finer P_{80} grind size of 50 µm improved Pb recovery; while an increased dosage of SEX increased rougher mass pull. PE26 was added to suppress organic carbon and reduce reagent consumption. This reagent dosing is sensitive and can negatively affected rougher flotation, reducing Pb recovery and increasing rougher mass pull.

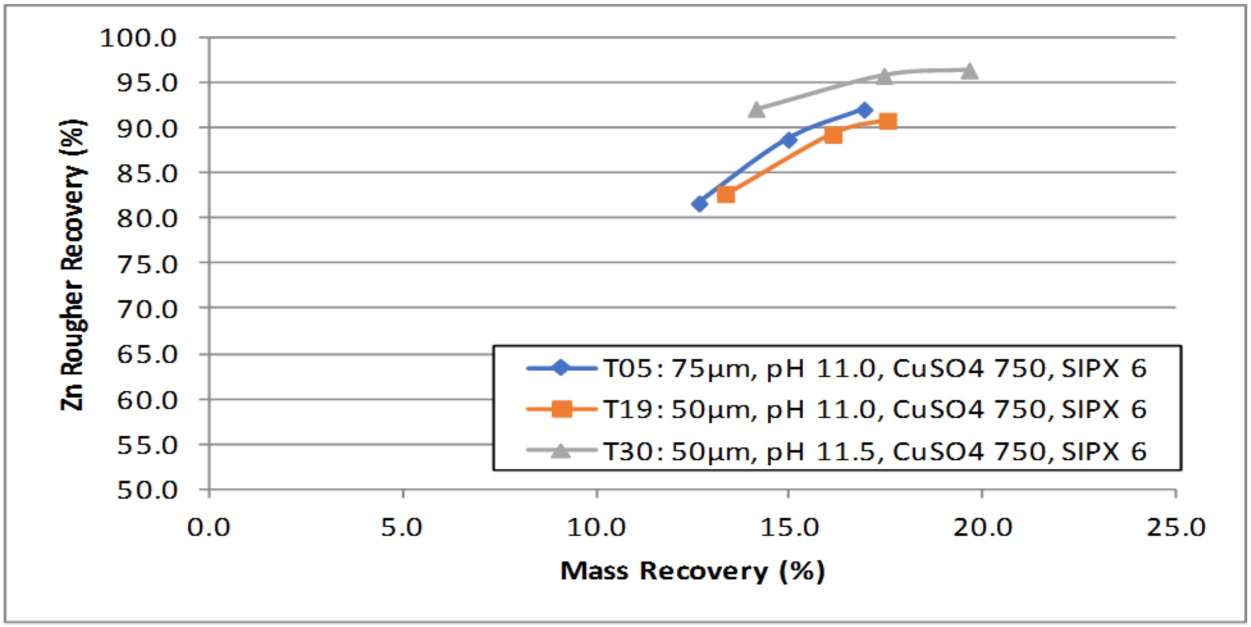
Figure 13-23: Pb Rougher Kinetic Testing for Composite 5



Source: Base Met (2018)

The Pb rougher tailings were then used as the feed for Zn flotation. The results for the Zn rougher circuit are shown in Figure 13-24. In contrast to the Pb results, the addition of PE26 in the Pb circuit improved Zn recovery considerably.

Figure 13-24: Zn Rougher Kinetic Testing for Composite 5

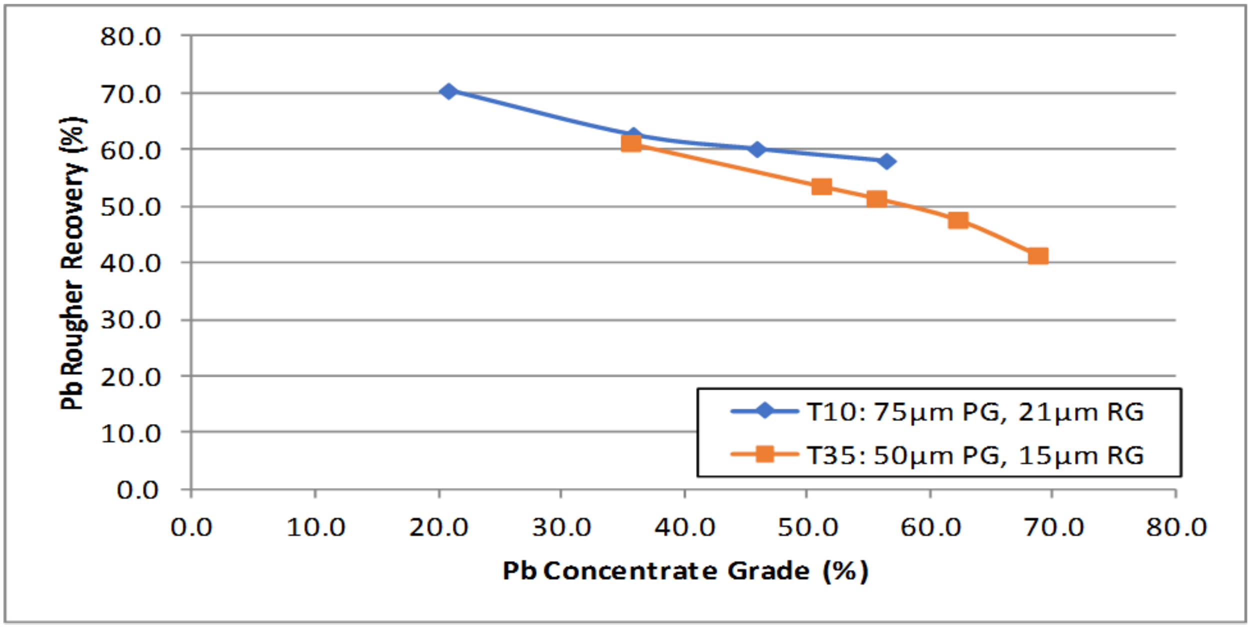


Source: Base Met (2018)

Two cleaner flotation tests were performed to determine if high concentrate grades could be achieved at high overall Pb and Zn recoveries. These tests were completed at primary P_{80} grind sizes of 75 µm and 50 µm. In the Pb circuit, rougher concentrate was reground to a P_{80} grind size of 15-21 µm and subjected to two stages (Test #10) or three stages (Test #35) of cleaner flotation. Pb rougher tailings were then used to feed the Zn circuit.

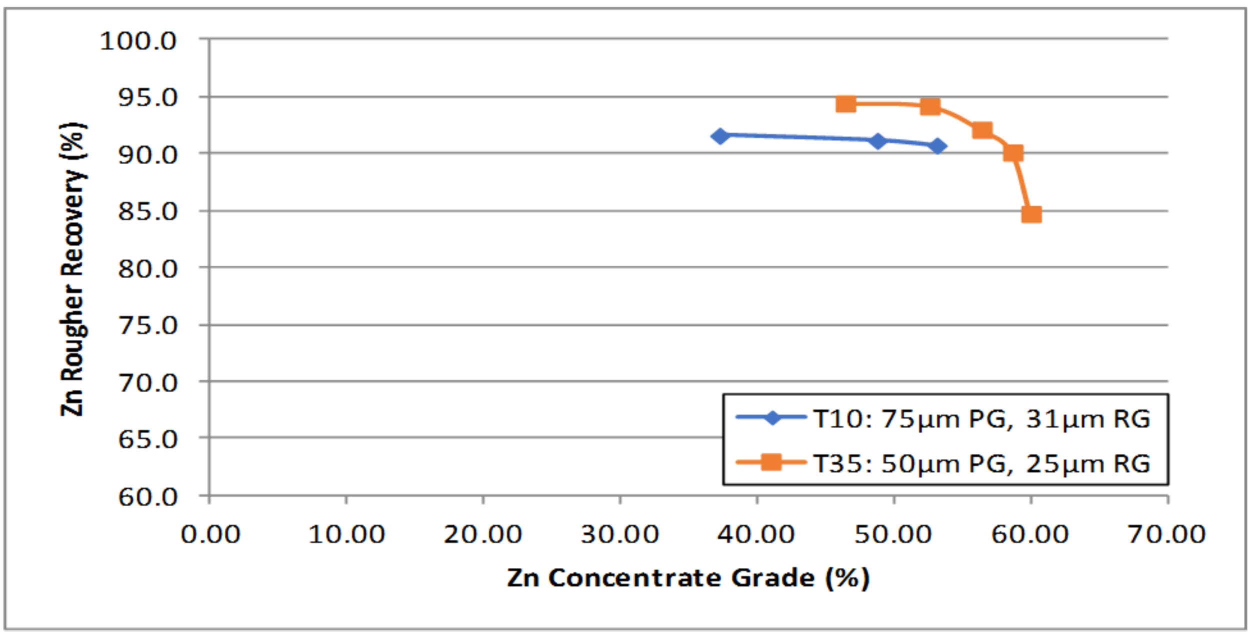
Zn rougher concentrate was reground to a P_{80} of 25-31 µm and subjected to three stages of cleaner flotation and one stage of cleaner scavenger flotation. Grade vs. recovery curves for Pb and Zn at the two primary grind sizes are shown in Figure 13-25 and Figure 13-26 respectively. For the Pb circuit, a finer P_{80} grind size of 50 µm resulted in lower Pb recoveries but higher concentrate grades than the 75 um grind. For the Zn circuit, a finer P_{80} grind size did result in a significant improvement of the grade vs. recovery curve.

Figure 13-25: Pb Grade vs. Recovery Curves for Composite 5



Source: Base Met (2018)

Figure 13-26: Zn Grade vs. Recovery Curves for Composite 5



Source: Base Met (2018)

13.6.6 Summary of Variability Flotation Testing

A summary of the cleaner flotation results for each composite is presented in Table 13-9. The tests operated at a target P_{80} grind size of 50 µm were chosen as the optimal results due to the positive results observed in the Zn circuit. It should be noted that these overall batch cleaner flotation tests confirm that high Pb and Zn final concentrate grades can be achieved. The recoveries from these tests are not representative of the overall combined circuit performance for the selected flowsheet. For these overall recovery estimates, locked cycle tests were completed later in the test program.

Table 13-9: A Summary of Cleaner Flotation Results

Composite ID	Test Number	Pb Flotation		Zn Flotation	
		Concentrate Grade (%)	Recovery (%)	Concentrate Grade (%)	Recovery (%)
Composite 1 TOM WEST	31	58.7	48.4	59.1	74.0
Composite 2 TOM WEST	32	70.5	64.3	57.3	59.0
Composite 3A / 3B TOM EAST	33	71.2	70.6	58.9	62.8
Composite 4 JASON MAIN	34	71.9	33.6	59.7	77.2
Composite 5 JASON MAIN	35	68.7	41.3	56.4	92.1

Source: Base Met (2018)

13.7 Base Met (2018) Global Composite Flotation Results

After developing flowsheet conditions through variability testing, global composites for the Tom and Jason deposits were created using the ratios shown Table 13-10. Jason is a blend of drill core from Jason Main. Drill core from Jason South was not available for the test program. The Blend Global composite represents the estimate LOM proportions of Tom and Jason based on the mine schedule.

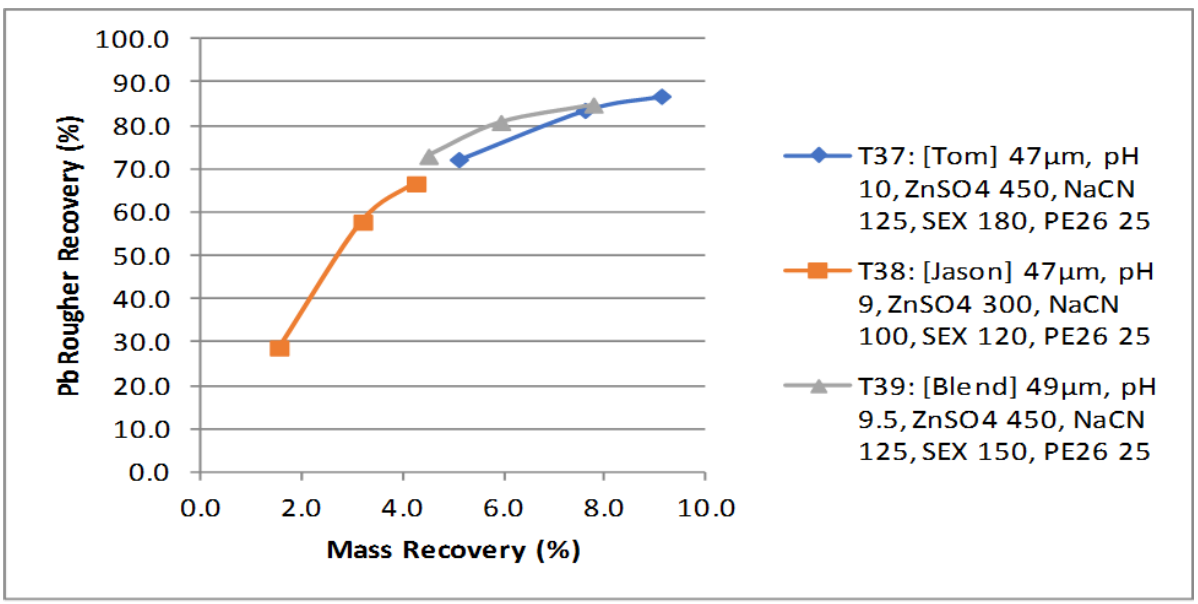
Table 13-10: Composition of Global Composites

	Tom Composite	Jason Composite	Blend Global Composite
Composite Composition			
Composite 1	40%	-	-
Composite 2	40%	-	-
Composite 3A / 3B Blend	20%	-	-
Composite 4	-	60%	-
Composite 5	-	40%	-
Tom Composite	-	-	65%
Jason Composite	-	-	35%
Head Grade			
Lead (%)	4.31	1.68	3.31
Zn (%)	7.40	7.35	7.20
Iron (%)	3.5	10.2	5.9
Silver (g/t)	65	4	41
Sulphur (%)	10.6	15.4	12.4
Carbon (%)	1.02	0.81	0.90
Total Organic Carbon (%)	0.58	0.56	0.60

Source: Base Met (2018)

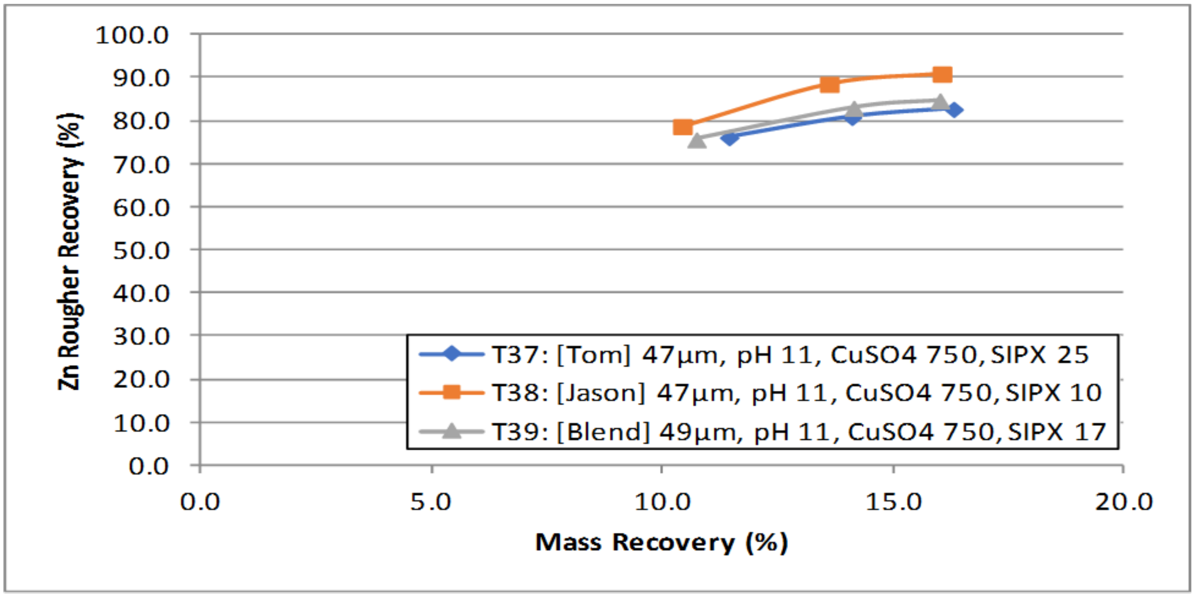
Initial rougher flotation tests were completed on all three composites to assess Pb and Zn recoveries at two minute intervals. The results are presented in Figure 13-27 and Figure 13-28.

Figure 13-27: Pb Rougher Kinetic Testing for the Global Composites



Source: Base Met (2018)

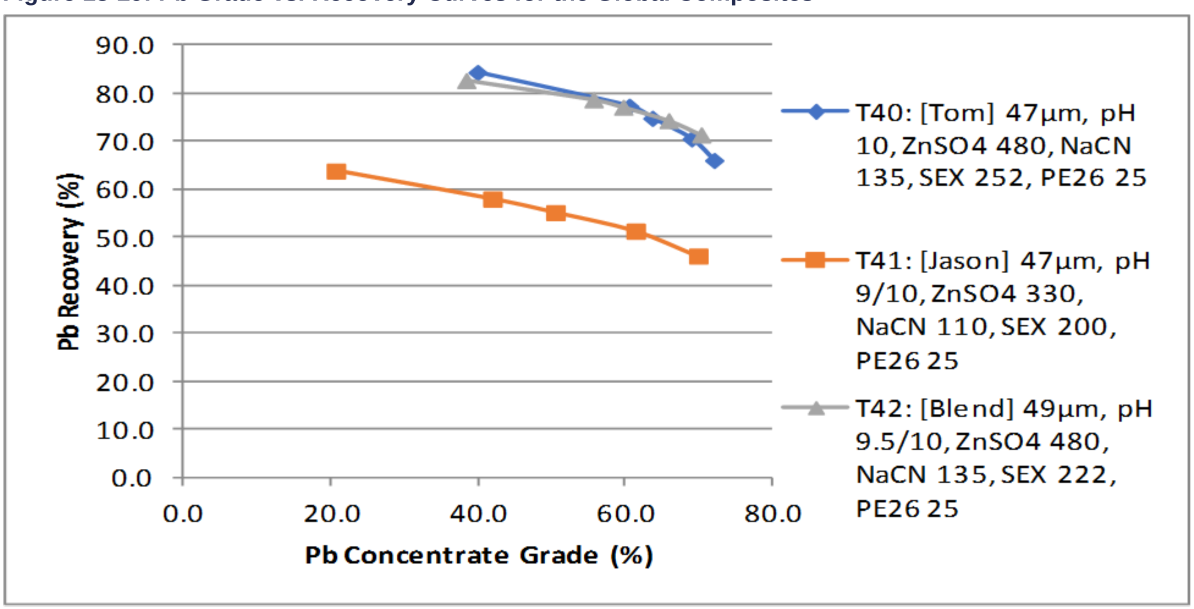
Figure 13-28: Zn Rougher Kinetic Testing for the Global Composites



Source: Base Met (2018)

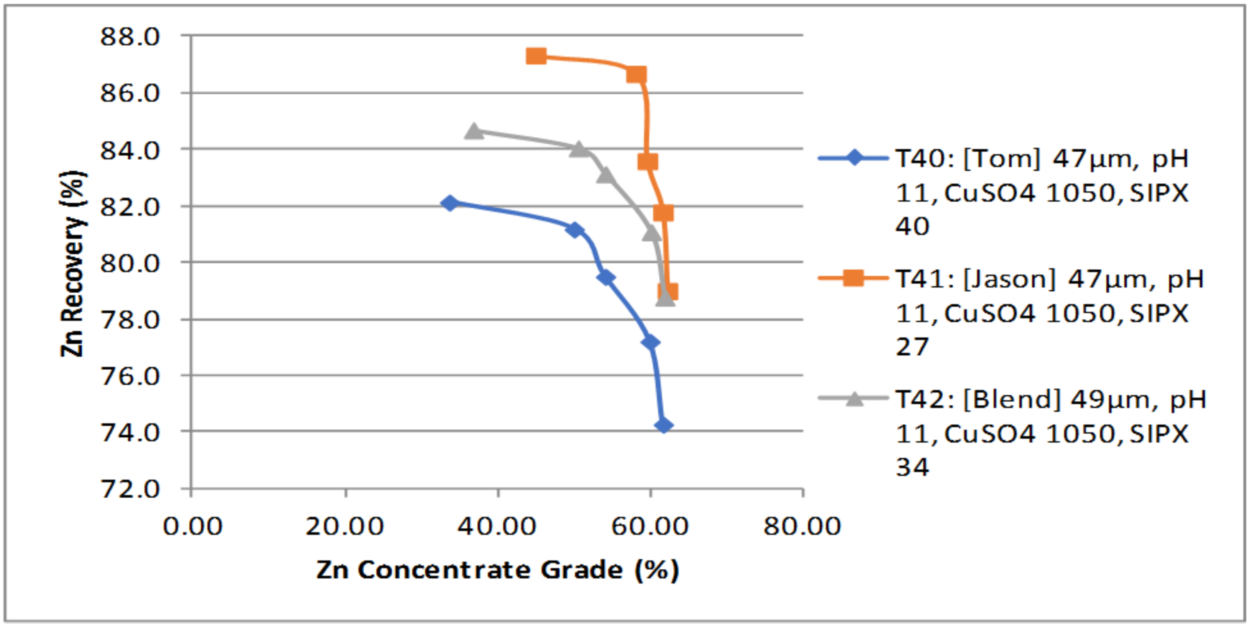
Cleaner flotation tests were performed on all three composites to determine if high concentrate grades could be achieved at high overall Pb and Zn recoveries. Grade vs. recovery curves for each composite are shown in Figure 13-29 and Figure 13-30. Test 41 results in Figure 13-29 for the Jason composite show much lower overall recovery, and this may be attributed to the low lead head grade. Variability testwork is required to determine the relationship between lead recovery and head grade.

Figure 13-29: Pb Grade vs. Recovery Curves for the Global Composites



Source: Base Met (2018)

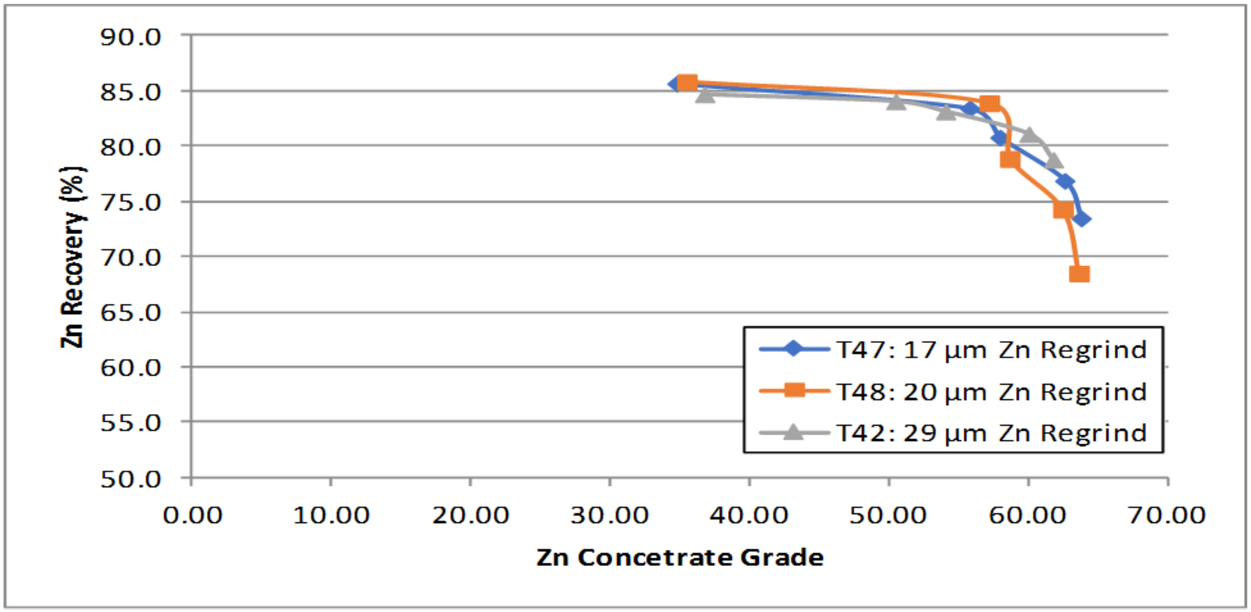
Figure 13-30: Zn Grade vs. Recovery Curves for the Global Composites



Source: Base Met (2018)

Two additional cleaner tests were completed on the Blend Global Composite to determine how Zn regrind size affected Zn concentrate grade. The Zn grade vs. recovery curves are show in Figure 13-31. A Zn regrind size of 17 µm produced slightly better results compared to the tests with coarser regrind sizes (T42, T48) with a recovery of 73.5% Zn and concentrate grade of 63.8% Zn. Since there was no significant improvement in the grade vs. recovery curve, the lower Zn regrind size does not warrant the additional regrinding cost. Based on the results from T42, T47 and T48 the regrind target chosen for design was 25 µm.

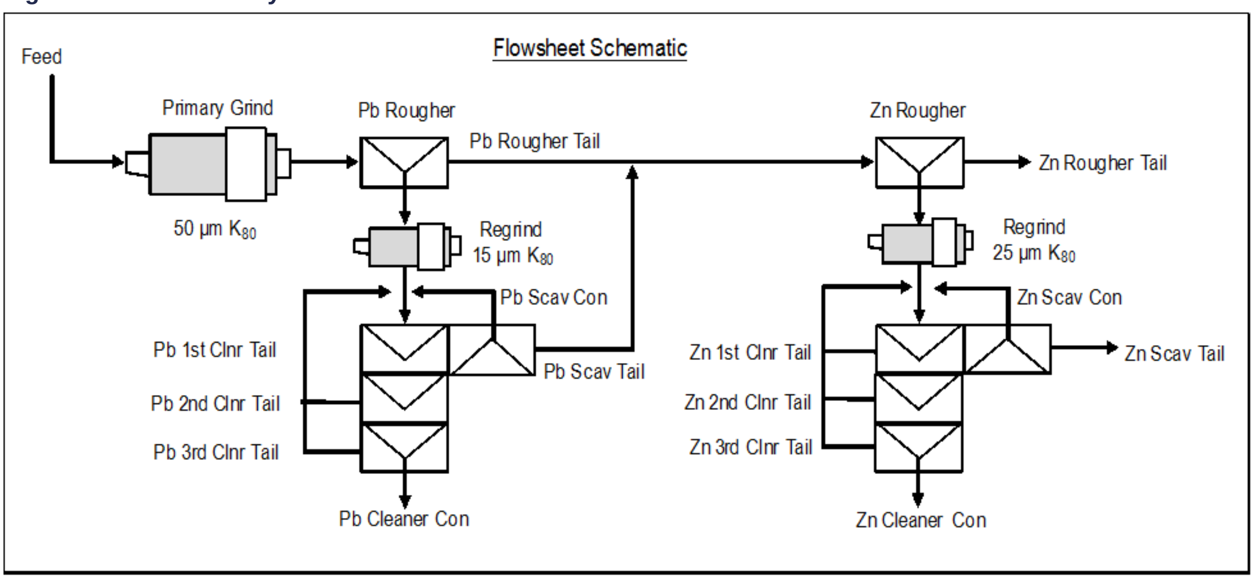
Figure 13-31: The Effect of Zn Regrind Size on Zn Concentrate Grade vs. Recovery Curves



Source: Base Met (2018)

Locked cycle flotation testing was initially carried out on the three global samples. Composite 1 was subsequently tested to include a sample at a lower Pb feed grade. An example of the test flowsheet used by Basemet is presented in Figure 13-32.

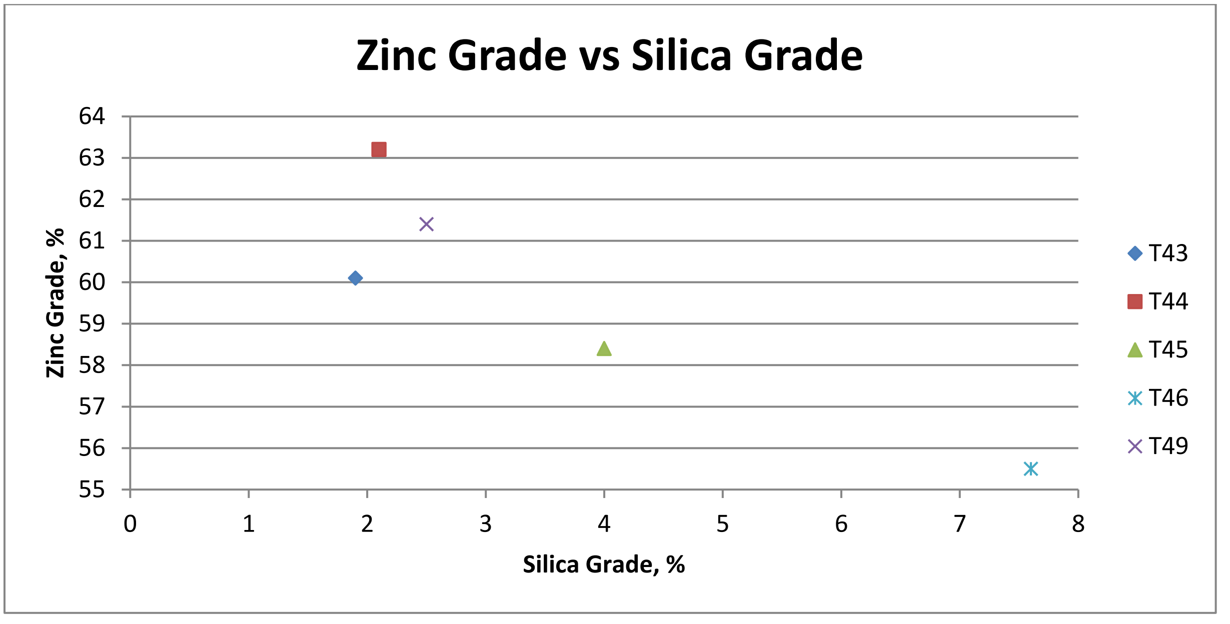
Figure 13-32: Locked Cycle Test Flowsheet



Source: Base Met (2018)

To investigate how silica content in the Zn concentrate is affected by Zn regrind size, an additional locked cycle test (T49) was carried out on the Blend Global Composite at a slightly lower Zn regrind size of 20 µm. Although the silica content was lower further optimization at different regrind sizes is required to determine what correlations exist. The results did show a relationship developing between zinc grade and silica grade in the final concentrate. The trend is shown below in Figure 13-33. The results indicate that a target grade of 58% and above has silica content below the levels that would results in smelter penalties.

Figure 13-33: Locked Cycle Testing – Zinc and Silica Grades in the Final Concentrate



Source: Base Met (2018)

The results from all locked cycle tests are summarized in Table 13-11.

Table 13-11: Locked Cycle Testing Results

Composite ID	Test Number	Pb Flotation		Zn Flotation	
		Concentrate Grade (%)	Recovery (%)	Concentrate Grade (%)	Recovery (%)
Tom Composite	43	69.1	74.4	60.1	85.5
Jason Composite	44	69.9	55.7	63.2	88.4
Blend Global Composite	45	61.5	75.4	58.4	88.9
Blend Global Composite	49	69.1	77.5	61.4	84.1
Composite 1	46	67.4	59.8	55.5	91.0

Source: Base Met (2018)

Reagent dosages were higher than some typical Pb-Zn ore benchmarks, possibly due to the carbon absorbing a portion of the reagents. Overall the reagents dosages were still generally towards the higher end but not outside the range of industry norms. There may be room for additional reagent and grind size optimization to reduce operating cost while achieving high metal recoveries.

13.8 Base Met (2018) Settling Results

Flocculant screening tests and settling rate tests were conducted on the flotation tailings from locked cycle testing. Magnafloc 1011, a high molecular weight anionic flocculent, was chosen for testing, providing the fastest settling rate with very clear overflow water. A summary of the results is show in Table 13-12.

Table 13-12: Settling Test Results

Composite	Sample	Flocculant (g/t)	Free Settling Rate (mm/min)	Final Density (% Solids)
Tom Composite	Final Tailings	0	28	67.7
		10	178	60.8
		20	389	57.3
		30	466	53.3
Jason Composite	Final Tailings	0	23	66.0
		10	128	58.3
		20	276	52.2
		30	331	51.3
Blend Global Composite	Final Tailings	0	29	67.9
		10	264	58.6
		20	467	55.4
		30	593	55.4
Composite 1	Final Tailings	0	32	73.1
		10	233	63.8
		20	357	58.8
		30	977	56.6

Source: Base Met (2018)

The samples had very fast settling rates with very clear overflow, ranging from 128 to 977 mm/min at 10 to 30 g/t of flocculant. Final densities of the thickened solids ranged from about 51% to 64% solids for tests with flocculant added, with decreasing density at higher flocculant dosages.

13.9 Base Met (2018) Concentrate Quality Results

The Pb and Zn concentrates from the locked cycle tests, as well as concentrates from the Composites 1 through 5 cleaner tests, were analyzed for minor elements. The results are summarized in Table 13-13 and Table 13-14. Impurity levels were generally low, except for silica. Elevated silica levels can affect some lead and zinc smelters, with a threshold of 5% causing concern. It should be noted that silica content was measured using sodium peroxide fusion - ICP analysis. Silica content in the Pb and Zn concentrates were well below the 5% threshold for all samples except Composite 1.

Table 13-13: Minor Element Analysis of Pb Cleaner Concentrate

Sample	As (ppm)	Bi (ppm)	C (%)	Cd (ppm)	Co (ppm)	Cr (ppm)	Cu (ppm)	Hg (ppm)	MgO (%)	Mn (ppm)	Mo (ppm)	Ni (ppm)	Sb (ppm)	Se (ppm)	SiO_2 (%)	V (ppm)
Composite 1	135	<2	3.01	93.8	12	120	232	1.0	<0.1	52	29	77	121	130	5.4	170
Composite 2	87	<2	1.31	558	16	20	92	60	<0.1	35	8	17	870	269	1.2	22
Composite 3A/3B	148	<2	2.10	460	24	48	234	44	<0.1	58	16	46	1,120	62	2.3	62
Composite 4	100	<2	2.60	94.8	22	40	408	11	<0.1	53	20	47	114	325	4.8	105
Composite 5	193	3	2.12	1,070	23	67	93	11	<0.1	33	22	50	285	64	3.8	97
Tom Comp	125	<2	1.88	444	18	57	199	43	<0.1	50	13	41	788	182	1.3	43
Jason Comp	150	<2	2.46	540	22	49	200	9	<0.1	58	23	50	155	151	2.3	51
Blended Global Comp	152	<2	2.40	488	24	93	195	40	<0.1	69	21	71	642	46	2.4	58

Source: Base Met (2018)

Table 13-14: Minor Element Analysis of Zn Cleaner Concentrate

Sample	As (ppm)	Bi (ppm)	C (%)	Cd (ppm)	Co (ppm)	Cr (ppm)	Cu (ppm)	Hg (ppm)	MgO (%)	Mn (ppm)	Mo (ppm)	Ni (ppm)	Sb (ppm)	Se (ppm)	SiO_2 (%)	V (ppm)
Composite 1	43	<2	0.85	989	5	70	936	8	<0.1	212	10	43	11	12	7.6	143
Composite 2	44	<2	0.28	>2000	36	30	721	153	<0.1	135	4	20	39	<5	2.7	31
Composite 3A/3B	36	3	0.42	>2000	30	19	883	439	<0.1	161	4	15	59	5	1.5	34
Composite 4	13	3	0.39	921	6	14	607	117	<0.1	209	3	9	3	6	1.2	27
Composite 5	45	6	0.20	>2000	8	30	460	97	<0.1	159	4	17	14	5	2.1	36
Tom Comp	13	<2	0.43	>2000	21	20	667	183	<0.1	132	3	15	39	<5	1.9	21
Jason Comp	15	<2	0.51	1,410	6	24	535	89	<0.1	158	4	16	8	<5	2.1	18
Blend Global Comp	24	4	0.68	1,870	20	54	712	155	<0.1	168	6	36	32	<5	4.0	29

Source: Base Met (2018)

13.10 Relevant Results

Based on the results from the Base Met (2018) test program, the process flowsheet will include primary crushing followed by a semi-autogenous grinding (SAG) mill / ball mill grinding circuit. The material will be ground to a P_{80} of 50 µm and fed to sequential Pb and Zn flotation. The Pb and Zn regrind circuits will be designed to produce P_{80} grind sizes of 15 µm and 25 µm respectively. Test results indicated that no significant benefit was observed for regrind sizes below these values. Refer to Section 17 for more detail on the process operations.

The grinding circuit will be designed using a Bond ball mill work index of 14.0 kWh/t, and grinding mill consumables will be estimated using a Bond abrasion index of 0.27. A preliminary estimate of Pb and Zn recoveries and concentrate grades are summarized in Table 13-15. These results are from locked cycle testing on the Blend Global Composite Test 45 (weighted average of cycles D & E) and were used for the economic analysis

Table 13-15: Preliminary Recovery Projections for Macmillan Pass Global

Product	Grade			Metal Recoveries (%)		
	Zinc (%)	Lead (%)	Silver (g/t)	Lead	Zinc	Silver
Feed	7.3	3.2	44	100	100	100
Lead Concentrate	8.9	**61.5**	688	**75**	5	59
Zinc Concentrate	**58.4**	2.2	88	7	**89**	22

Source: Base Met (2018)

14 Mineral Resource Estimate

14.1 Introduction

During the period November 2017 to January 2018, CSA Global carried out a mineral resource estimate (MRE) update study for the Tom and Jason deposits at the Macmillan Pass Zinc-Lead-Silver Project. In the opinion of CSA Global, the resource evaluation reported herein is a reasonable representation of the zinc, lead, silver mineral resources at the deposits based on the available information. The updated MRE has an effective date of 10 January 2017 and is reported in accordance with the Canadian Securities Administrators' NI 43-101. The MRE is generated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (CIM Council, 2003).

The MRE for the Tom and Jason deposits has been prepared by L. McGarry, CSA Senior Resource Geologist and a Qualified Person for the reporting of Mineral Resources as defined by NI 43-101. Mr. McGarry is responsible for the geological domaining, block modelling, MRE studies presented in this Report section.

Previous NI 43-101 MREs generated for the deposits in 2007 are described in Section 6.3. The current MRE presented in this Report supersedes all past estimates and benefits from the additional information generated since the 2007 MREs (summarized in Section 14.12.2 '*Comparison With Previous Resource Estimate*') which includes geologic data from an exploration core drilling programs undertaken by Fireweed in 2017 and Hudbay in 2011.

Reported Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part, of a Mineral Resource will be converted into a Mineral Reserve.

14.2 Informing Data and Validation

14.2.1 Deposit Drill Data

CSA Global has relied on the following drillhole data for the deposits provided by Fireweed by way of a digital data export containing Microsoft Excel spreadsheets transferred to CSA Global in December 2017.

14.2.1.1 Tom

Hudbay – 96 surface core drillholes completed between 1952 and 1980 totalling 14,587 m:

- Holes TS001 to TS036 were EX size, holes TS037 to TS062 were BQ size, all others were NQ size;
- Drilling focused on Tom West and Tom East deposits and comprised predominantly angled holes drilled on a north-northwest orientated grid with collar spacings of 60 m to 120 m along strike;
- At Tom West, holes typically intercepted mineralized horizons at an angle of 45° to 90°, with downhole intercepts equal to approximately 100% to 140% of horizon true thickness;
- At Tom East holes typically intercepted mineralized horizons at an angle of 30° to 45°, with downhole intercepts equal to approximately 140% to 200% of horizon true thickness;

- Downhole survey data derived from acid dip tests is available for 73 holes, from Sperry Sun tests for five holes and from Tropari tests for 18 holes. Location errors for Hudbay holes without azimuth data may be ±10 m per 100 m of vertical depth. For Hudbay holes that intercept Tom West at 200 m below surface, pierce point location errors of up to 20 m might be expected;
- Lithology data is available for all drillholes. Recovery data is available for five drillholes; and
- 1,131 samples were collected from 63 drillholes representing a combined length of 1326.52 m.

Hudbay – 80 underground core drillholes completed between 1970 and 1982 totaling 5,235 m:

- Holes TU001 to TU64 were AX size, holes TU064 to TU71 were BQ size, TU072 to TU077 were AQ size, all others were NQ size;
- Drilling comprised predominantly short horizontal holes drilled perpendicular to the strike of the deposit at 30 m collar spacings from adits at Tom West and Tom East. Longer angled holes were drilled from exploratory drifts;
- Horizontal underground holes typically intercepted mineralized horizons at an angle of 65°, with downhole intercepts equal to 110% of horizon true thickness. Angled underground holes had comparable interception angles to surface drillholes;
- Negatively angled holes have downhole survey data derived from acid dip tests for 14 holes and from Tropari tools for four holes. Horizontal holes do not have survey data;
- Lithology data is available for all drillholes. Recovery data is available for 20 drillholes; and
- 1,894 samples were collected from 61 drillholes representing a combined length of 902.77 m.

Cominco – 23 surface core drillholes completed between 1988 and 1981 totalling 11,952 m:

- Drilling comprised widely spaced angled HQ and NQ holes that targeted the southeast of Tom West and the Tom Southeast deposit areas and typically intercepted mineralized horizons at an angle of 60°;
- Downhole survey data derived from a Sperry Sun device is available for 19 holes;
- Lithology data is available for all drillholes. Recovery data is available for two drillholes; and
- 307 samples were collected from 14 drillholes representing a combined length of 436 m.

Hudbay – 11 HQ size surface core drillholes completed in 2011 totalling 1,823 m:

- Drilling infilled between earlier Hudbay holes at Tom West, with hole orientations and horizon interception angles comparable to earlier drilling;
- Downhole survey data derived from a Reflex multi-shot device is available for all holes;
- Lithology data is available for all holes and includes: major and minor rock types, mineralization, alteration and structural data. RQD data is available for all holes; and
- 706 samples were collected from 11 drillholes representing a combined length of 649 m.

Fireweed – seven HQ size surface core drillholes completed in 2017 totalling 1,823 m:

- Drilling comprised six angled holes at Tom West and one at Tom East that infilled between, and stepped out from, earlier Hudbay holes. Hole orientations and horizon interception angles were comparable to earlier drilling;
- Down hole survey data, derived from a multi-shot Reflex device is available for all holes;
- Lithology data is available for all holes and includes: major and minor rock types, mineralization, alteration and structural. Recovery and RQD data is available for all holes; and
- 531 samples were collected from seven drillholes representing a combined length of 467 m.

14.2.1.2 Jason

Ogilvy Joint Venture – 56 surface core drillholes completed between 1975 and 1979 totalling 9,279 m:

- Holes were of unknown core size;
- Drilling comprised angled holes drilled on a west-northwest orientated grid at 80 m to 160 m collar spacings and was focused on the Jason Main Zone and shallow portions of the Jason South deposits;
- Holes were orientated perpendicular to the strike of the deposits and typically intercept mineralized horizons at an angle of 20° to 40, with downhole intercepts equal to approximately 155% to 300% of horizon true thickness;
- Downhole survey data derived from acid dip tests is available for eight holes and from an unknown single shot device for 30 holes;
- Lithology data is available for 54 drillholes. Recovery data is available for two drillholes; and
- 933 samples were collected from 37 drillholes representing a combined length of 1,229 m.

Pan Ocean – 48 surface core drillholes completed between 1980 and 1981 totalling 20,225 m:

- Holes were of unknown core size;
- Drilling comprised angled and wedged holes drilled at variable spacings predominantly focused on the deeper portions of the Jason South deposit;
- Holes were orientated at an acute angle to the strike of the deposit and typically intercepted mineralized horizons at an angle of 40° to 90°, with down hole intercepts equal to approximately 100% to 160% of horizon true thickness;
- Downhole survey data from either gyroscope, single or multi-shot devices are available for all holes;
- Lithology data is available for 46 drillholes. Recovery data is available for five drillholes; and
- 1,672 samples were collected from 38 drillholes representing a combined length of 2,094 m.

Aberford – five surface core drillholes completed in 1982 totalling 3,198 m:

- Holes were of unknown core size;
- Drilling infilled between earlier Pan Ocean holes and Ogilvy holes at Jason South, with hole orientations and horizon interception angles that are comparable to Pan Ocean drilling;

- Downhole survey data from either gyroscope, single or multi-shot devices are available for all holes;
- Lithology data is available for all holes. Recovery data is available for two drillholes; and
- 234 samples were collected from five drillholes representing a combined length of 264 m.

Phelps Dodge – 20 surface core drillholes completed in 1991 totalling 5,221 m:

- Drilling comprised angled exploration holes drilled at variable spacing throughout the property. Holes did not intercept Jason Main Zone or Jason South mineralized horizons.

Fireweed – seven HQ3 size diamond drillholes completed by Fireweed in 2011 totalling 1,823 m:

- Drilling comprised angled holes that infilled between earlier holes at Jason Main Zone. Hole orientations and horizon interception angles were comparable to earlier drilling;
- Downhole survey data, derived from a multi-shot reflex device, is available for all drillholes;
- Lithology data is available for all holes and includes: major and minor rock types, mineralization, alteration and structural. Recovery and RQD data is available for all holes; and
- 531 samples were collected from seven drillholes representing a combined length of 466.55 m.

The current grid system used is NAD83 UTM Zone 9N. Drillhole azimuths are recorded in True North. Measurements are in metric units.

All drill data was imported into Micromine software and interrogated via Micromine validation functions prior to constructing a drillhole database for the deposit. The resulting database contains all available drilling and sampling data for the project. Key fields within these critical drillhole database data files are validated for potential numeric and alpha-numeric errors. Data validation cross-referencing collar, survey, assay and geology files was performed to confirm drillhole depths, inconsistent or missing sample/logging intervals and survey data. The data was validated – checked for logical or transcription errors such as overlapping intervals.

CSA Global has reviewed sample collection methodologies adopted by Fireweed and previous operators and is satisfied that they are of a standard that allow the estimation of resources under CIM guidelines and that mineral resource databases for the Tom and Jason deposits fairly represent the primary information available to Fireweed.

14.2.2 Historical Collar Location Maps

Drillhole plan maps generated by previous operators were georeferenced and cross-checked against collar traces plotted from database records. At Tom, historical hole TS88-004W is recorded as having a downhole depth of 641 m, which is the same as TS88-004. Yet lithology logs only extend to a depth of 372.80 m and a surface plan map generated by Hudbay in 1993 shows TS88-004W with a much shorter drillhole trace. On the balance of probabilities, it is more likely that this hole does not intercept the mineralized horizon. The un-sampled interval in this hole is ignored.

14.2.3 Topography

At Jason a digital elevation model (DEM) with a lateral resolution of 5 m and vertical resolution ±1 m was provided by Fireweed. The DEM was generated by an airborne LiDAR survey completed in 2017. At Tom a DEM was generated from contour strings spaced at 10 m intervals, provided by Fireweed in an ArcGIS Shape file.

14.2.4 Model of Historical Workings

A three-dimensional model of underground workings at the Tom project in DXF format was provided by Fireweed. The model represents exploratory underground development and aligns with projected mineralized intervals throughout the Tom West and East deposits and aligns with underground drillhole collar positions. The digital model of underground workings supports the drillhole database information. The position of historical mine workings with interpreted mineralization domains is shown in Figure 14-4.

14.3 Geological Interpretation

14.3.1 Preliminary Statistical Assessment

Descriptive statistical analysis of assay data was undertaken for the identification of assay populations that may represent separate styles of mineralization. Specifically, this analysis was undertaken to estimate the natural cut-off grades that define mineralized units and to determine the distribution parameters for zinc, lead and silver.

At the Tom deposit:

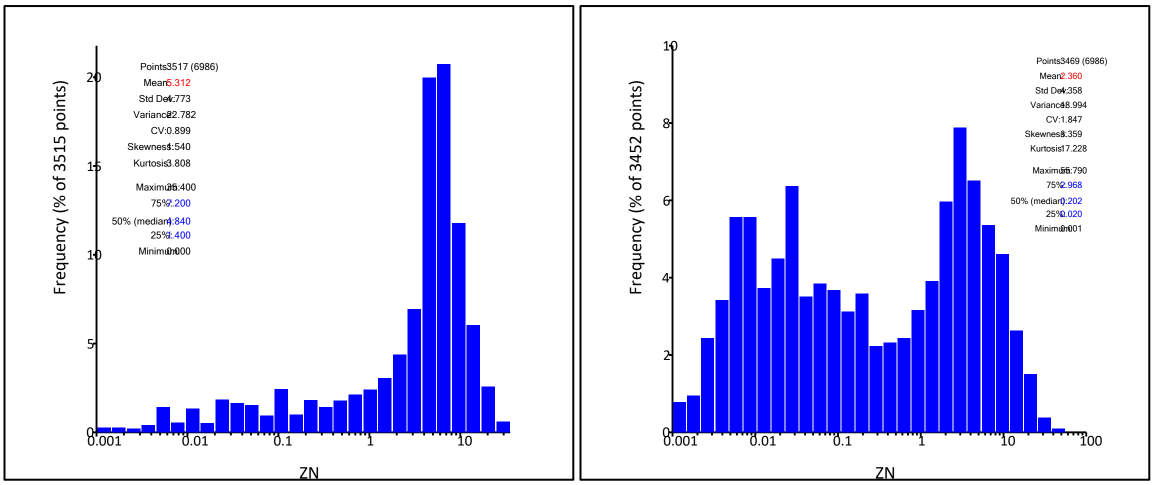
- Zinc grades (Figure 14-1) have a negatively skewed distribution. Above a grade of approximately 2% Zn, there is a well-defined higher-grade population representing Tom West and East mineralized horizons and a tail of low zinc grades representing wall rock and interstitial waste samples;
- Lead grades (Figure 14-2) at Tom show multiple overlapping populations, low grades are associated with wall rock and interstitial waste samples, and higher grade associated with Tom East and the core of Tom West; and
- Silver grade populations are not well defined (Figure 14-3); a peak at 1 g/t Ag is associated with the lower limit of detection.

At the Jason deposit:

- Zinc grades are bimodal, with apparently overlapping populations. There is a well-defined higher-grade population above 1% Zn. Relative to samples at Tom, the larger lower-grade population at Jason results from more wall rock samples and sampling of low-grade mineralization zones at Jason South;
- Lead grades show a similar distribution to zinc, with the addition of a high-grade lead population above a grade of 10% Pb associated with the “massive pyrite” facies encountered at Jason South; and
- Silver grades show a positively skewed distribution with a long high-grade tail.

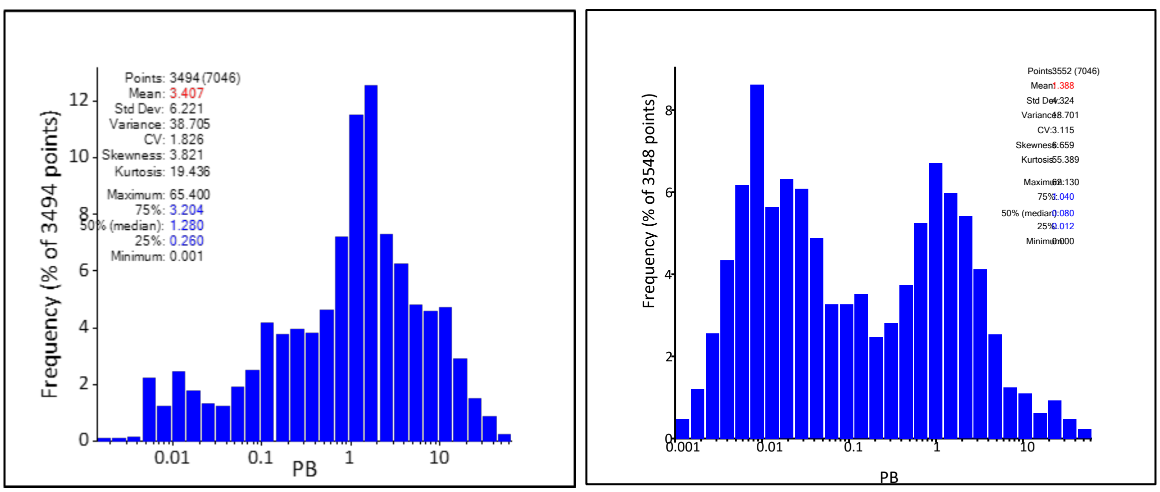
At Tom, assayed zinc, lead and silver grade populations can be related to sequences identified by Goodfellow, 1991 and discussed in Section 7, specifically: vent facies (15–30% Pb+Zn, Ag between 150 g/t and 200 g/t); pink facies (ranging from 10–30% combined Pb and Zn); gray facies (range 4–5% Pb+Zn); and black facies (4–10% Pb+Zn). At both deposits, a population boundary between mineralized and wall rock sample populations might be discerned between 0.5% and 1.0% Zn and Pb. At both deposits, it is not possible to determine a geologically representative modelling cut-off grade for zinc, lead and silver, due to the presence of mixed mineralization facies.

Figure 14-1: Histogram of Raw Zinc Assay Data (% Zn) from the Tom Deposit (L) and Jason Deposit (R)



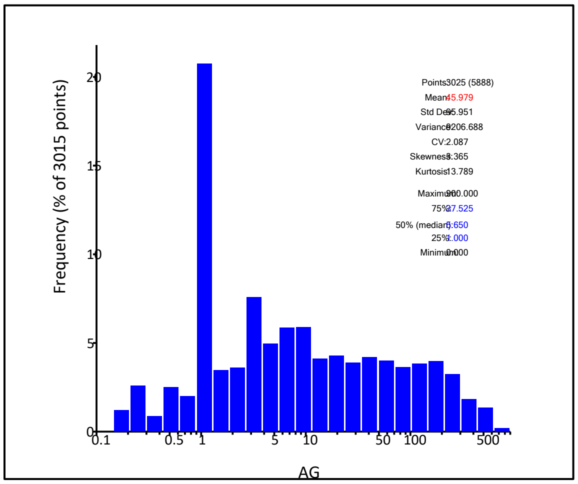
Source: CSA Global (2018)

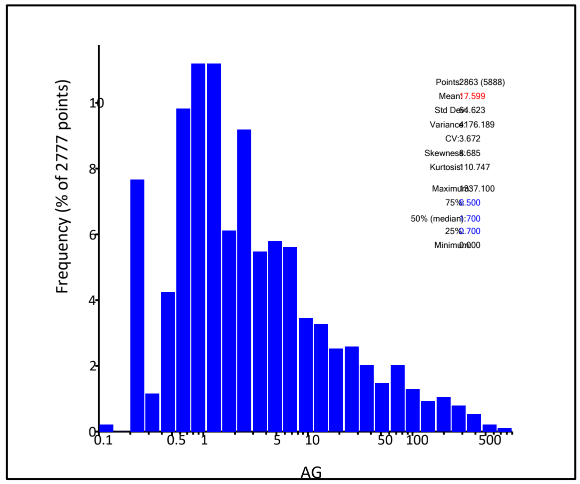
Figure 14-2: Histogram of Raw Lead Assay Data (% Pb) from the Tom Deposit (L) and Jason Deposit (R)



Source: CSA Global (2018)

Figure 14-3: Histogram of Raw Silver Assay Data (g/t Au) from the Tom Deposit (L) and Jason Deposit (R)





Source: CSA Global (2018)

14.3.2 Lithology and Mineralization Modelling

Lithological and structural features were defined from logged and interpreted geology.

At both deposits, cross section views were displayed in Micromine software together with drillhole traces colour-coded according to lithology codes and annotated with zinc, silver and lead grades.

Domains were modelled based on mineralized horizon intervals coded as “Exhalite” or sulphide units in 2011 and 2017 logs, “Ore Zone” in historical logs, or intervals with the lithology qualifier code “mineralized”. At present, it is not possible to model the individual mineralization facies described in Section 7 using logged geology.

Interpretation outlines were generated for the Tom West and Tom East deposits on sections spaced at 30 m and orientated towards 335°. Interpretation outlines were generated for Tom Southeast on sections spaced at 30 m and orientated towards 65°.

At Jason Main Zone and Jason South interpretation outlines were generated on section at 12.5 m to 25 m spacing orientated to 285°.

The following techniques were employed whilst interpreting the mineralization:

- Each cross section was displayed on screen with a clipping window equal to a half distance from the adjacent sections;
- All interpreted strings were snapped to drillhole intervals;
- Internal waste within the mineralized envelopes was not interpreted and modelled. It was included in the interpreted envelopes or split using bifurcation techniques where supported by surrounding drill information;
- If a mineralized envelope did not extend to the adjacent drillhole section, it was projected halfway to the next section, and terminated. The general direction and dip of the envelopes was maintained, although the lens thickness was reduced from the last known intersection;

- Where no drillhole was present down dip, the mineralization was extended up to 150 m down dip; and
- If a mineralized horizon extended to the topography surface, it was extended, at the same width as the last drillhole, above the surface to ensure there would not be any gaps between the horizon and the topography surface when the block model was built.

Interpreted polygons were used to generate three-dimensional solid wireframes for the mineralized envelopes. To reduce the incidence of irregularly shaped (long and thin) and orientated triangles, wireframes are constructed using an equiangular triangulation method. Additional nodes were added to polygons to generate regularly spaced wireframe triangles with edges of 30 m to 60 m.

At Tom, three domains were modelled with variable volumes and drill densities.

Tom West:

- Modelled as a single mineralized horizon ranging in thickness from 10 m to 60 m, with a strike of 340° and a dip of 60°. Logged mineralized horizons typically have a grade greater than 2% Zn. Where shoulder intervals of argillite or mudstone had grades above 2% Zn, polylines were extended to incorporate this material;
- The horizon undulates along strike and down dip;
- Northern model extent is limited by insufficient drilling north of 7,004,230 mN, beyond which the mineralized horizon continues for a further 500 m becoming thinner with lower metal grades;
- Southern model extent at surface is limited to 7,003,800 mN, below which the limit of the model plunges southward at an angle of 65°. This plunging trend is exhibited by logged “sulphide” intervals associated with vent proximal facies and high lead and silver grades above 10% and 200 g/t respectively;
- To the south below a depth of 350 m, the deposit is folded about the Tom anticline. Turning northward the mineralized horizon is intercepted by a small number of deep Cominco holes; and
- The horizon is unconstrained at depth along the entire modelled strike extent.

Tom Southeast:

- Interpreted to be a thinner continuation of the Tom horizon northward from the Tom West domain. The Tom Southeast domain ranges in thickness from 1.5 m to 5 m, with a strike of 15° and a dip of 45°;
- The northern extent of the horizon is limited to 7,003,850 mN, where the horizon become thinner and lower grade and is constrained by drillhole TS082; and
- The southern extent of the Tom Southeast horizon is constrained by barren intervals in drillholes TS90-011 and TS89-005. The horizon is unconstrained at depth.

Tom East:

- Modelled as a single lens shaped sequence of folded and sheared mineralized horizons ranging in thickness from 5 m to 30 m. The lens has a strike of 340°, a dip of 60°, and plunges 55° to the north;

- Logged mineralized horizons typically have a grade greater than 2% Zn, 2% Pb and 10 g/t Ag and show a contrasting character to Tom West; and
- The deposit is open in the up-plunge direction but is constrained by drilling in all other directions.

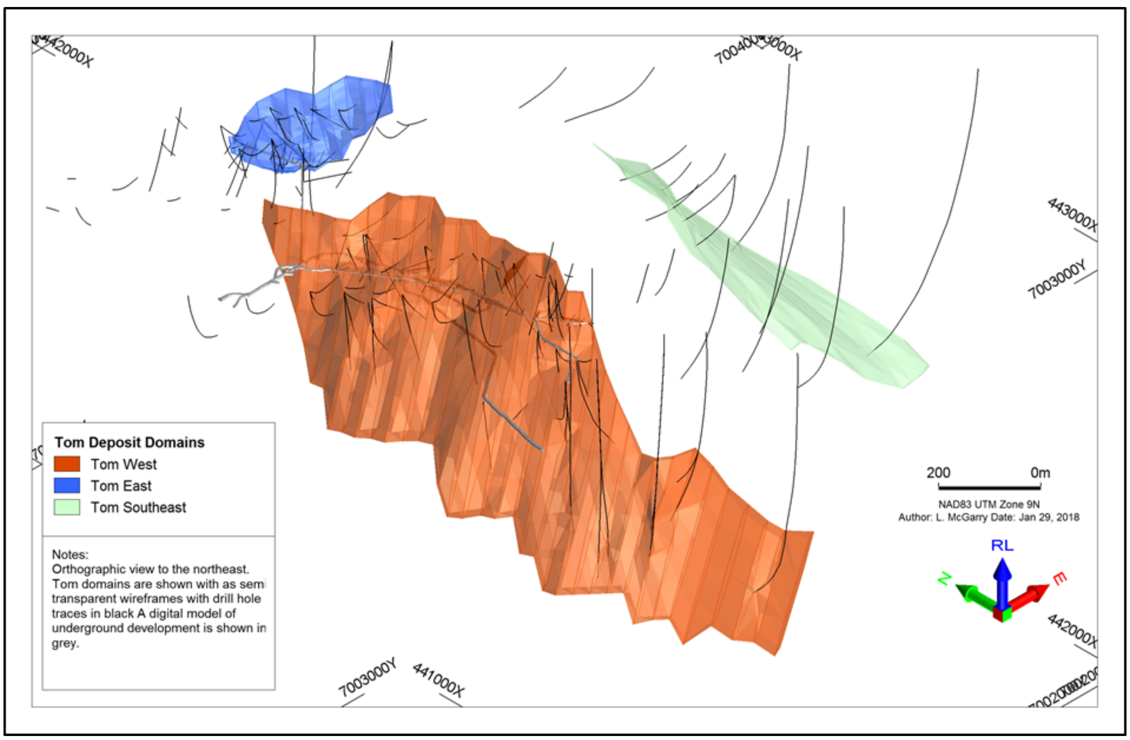
Modelled units at Tom are listed in Table 14-1 with dimensions and approximate drillhole spacing. Figure 14-4 shows Tom modelled domains in a 3D view to the northwest.

Table 14-1: Tom Domain Details

Domain Name	Code	Volume (m^3)	Planar Area (m^2)	Strike Extent (m)	Down Dip Extent (m)	Average Thickness (m)	No. of Drillholes	No. of Samples	Average Drillhole Spacing (m)
Tom									
Tom West	TMZ	11,060,000	460,000	1300	700	24.00	93	2,066	70
Tom East	TEA	810,000	70,000	350	375	11.00	9	45	90
Tom Southeast	TSE	580,000	190,000	300	700	3.00	32	555	75

Source: CSA Global (2018)

Figure 14-4: Tom Wireframe Domains – Orthographic View to the Northeast



Source: CSA Global (2018)

At Jason, three domains were modelled with variable volumes and drill densities.

Jason Main Zone:

- Modelled as a vertical mineralized horizon ranging in thickness from 5 m to 20 m, with a strike of 285°;
- Divided by an offset at 437,000 mE, east of which, the horizon is interpreted to dip southward with lower average zinc grades (<5% Zn);
- The western extent of Jason Main Zone model is limited to 436,000 mE by barren hole JS79-049. Jason horizon is encountered 3 km further to the west-northwest but was not is not modelled in this study; and
- The horizon is unconstrained at depth along the entire modeled strike extent.

Jason South:

- Modelled as two lens shaped horizons ranging in thickness from 5 m at deposit edges to 40 m;
- At depth horizons are gently folded about the Jason syncline. Turning northward, mineralized horizons are interpreted to continue toward the Jason Main Zone on the northern limb of the Jason syncline;
- Horizons are bisected by the north-south trending, steeply dipping Jason Fault, resulting in two in the footwall domains and two hangingwall domains; and
- Within the two footwall horizons approaching the fault, lead and silver grades increase. Possibly associated with a separate style of lead silver-rich mineralization also encountered to the south in holes JS82-086 and JS82-086W1. An area unconstrained by drilling and not included in the resource model.

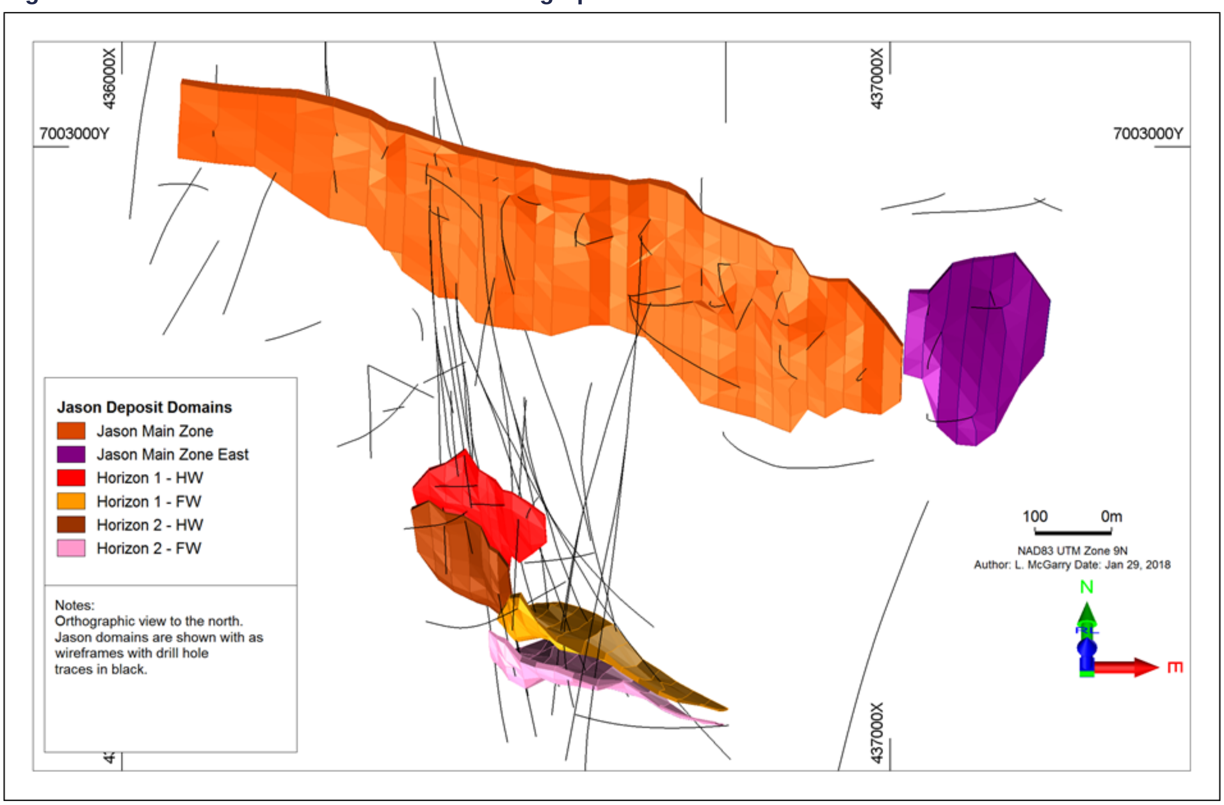
Modelled units at Jason are listed in Table 14-2 with dimensions and approximate drillhole spacing. Figure 14-5 shows Jason modelled domains in a 3D view to the northwest.

Table 14-2: Jason Domain Details

Domain Name	Code	Volume (m^3)	Planar Area (m^2)	Strike Extent (m)	Down Dip Extent (m)	Average Thickness (m)	No. of Drillholes	No. of Samples	Average Drillhole Spacing (m)
Jason									
Jason Main Zone	JMZ (201)	3,660,000	360,000	980	600	10.00	31	570	110
Jason Main Zone East	JMZ (201)	360,000	50,000	175	400	7.00	5	42	100
Jason Sth Hor. 1 HW	H2H (211)	560,000	50,000	125	400	10.50	7	136	85
Jason Sth Hor. 1 FW	H2F (210)	1,170,000	70,000	300	300	18.00	9	273	90
Jason Sth Hor. 2 HW	H1H (213)	650,000	80,000	150	620	7.75	10	251	90
Jason Sth Hor. 2 FW	H1F (212)	1,140,000	80,000	300	475	13.75	11	121	85

Source: CSA Global (2018)

Figure 14-5: Jason Wireframe Domains – Orthographic View to the North



Source: CSA Global (2018)

14.4 Dry Bulk Density

Dry bulk density values were derived for each sample that contained Zn, Pb Ba and values. Bulk Density = (0.0301 x (%Zn + %Pb + %Ba + %Fe)) + 2.4353.

The formula was provided in the "Technical Report on the Tom and Jason Deposits, Yukon Territory, Canada. Prepared for Hudbay Minerals Inc. by RPA Scott Wilson Roscoe Postle Associates" in May 2007.

Additional bulk density determinations, obtained in 2017 and 2011 were compiled by Fireweed and were used to check the RPA regression equation which was found to be appropriate for use in this study.

14.5 Sample Compositing

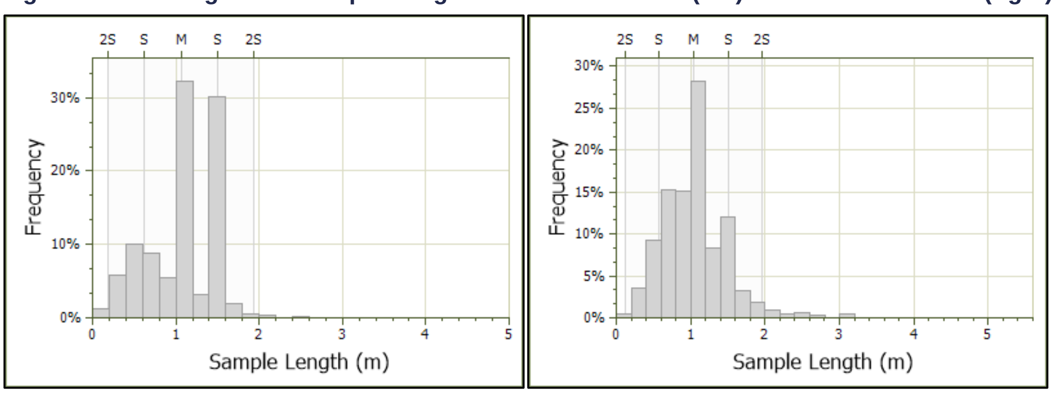
14.5.1 Sample Length Analyses

To generate representative length-weighted composites a sample length analysis was conducted. As shown in Figure 14-6, of the 2,666 assays that fall within Tom wireframe models, 32 % of samples are 1 m in length, and 30% are 1.5 m long. Of the 1,393 assays that fall within Jason wireframe models, 28 % of samples are 1 m in length.

To ensure equal sample support, and to avoid splitting assay intervals, a composite interval length of 1 m was selected. This length is equal to the most common assay interval for domained Tom and Jason samples. Domained assays were regularized using the length-weighted averages of zinc, lead and silver grades and density values.

Composites that were less than 0.3 m in length were discarded to not introduce a short sample bias into the estimation process.

Figure 14-6: Histogram of Sample Lengths on Tom Domains (left) and Jason Domains (right)



Source: CSA Global (2018)

14.5.2 Treatment of Un-sampled Intervals

Within mineralized horizons, historical underground drillholes of Ax core diameter regularly contained 1 m to 10 m intervals of no recovery. Recoveries for recent drilling were much better.

Unrecovered core was perceived as mineralized material by past operators. Historically at Tom, Hudbay assigned an average of preceding and subsequent sample grades to an unrecovered interval within mineralized horizons.

CSA Global elected to treat these intervals as null and exclude them from the composite file rather than assigned a zero grade. This approach is in line with how intervals of no recovery were treated by Hudbay who drilled and logged the holes.

14.6 Statistical Analyses

Before undertaking the resource estimate, univariate statistical assessment of composited assay data was undertaken. Exploration sample data were statistically reviewed, and variograms were calculated to determine spatial continuity for composited sample zinc, lead, silver values and density values.

Statistical analysis was carried out using Snowden Supervisor 8.7 software.

14.6.1 Summary Statistics – Sample Assays

Histograms for the major Tom deposits are presented in Figure 14-7 to Figure 14-9. Statistics for each Tom domain are presented in Table 14-5. The following features are observed:

- The Tom West domain has a mean grade of 6.11% Zn, 2.62% Pb and 30.34 g/t Ag. Zinc and lead both have coefficient of variation values below one ("CV" is the ratio of the standard deviation to the mean). Zinc and lead populations show a tendency to a negatively skewed distribution. Silver grades shows a positively skewed distribution of mixed grade populations and greater variability with a CV of 2.02. A large CV indicates a large spread of values about the mean and that capping of high values samples may be required;
  - The linear correlation between zinc and lead or silver grades is not strong with coefficients of 0.35 and 0.43 respectively. However, silver and lead show a strong correlation with coefficient of 0.90, suggestive of a separate style of mineralization compared to zinc;
  - Zinc, lead and silver grades have a positive correlation with sample easting coordinates; for zinc the coefficient is low at 0.25. For lead and silver, correlation increases to 0.38 and 0.34 representing a stronger spatial control to grades, which are higher in the eastern, shallow and southern edges, of the deposit;
- The Tom East domain has a mean grade of 8.59% Zn, 9.77% Pb and 128.85 g/t Ag. Lead and silver have CV values close to one. Zinc, lead and silver populations show positively skewed distributions;
  - There is no strong linear correlation between zinc and lead or silver grades with coefficients of 0.40 and 0.50 respectively. However, silver and lead show a strong correlation with a correlation coefficient of 0.90;
  - There is no clear spatial control to metal grades within the domain; and
- The Tom Southeast domain has comparable average metal grades to the Tom West deposit. The limited number of samples for this domain prevent a detailed statistical analysis.

Histograms for the major Jason deposits are presented in Figure 14-10 to Figure 14-12. Statistics for each Tom domain are presented in Table 14-6. The following features are observed:

- The Jason Main Zone domain has a mean grade of 6.57% Zn, 1.31% Pb and 2.20 g/t Ag. Zinc and lead populations show a tendency to a negatively skewed logarithmic distribution with CVs of 0.7

and 1.03 respectively. Silver grades show a positively skewed distribution toward lower silver grades with a CV of 2.59;

  - The linear correlation between zinc and lead show a strong correlation with coefficient of 0.60. Silver shows a poor linear correlation with both lead and silver;

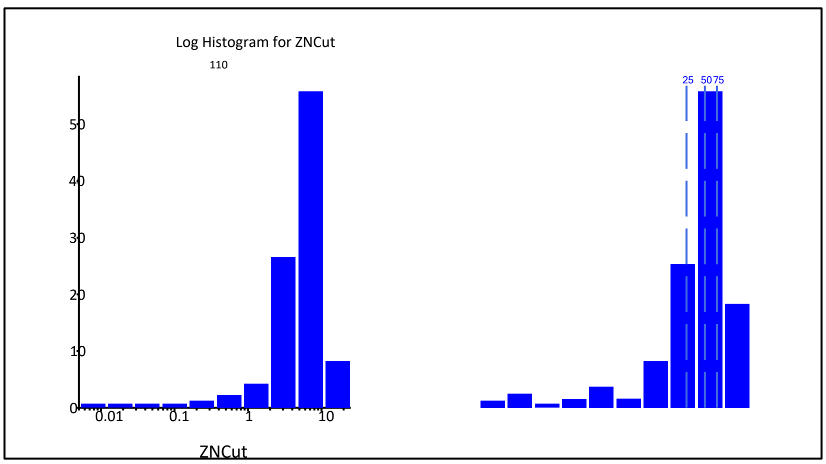
- The Jason South domains have a mean zinc grades that range from of 1.97% Zn to 6.06% Zn, lead grades that range from of 1.68% Pb to 5.71% Pb and silver grades that range from of 13.00 g/t Ag to 99.42 g/t Ag. As shown in Table 14-6, CVs at Jason South are higher than other deposits indicating the presence of mixed grade populations and mineralization styles. Domains in the footwall of the Jason Fault tend to have higher average lead and silver grades; and

  - There is no strong linear correlation between zinc and lead or silver grades with correlation coefficients of 0.21 and 0.28 respectively. However as with other domains, silver and lead show a stronger correlation with a correlation coefficient of 0.83.

14.6.2 Summary Statistics – Densities

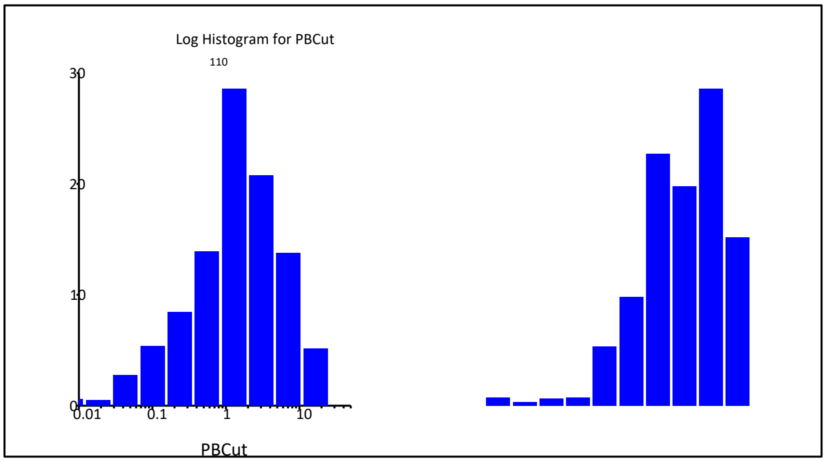
Summary statistics for Tom are presented in Table 14-3 and for Jason in Table 14-4. Density values range from a minimum of 1.37 t/m^3 at Tom West to 5.01 t/m^3 at Jason Main Zone. The application of a lower limit to density values was investigated. After consideration of the occasionally porous nature of rock at the deposits it was decided to retain untreated composite values.

Figure 14-7: Histogram of Capped Composite Zinc Grades for Tom West (left) and Tom East (right)



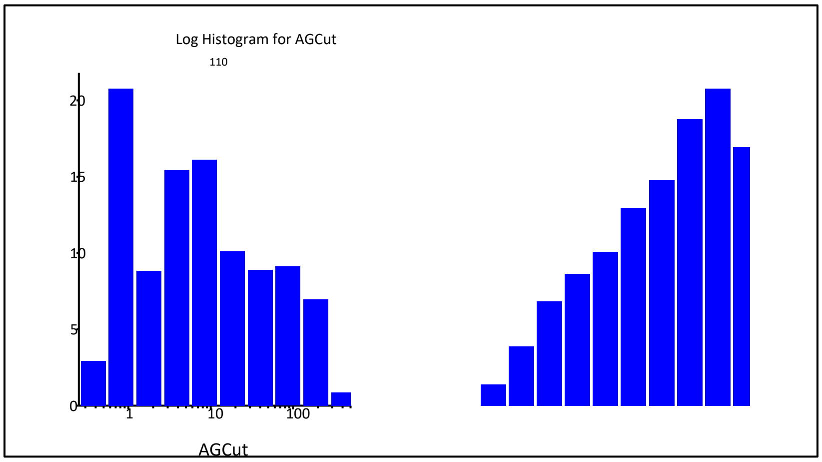
Source: CSA Global (2018)

Figure 14-8: Histogram of Capped Composite Lead Grades for Tom West (left) and Tom East (right)



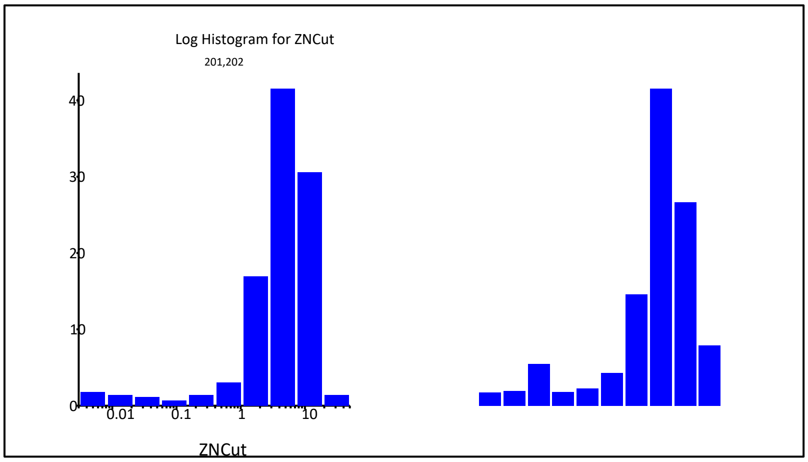
Source: CSA Global (2018)

Figure 14-9: Histogram of Capped Composite Silver Grades for Tom West (left) and Tom East (right)



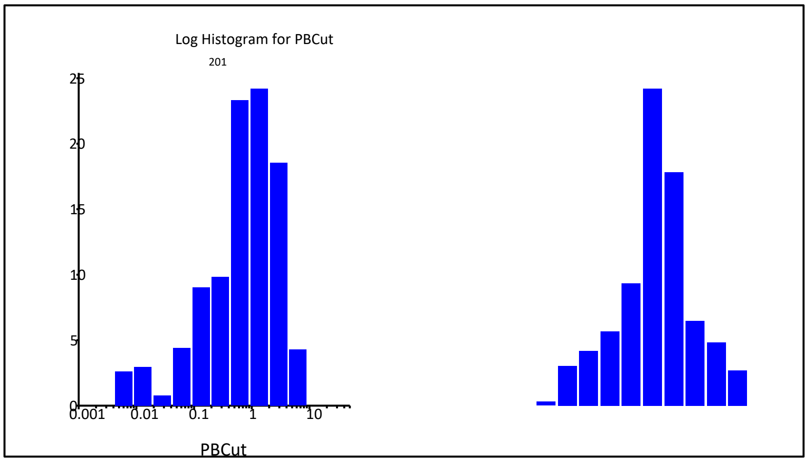
Source: CSA Global (2018)

Figure 14-10: Histogram of Capped Composite Zinc Grades for Jason Main (right) and Jason South (left)



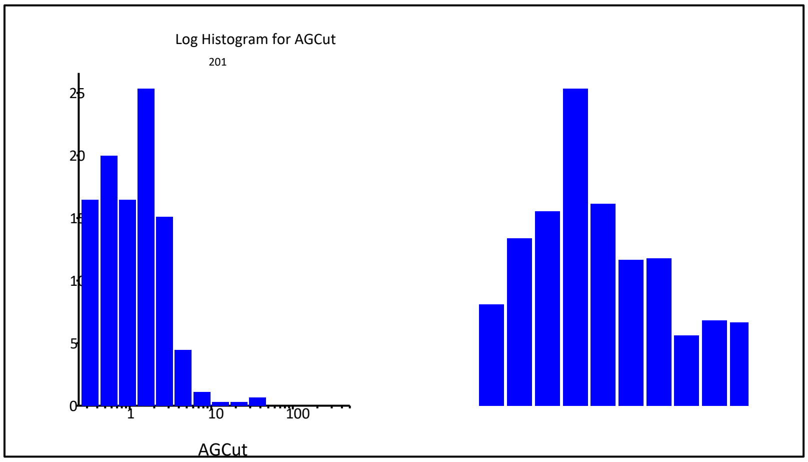
Source: CSA Global (2018)

Figure 14-11: Histogram of Capped Composite Lead Grades for Jason Main (right) and Jason South (left)



Source: CSA Global (2018)

Figure 14-12: Histogram of Capped Composite Silver Grades for Jason Main (right) and Jason South (left)



Source: CSA Global (2018)

Table 14-3: Tom Density Summary

Domain	Count	Minimum	Maximum	Mean	Standard Deviation	Uncut CV
TMZ	1138	1.37	4.73	3.37	0.38	0.11
TEA	239	1.61	4.98	3.13	0.52	0.17
TSE	24	2.59	3.36	2.92	0.21	0.07

Source: CSA Global (2018)

Table 14-4: Jason Density Summary

Domain	Count	Minimum	Maximum	Mean	Standard Deviation	Uncut CV
JMZ	647	1.89	4.65	3.18	0.55	0.17
H1H	96	2.53	4.30	3.29	0.46	0.14
H1F	274	2.61	5.01	3.44	0.51	0.15
H2F	273	2.70	4.51	3.42	0.38	0.11
H2H	140	2.48	3.88	3.22	0.33	0.10

Source: CSA Global (2018)

14.6.3 Grade Capping

A review of high-grade samples was undertaken to ensure that extreme grades were treated appropriately during grade interpolation. Although extreme grade outliers within the grade populations of variables are real, they are potentially not representative of the volume they inform during estimation. If these values are not capped, they have the potential to result in significant grade over-estimation on a local basis.

In general, very high-grades (at the Tom and Jason deposits these are typically greater than 20% Zn and Pb and 500 g/t Ag) are located within the higher-grade portions of the deposit and their influence is well constrained by surrounding samples. Accordingly, a relaxed approach to the application of capping values was taken.

The capping strategy was applied based on the following method:

- Probability plots were reviewed to identify inflection points at the upper end of zinc and lead grade distributions on a domain by domain basis;
- Inflection points were rounded to the nearest 5% zinc and lead interval to identify a capping value; and
- A capping value of 600 g/t Ag was applied to all domains.

In addition, sample data were sorted into descending order and several capping scenarios applied to see what effect the capping value would have on the mean, standard deviation and CV, as well as the loss of metal from the sample population.

At Tom, the capping thresholds presented in Table 14-5 were selected, resulting in the capping of five zinc, two lead and four silver composite assay values prior to estimation.

At Jason, the capping thresholds presented in Table 14-6 were selected, resulting in the capping of nine zinc, seven lead and three silver composite assay values prior to estimation.

Table 14-5: Tom Deposit Composite Summary

Domain	Count	Min.	Max.	Mean	Standard Deviation	Uncut CV	Capping Value	No. Capped	Capped Mean	Capped Standard Deviation	CV
Zn %											
TMZ	2,283	0.02	26.538	6.11	3.41	0.56	25	5	5.95	3.40	0.56
TEA	590	0.008	35.4	8.59	5.79	0.67	30	2	8.58	5.73	0.67
TSE	38	0.037	15.77	6.04	4.33	0.72	-	-			
Pb %											
TMZ	2,252	0.005	42.358	2.62	3.48	1.33	25	3	2.58	3.40	1.30
TEA	596	0.012	58.204	9.77	10.33	1.06	50	2	9.75	10.26	1.05
TSE	38	0.02	15.12	3.06	4.11	1.34	-	-			
Ag g/t											
TMZ	1,830	0.00	718.87	30.34	61.35	2.02	600	1	28.11	60.68	2.00
TEA	592	1.4	670.6	128.85	137.80	1.07	600	3	128.65	137.07	1.07
TSE	38	0.34	187.01	32.70	44.84	1.37	-	-			

Source: CSA Global (2018)

Table 14-6: Jason Deposit Composite Summary

Domain	Count	Min.	Max.	Mean	Standard Deviation	Uncut CV	Capping Value	No. Capped	Capped Mean	Capped Standard Deviation	CV
Zn %											
JMZ	688	0.008	26.284	6.57	4.76	0.72		-			
H1H	106	0.005	37.64	6.06	6.63	1.09	20	5	5.56	4.76	0.86
H1F	274	0.005	55.79	4.88	5.54	1.13	20	4	4.63	4.12	0.89
H2F	263	0.01	8.233	1.97	1.55	0.79		-			
H2H	140	0.235	16.5	3.02	2.61	0.86		-			
Pb %											
JMZ	688	0.008	9.83	1.31	1.36	1.03		-			
H1H	106	0.027	22.85	2.93	4.03	1.38	20	1	2.90	3.91	1.35
H1F	274	0.01	52.18	5.71	9.42	1.65		-			
H2F	273	0.03	46.70	3.39	7.42	2.19	40	2	3.34	7.17	2.15
H2H	140	0.082	9.53	1.68	1.47	0.88		-			
Ag g/t											
JMZ	485	0	66.50	2.20	5.70	2.59		-			
H1H	102	0.7	313.00	33.08	59.74	1.81		-			
H1F	273	0.7	1168.14	99.42	165.46	1.66	600	7	93.28	93.28	1.00
H2F	273	1	360.00	27.80	53.48	1.92		-			
H2H	140	0.7	95.01	13.00	15.61	1.20		-			

Source: CSA Global (2018)

14.7 Geostatistics

The Tom West and Tom East and Jason Main Zone domains have a sufficient number of samples to generate meaningful grade variation models for composited and capped zinc, lead and silver grades.

The Tom Southeast domain and individual Jason South domains did not have sufficient data to allow variography. At Tom Southeast, grade variation models were derived from Tom West variation models aligned to the deposit orientation. At Jason South, sample grade variation models were generated from samples within all Jason South domains.

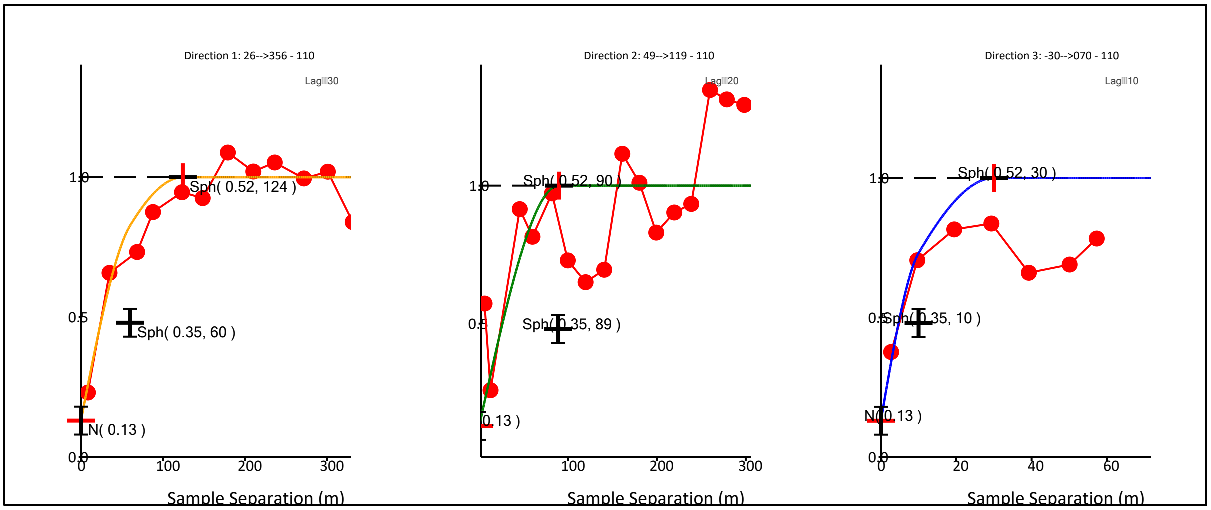
Composite zinc, lead and silver values underwent a normal score transform prior to being assessed for anisotropy, or directional dependence. Maps of zinc, lead and silver value continuity were used to investigate the strike, dip and pitch direction axis of the major mineralization domains.

The grade variation between sample pairs orientated along each direction axis +/-15° was reviewed using semi-variogram charts. Example zinc, lead and silver semi-variogram charts for the Tom West domain are shown in Figure 14-13, Figure 14-14 and Figure 14-15. Sample pairs are grouped by their separation distance, or “lag interval” on the X-axis. For each lag interval assessed, half of average variance value of paired samples is plotted on the Y-axis. The resulting empirical semi-variogram chart can show if there is a relationship that can be modelled between grade variance and distance along each axis. Normal score variograms are back transformed to give the semi-variogram parameters presented in Table 14-7.

For all domains, semi-variogram charts for zinc, lead and silver were modelled using two spherical functions. The semi-variogram models described in Table 14-7 are sufficiently well defined to allow meaningful kriging calculations.

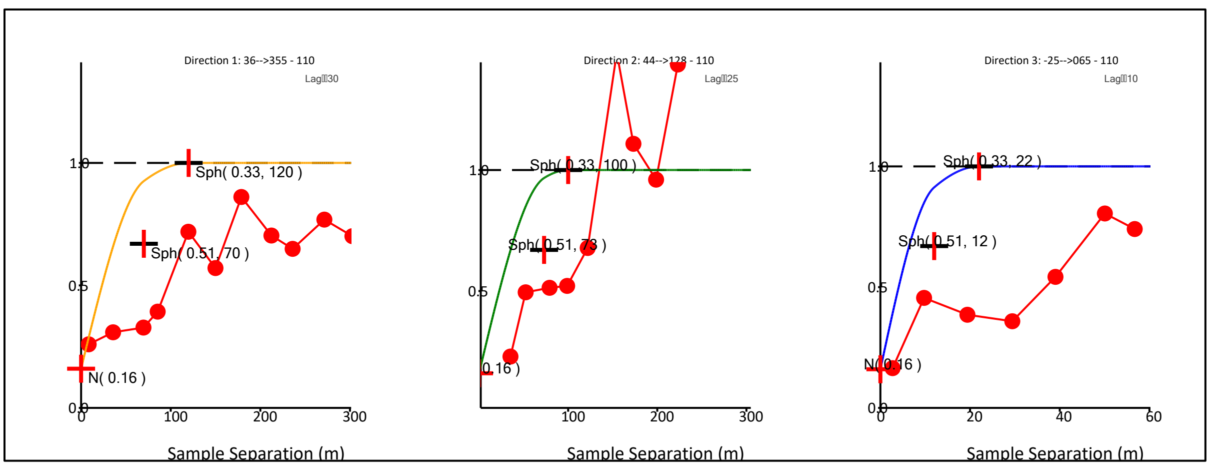
Ellipses were visualized in Micromine and compared with deposit orientations and apparent mineralization trends. Variogram models are used to define the size of the search ellipse during estimation.

Figure 14-13: Example Major and Semi-Major and Minor Axis Variograms for Zinc at Tom West



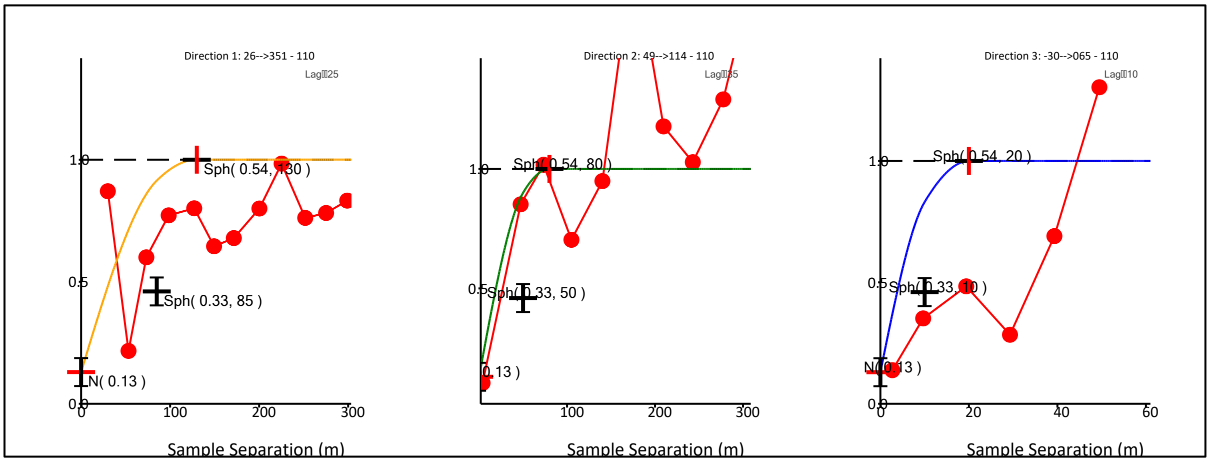
Source: CSA Global (2018)

Figure 14-14: Example Major and Semi-Major and Minor Axis Variograms for Lead at Tom West



Source: CSA Global (2018)

Figure 14-15: Example Major and Semi-Major and Minor Axis Variograms for Silver at Tom West



Source: CSA Global (2018)

Table 14-7: Modelled Semi-Variogram Parameters for Tom Deposit Grade Interpolation

Domain	Element	Ellipse rotation			Nugget Value	Partial Sill	Range (m)		
		z	y	x			Major	Semi-major	Minor
TMZ	Zn	119	-49	139	0.15	0.38	60	85	10
						0.47	120	90	30
	Pb	128	-44	126	0.21	0.56	70	73	12
						0.23	120	100	22
	Ag	114	-49	139	0.2	0.5	85	50	10
						0.31	130	80	20
TEA	Zn	-71	-49	-41	0.14	0.49	71	30	5
						0.37	75	70	10
	Pb	-66	-45	-45	0.16	0.3	80	70	5
						0.54	90	90	15
	Ag	-61	-48	-51	0.15	0.28	50	30	5
						0.56	70	60	15
TSE	Zn, Pb, Ag	-65	-50	0	0.14	0.48	60	60	5
						0.38	80	80	15

Source: CSA Global (2018)

Table 14-8: Modelled Semi-Variogram Parameters for Jason Deposit Grade Interpolation

Domain	Element	Ellipse Rotation			Nugget Value	Partial Sill	Range (m)		
		z	y	x			Major	Semi-major	Minor
JMZ	Zn	105	-50	90	0.14	0.46 0.4	94 110	110 105	10 30
	Pb	105	-50	90	0.15	0.61 0.24	60 110	75 90	10 30
	Ag	105	-50	90	0.28	0.62 0.1	60 110	75 90	10 30
JST	Zn	140	0	65	0.21	0.48 0.31	30 80	7 72	10 20
	Pb	140	0	65	0.22	0.55 0.23	30 80	7 72	10 20
	Ag	140	0	65	0.16	0.61 0.23	30 80	7 72	10 20

Source: CSA Global (2018)

14.8 Block Model

14.8.1 Block Model Construction

Separate block models were constructed for Tom East, Tom West, Tom Southeast, Jason Main Zone and Jason South deposits using Datamine Studio RM software. Block models encompassed the full extent of each deposit area.

Block models were rotated into the plane of each deposit and use a parent cell size of 15 m in the along strike and down dip directions and 5 m in the across strike direction with sub-celling to 3.75 m x 3.75 m x 1.25 m to maintain the resolution of the mineralized horizons.

The along strike and down dip parent cell size was selected based on approximately one third of the average drill section spacing in better drilled areas of each deposit. The model cell dimensions in the across strike directions was selected to provide sufficient resolution to honour grade variation across the mineralized horizons. Block model parameters are presented in Table 14-9.

14.8.2 Assignment of Strike and Dip Orientations Dynamic Anisotropy

Model blocks were coded with an estimated strike and dip value representing the orientation of the mineralized horizon at the location of the block. Strike and dip angles are subsequently used to dynamically adjust search ellipse anisotropy for the estimation of each block. In this way, local undulations and folds are represented in the block model and the banded nature of mineralized horizons are preserved.

True strike and dip orientations are extracted from wireframe model triangles. Orientation angles are not extracted from triangles at wireframe edges that are flat or triangles orientated perpendicular to the deposit. Model blocks are assigned strike and dip angles from the five nearest wireframe triangles using the Inverse

Distance Weighting (IDW) estimation method for angles using an omni-directional search ellipse with a range of 30 m.

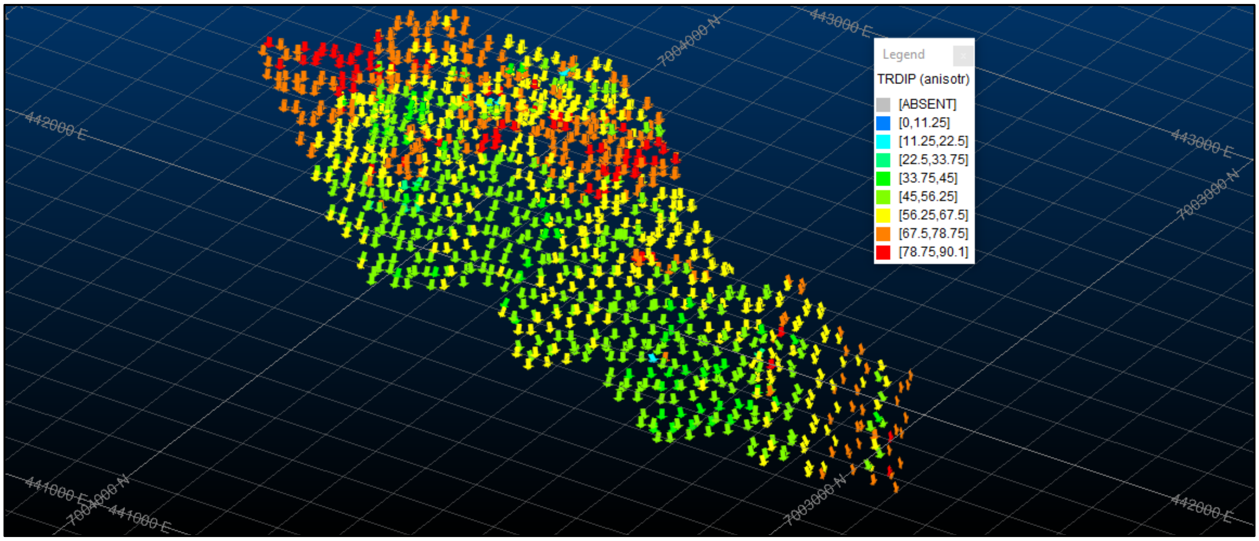
An example of resultant orientation points for the Tom West domain is shown in Figure 14-16.

Table 14-9: Block Model Parameters

Item	X	Y	Z
Model	**Tom West**		
Origin	442167.6	7003284	869.7237
Parent block size	15	15	3
No. of sub-blocks	4	4	4
Rotation around axis	0°	65°	340°
Model	**Tom East**		
Origin	442020.8	7004525	1341.4
Parent block size	15	15	3
No. of sub-blocks	4	4	4
Rotation around axis	0°	70°	150°
Model	**Tom Southeast**		
Origin	442562.7	7003786	1404.9
Parent block size	15	15	3
No. of sub-blocks	4	4	4
Rotation around axis	0°	45°	210°
Model	**Jason Main Zone**		
Origin	437194.2	7002565	682.5
Parent block size	3	15	15
No. of sub-blocks	4	4	4
Rotation around axis	0°	0°	285°
Model	**Jason South**		
Origin	436364.7	7002534	427.9365
Parent block size	15	15	3
No. of sub-blocks	4	4	4
Rotation around axis	0°	65°	120°

Source: CSA Global (2018)

Figure 14-16: Tom West Wireframe Orientation Points Shown as Arrows Aligned to Dip Direction and Colour Coded by Dip



Source: CSA Global (2018)

14.9 Estimation

14.9.1 Grades

Tom and Jason mineralization domain shell contacts are interpreted as hard boundaries for grade interpolation, such that zinc, lead and silver grades in one domain cannot inform blocks in another domain.

Both the Ordinary Kriging (OK) method and IDW techniques are considered appropriate methods for estimating block grades at the Tom and Jason Deposits where mineralization has a locally variable nature. In this scenario, the OK method and the utilization of a local mean within the search neighbourhood is preferred. The OK interpolation utilized the variogram models contained in Table 14-7 for Tom and Table 14-8 for Jason.

For validation purposes, an Inverse Distance Weighted interpolation was undertaken, whereby samples were weighted proportionally to the inverse of their distance from the block raised by a power of two (IDW^2). The IDW^2 used the same search ellipse and sample constraint parameters as the OK interpolation.

For both OK and IDW^2 estimates the search ellipse and detailed in Table 14-10 were used. For the estimation of block grades search ellipses were aligned with the dominant orientation of the mineralization using dynamic anisotropy.

Table 14-10: Estimation Search Ellipse Ranges

Domain	Major (m)	Semi-major (m)	Minor (m)
TMZ – Zn, Pb, Ag	60	60	15
TEA – Zn, Pb, Ag	40	40	15
TSE – Zn, Pb, Ag	60	60	15
JMZ – Zn, Pb, Ag	60	60	15
JST – Zn, Pb, Ag	40	40	20
ALL – BD	60	60	20

Source: CSA Global (2018)

Grades were interpolated in three passes at half, one and two times the variogram ellipse range.

Table 14-11: Estimation Run Parameters

Interpolation Run #	1	2	3
Search Radii	1 × range	2 × range	4 × range
No. of Sectors	1	1	1
Minimum No. of Drillholes	2	2	2
Maximum No. of Samples per Hole	4	4	4
Minimum No. of Samples (Total)	8*[1]	8*[1]	8*[1,2]
Maximum No. of Samples (Total)	12	12	12
Discretization	3×3×3	3×3×3	3×3×3

*[1] Minimum of 6 for the Tom Southeast Domain.
*[2] Minimum of 4 for density estimates.
Source: CSA Global (2018)

Data used to interpolate grade into the Tom and Jason deposit block models contains locally clustered drillhole samples that may unduly influence or bias block grades. To address this issue, a restriction of 12 samples was applied that limits the maximum number of samples used to estimate block grades.

14.9.2 Densities

Densities were estimated from composite intervals using the IDW^2 method. Estimation was undertaken in three runs using a dynamic search ellipse detailed in Table 14-10 for “BD” and using the parameters described in Table 14-11.

It was decided not to calculate densities from estimated zinc, lead and barium grades using a regression equation, since the availability of barium assays is limited in many domains. This would result in poor estimates that may significantly affect the accuracy of density estimates on a per block basis.

14.10 Model Validation

Validation of the grade estimates was completed by:

- Visual checks on screen in cross-section and plan view to ensure that block model grades and assigned densities honour the input sample composites;

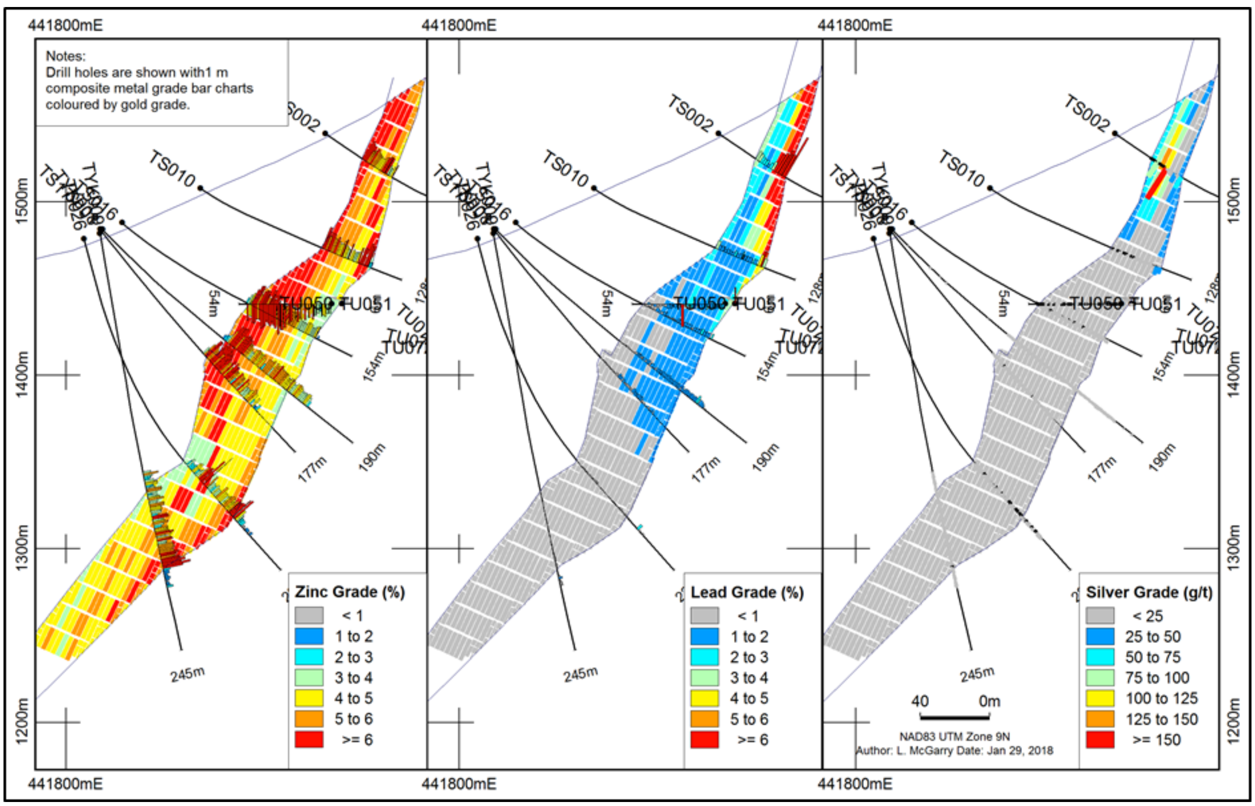
- Comparison of sample and block grades and densities;
- Generation of swath plots to compare input and output grades in a semi-local sense, by easting, northing and elevation; and
- Investigation of the global change in support for metal grades.

14.10.1 Visual Validation

Block grades correlate very well with input sample grades. The distribution and tenor of grades in the composites is well honoured by the block model and is appropriate considering known levels of grade continuity and the variogram. Poorly informed deposit areas with widely spaced samples are more smoothed but expected.

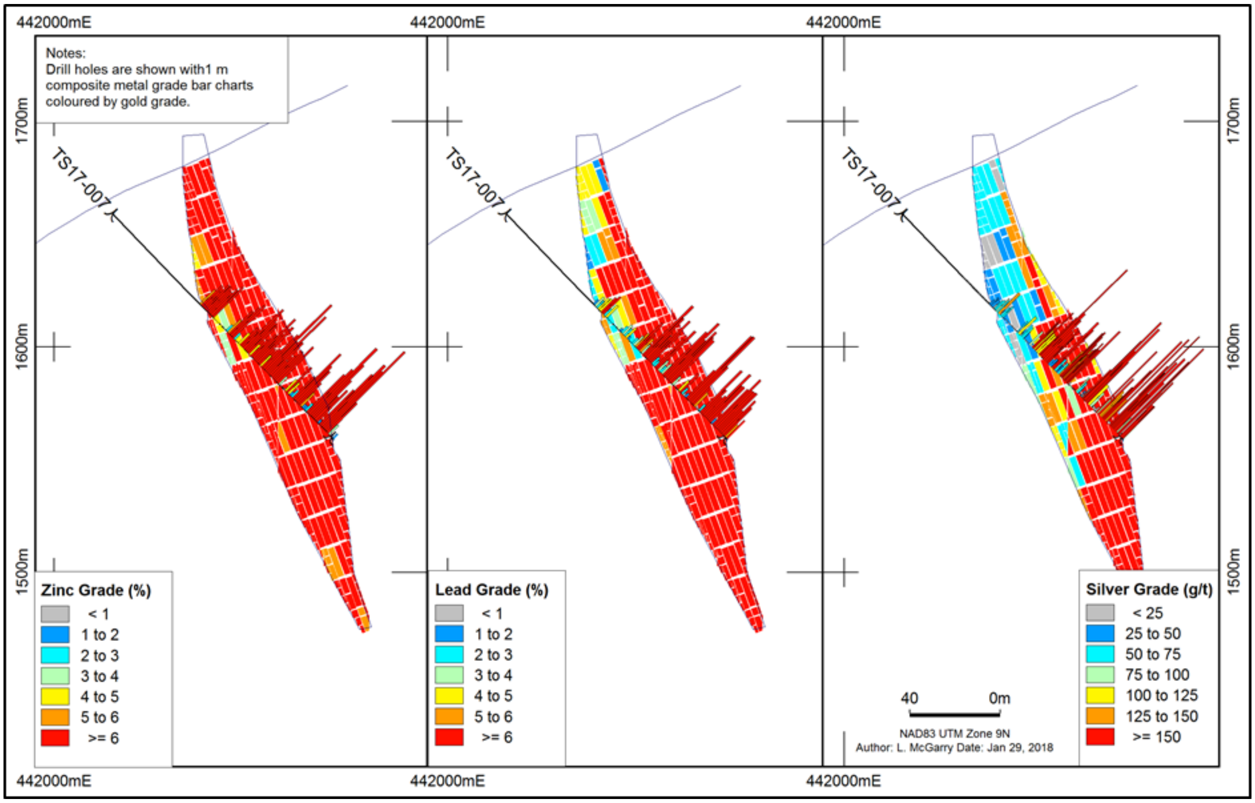
Cross section views of block models coloured by zinc, lead and silver are shown for Tom West in Figure 14-17, Tom East in Figure 14-18, Jason Main Zone in Figure 14-19, and Jason East in Figure 14-20.

Figure 14-17: Section Plots for Zinc, Lead and Silver at Tom West



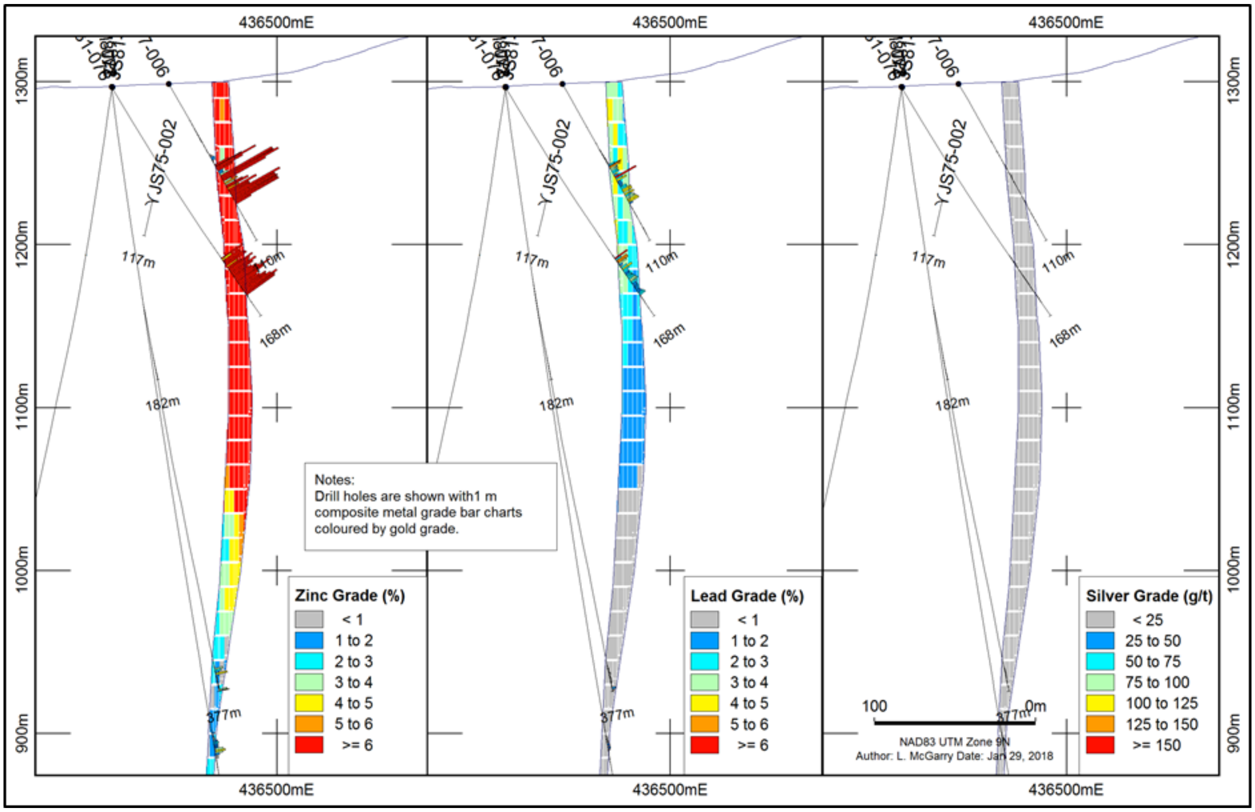
Source: CSA Global (2018)

Figure 14-18: Section Plots for Zinc, Lead and Silver at Tom East



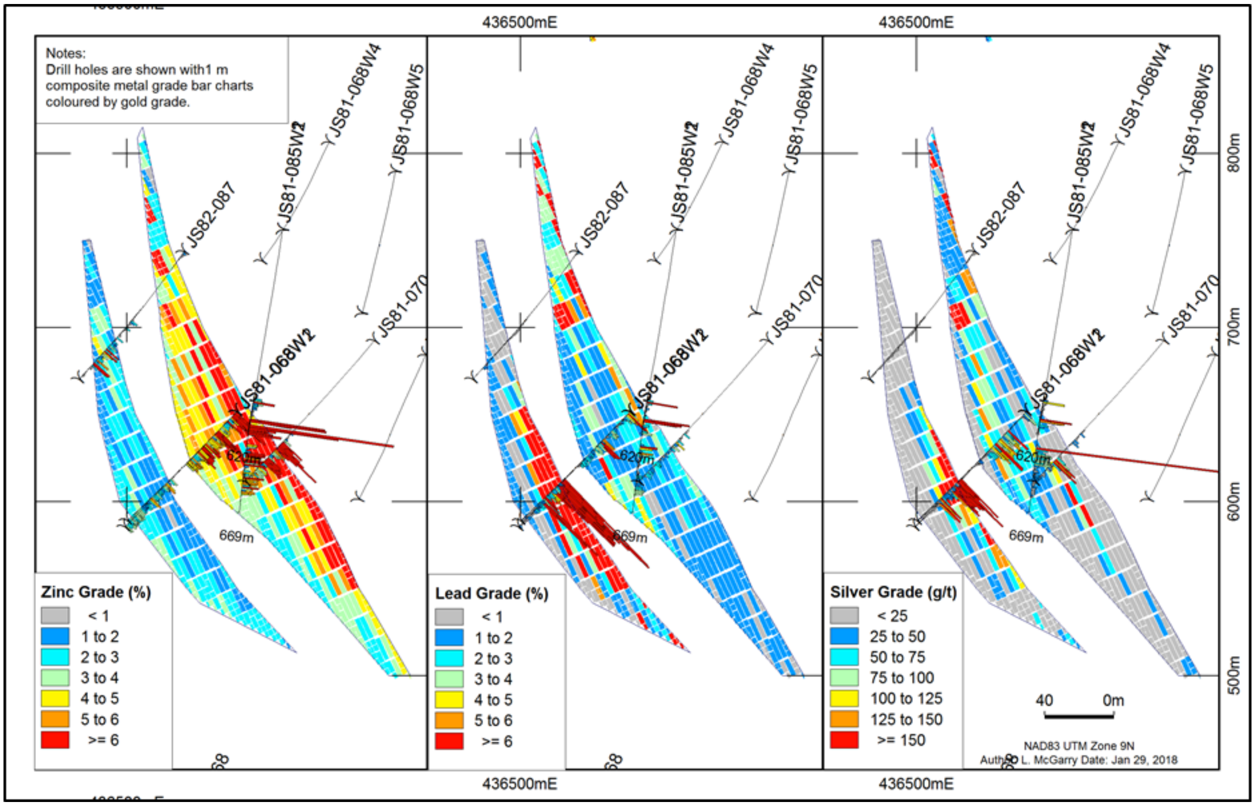
Source: CSA Global (2018)

Figure 14-19: Section Plots for Zinc, Lead and Silver at Jason Main Zone



Source: CSA Global (2018)

Figure 14-20: Section Plots for Zinc, Lead and Silver at Jason East, H1F and H2F Domains



Source: CSA Global (2018)

14.10.2 Comparison of Means

A check was conducted to test that the mean of the input data was close to the block model mean. The check compared the average zinc, lead and silver grades in composite drillhole samples and in model blocks for each resource estimate domain. To account for locally clustered sample data, sample data were de-clustered using the procedure explained by Clayton and Journel, 1998, with de-clustering variable cell sizes deduced on a domain by domain basis.

The test demonstrated that the grades for the de-clustered mean input composites and both the OK and IDW block models are comparable as shown in Table 14-12. Larger differences are seen for domains with greater grade variance, and or fewer samples. The difference between OK and IDW3 estimates are comparable:

- At Tom West, de-clustered composite and modeled zinc grades show good agreement. Average lead and silver model grades are significantly less than input composites. This is due to a greater drill density in the lead and silver rich shallow and southern edges of the deposit, relative to the lower grade but more extensive deeper and northern portions of the deposit;
- At Tom East, there is reasonable agreement between the de-clustered composite and block model zinc, silver and lead grades which are within 10% of each other;

- At Jason Main Zone, there de-clustered composite and modeled zinc and lead grades show good agreement. For silver, average block model grades are significantly higher than the average grade input composites, due to the high variance of silver assays in this domain where a small number of higher grade samples inform proportionally larger number of blocks than low grade samples; and
- At Jason South, there is reasonable agreement between the de-clustered composite and block model zinc grades except for the hangingwall horizon two. For horizon two, repeatability of composite lead and silver grades is also poor.

14.10.3 Swath Plots

Sectional validation plots compare the de-clustered grades of composites (blue line) and OK (black line), IDW3 (grey line) that fall within 10 m easting, northing and 15 m elevation slices. The plot will identify slices that contain high-grade samples and low-grade blocks, or vice versa, which might indicate a problem with the estimation technique.

For all domains, block grades estimated by OK and IDW3 have a smoother profile relative to input samples. Where there are more samples, good agreement is seen between the trends of input composites and block grades estimated by each technique. The OK profile is slightly smoother than IDW. Both models reflect drillhole data on a local basis.

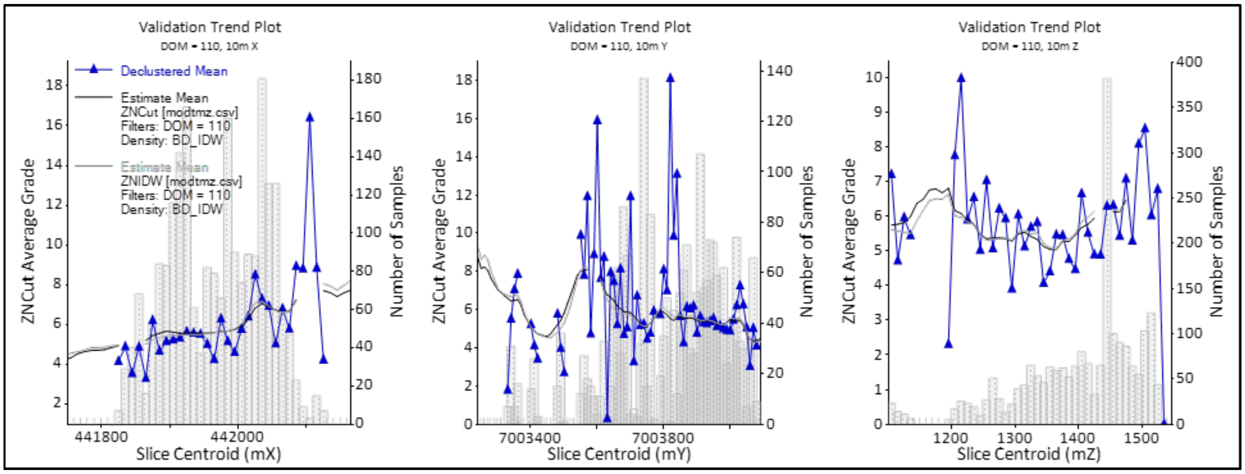
Example swath plots for zinc are show for Tom West (Figure 14-21), Tom East (Figure 14-22) and Jason Main Zone (Figure 14-23).

Table 14-12: Comparison of Means

Domain	Sample count	Sample data		Block model		% difference	
		Raw	De-clustered	OK Mean	IDW^3 Mean	De-clustered vs OK	De-clustered vs IDW^3
Zn %							
TMZ	2283	6.11	5.98	5.87	5.86	-1.7	-1.9
TEA	590	8.58	8.72	9.48	9.48	8.8	8.8
JMZ	688	6.57	6.24	6.38	6.38	2.3	2.2
H1F	274	4.63	4.34	4.54	4.53	4.7	4.4
H1H	106	5.56	6.24	6.02	6.35	-3.6	1.7
H2F	263	1.97	1.85	1.92	1.86	4.1	0.4
H2H	140	3.02	3.17	4.00	3.65	26.4	15.3
Pb%							
TMZ	2252	2.61	2.83	1.95	1.91	-31.0	-32.6
TEA	596	9.76	11.02	11.28	10.90	2.3	-1.1
JMZ	688	1.31	1.21	1.19	1.16	-1.8	-4.0
H1F	274	5.71	7.30	7.89	7.48	8.1	2.5
H1H	106	2.90	3.19	3.58	3.59	12.4	12.7
H2F	273	3.34	2.83	3.79	3.37	34.0	18.9
H2H	140	1.68	1.77	2.04	2.08	15.1	17.3
Ag g/t							
TMZ	1830	30.27	27.41	21.34	21.55	-22.2	-21.4
TEA	592	128.84	146.69	147.46	144.18	0.5	-1.7
JMZ	485	2.20	3.63	4.97	5.09	36.8	40.0
H1F	273	93.28	114.59	119.46	115.08	4.2	0.4
H1H	102	33.11	39.63	40.45	42.01	2.1	6.0
H2F	273	27.88	23.85	31.33	28.71	31.4	20.4
H2H	140	13.00	14.70	16.81	17.65	14.3	20.0

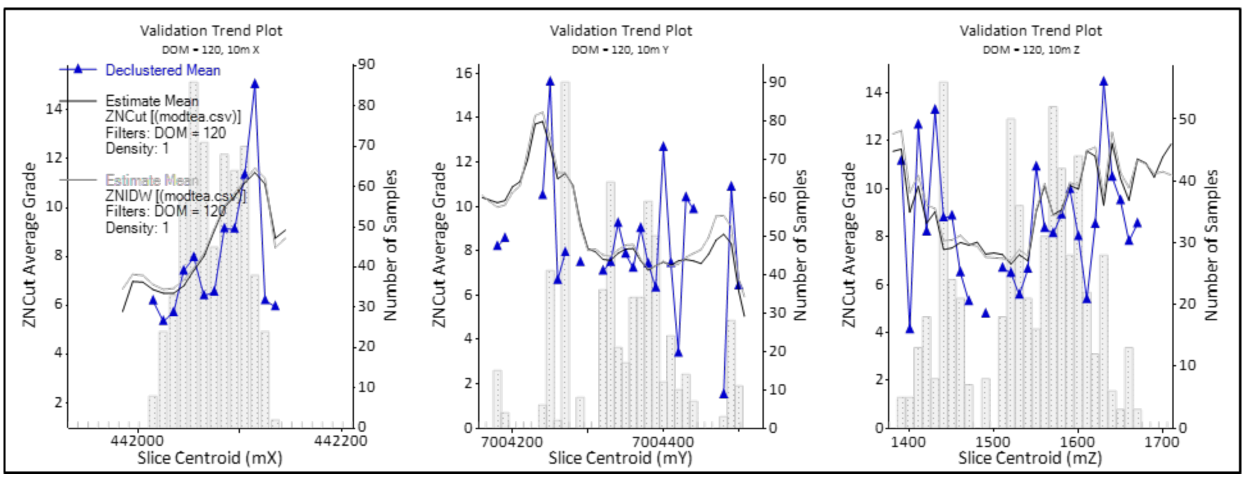
Source: CSA Global (2018)

Figure 14-21: Swath Plots for Zinc at Tom West



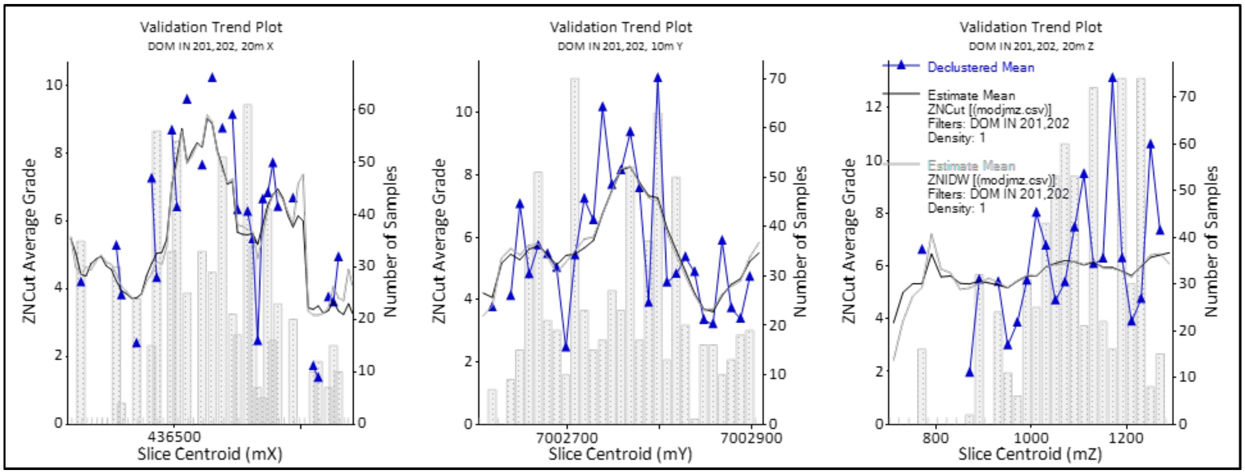
Source: CSA Global (2018)

Figure 14-22: Swath Plots for Zinc at Tom East



Source: CSA Global (2018)

Figure 14-23: Swath Plots for Zinc at Jason Main Zone



Source: CSA Global (2018)

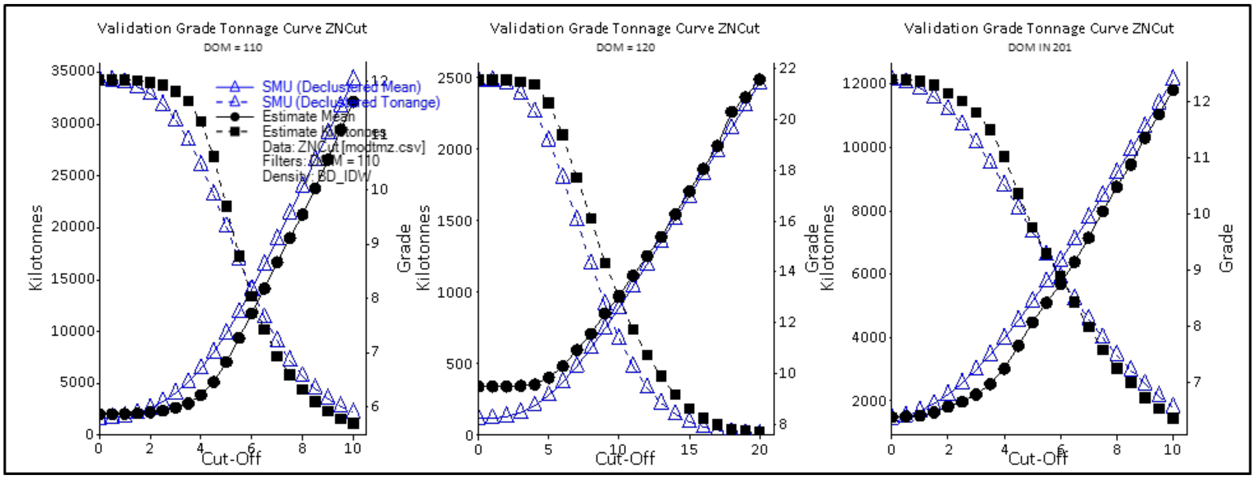
14.10.4 Global Change of Support

The Global Change of Support (GCOS) assessment compares the estimated block model grade and tonnage curves, to the theoretical grade and tonnage curves deduced from sample distributions. The sample grade and tonnage curves are adjusted to account for the decrease in variability that is expected for grades between Selective Mining Unit (SMU). This decrement in variability is known as support effect. Estimates were validated by comparing global theoretical grade-tonnage curves in SMU support with global theoretical grade-tonnage calculated with OK estimates.

Example zinc grade tonnage curves are shown for the Tom West, Tom East and Jason Main Zone deposits in Figure 14-24, examples for lead are shown in Figure 14-25 and examples for silver in Figure 14-26. Jason South domains had relatively few samples on a domain basis and GCOS was not undertaken for these domains.

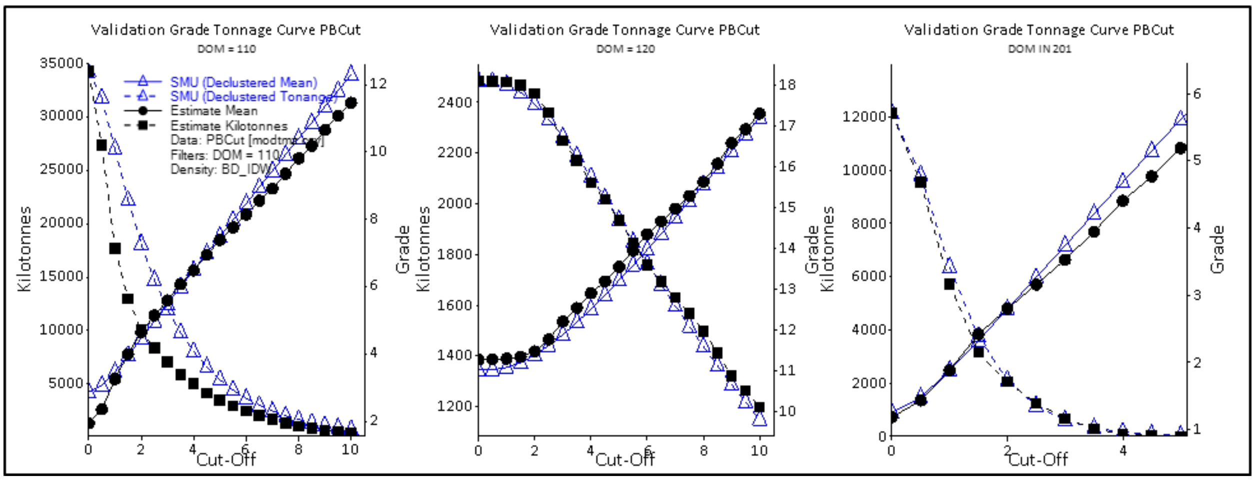
The OK estimation technique returns average block grades and tonnages that are similar to theoretical de clustered SMU grades-tonnage curves and are typically within +/-10% at a nominal cut off of 5% Zn and 5% lead.

Figure 14-24: GCOS Plots for Zinc at the Tom West, Tom East and Jason Main Zone Deposits



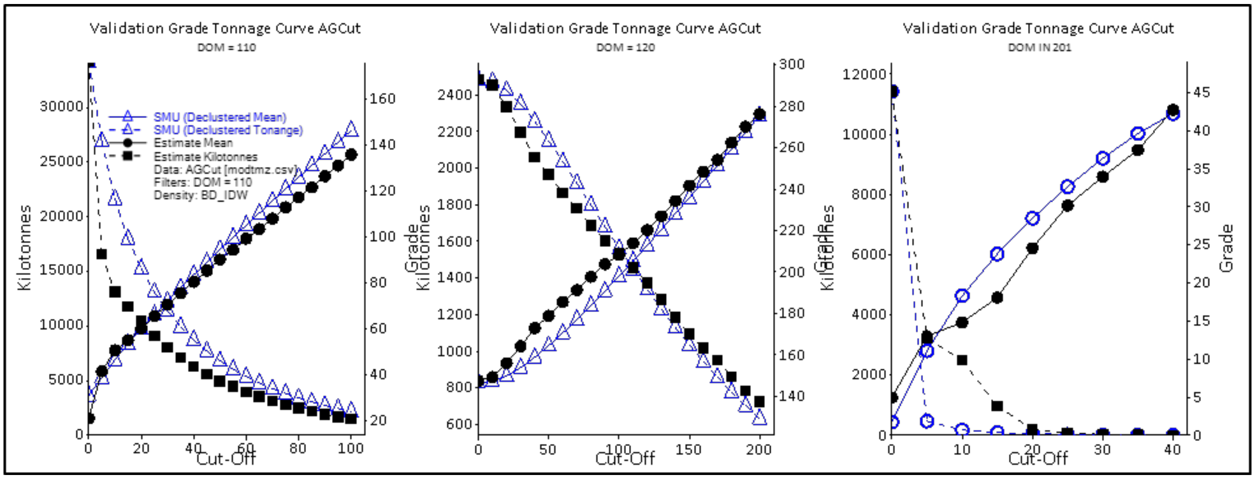
Source: CSA Global (2018)

Figure 14-25: GCOS Plots for Lead at the Tom West, Tom East and Jason Main Zone Deposits



Source: CSA Global (2018)

Figure 14-26: GCOS Plots for Silver at the Tom West, Tom East and Jason Main Zone Deposits



Source: CSA Global (2018)

14.10.5 Densities

A check was conducted to test that the mean density of the input data was close to the block model density mean. Results showed that on average composite means densities are within +/-2% for average block model densities.

Table 14-13: Comparison of Means for Densities

Domain	Sample Count	Raw	De-clustered	Block Mean IDW^2	De-clustered vs IDW^3
TMZ	1069	3.37	3.31	3.32	0.4
TEA	239	3.13	3.19	3.25	1.6
JMZ	593	3.18	3.18	3.19	0.5
H1F	272	3.43	3.46	3.45	-0.4
H1H	96	3.30	3.19	3.24	1.5
H2F	273	3.42	3.34	3.39	1.5
H2H	140	3.22	3.14	3.17	0.8

Source: CSA Global (2018)

14.11 Mineral Resource Classification

The resource estimate is prepared in accordance with CIM Definition Standards- For Mineral Resources and Mineral Reserves, adopted by the CIM Council on 10 May 2014 where:

An Inferred Mineral Resource as defined by the CIM Standing Committee is "*that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.*

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."

An Indicated Mineral Resource has a higher level of confidence than that applying to an Inferred Mineral Resource. It may be converted to a Probable Mineral Reserve. An Indicated Mineral Resource as defined by the CIM Standing Committee is *"that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.*

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve." and,

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve. A Measured Mineral Resource, as defined by the CIM Standing Committee is "*that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.*

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve."

Mineral resources that are not mineral reserves do not account for mineability, selectivity, mining loss and dilution and do not have demonstrated economic viability. These MREs include Inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these Inferred and Indicated mineral resources will be converted to the Indicated and Measured categories through further drilling, or into mineral reserves, once economic considerations are applied.

Classification, or assigning a level of confidence to Mineral Resources, is undertaken in strict adherence to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM Council, 2014). MREs for the Tom and Jason deposits were prepared by L. McGarry, CSA Senior Resource Geologist and Qualified Person for the reporting of Mineral Resources as defined by NI 43-101.

14.11.1 Reasonable Prospects of Economic Extraction

CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on 10 May 2014 require that resources have "reasonable prospects for economic extraction". This generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account possible extraction scenarios and processing recoveries.

To define reasonable prospects of economic extraction the in-ground value of each block was calculated using estimated factors for: metallurgical recoveries, assumed metal prices and smelter terms including payable factors, treatment charges and refining charges. These factors were estimated by consulting mining engineers retained by Fireweed and deemed by the Author to be reasonable. No penalties were included.

- Metal price assumptions were: US$1.17/lb zinc, US$0.99/lb lead, and US$16.95/oz silver;
- An exchange rate of US$1 = C$1.24 was used; and
- Metal recovery assumptions were: 79% for zinc, 82% for lead and 85% for silver (whereby 12% is recovered from zinc concentrate and 73% is recovered from lead concentrate) based on metallurgical tests carried out by Hudbay in 2012.

Based on these assumptions the formula for the NSR on each block was calculated as:

NSR $/t CAD =

$16.16 * Zn(%)	(Zn NSR from Zn concentrate)
+ $16.08 * Pb(%)	(Pb NSR from Pb concentrate)
+ $0.05853 *Ag(g/t) - $61.46 * Zn(%)	(Ag NSR from Zn concentrate, only if >0)
+ $0.4470 * Ag(g/t) - $36.07 * Pb(%)	(Ag NSR from Pb concentrate, only if >0)

The zinc equivalent (ZnEq) calculation was performed as: ZnEq = NSR/C$16.16.

An NSR cut-off grade of C$65 was selected for the reporting of mineral resource blocks after consideration of potential underground mining costs in the Yukon.

Note, these parameters and assumptions were made with an effective date of 10 January 2018 and used to calculate the MRE before the engineering and other work was done for this PEA Report, and as such, differ from those parameters and assumptions used elsewhere in this Report.

14.11.2 Resource Classification Parameters

The MRE is classified in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on 10 May 2014. Resource classification parameters are based on the validity and robustness of input data and the Qualified Person's judgement with respect to the proximity of resource blocks to sample locations and the kriging variance recorded during grade estimation.

At the Tom and Jason deposits, a sizable proportion of sampling is historical and has undergone variable amounts of QAQC sampling. Overall, sample data is considered to be of reasonable quality. CSA Global are confident that core samples and the zinc, lead and silver assays derived from them are representative of the material drilled and can be used in resource estimation studies.

The following is considered when classifying resources at Tom:

- Reliable down hole surveys are not available for the majority of historical drilling at Tom. Recent drillholes allow modeling of horizon intercepts with greater spatial accuracy;
- The majority of historical Hudbay drilling utilized small drill core diameters with small sample volumes;

- Intervals of poor drill core recovery were encountered in underground AX drillholes. For the 2011 and 2017 drilling campaigns, high core recoveries provide confidence that core samples, and the assay values derived from them, are representative of the material drilled and suitable for inclusion in resource estimation studies;
- Lithology domain and grade continuity are well established where drill density is greater than 40 m x 40 m; however, there remain portions of the deposit where sample density is insufficient to establish continuity beyond an Inferred level, specifically:
    - at Tom West below a depth of 200 m and in the folded southern portion of the deposit, and where unresolved sub-domains of lead and silver rich vent proximal mineralization occur;
    - at the peripheries of Tom East and below a depth of 100 m to 150 m; and
    - throughout Tom South East domains.

The following is taken into account when classifying resources at Jason:

- Lithology domain and grade continuity are well established where drill density is greater than 40 m x 40 m; however, there remain portions of the deposit where sample density is insufficient to establish continuity beyond an Inferred level, specifically:
    - at Jason Main Zone below a depth of 250 m and west of 436,450 mE and east of 436,950 mE; and
    - throughout Jason South, where unresolved fault offsets and sub-domains of lead and silver rich vent proximal mineralization occur; and where sample numbers on a per domain basis remain low and prevent a comprehensive model validation.

At both deposits:

- Surveying of historic drill collars in 2017, described in Section 12, provided more accurate location data for modeling of resources;
- Check sampling was undertaken on drill core assays from historical drill programs at Tom and Jason. The results presented in Table 12-1 verified that the historical assays could be used in a MRE;
- Digital lithology files have sufficient information to enable broad interpretations of geology. However, there are many internal dilution zones that are not yet properly defined. Core logging practices and lithology codes for recent drilling are inconsistent with earlier campaigns. It has not been possible to model the separate mineralization facies identified in Section 7 based on logged geology. Within major lithological units, statistical evaluation of assay grades indicates the presence of mixed mineralization styles, particularly at Jason South and in portions of the Tom West deposit; and
- The estimation and modelling technique is considered robust after consideration of the validation exercised undertaken as part of this study.

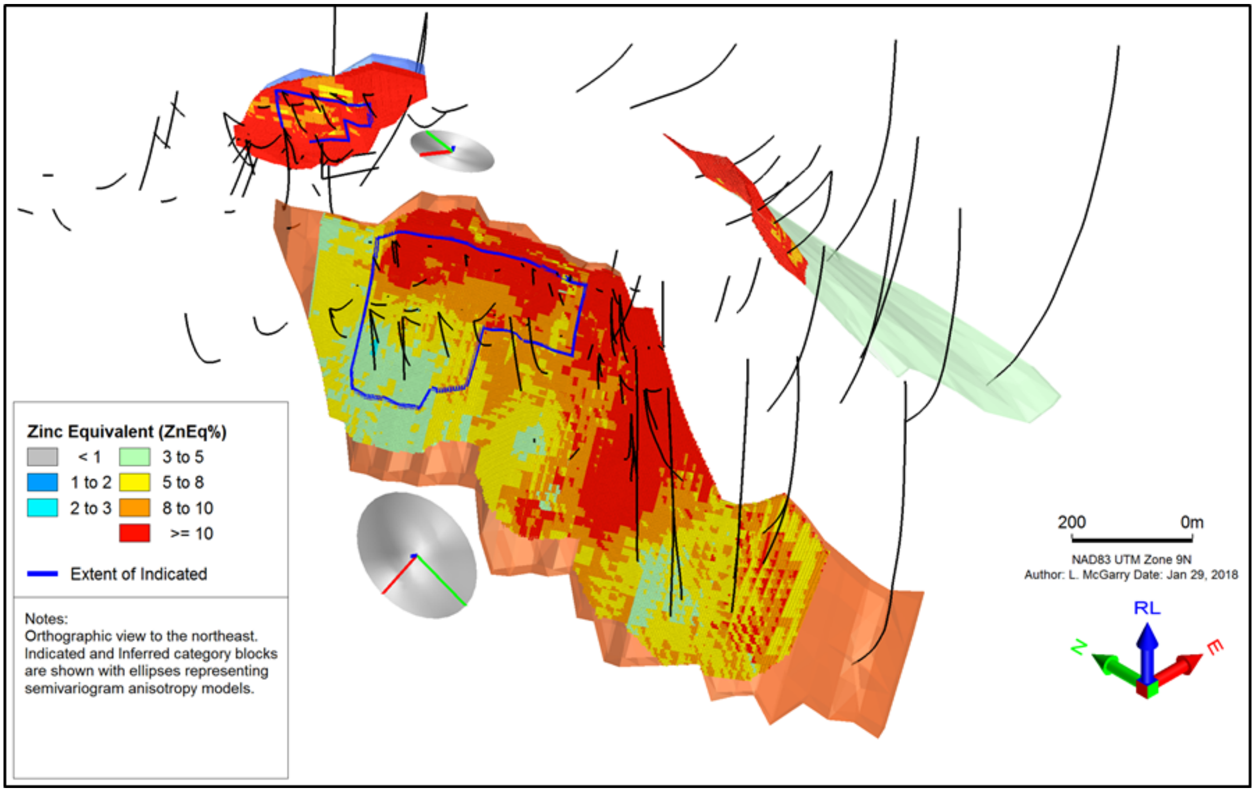
Resource classification was undertaken using classification boundary strings assigned to the block model in a cookie cutter fashion. Strings define a region of blocks that, on average, met the following criteria:

- Indicated Resources are defined by blocks that are within 50 m of composites with density values from drillholes greater than Bx size, completed after 1980 with good downhole survey data.

Indicated resource are in well drilled portions of the deposit where blocks are generally less than 40 m from the nearest drillhole and have a Kriging Variance of <60% and good geological continuity in Tom West, Tom East and Jason Main Zones. The extent of Indicated Resources is shown in Figure 14-27 and Figure 14-28; and

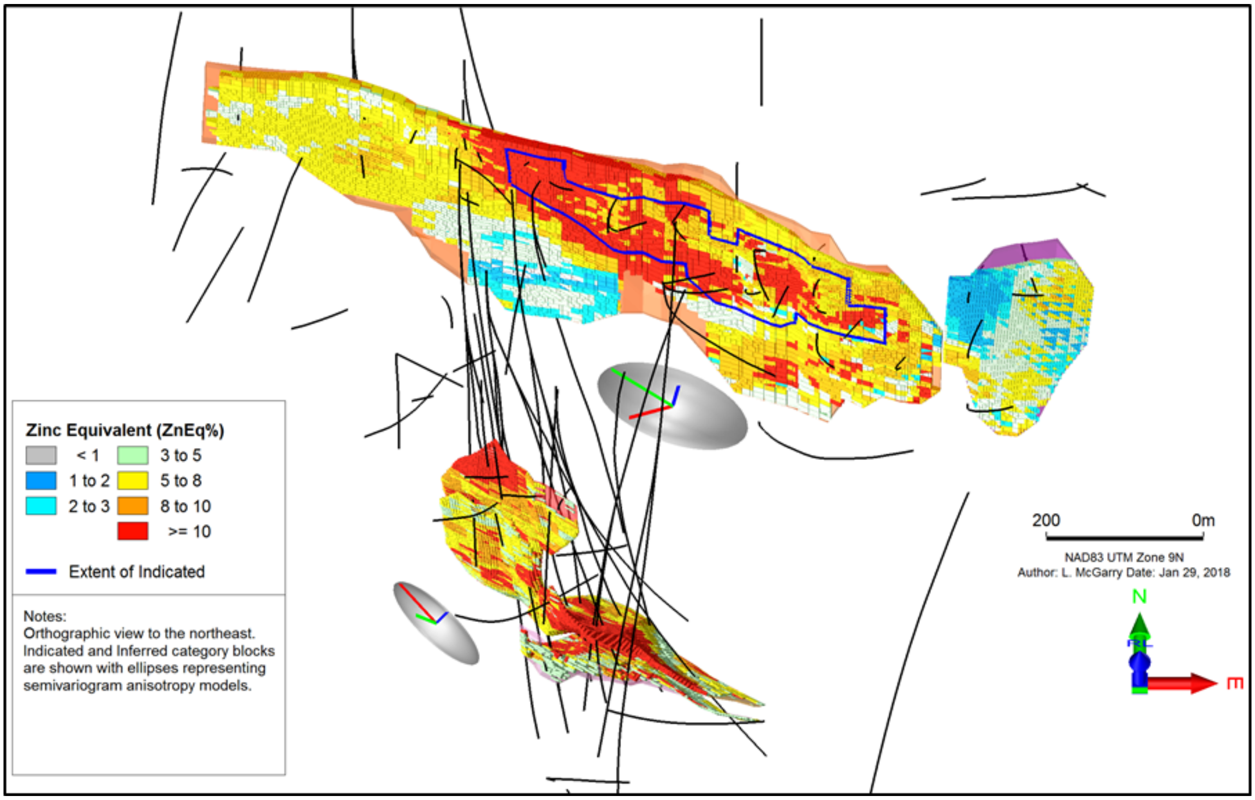
- Inferred Resources are defined by run 2 and run 3 blocks within 100 m of a drillhole (i.e. the typical variogram range) and in areas of poor core recovery, or that are geologically complex. All Tom Southeast and Jason South blocks are classified as Inferred. At Tom Southeast, inferred blocks are constrained to within 60 m of a drillhole. The extent of Inferred resource blocks is shown in Figure 14-27 and Figure 14-28.

Figure 14-27: Tom Block Models – Orthographic View to the Northeast



Source: CSA Global (2018)

Figure 14-28: Jason Block Models – Orthographic View to the Northeast



Source: CSA Global (2018)

14.12 Mineral Resource Reporting

Resources are reported in adherence to NI 43-101 Standards of Disclosure for Mineral Projects (Canadian Securities Administrators, 2011), and to the CIM Definition Standards on Minerals Resources and Reserves (CIM Council, 2014). The MRE is summarized by resource category in Table 14-14 and by resource domain in Table 14-15. The Mineral Resource has been reported above an NSR cut-off grade of C$65 and has an effective date of 10 January 2018.

Table 14-14: Macmillan Pass MRE as at 10 January 2018

	Tonnes	ZnEq %	Zn %	Pb %	Ag g/t	B lbs Zn	B lbs Pb	Moz Ag
Indicated	11.21	9.61	6.59	2.48	21.33	1.63	0.61	7.69
Inferred	39.47	10.00	5.84	3.14	38.15	5.08	2.73	48.41

Notes:

- The Mineral Resources in this disclosure were estimated by Leon McGarry, P.Geo
- The effective date of this Mineral Resource is 10 January 2018.
- Numbers have been rounded to reflect the precision of an Inferred and indicated MRE.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability but are required to have reasonable prospects for eventual economic extraction.
- The in-ground NSR values were calculated using estimated metallurgical recoveries, assumed metal prices and smelter terms including payable factors, treatment charges and refining charges. No penalties were included. Metal price assumptions were: US$1.17/lb Zn, US$0.99/lb Pb, and US$16.95/oz Ag and an exchange rate of US$1 = C$1.24. Metal recovery assumptions were: 79% Zn, 82% Pb and 85% Ag (12% to Zn concentrate and 73% to Pb concentrate). Based on these assumptions the formula for the NSR on each block was calculated as: NSR C$/t = $16.16 * Zn(%)+ $16.08 * Pb(%) + $0.05853 *Ag(g/t) - $61.46 * Zn(%) + $0.4470 * Ag(g/t) - $36.07 * Pb(%).
- The ZnEq calculation was performed as: ZnEq = NSR/C$16.16.
- The Mineral Resources in this news release were estimated using current CIM standards, definitions and guidelines. The Author estimated the resources by OK.
- The Tom and Jason database was audited in its entirety and has 6,986 samples assayed for zinc, 7,031 for lead and 5,888 for silver. Samples are collected from 249 exploration drillholes including duplicate and blank samples plus 111 assays from 2017 re sampled and re-assayed historical drill core. There are also 1,129 samples with density measurements in the database. During that work, CSA Global found the QAQC on the analytical data to support a qualitatively reasonable set of drill data.
- QAQC protocols were carried out to assess the quality of the drilling assay results and the confidence that can be placed in the assay data. The QAQC data available for Tom and Jason demonstrate the analytical data are of sufficient quality to be used in estimating mineral resources.
- Nine mineral domains were modeled from drillholes spaced at 30 m to 100 m. Within each domain, assays were regularized to 1 m intervals. Capping values between 20% and 50% were applied to zinc and lead grades, and 600 g/t applied to silver grades. Geostatistical analysis identified grade continuity ranges of between 70 m and 120 m within the plane each domain. Metal grades and bulk density values were interpolated into rotated block models with dimensions of 15 m in the along strike and down dip directions and 3 m in the across strike direction.
- Indicated Resources are defined in areas that are less than 40 m from the nearest drillhole and within 50 m of samples with assigned density values collected from a drillholes completed after 1980. Inferred Resources are defined within 100 m of a drillhole, and in areas of greater geological complexity or poor core recovery.

Source: CSA Global (2018)

Table 14-15: Macmillan Pass MRE Reported by Domain as at 10 January 2018

	Tonnes (Mt)	ZnEq %	Zn %	Pb %	Ag g/t	B lbs Zn	B lbs Pb	Moz Ag
Indicated								
Tom West	7.91	8.39	5.85	2.08	18.31	1.02	0.36	4.65
Tom East	0.81	20.29	8.74	8.62	110.00	0.16	0.15	2.85
Jason Main Zone	2.49	10.04	8.25	1.76	2.22	0.45	0.10	0.19
Indicated Total	**11.21**	**9.61**	**6.59**	**2.48**	**21.33**	**1.63**	**0.61**	**7.69**
Inferred								
Tom West	21.25	8.82	**5.97**	2.17	25.97	2.80	1.02	17.75
Tom East	1.68	27.35	9.86	12.86	170.00	0.37	0.48	9.17
Tom Southeast	0.29	11.51	7.08	3.56	34.84	0.05	0.02	0.33
Jason Main Zone	7.31	7.47	6.23	1.07	6.95	1.00	0.17	1.63
Jason South	8.93	11.56	4.41	5.28	68.01	0.87	1.04	19.53
Inferred Total	**39.47**	**10.00**	**5.84**	**3.14**	**38.15**	**5.08**	**2.73**	**48.41**

See notes in Table 14-15.
Source: CSA Global (2018)

A series of cut-off grade scenarios are presented in Table 14-16. CSA Global considered that each cut-off grade scenario has a reasonable prospect of economic extraction given appropriate variations to metal price and mining cost assumptions identified in the preceding section "*Reasonable Prospects of Economic Extraction*".

Table 14-16: Macmillan Pass MRE Reported by NSR Cut-off as at 10 January 2018

NSR cut-off (C$/t)	Tonnes (Mt)	ZnEq %	ZN %	Pb %	Ag g/t
Indicated					
$45	11.43	9.49	6.52	2.44	20.96
$65 (Base Case)	**11.21**	**9.61**	**6.59**	**2.48**	**21.33**
$85	10.30	10.04	6.81	2.65	22.92
$105	7.63	11.48	7.49	3.22	29.77
Inferred					
$45	43.14	9.44	5.55	2.95	35.45
$65 (Base Case)	**39.47**	**10.00**	**5.84**	**3.14**	**38.15**
$85	33.18	11.01	6.26	3.56	43.99
$105	24.48	12.83	6.82	4.50	56.34

See notes in Table 14-15.
Source: CSA Global (2018)

14.12.1 Factors that May Affect the Mineral Resource

CSA Global is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that could potentially affect this mineral resource estimate.

The mineral resources may be affected by a future conceptual study assessment of mining, processing, environmental, permitting, taxation, socio-economic and other factors.

Additional technical factors which may affect the MREs include:

- Metal price and valuation assumptions;
- Changes to the technical inputs used to estimate zinc, lead and silver content (e.g. bulk density estimation, and grade model methodology);
- Geological interpretation (revision of deposit models and the modeling of internal waste domains e.g. dikes and structural offsets such as faults and shear zones);
- Changes to geotechnical and mining assumptions, including the minimum mining thickness; or the application of alternative mining methods such as open pit mining; and
- Changes to process plant recovery estimates if the metallurgical recovery in certain domains is lesser or greater than currently assumed.

There is insufficient information at this early stage of study at 10 January 2018 to assess the extent to which the resources might be affected by these factors.

14.12.2 Comparison with Previous Mineral Resource Estimates

These new current MREs represent a substantial increase over the previous historic 2007 publicly reported MREs (see Fireweed news release dated 1 June 2017). The increase is mainly due to:

- Results from an additional 25 drillholes (11 by Hudbay Minerals in 2011 and 14 by Fireweed in 2017 – see Fireweed news release dated 27 December 2017 for details), a number of which cut wider and/or higher-grade intersections than predicted by historic drill results, and others which were step-outs that expand on previous drilling;
- More accurate survey coordinates for historic drillholes than were available for the 2007 report; and
- New bulk density determinations obtained during the 2017 field season that, when coupled with both new and historical assay data, allowed better estimates of rock bulk density than were available for the 2007 report.

Table 14-17: 2017 Macmillan Pass MRE Comparison with 2007 Estimate

Domain	2007				Current				Change			
	Mt	Zn (%)	Pb (%)	Ag (g/t)	Mt	Zn (%)	Pb (%)	Ag (g/t)	Mt	Zn (%)	Pb (%)	Ag (g/t)
Indicated												
Tom	4.98	6.64	4.36	47.77	8.7	6.12	2.68	26.80	+75%	-8%	-38%	-44%
Jason	1.45	5.25	7.42	86.68	2.5	8.25	1.76	2.22	+72%	+57%	-76%	-97%
Inferred												
Tom	13.55	6.68	3.1	31.77	23.2	6.27	2.96	36.49	+71%	-6%	-5%	+15%
Jason	11	6.75	3.96	36.42	16.2	5.23	3.39	40.53	+47%	-23%	-14%	+11%

CSA Global have not verified these historic 2007 resource estimates and are not treating these historical estimates as current mineral resources. While these estimates were prepared in accordance with National Instrument 43-101 and the “Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Mineral Reserves Definition Guidelines” in effect at the time (2007), there is no assurance that they are in accordance with current standards, and these historical resource estimates should not be regarded as consistent with current standards or unduly relied upon as such. CSA Global include these historical estimates in this report for purposes of comparison to the current resource only.

Source: CSA Global (2018)

15 Mineral Reserve Estimates

15.1 Mineral Reserve Non-Compliance

No Mineral Reserve has been established at the Macmillan Pass project to date.

Mineral resources are not mineral reserves and have no demonstrated economic viability. **This preliminary economic assessment does not support an estimate of mineral reserves, since a pre-feasibility or feasibility study is required for reporting of mineral reserve estimates.** This report is based on mine plan tonnage (mine plan tonnes and/or mill feed).

Mine plan tonnes were derived from the resource model described in the previous section. Measured, indicated and inferred mineral resources were used to establish mine plan tonnes.

Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that will enable them to be categorized as mineral reserves, and there is no certainty that all or any part of the mineral resources or mineral resources within the PEA mine plan will be converted into mineral reserves.

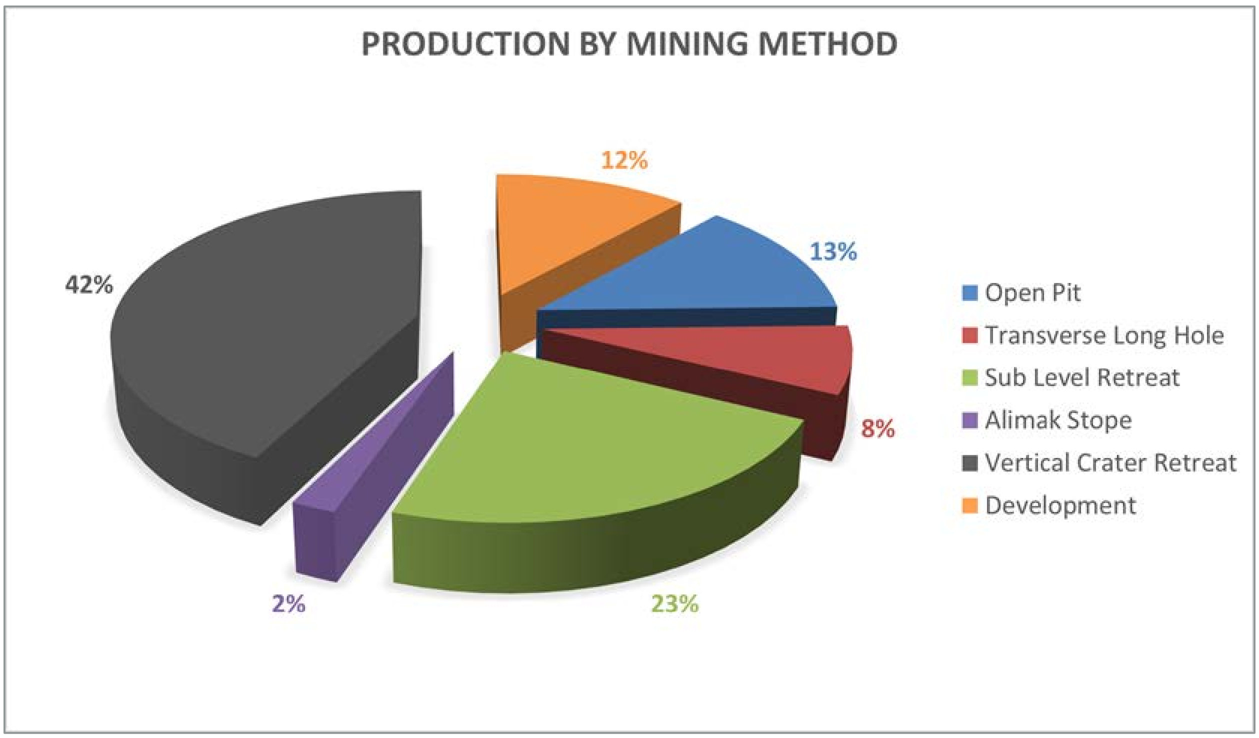
16 Mining Methods

The Macmillan Pass mineable resource will be extracted from both Tom and Jason deposits using a combination of mining methods, including:

- Open Pit Mining (OPM);
- Longhole stoping (LHS);
- Vertical Crater Retreat (VCR);
- Sub-level retreat (SLR);
- Alimak Stoping (ALS); and
- Development and cross cuts (XCO).

Table 16-1 below outlines a summary of mining methods proposed at Macmillan Pass.

Figure 16-1: Mine Production by Method



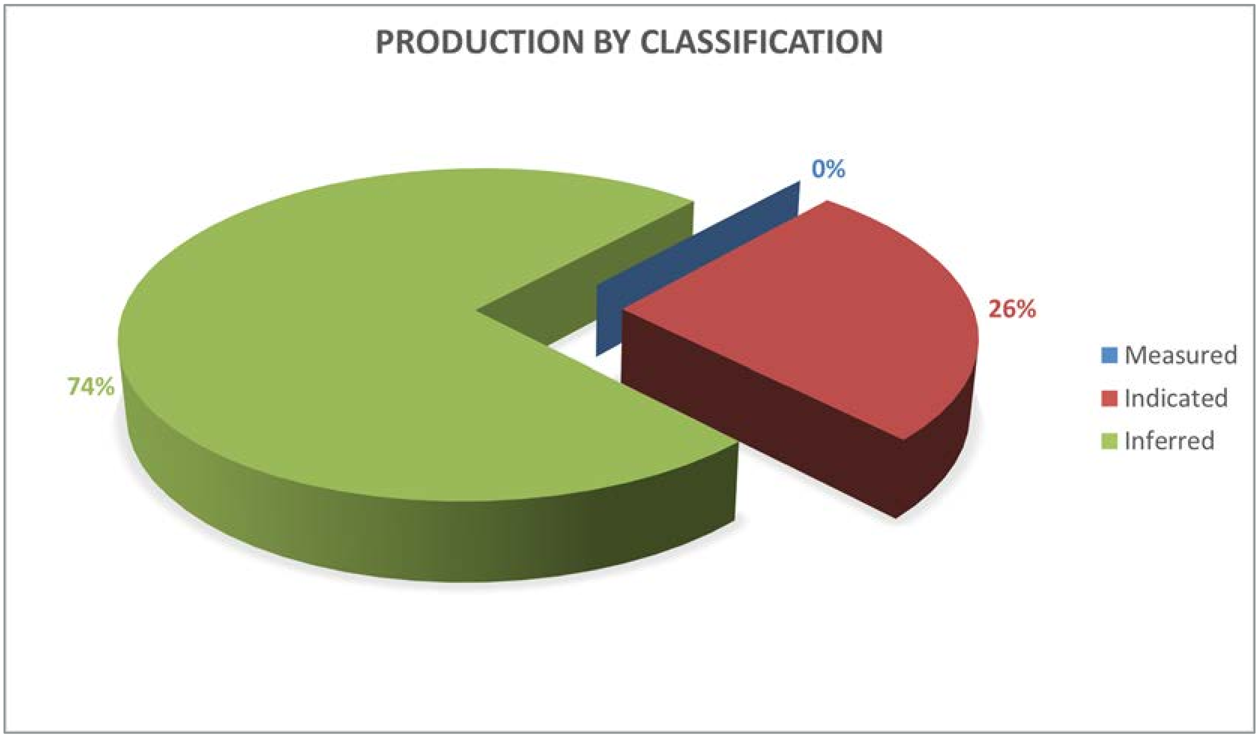
Source: JDS (2018)

The Fireweed deposit will be initially mined by open pit methods while underground development is occurring. As the open pit resources are depleted the underground operation will ramp up to sustain the nominal throughput of 5,000 tonnes per day to the mill.

Existing surface roads and underground development will be rehabilitated and utilized as part of the mine plan, with the addition of new roads and development as necessary to execute the production schedule.

Measured, Indicated and Inferred Mineral Resources were included in the mine design and schedule optimization process. The PEA mine plan tonnes per mineralization classification is shown in Table 16-2 below. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves, and there is no certainty that the inferred resources would be upgraded to a higher resource category.

Figure 16-2: Mine Tonnes by Resource Classification



Source: JDS (2018)

16.1 Mine Plan Tonnes Estimation Process

To determine the mine plan tonnes at Macmillan Pass, the following process was utilized:

- Analyze the geologic resource model for geometric properties, such as mineralized zone width, depth, length, and continuity;
- Select the mining methods best suited for the deposit based on geometry, economics, and geotechnical parameters;
- Determine an economic cut-off grade based on expected operating cost, mining recovery, mining dilution, and commodity price assumptions;
- Identify the blocks in the model that are above cut-off, and design production stope shapes around these blocks;

- Query the production stope shapes for in-situ tonnage and grade data, apply mine dilution, and check the diluted stope grades against the cut-off grade, removing all stopes that fall below cut-off; and
- Develop a mine plan around the economically viable production stopes and run economic models on various production scenarios.

16.2 Deposit Characteristics

Fireweed hosts five zinc-lead-silver-barite deposits, known as Tom West (TMZ), Tom East (TEA), Tom Southeast (TSE), Jason Main (JMZ), and Jason South (JST). These zones are SEDEX type deposits, subvertical in nature, with a long planar strike. Tom and Jason deposits are separated from one another by approximately 4.5 km and share two distinct orientations. Tom deposits strike approximately 335 degrees while Jason deposits strike 285 degrees.

Each zone is variable in size, with average dimensions listed below in Table 16-1.

Table 16-1: Average Dimensions of Potentially Mineable Resource

Zone	Units	Height	Dip	Width			Strike			Outcropping
				Min	Max	Average	Min	Max	Average	
TMZ	m	700	60	2.4	43.2	24	41	687	491	YES
TEA	m	375	65	3.1	15.7	11	50	248	193	YES
TSE	m	700	45	1.2	7.6	4	31	337	265	YES
JMZ	m	600	85	1.9	11.7	10	20	957	406	YES
JST	m	650	65	2.1	25.2	12	11	317	151	NO

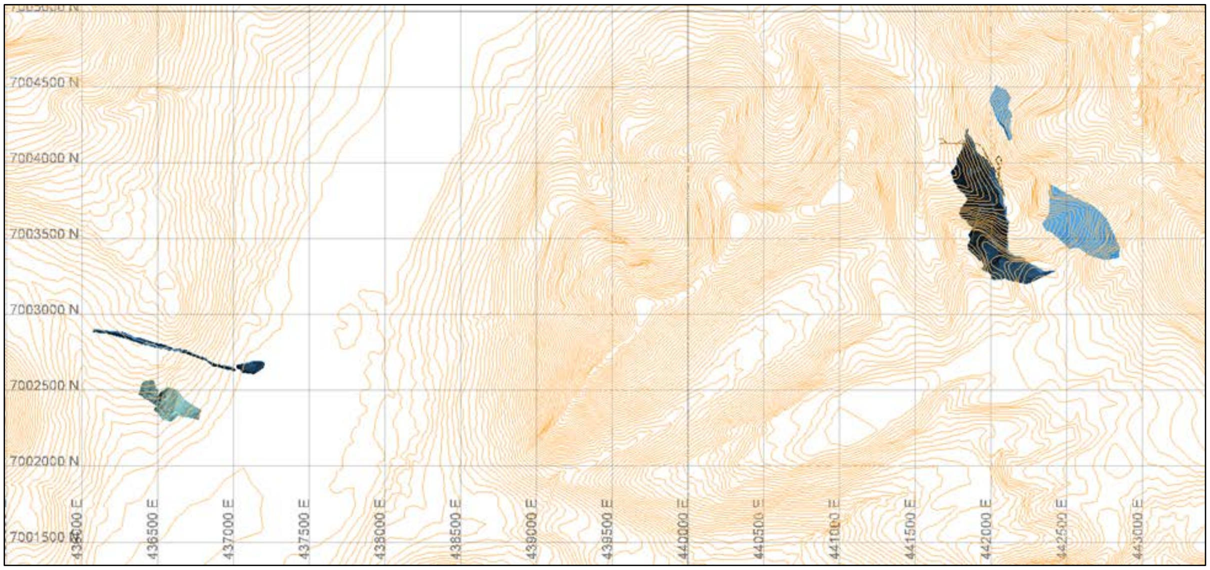
Source: JDS (2018)

Of the five principal mineral zones, all but JST contain mineral resources outcropping to surface, which suggests open pit mining as a potentially viable mining method.

Tom West contains existing underground development which has been abandoned for several years and will require extensive rehabilitation for future use.

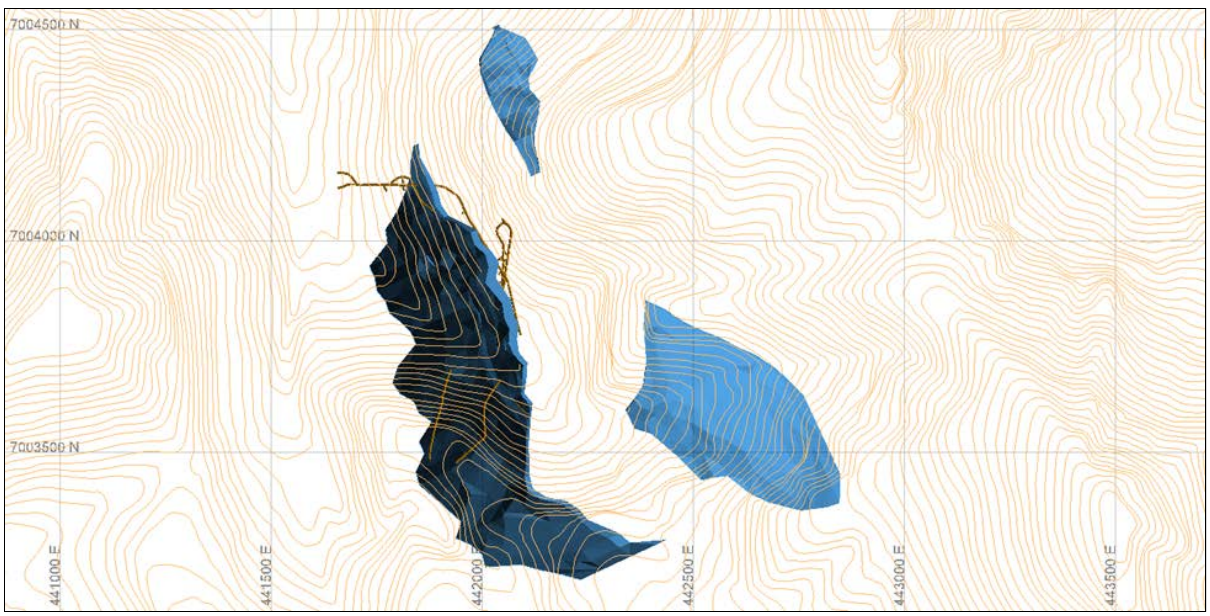
Figure 16-3 through Figure 16-7 depicts the general orientation of the Tom and Jason deposits.

Figure 16-3: Tom (Right) and Jason (Left) Deposits – Plan View



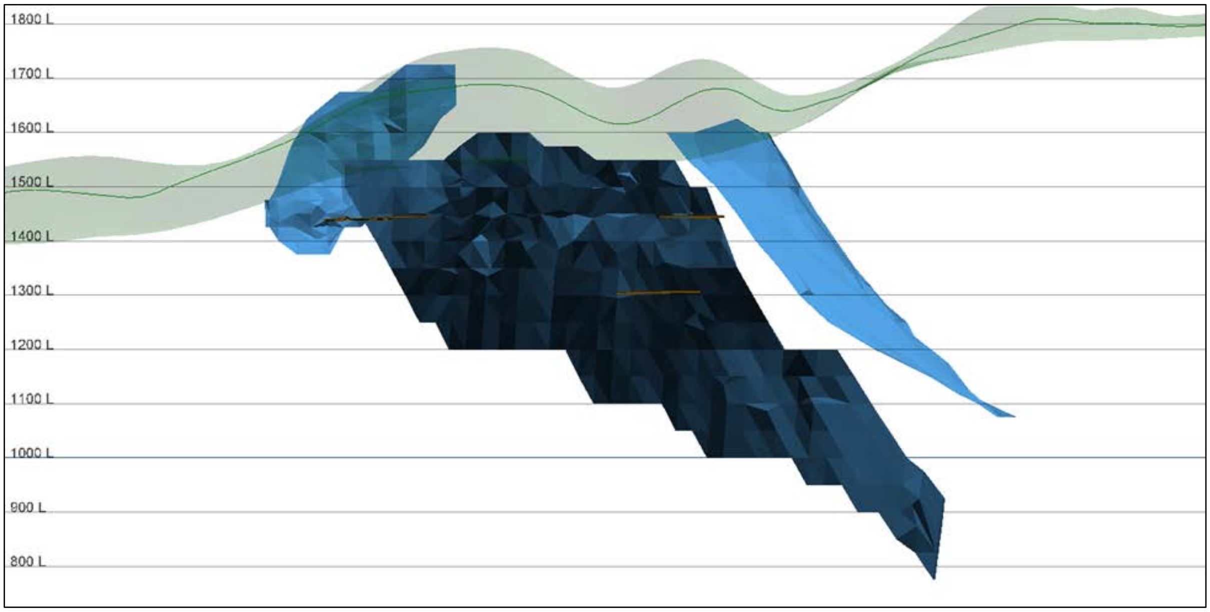
Source: JDS (2018)

Figure 16-4: Tom Zone – Plan View



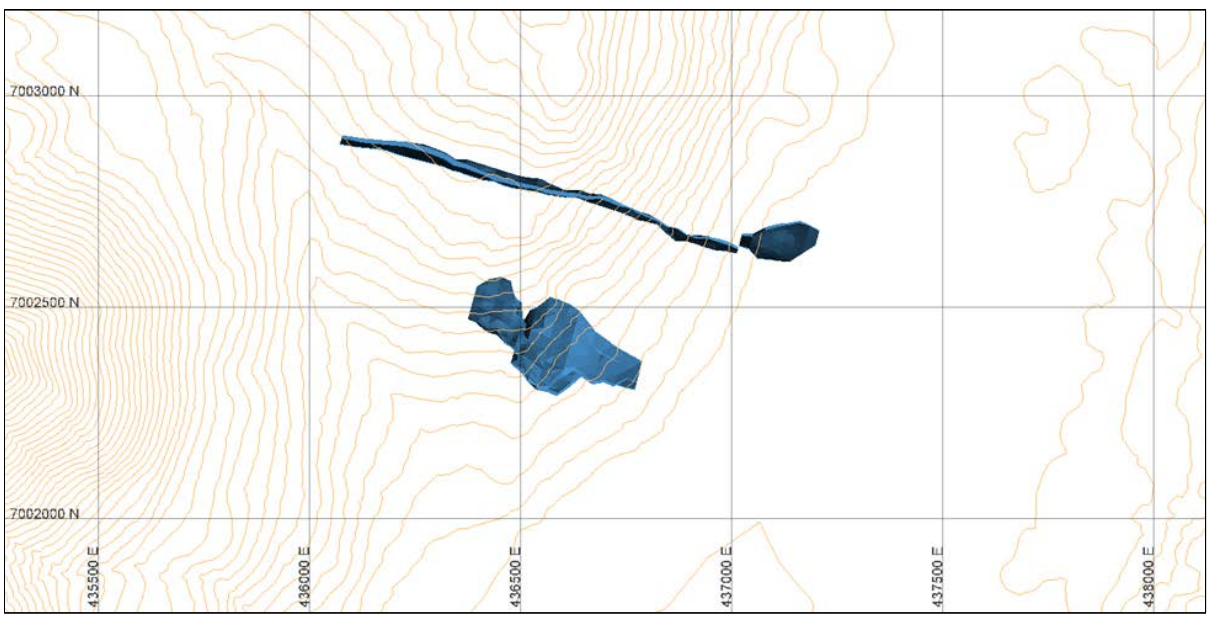
Source: JDS (2018)

Figure 16-5: Tom Zone – Long Section Looking Northeast



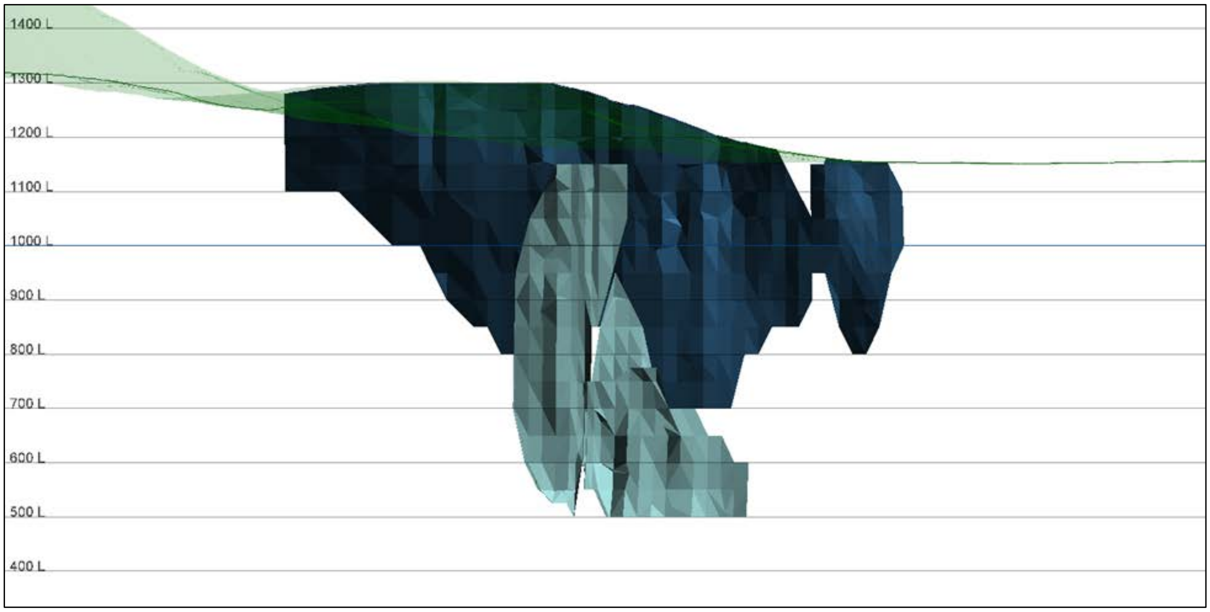
Source: JDS (2018)

Figure 16-6: Jason Zone – Plan View



Source: JDS (2018)

Figure 16-7: Jason Zone – Long Section Looking North



Source: JDS (2018)

16.3 Mining Method Selection

Given the outcropping nature and deep vertical extension of both Tom and Jason deposits, several zones were reviewed for both open pit and underground potential.

16.3.1 Open Pit Methods

The Jason Main (JMZ) and Tom West (TMZ) deposits will initially be mined via conventional truck and shovel surface mining methods and then will transition to underground mining. Given the current understanding and extent of potentially acid-generating mine waste, the open pits were purposely limited in size as any waste not used as UG backfill will be re-handled and placed back in the mined out pits at closure. In addition these starter pits reduce up-front capital costs.

16.3.2 Underground Mining Methods

Longhole (LH) stoping will be used at Macmillan Pass as the principal mining method for its high productivity, low cost, selectiveness, and successful history of application for deposits of this nature.

In the planned longhole stopes at Macmillan Pass, a top and bottom drift delineate the stope and a dedicated longhole drilling machine drills blast holes between the two drifts. The drill holes are loaded with explosives and the stope is blasted, with broken material falling to the bottom drift for extraction. In LH stopes, remote controlled load haul dump machines (LHD) are required to remove the blasted material from the stope once blasting commences.

Several variations of longhole stoping will be applied at Macmillan Pass, specific to the geotechnical and geometric properties of the resource zones.

16.3.2.1 Vertical Crater Retreat

Where ground conditions are good and the resource is thick (greater than 20 m), such as Tom West and Jason South, vertical crater retreat (VCR) LH stoping will be used. VCR mining is an open stoping, bottom-up mining method that involves vertically drilling large-diameter holes into the deposit from the top, and then blasting horizontal slices of the deposit into an undercut. The broken muck is left in the stope as it is mined upwards until the stope is fully broken. During stoping a portion of the mineralized zone will be mucked out of the stope to ensure a small void is kept to allow further blasting of horizontal slices. VCR stopes are mined in a primary / secondary fashion whereby primary stopes are mined and filled with an engineered structural backfill before adjacent stopes are mined and filled with loose fill. VCR is similar to shrinkage mining for the fact that the stope stays full of broken muck during mining to provide wall support during stoping. Other advantages to VCR are related to an increase in level spacing which incur fewer drill set-ups and muck points, increasing safety and efficiency in the stoping process. A disadvantage to VCR mining is the large preparation costs and long wait for muck release given the stope stays full until final draw down. VCR stopes are also less selective than other mine methods and may incur high dilution in irregular ore bodies.

16.3.2.2 Transverse Longhole

Where ground conditions are good and the resource is of average thickness (greater than 4 m but less than 20 m), such as that of Tom East, transverse longhole sub-level stoping (LHS) will be used. LHS mining is an open stoping, bottom-up mining method that involves vertically drilling large-diameter holes into the deposit from top or bottom, and then blasting vertical slices of the deposit into an undercut. Broken muck is removed immediately before additional blasts are made. LHS stopes are mined in a primary / secondary fashion whereby primary stopes are mined and filled with an engineered structural backfill before adjacent stopes are mined and filled with loose fill. Transverse LH stoping requires a footwall access to be driven in parallel to the resource to provide cross cuts entries evenly spaced along strike. As such, this method is more expensive than longitudinal LH stoping but offers higher productivity and selectivity given the ability to mine several high grade stopes at once.

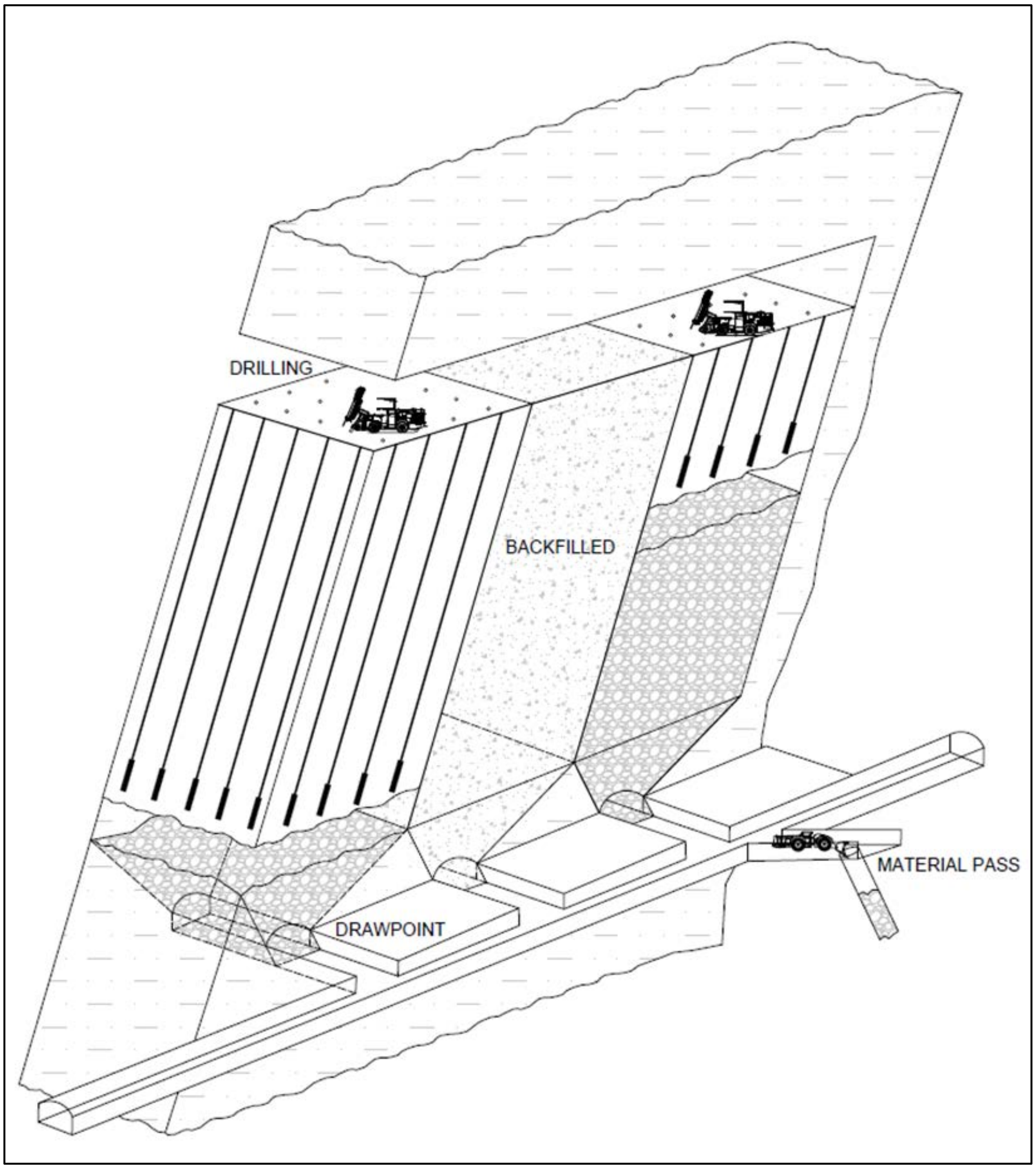
16.3.2.3 Sub-Level Retreat

Where ground conditions are poor and the resource is of average thickness (greater than 4 m but less than 20 m), such as that of Jason Main, sub-level retreat stoping (SLR) will be used. SLR mining is an open stoping, top-down mining method that involves vertically drilling large-diameter holes into the deposit from the bottom, and then blasting vertical slices of the deposit into an undercut. Broken muck is removed immediately before additional blasts are made. SLR stopes are mined in a longitudinal retreat fashion whereby stopes are mined in series along strike, retreating from the outside-in towards the level access. As SLR stopes are mined they are simultaneously backfilled with loose material contained in the stopes above and adjacent. Loose backfill is deposited from either a glory hole on surface direct to the stope locations, or from an LHD which trams waste to an access point above the stope being mined. SLR mine method is well suited to deposits with poor ground conditions because the stopes are never mucked empty and allowed to fail. Disadvantages to SLR mining include high backfill dilution, low mineral recovery, and limited selectivity and productivity. Advantages to SLR include the ability to longhole mine an otherwise cut and fill zone and to avoid requirements for structural backfill, which keeps mining costs low.

16.3.2.4 Alimak Stoping

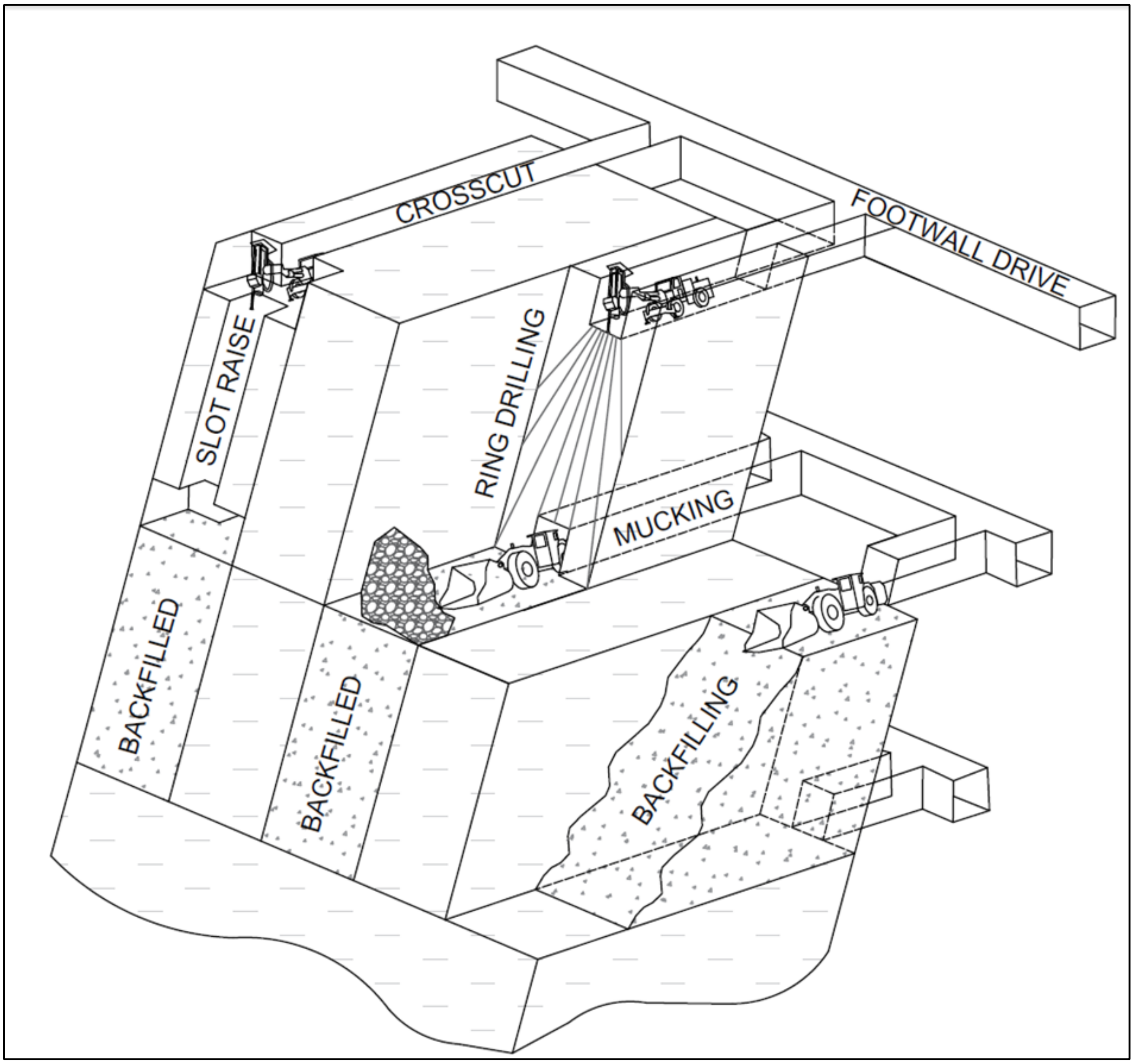
One portion of Jason South will utilize Alimak longhole stoping (ALS). ALS mining is an open stoping, bottom-up mining method that involves laterally drilling large-diameter holes into the mineralized zone from a central raise, and then blasting horizontal slices into an undercut. Broken muck may be mucked immediately or left in the stope before additional blasts are made, depending on geotechnical constraints. ALS stopes are mined in a primary / secondary fashion whereby primary stopes are mined and filled with an engineered structural backfill before adjacent stopes are mined and filled with loose fill. ALS is one of the highest cost, lowest productivity LH stoping methods only to be used where capital development precludes access for alternate LH mine methods. ALS mine method is well suited to deposits with a large vertical extent but small strike length, and where development of sill drives is cost prohibitive.

Figure 16-8: Vertical Crater Retreat Mine Method



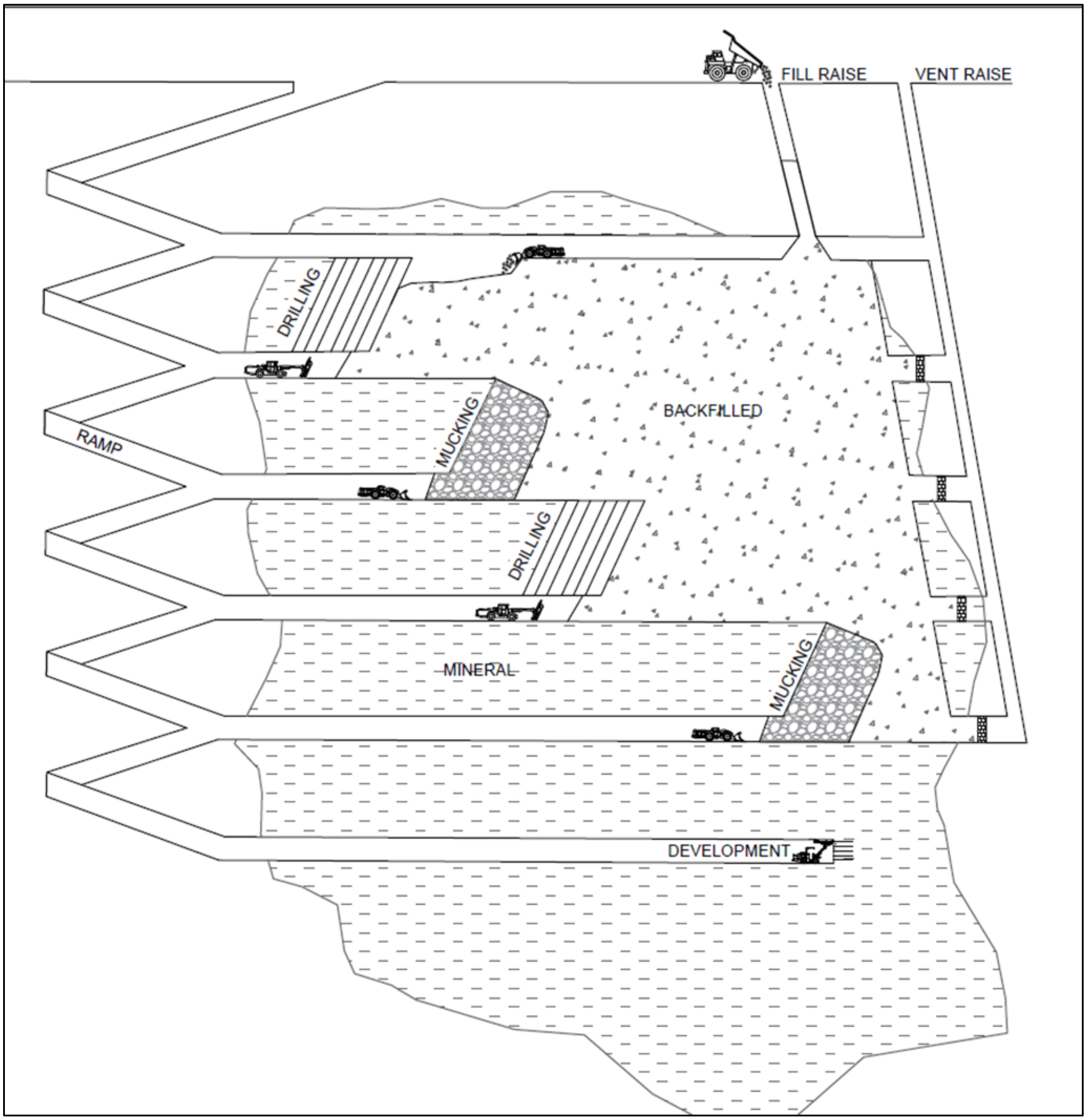
Source: JDS (2018)

Figure 16-9: Transverse Longhole Stope Mine Method



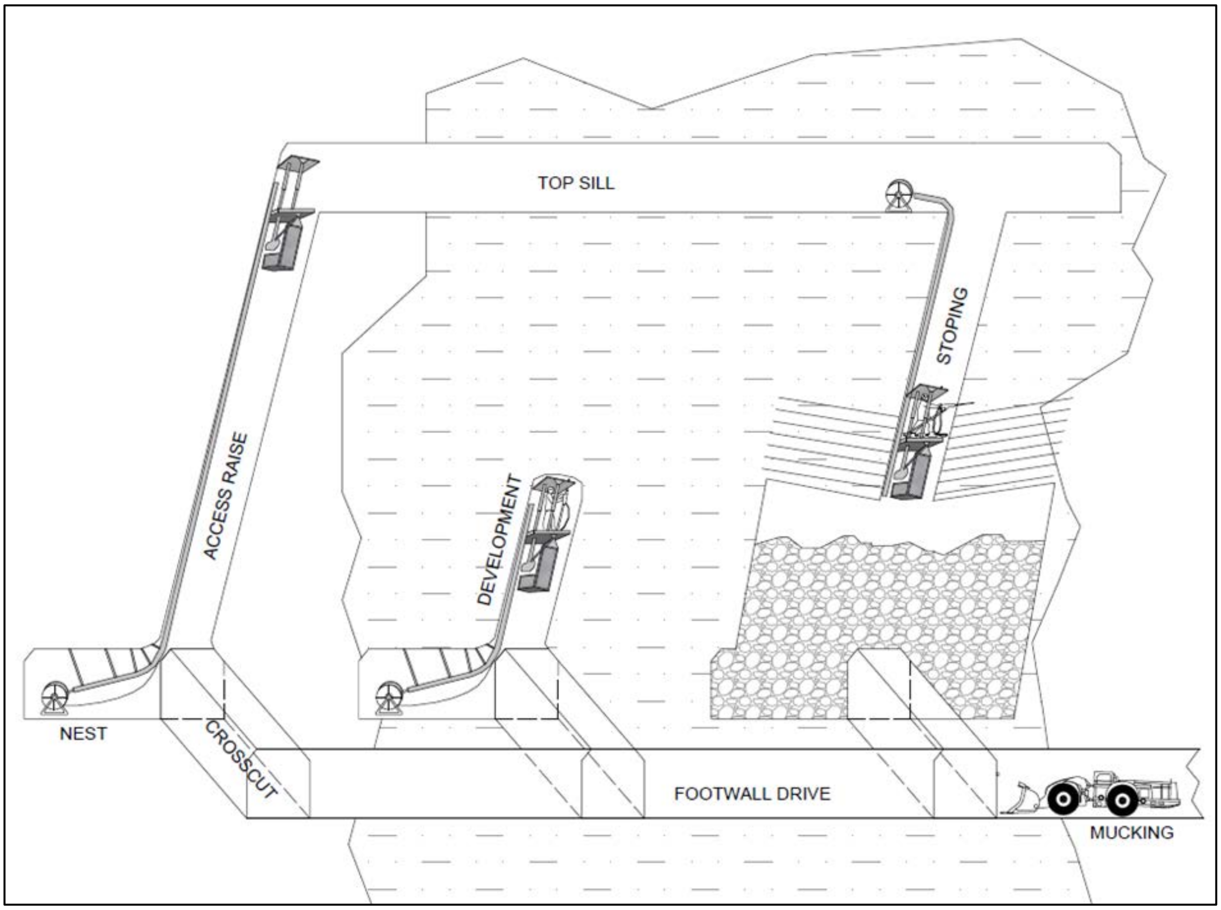
Source: JDS (2018)

Figure 16-10: Sub-Level Retreat Stope Mine Method



Source: JDS (2018)

Figure 16-11: Alimak Stope Mine Method



Source: JDS (2018)

16.4 Geotechnical Considerations

16.4.1 Geotechnical Characterization

Geotechnical specific drilling and testing programs have not yet been carried out for the project. In order to estimate geotechnical design parameters for the PEA, JDS has relied primarily upon rock quality designation (RQD) and core recovery data collected by Fireweed during their 2017 resource core logging program as well as core photographs from select drillholes. In particular the following drillholes were reviewed:

- TS17-001 through TS17-006 (Tom West Zone);
- TS17-007 (Tom East Zone); and
- JS17-001 through JS17-007 (Jason Main Zone).

Due to the lack of recent drilling at the Jason South Zone, there is very limited information available that is relevant to geotechnical characteristics of the deposit. Assessment of the Jason South Zone was limited to review of core photographs for a few intervals of split core. No information was available for the Tom Southeast Zone.

Using the Fireweed drillhole database, RQD and core recovery values were averaged for the immediate approximately 20 m of hanging wall and footwall as well as the mineralized zone. Table 16-2 and Table 16-3 contain summaries of the average RQD and core recovery, respectively, for the HW, FW and Mineralized Zones at Tom and Jason.

Table 16-2: Average RQD for HW, FW and Mineralized Zone (%)[A]

Deposit	Hanging Wall	Ore Zone	Footwall
Tom West Zone	58 (Fair)	80 (Good)	64 (Fair)
Jason Main Zone	41 (Poor)	30 (Poor)	52 (Fair)

[A] Descriptions according to (Deere, 1989) guidelines
Source: Fireweed, JDS (2018)

Table 16-3: Average Core Recovery for HW, FW and Mineralized Zone (%)

Deposit	Hanging Wall	Ore Zone	Footwall
Tom West Zone	93	97	91
Jason Main Zone	93	86	96

Source: Fireweed, JDS (2018)

High level estimates of rock mass quality were made according to the Barton Q’ rock mass rating system (Barton & Grimstad, 1994). The estimates were made using the average RQD values for each zone and applying reasonably conservative estimates of the number of joint sets and joint condition parameters, Jr and Ja based on the core photographs and experience in similar geologic environments. The estimates of Q’ for each zone are summarized in Table 16-4 along with the Bieniawski (1976) rock mass rating (RMR).

Table 16-4: Average Rock Mass Quality (Q’ & RMR 76) HW, FW and Mineralized Zone

Deposit	Q’ [A]		RMR 76 [B]	
	Hanging Wall / Footwall	Ore Zone	Hanging Wall / Footwall	Ore Zone
Tom West Zone	3.3 (Poor)	7.5 (Fair)	55 (Fair)	62 (Good)
Jason Main Zone	1.7 (Poor)	1.7 (Poor)	49 (Fair)	49 (Fair)

A Q’ is calculated by setting the Joint Water Factor (Jw) and Stress Reduction Factor (SRF) both equal to 1 in the Q equation.
B RMR calculated using Bieniawski (1976) equation (RMR = 9 ln Q’ + 44).
Source: JDS (2018)

An approximately 4.5 m by 5.5 m exploration adit also exists at the Tom West deposit but is blocked off to control water flow approximately 100 m inside the portal. The portal was not accessible as part of the PEA however recent photographs taken by Fireweed within the approximately 100 m of open adit were reviewed and have been considered in the rock quality estimations.

16.4.2 Anticipated Ground Conditions

Tom Deposit

The Tom mineralized zones are typically located within Unit 3B, towards or sometimes at the base of the unit. Unit 3B is described as a carbonaceous, dark grey to black, finely laminated mudstones and is exposed at the portal and within the existing adit.

The sedimentary rock formations at Tom typically have a sub-vertical, east-west trending foliation which is the dominant fabric in the area. Bedding planes typically parallel the deposit hanging wall, striking N-NE and steeply southwest dipping at Tom West and steeply northeast and east dipping at the Tom East and Southeast Zones, respectively.

The mineralized zones at Tom appear to be of generally good quality with relatively few fractures compared to the hangingwall (HW) and footwall (FW) rocks. The average Q' value of 7.5 suggests that the rock mass can be classified as 'Fair' quality according to Barton (2002). Based on the available information, the HW and FW appear to have a significantly higher number of fractures and possible fault zones. An average Q' value of 3 was estimated for the HW and FW rocks which classifies as 'Poor' quality rock mass quality according to the Barton (2002) Q system.

Bedding forms planes of weakness in the HW and FW sedimentary units. These planes of weakness are anticipated to limit the maximum area of HW that can be open at any one time, prior to backfilling. Based on review of photographs of the exploration adit, the bedding planes and orthogonal foliation trend may create blocky ground in larger excavations. The approximately 4.5 m high by 5.5 m wide adit did not appear to have encountered ground control issues and was developed with minimal ground support.

Jason Deposit:

The dominant foliation at Jason strikes SE and dips steeply towards the SW. The sedimentary units parallel the E-SE mineralization trend with the Jason Main Zone being sub-vertical and Jason South dipping steeply towards the NE. Bedding planes typically parallel the deposit hanging wall and are expected to heavily influence the maximum stope dimensions. Geologic mapping of the Jason area demonstrates that the deposit is structurally complex with several major fault structures along and within the mineralized zone.

Based on review of the RQD database and core photographs, the mineralized zone at Jason Main Zone is heavily fractured and generally poor quality. The average Q' value of 1.7 suggests that the mineralized zone as well as the immediate HW and FW rock mass can be classified as 'Poor' quality at Jason Main, according the Barton (2002) system.

Significantly less is known about ground conditions at Jason South. The limited information available suggests that the Jason South rock mass quality is likely better than Jason Main. Based on review of historic core photographs for select intervals of split core, it has been assumed for the PEA that the Jason South rock quality is similar to that of the Tom West Zone.

16.4.3 Stope Dimensions

Limiting stope dimensions for the LH stoping areas were estimated using the Potvin and Hadjigeorgiou (2001) empirical stope design method for the average rock mass conditions anticipated. Using a level height of 20 m, the maximum unsupported stope lengths and widths were estimated and provided as inputs to the mine design.

Empirical charts such as the Potvin and Hadjigeorgiou (2001) method were developed based on traditional open stoping methods where the stope will remain fully open during mining (ex. longhole open stoping). The use of such empirical methods to determine wall dimensions for the VCR and SLR mining areas would lead to overly conservative results given theses stopes will remain partially full during mining providing confinement to stope walls.

16.4.4 Ground Support

Based on the anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings, ground support recommendations were developed according to the Barton (2002) criteria. The Q-system also takes into account the life and use of the opening (ex. man-entry or equipment only) with the excavation support ratio (ESR) parameter. The ESR is used to adjust the design span which in effect imposes a higher factor of safety on critical structures with long life (ex. an underground nuclear power station with an ESR of 0.5 to 0.8) than on temporary tunnels (ex. temporary mine workings with an ESR rating of 2 to 5). The ground support recommendations include the following.

- Temporary and permanent waste development (5 to 6 m x 5 to 6 m):
  - 2.4 m long #7 resin bolts on 1.5 m ring spacing and 1.5 m within the ring with 6 gauge welded wire mesh in back to within 1.5 m of floor; and
  - Assume 15% of the total permanent waste development will require 5 cm of shotcrete in addition to bolting. No shotcrete in temporary waste development.
- Temporary ore development (5 m x 5 m):
  - 2.4 m long #7 resin bolts on 1.5 m ring spacing and 1.5 m within the ring with 6 gauge welded wire mesh in back to floor; and
  - Assume 50% of total sill development will require 6 cm of shotcrete in addition to the above for Jason Main zone sills. No shotcrete required in other deposit ore development.
- Tom West VCR top cuts (30 x 30 m):
  - 10 m long twin strand cables on 2 x 2 m spacing; and
  - 1.5 m split sets and mesh on back and walls to 1.5 m from floor.

Cable bolts spacing and lengths for the VCR 30 x 30 m top cuts were estimated using empirical methods from Hutchinson and Diederichs (1996).

16.4.5 Open Pit Slope Angles

Overall slope angles were estimated for the open pits using the Haines and Terbrugge (1991) empirical design chart which is based on the Laubscher (1990) mining rock mass rating (MRMR) system and the slope height. The method is commonly used for estimating slope angles for preliminary or conceptual project stages.

The RMR values summarized in Table 16-4 were discounted approximately 15% to account for the rock mass weathering, discontinuity orientation, stress and blasting factors incorporated into Laubscher's MRMR system. Based on the discounted MRMR values, the following maximum overall slope angles were estimated for the PEA:

- 45° for the Jason South pit slopes up to 100 m in height;
- 45° for Tom West pit slopes up to 100 m in height; and
- 42° for Tom West pit slopes between 100 m and 200 m in height.

16.5 Mine Design Parameters

16.5.1 Mine Dilution & Recovery

Underground mine dilution was estimated based on each stope's individual dimensions to estimate unplanned over break experienced during mining operations. The rock quality at Macmillan Pass is variable between mineralized zones and over break parameters selected for each mining method reflect this. Each stope received unique dilution parameters depending on the geotechnical conditions and orientation and sequence within the mine plan. Three types of dilution were evaluated for each stope:

- Floor Dilution – Mucking backfill from the stope below;
- Wall Dilution – Over breaking into adjacent stopes; and
- Hanging Wall / Footwall Dilution – Over breaking into the hanging wall and footwall;

Hanging wall and footwall dilution grade was assigned to all applicable stopes. Average dilution grades were derived for each zone by querying the resource block model for material inside the diluted stope shape, but outside the undiluted stope shape.

Open pit mine dilution was estimated based on the geometry of the resource, geotechnical considerations for wall sloughing, and the size of excavating equipment planned for mucking material.

Mine dilution parameters are listed in Table 16-5.

Table 16-5: Dilution Parameters by Mining Method

Mine Method	Average	JMZ	JST	TMZ	TEA
Open Pit	7%	10%	-	5%	-
Transverse Longhole	20%	-	-	-	20%
Vertical Crater Retreat	17%	-	19%	14%	-
Sub-Level Retreat	34%	36%	-	34%	31%
Alimak Stope	31%	-	31%	-	-
Wall Slashing	4%	3%	2%	3%	7%
Development	18%	-	18%	18%	-
Total	**20%**	**32%**	**19%**	**18%**	**19%**

Source: JDS (2018)

The Macmillan Pass mine plan does not mine stopes to the mineral boundary, as grade is often distributed from hanging wall (HW) to footwall (FW) with higher grade at the centre and lower grade along the contact. JDS queried the production stopes for overbreak material on the HW and FW to determine in-situ grades for dilution. Total combined dilution grades by zone are located in Table 16-6 below, inclusive of all mineral and waste contained within the mined dilution tonnes. Open pit mining did not assume dilution grade.

Table 16-6: Mine Dilution Grades

Dilution Grade	Total	JMZ	JST	TMZ	TEA
Ag (g/t)	10	1	20	6	23
Pb (%)	0.9	0.5	1.6	0.5	1.8
Zn (%)	1.7	2.2	1.6	1.5	1.5

Source: JDS (2018)

Mine recovery was calculated under the following mine assumptions:

- 95% mining recovery for all open pit mining;
- 95% mining recovery for all underground stopes in good ground;
- 85% mining recovery of crown pillar removal at the end of the mine life; and
- 85% recovery for all underground stopes in poor ground (Jason Main sub-level retreat).

16.5.2 Cut-off Grade Criteria

Cut-off grade parameters were prepared for each mineralized zone with key difference in estimated mine dilution, recovery, and sustaining capital cost. Metallurgical performance was assumed equal between all zones for cut-off grade preparation.

NSR cut-off grade calculation criteria are summarized in Table 16-7.

Table 16-7: Cut-off Grade Parameters

Zone	Units	TMZ	TEA	TSE	JMZ	JST
Metal Price						
Zn	$USD/lb	$1.17	$1.17	$1.17	$1.17	$1.17
Pb	$USD/lb	$0.99	$0.99	$0.99	$0.99	$0.99
Ag	$USD/oz	$16.95	$16.95	$16.95	$16.95	$16.95
Exchange Rate	$CAD/$USD	0.78	0.78	0.78	0.78	0.78
Zn Concentrate						
Zn Recovery	% Zn	89%	89%	89%	89%	89%
Ag Recovery	% Ag	22%	22%	22%	22%	22%
Zn Concentrate Grade	%	58%	58%	58%	58%	58%
Moisture Content	%	8%	8%	8%	8%	8%
Zn Minimum Deduction	%Zn/tonne	8%	8%	8%	8%	8%
Zn Payable	% Payable	85%	85%	85%	85%	85%
Ag Minimum Deduction	g/t Ag	93.31	93.31	93.31	93.31	93.31
Ag Payable	% Payable	70%	70%	70%	70%	70%
Zn Treatment Charge	$USD/dmt	$210.00	$210.00	$210.00	$210.00	$210.00
Ag Refining Charge	$CAD/Ag oz	$0.50	$0.50	$0.50	$0.50	$0.50
Transport Costs	CAD$/wmt	$217.37	$217.37	$217.37	$217.37	$217.37
Hg Penalty Base	%	0%	0%	0%	0%	0%
Hg Penalty per 0.01%	$USD	$1.75	$1.75	$1.75	$1.75	$1.75
Royalties	%NSR	0%	0%	0%	0%	0%
Pb Concentrate						
Pb Recovery	% Pb	75%	75%	75%	75%	75%
Ag Recovery	% Ag	59%	59%	59%	59%	59%
Pb Concentrate Grade	%	62%	62%	62%	62%	62%
Moisture Content	%	8%	8%	8%	8%	8%
Pb Minimum Deduction	%Pb/tonne	3%	3%	3%	3%	3%
Pb Payable	% Payable	95%	95%	95%	95%	95%
Ag Minimum Deduction	g/t Ag	50	50	50	50	50
Ag Payable	%	95%	95%	95%	95%	95%
Pb Treatment Charge	$CAD/dmt	$170.00	$170.00	$170.00	$170.00	$170.00
Ag Refining Charge	$/Ag oz	$1.25	$1.25	$1.25	$1.25	$1.25
Transport Costs	CAD$/wmt	$217.37	$217.37	$217.37	$217.37	$217.37
Royalties	%NSR	0%	0%	0%	0%	0%
Underground Cost Estimate						
Opex - Mining	$CAD/tonne	$43.00	$43.00	$43.00	$43.00	$43.00
Opex - Processing	$CAD/tonne	$21.28	$21.28	$21.28	$21.28	$21.28
Opex - G&A	$CAD/tonne	$11.08	$11.08	$11.08	$11.08	$11.08
Capex - Sustaining	$CAD/tonne	$17.42	$16.42	$16.28	$17.89	$16.28
Opex - Total	$CAD/tonne	$92.78	$91.78	$91.64	$93.25	$91.64
Mine Dilution	%	10%	11%	22%	22%	12%
Mine Recovery	%	95%	95%	86%	86%	95%
Underground Cut-Off						
Cut-Off - NSR	$CAD/tonne	$110.00	$110.00	$130.00	$130.00	$110.00
Cut-Off - Equivalent Zn	%	4.9%	4.2%	5.9%	6.5%	4.1%
Open Pit Cost Estimate						
Opex – Waste Mining	$CAD/tonne	$3.80	-	-	$3.80	-
Opex – Mineral Mining	$CAD/tonne	$3.80	-	-	$4.20	-
Estimated Strip Ratio	Wt : Ot	6.0	-	-	5.0	-
Opex - Processing	$CAD/tonne	$3.80	-	-	$4.20	-
Opex - G&A	$CAD/tonne	$11.08	-	-	$11.08	-
Opex - Total	$CAD/tonne	$58.96	-	-	$55.56	-
Mine Dilution	%	5%	-	-	10%	-
Mine Recovery	%	95%	-	-	95%	-
Open Pit Cut-Off						
Cut-Off - NSR	$CAD/tonne	$65.17	-	-	$64.33	-
Cut-Off - Equivalent Zn	%	2.9%	-	-	3.2%	-

Source: JDS (2018)

16.5.3 Underground Stope Criteria

Stope design criteria are summarized in Table 16-8.

Table 16-8: Production Stope Design Criteria (Max. Dimensions)

Mine Method	Stope Width (m)	Stope Height (m)	Stope Length (m)	Dip (°)
Transverse Longhole Stoping	10	20	20	45-90
Vertical Crater Retreat Stoping	20	40	30	45-90
Sub-Level Retreat Stoping	10	20	10	45-90
Alimak Longhole Stoping	20	60	10	45-90
Development Drifting	5	5	N/A	0-45

Source: JDS (2018)

16.5.4 Crown Pillar Design Criteria

Crown pillars will be left for removal at the end of the mine life to transition from open pit to underground mining. Crown pillars will vary in thickness from 15 to 30 m depending on the vein width mined.

16.5.5 Open Pit Design Criteria

Open pit bench design criteria are summarized in Table 16-9.

Table 16-9: Open Pit Design Criteria

Item	Overall Pit Slope (Degrees)	Bench Height (m)	Exclusions
Tom West	42	10	Streams & UG Workings
Jason Main	45	5	Cap Waste for UG Backfill needs

Source: JDS (2018)

16.6 Potentially Mineable Resource

The potentially mineable resource for Macmillan Pass is a product of multiple runs of the Vulcan Underground Stope Optimizer© and the Datamine Open Pit NPV Scheduler© software. Optimized designs were adjusted for various controls established in the design criteria to ensure final products are of sound and mineable quality.

Table 16-10 and Table 16-11 outlines the diluted, recoverable, mineable tonnage used for mine planning purposes.

Table 16-10: Mine Tonnes by Method

Item	Units	Total	Jason Main	Jason South	Tom West	Tom East
Open Pit						
Tonnes - Mineral	kt	**4,229**	1,448	-	2,781	-
Tonnes - Waste	kt	**20,934**	5,079	-	15,855	-
Grades - Silver	g/t	**27**	5.6	-	38.1	-
Grades - Lead	%	**2.9**	1.4	-	3.8	-
Grades - Zinc	%	**6.2**	5.9	-	6.4	-
Underground						
Tonnes - Mineral	kt	**28,427**	5,045	6,830	13,381	3,171
Tonnes - Waste	kt	**3,745**	847	807	1,277	814
Grades - Silver	g/t	**46**	1.6	73.0	35.2	103.7
Grades - Lead	%	**3.7**	1.2	5.4	2.8	7.9
Grades - Zinc	%	**5.2**	6.2	3.6	5.3	6.6
Total						
Tonnes - Mineral	kt	**32,656**	6,493	6,830	16,162	3,171
Tonnes - Waste	kt	**24,679**	5,926	807	17,132	814
Grades - Silver	g/t	**44**	2.5	73.0	35.7	103.7
Grades - Lead	%	**3.6**	1.3	5.4	2.9	7.9
Grades - Zinc	%	**5.3**	6.1	3.6	5.5	6.6

Source: JDS (2018)

Table 16-11: Mine Tonnes by Classification

Item	Units	Total	Jason Main	Jason South	Tom West	Tom East
Indicated						
Tonnes	kt	**8,523**	2,713	-	4,804	1,006
Grades - Silver	**g/t**	**24.8**	1.6	-	26.2	81.1
Grades - Lead	**%**	**23.7**	1.4	-	2.6	6.4
Grades - Zinc	**%**	**2.7**	6.3	-	5.5	6.5
Inferred						
Tonnes	kt	**24,133**	3,781	6,830	11,358	2,164
Grades - Silver	**g/t**	**50.1**	3.3	73.0	39.7	114.3
Grades - Lead	**%**	**30.0**	1.2	5.4	3.1	8.6
Grades - Zinc	**%**	**3.3**	5.9	3.6	5.5	6.6

Source: JDS (2018)

16.7 Mine Design

16.7.1 Open Pit Mine Design

For the TMZ and JMZ deposits, the ultimate pit shell limits were derived using NPV Scheduler software, utilizing the industry standard Lerch-Grossman algorithm, and input parameters described in this report. Open pit / underground cross-over (UGX) analysis were undertaken. The UGX optimization also takes into account the estimated UG mining cost and determines where the transition from OP to UG should occur.

The resultant pit shell for TMZ contain 2.8 Mt of mineralized material, 15.9 Mt of waste for an average strip ratio of 5.7:1. This pit shape represents the maximum NPV shell given the aforementioned parameters and constraints. At Tom West, the pit optimization was limited to a bounding area in order to maintain a safe working distance from all significant water ways.

The JMZ pit shell contains 1.5 Mt of mineralized material and 5.1 Mt of waste rock at an average strip ratio of 3.5:1. Given the current understanding and extent of potentially acid-generating mine waste, as well as the amount of waste needed for UG backfill, the Jason Main pit shell selection is purposely limited in size so as to limit the amount of waste that will be re-handled and placed back in the mined out pit at closure. As such, this selected ultimate pit shape (as labelled Pit 40) represents a smaller revenue shell as shown by the series of pit shells generated in the cross-over optimization in Table 16-12. A small satellite pit and narrow eastern extension of the pit shell were excluded from the final results at JMZ as they were deemed to be unrecoverable once practical mining considerations are taken into account (i.e. minimum mining widths, ramp accesses, etc.).

Open pit mining accounts for 13% or 4.2 Mt of the total 32.7 Mt of material mined and processed.

Table 16-12: Jason Main Pit Optimization Results

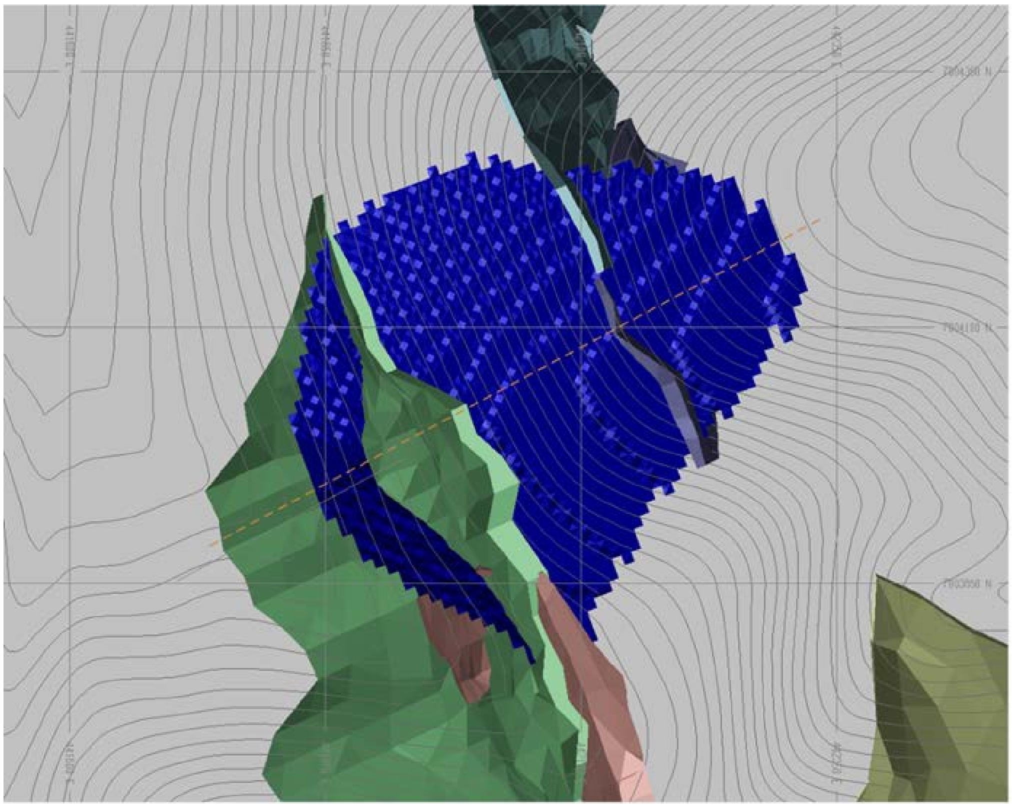
Pit #	RevFac	Life	Total Millfeed									Waste	Strip	Total	TCF*	NPV *
		(yrs)	(Mt)	Zn (%)	Pb (%)	Ag (g/t)	NSR ($/t)	Zn (M Lb)	Pb (M Lb)	Ag (M Oz)	NSR (M $)	(Mt)	Ratio	(Mt)	(M $)	(M $)
Pit 14	0.60	0.0	0.1	4.46	0.85	1.9	92	9	2	0.0	8	0.1	0.8	0.2	4.6	4.6
Pit 15	0.61	0.1	0.1	4.48	0.87	2.2	93	10	2	0.0	9	0.1	0.8	0.2	5.2	5.2
Pit 16	0.62	0.1	0.1	4.52	0.90	3.1	94	15	3	0.0	14	0.1	0.9	0.3	7.9	7.9
Pit 17	0.63	0.1	0.2	4.54	0.89	3.4	95	16	3	0.0	15	0.2	1.0	0.3	8.8	8.8
Pit 18	0.64	0.1	0.2	4.59	0.88	3.6	95	21	4	0.0	20	0.2	1.2	0.4	11.1	11.1
Pit 19	0.65	0.1	0.2	4.62	0.88	3.7	96	22	4	0.0	21	0.3	1.2	0.5	12.0	12.0
Pit 20	0.66	0.1	0.3	4.65	0.87	4.0	96	27	5	0.0	25	0.3	1.3	0.6	14.4	14.3
Pit 21	0.67	0.2	0.3	4.72	0.85	4.1	97	31	6	0.0	29	0.4	1.4	0.7	16.3	16.3
Pit 22	0.68	0.2	0.3	4.76	0.85	4.2	98	35	6	0.0	32	0.5	1.5	0.8	18.4	18.4
Pit 23	0.69	0.2	0.4	4.83	0.83	4.5	99	38	7	0.1	35	0.5	1.5	0.9	20.3	20.2
Pit 24	0.70	0.2	0.4	4.86	0.82	4.6	99	42	7	0.1	39	0.6	1.6	1.0	22.2	22.1
Pit 25	0.71	0.2	0.4	4.90	0.83	4.7	100	47	8	0.1	43	0.7	1.7	1.2	24.8	24.7
Pit 26	0.72	0.3	0.5	4.94	0.82	4.8	101	52	9	0.1	48	0.9	1.8	1.3	27.4	27.2
Pit 27	0.73	0.3	0.5	4.97	0.84	4.9	102	57	10	0.1	53	1.0	1.9	1.5	30.2	30.0
Pit 28	0.74	0.3	0.6	5.02	0.85	5.1	103	62	11	0.1	58	1.1	2.0	1.7	33.1	32.8
Pit 29	0.75	0.3	0.6	5.04	0.85	5.0	103	64	11	0.1	60	1.2	2.0	1.7	34.1	33.8
Pit 30	0.76	0.3	0.6	5.06	0.86	5.2	104	71	12	0.1	66	1.4	2.2	2.0	37.2	36.9
Pit 31	0.77	0.4	0.7	5.09	0.88	5.2	105	77	13	0.1	72	1.6	2.3	2.3	40.9	40.5
Pit 32	0.78	0.4	0.8	5.19	0.96	5.7	108	89	16	0.1	84	1.8	2.4	2.6	48.5	48.0
Pit 33	0.79	0.5	0.9	5.25	1.01	5.8	110	99	19	0.2	94	2.1	2.5	3.0	54.5	53.8
Pit 34	0.80	0.5	0.9	5.29	1.05	5.9	111	107	21	0.2	102	2.3	2.5	3.2	59.4	58.7
Pit 35	0.81	0.6	1.0	5.36	1.12	6.1	113	124	26	0.2	119	2.9	2.7	3.9	69.6	68.6
Pit 36	0.82	0.6	1.2	5.40	1.15	6.1	115	139	30	0.2	133	3.4	2.9	4.6	77.8	76.6
Pit 37	0.83	0.7	1.2	5.41	1.17	6.1	115	149	32	0.2	143	3.9	3.1	5.1	83.1	81.7
Pit 38	0.84	0.7	1.3	5.46	1.21	6.0	116	156	35	0.2	151	4.1	3.1	5.4	88.2	86.7
Pit 39	0.85	0.8	1.4	5.49	1.23	5.9	117	168	38	0.3	163	4.6	3.3	6.0	94.7	93.0
Pit 40	**0.86**	**0.8**	**1.5**	**5.53**	**1.28**	**5.8**	**119**	**182**	**42**	**0.3**	**177**	**5.1**	**3.4**	**6.6**	**102.8**	**100.8**
Pit 41	0.87	0.8	1.5	5.56	1.30	5.6	120	189	44	0.3	185	5.4	3.5	7.0	107.5	105.4
Pit 42	0.88	0.9	1.6	5.58	1.32	5.6	120	203	48	0.3	198	6.1	3.7	7.7	114.5	112.0
Pit 43	0.89	0.9	1.7	5.61	1.34	5.5	121	211	51	0.3	206	6.5	3.8	8.2	119.5	116.8
Pit 44	0.90	0.9	1.7	5.62	1.35	5.4	121	215	51	0.3	210	6.6	3.8	8.4	121.3	118.5
Pit 45	0.91	1.0	1.9	5.63	1.36	5.3	121	230	56	0.3	225	7.6	4.1	9.4	128.6	125.4
Pit 46	0.92	1.0	1.9	5.64	1.36	5.3	122	236	57	0.3	230	7.9	4.2	9.8	131.3	128.0
Pit 47	0.93	1.1	2.0	5.68	1.38	5.2	123	247	60	0.3	242	8.4	4.3	10.4	137.4	133.8
Pit 48	0.94	1.1	2.0	5.71	1.40	5.1	123	257	63	0.3	252	9.1	4.4	11.1	143.0	139.1
Pit 49	0.95	1.1	2.1	5.71	1.40	5.1	123	263	64	0.3	258	9.5	4.5	11.6	145.2	141.1
Pit 50	0.96	1.2	2.2	5.73	1.41	5.1	124	274	67	0.4	269	10.2	4.7	12.4	150.6	146.2
Pit 51	0.97	1.2	2.2	5.74	1.41	5.0	124	281	69	0.4	275	10.6	4.8	12.8	153.8	149.2
Pit 52	0.98	1.3	2.3	5.76	1.42	4.9	125	294	73	0.4	288	11.6	5.0	13.9	159.6	154.5
Pit 53	0.99	1.3	2.4	5.77	1.42	4.9	125	302	74	0.4	296	12.1	5.1	14.5	163.0	157.7
Pit 54	1.00	1.3	2.4	5.80	1.43	4.8	125	313	77	0.4	307	12.9	5.3	15.4	168.3	162.6

*Note that TCF and NPV values do not account for any initial or sustaining capital costs
Source: JDS (2018)

Table 16-10 summarizes the combined tonnages and grades contained within the planned ultimate starter pit shells for the two deposits. Given the relatively small size of both of these starter pits, no internal phase designs were considered for this study.

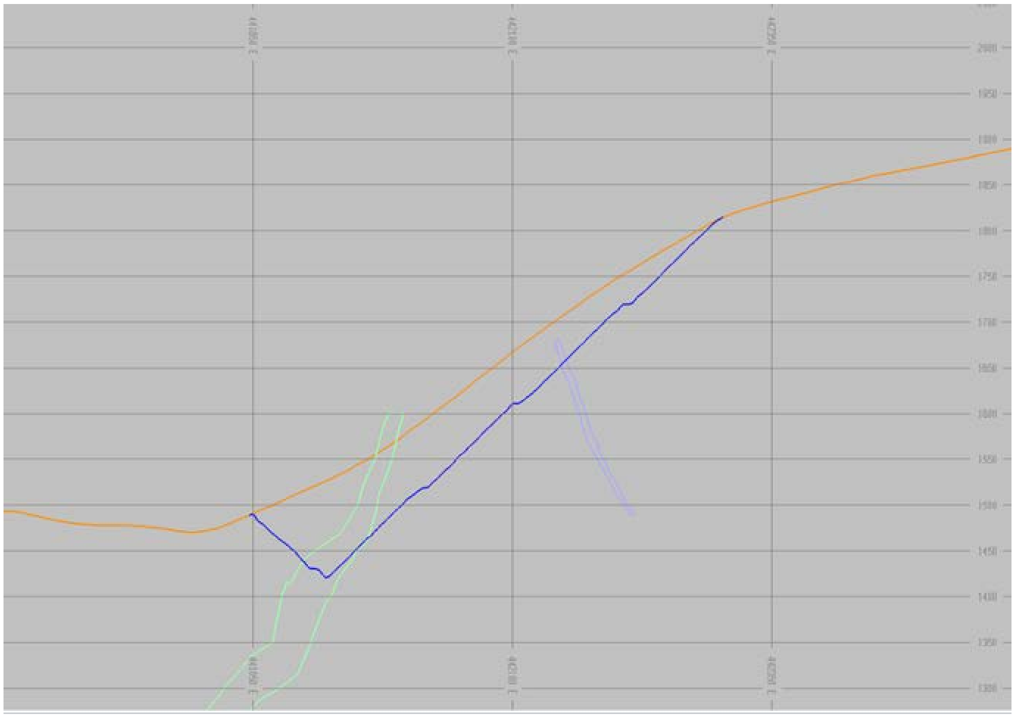
Table 16-12 to Table 16-15 represent plan and section views of the planned ultimate pit shapes.

Figure 16-12: Tom West Pit Shell - Plan View



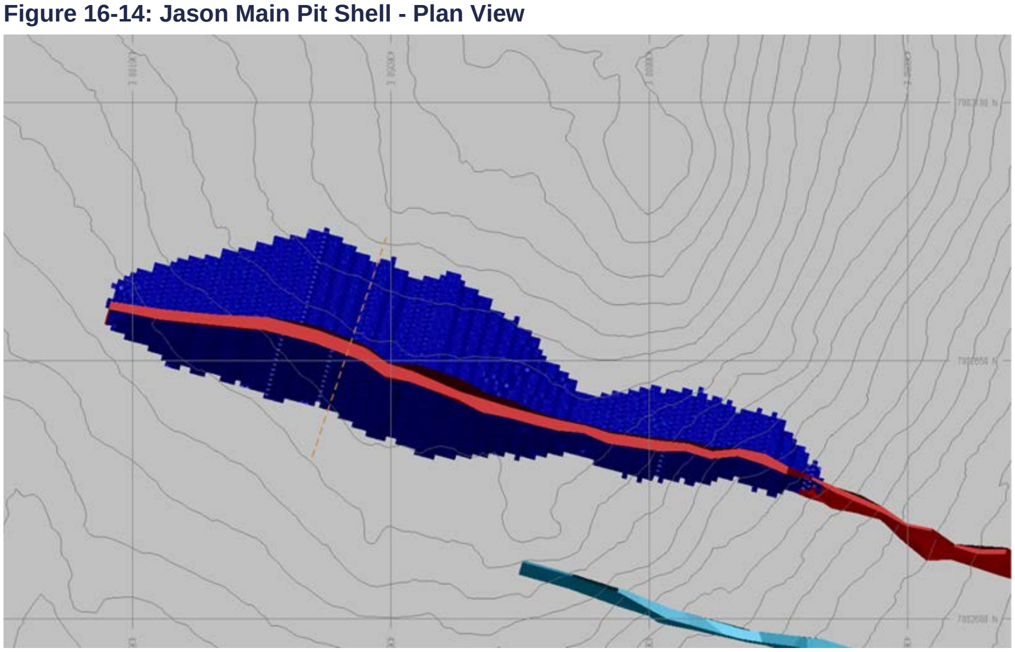
Source: JDS (2018)

Figure 16-13: Tom West Pit Shell - Typical Cross Section



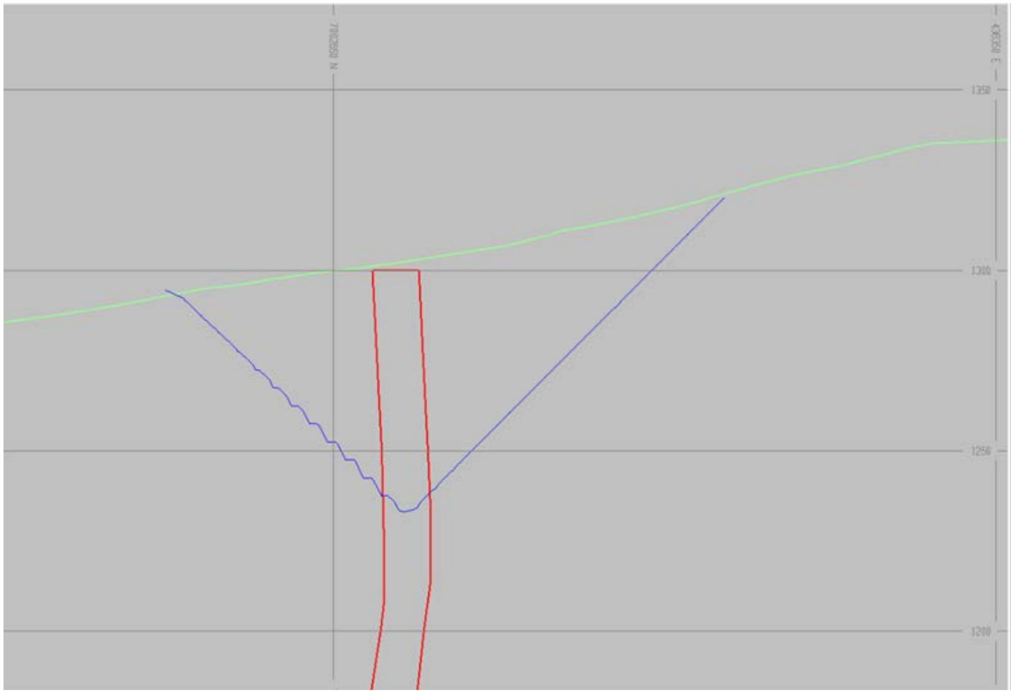
Source: JDS (2018)

Figure 16-14: Jason Main Pit Shell - Plan View



Source: JDS (2018)

Figure 16-15: Jason Main Pit Shell - Typical Section



Source: JDS (2018)

16.7.2 Underground Mine Design

16.7.2.1 Mine Access

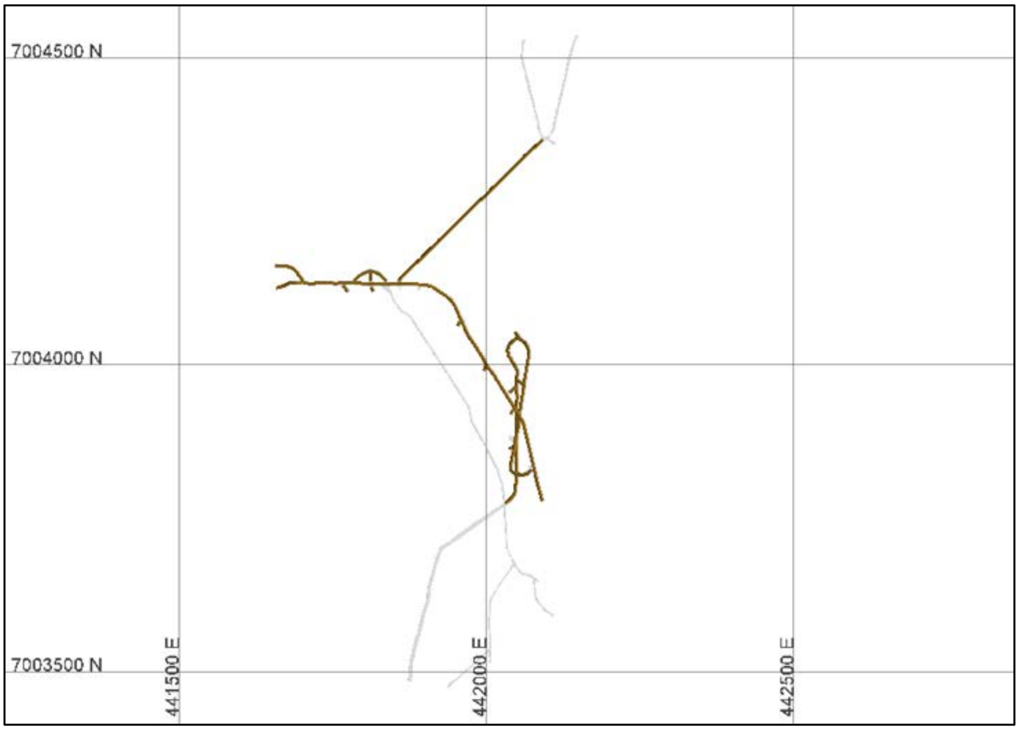
The Macmillan Pass deposits consist of mineable resources extending more than 500 vertical metres. The underground mine will be accessed from a series of portals, both new and existing.

A 3,420 m long exploration decline currently extends from a portal at Tom West at 1,435 m elevation, well situated near the top of Tom West and near the bottom of Tom East. The existing decline was driven in stages between 1969 and 1982 and permitted to flood in 1988. The decline was driven at approximate dimensions and grade of 4.5 m x 4.5 m and -15% respectively.

The life of mine plan at Tom includes the rehabilitation of the Tom portal and decline for access to Tom West and Tom East production horizons. Portions of the existing development will be slashed in the walls and back to accommodate the haulage fleet selected for the mine plan. 1.9 km of the available 3.4 km drift will be rehabilitated and used as part of the mine plan. As space for equipment and material storage become an issue additional drifts may be rehabilitated for use. During mine operation the existing Tom portal will be used to provide ventilation to the underground workings, as well as access for haul trucks bringing cemented waste rock fill to empty stope voids.

Figure 16-16 below depicts the total existing development at Tom, with planned rehabilitation (brown) and remaining (grey) to be left untouched.

Figure 16-16: Existing Development at Tom. Brown to be Re-habilitated



Source: JDS (2018)

One additional portal is planned at the Tom plant site. This portal will provide access to a 2.2 km incline driven to the 1,280 m elevation, terminating below the historic development. This incline will serve multiple purposes including:

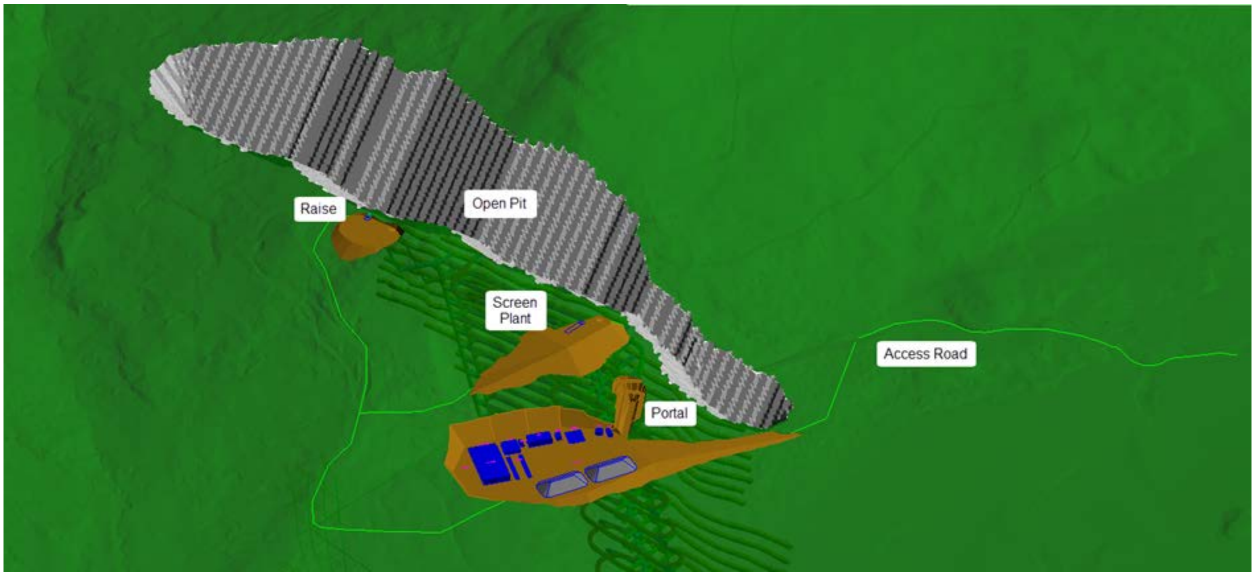
- Man and material entry to the mine;
- Conveyor haulage from underground crusher to the plant site direct; and
- Natural drainage for all development above the 1,280 m elevation.

The Tom incline is currently envisioned to be driven 5.5 m high by 6.5 m wide to accommodate a suspended 2.0 m conveyor and 50 tonne haul trucks, capable of transporting mineral and waste from the mine. The incline will be driven at +1.5% to promote natural dewatering of the mine above 1,280 m elevation. With average expected groundwater inflow of 500 gpm (31.5 L/s) this will alleviate what would otherwise be a large pumping requirement. The Tom incline will connect to the existing Tom infrastructure to establish ventilation circuits and access to production levels.

All Jason resources will be accessed from a single portal driven into the hillside at 1,180 elevation, south-east of the Jason Main zone. The portal will service a 5.5 m wide by 5.0 m high decline that feeds a spiral ramp connecting all production horizons in Jason Main and Jason South.

Figure 16-17 depicts the Jason portal location adjacent to the open pit.

Figure 16-17: Jason Portal Location



Source: JDS (2018)

16.7.2.2 Production Rate Selection

The Macmillan Pass mine plan has been sized for a maximum 5,000 t/d operation. Cycle times of the different mining methods were considered along with the tonnes per vertical metre and layout of the mine in determining the production rate.

The mine plan was created using scheduling software Minemax iGantt©. The scheduling rates used are shown in Table 16-13.

Table 16-13: Scheduling Rates Used for Mine Scheduling

Heading	Units	Rate
Lateral Development		
Rehabilitation	m/d	8.0
Ramp / Incline / Access / Footwall / Cross-cut	m/d	4.0
Auxiliary / Sump / Electrical / Vent Drives	m/d	4.0
Shop	m/d	2.0
Vertical Development		
Muck Pass	m/d	2.0
Vent Raise	m/d	2.0
Escape Way	m/d	2.0
Alimak Stope Raise	m/d	2.0
Mine Production		
Longhole Stope	t/d	200
Sub-Level Retreat	t/d	450
Wall Slash	m/d	8.0
Alimak Stope	t/d	285
Vertical Crater Retreat	t/d	575

Source: JDS (2018)

16.7.2.3 Production Sequencing

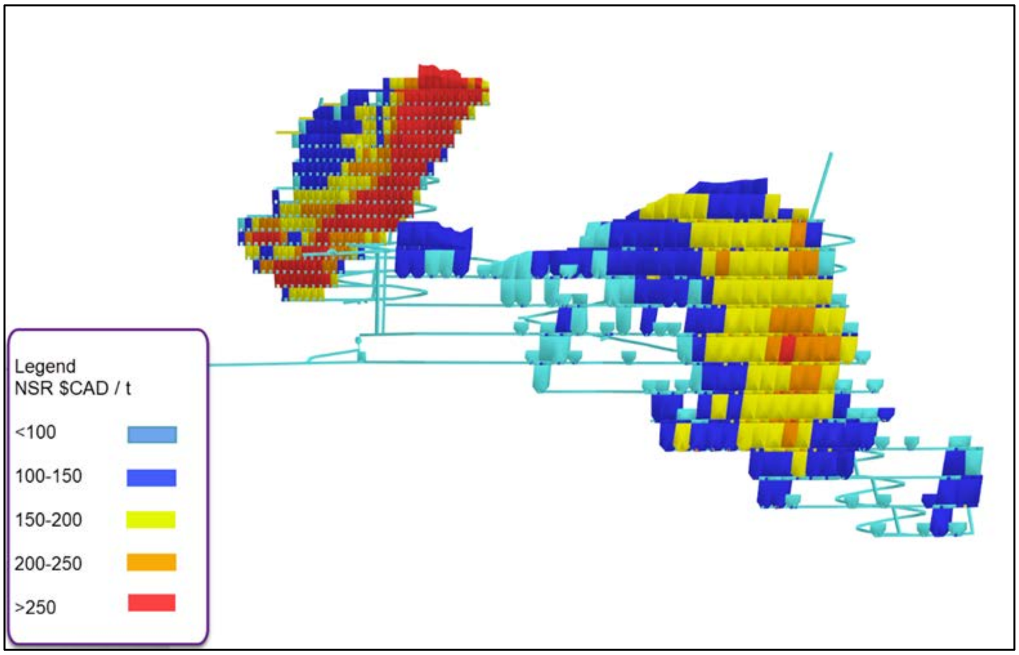
Mineral zones were sequenced in order to prioritize highest grade, lowest mine operating costs, and existing access development.

Stope sequencing is dependent on mining method. All stopes aside from those in Jason Main are mined from bottom up in a primary / secondary fashion, whereby a primary stope is mined and backfilled before the adjacent stopes are mined. Two primary stopes stacked vertically will be mined and backfilled before the first secondary stope is mined, ensuring that the top sill of the secondary stope is encased in fully backfilled stopes on both sides.

Jason Main is mined from top to bottom in retreat fashion, whereby stopes are mined in series one after another. Stopes are staggered such that the bottom sill is always on top of in-situ ground while the upper sill is buried in loose fill.

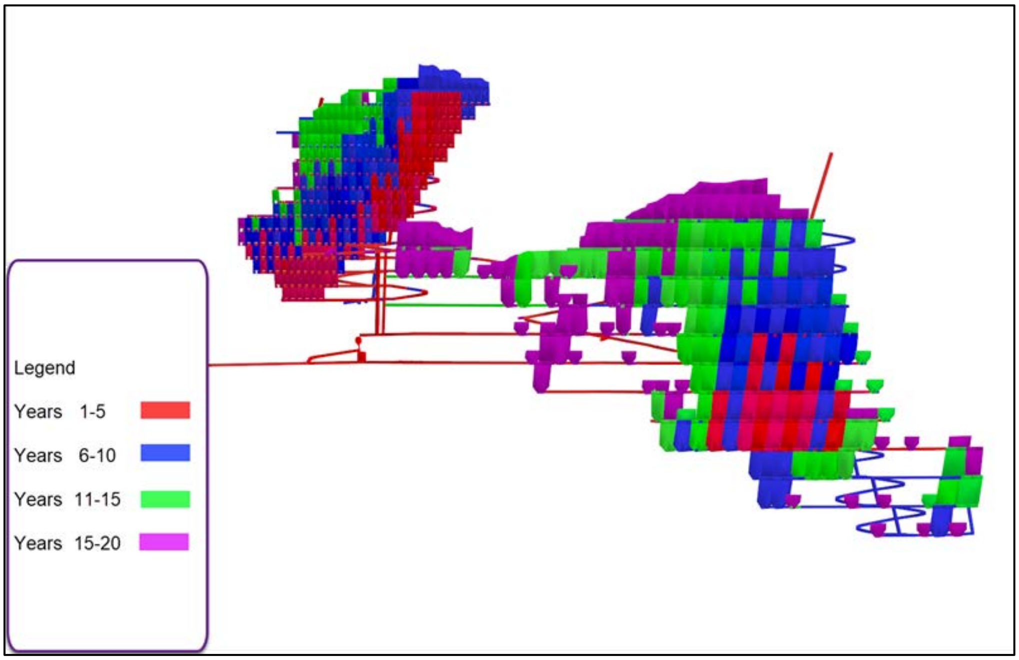
Figure 16-18 through Figure 16-21 depict the mine plans in both NSR value and annual schedule.

Figure 16-18: Tom Zone Mine Plan by NSR $/t



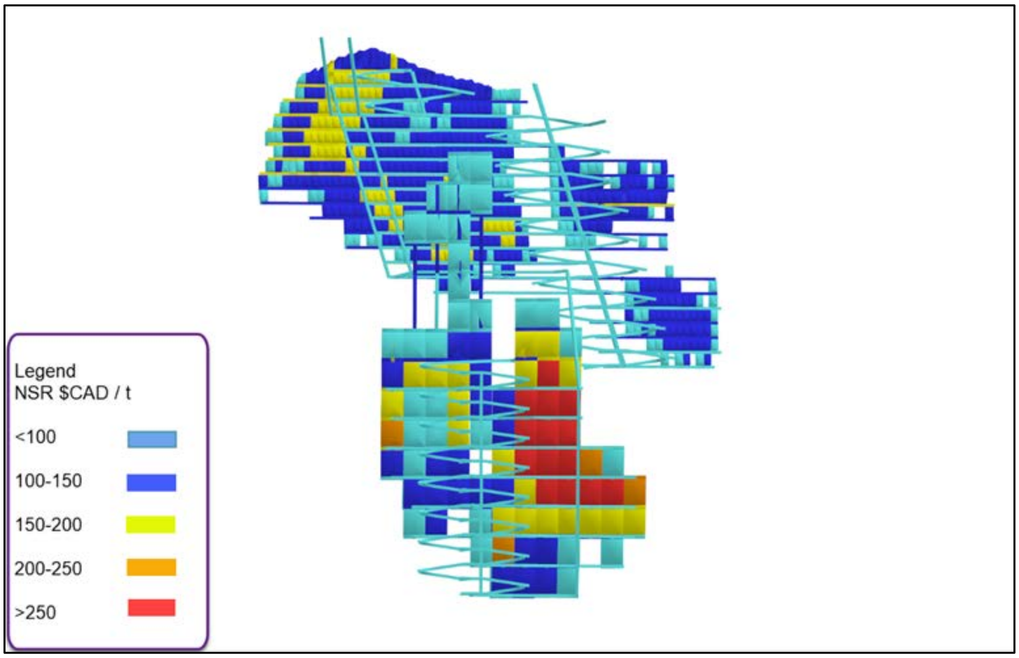
Source: JDS (2018)

Figure 16-19: Tom Zone Mine Plan by Schedule



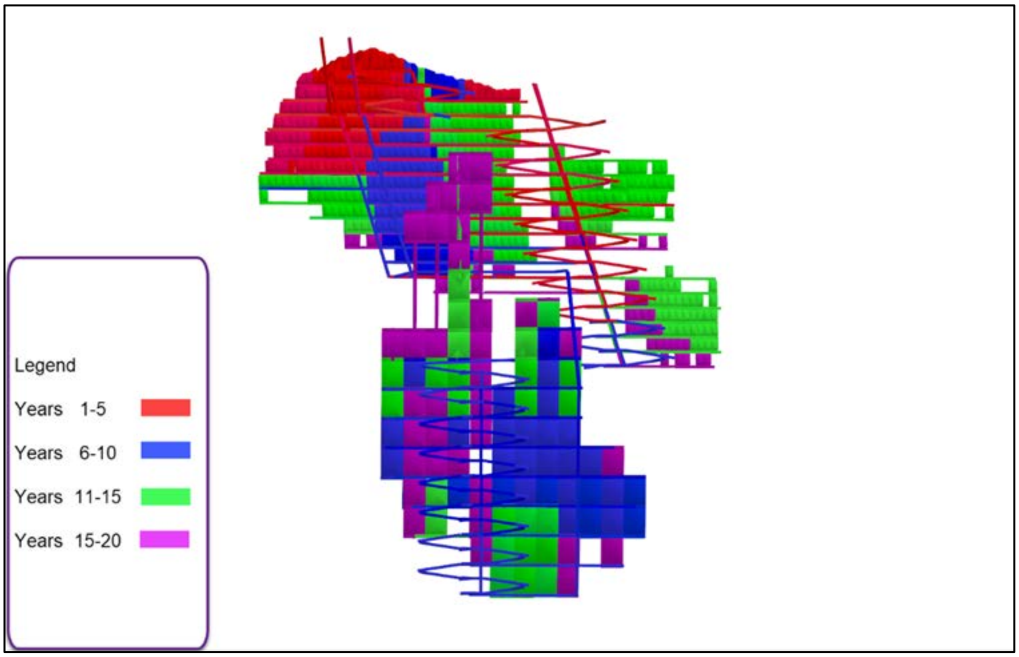
Source: JDS (2018)

Figure 16-20: Jason Zone Mine Plan by NSR $/t



Source: JDS (2018)

Figure 16-21: Jason Zone Mine Plan by Schedule



Source: JDS (2018)

16.7.2.4 Underground Mine Development

16.7.2.4.1 Lateral Development

The ramp is envisioned to be designed at a 5.5 m x 5.0 m arched profile to accommodate fully loaded 50 t haul trucks and 42” (1.0 m) round vent ducting. Footwall drifts would also be driven at 5.5 m x 5.0 m to allow haul truck access to the stope cross-cuts. Cross-cuts would be driven flat back style 5.0 m x 5.0 m to accommodate remote LHD entry.

16.7.2.4.2 Vertical Development

Muck passes at 2.4 m by 2.4 m profile are planned in both the Tom and Jason zones. A grizzly would be installed at the top of the muck pass to remove oversize blasted material. The bulk of blasted material would be sent down the muck passes, either direct to the loading pocket at the Tom underground crusher, or to loading stations where haul trucks would be loaded and haul to the crusher or out of the mine.

Ventilation raises at 4.0 m by 4.0 m profile would be established to provide fresh air for each of the mining zones. All raises would be driven with the use of contract Alimak raise climbers. Where raises are short enough, LH drop raises would be utilized.

Total lateral and vertical development over the mine life is summarized in Table 16-14.

Table 16-14: Development Schedule

Development Type	Lateral				Vertical		
	Waste CAPEX	Waste OPEX	Mineral OPEX	Total	Waste CAPEX	Mineral OPEX	Total
Year	m	m	m	m	m	m	m
-1	-	-	-	-	-	-	-
1	5,110	1,093	1,438	7,641	263	-	263
2	7,436	343	-	7,779	1,081	-	1,081
3	4,269	955	2,445	7,669	411	-	411
4	3,539	1,116	2,968	7,623	255	-	255
5	3,014	1,020	3,567	7,601	342	-	342
6	4,650	1,025	2,000	7,675	483	-	483
7	3,316	1,037	3,273	7,626	513	132	645
8	1,503	1,731	4,323	7,557	124	-	124
9	4,134	821	2,666	7,621	677	-	677
10	2,836	1,434	3,303	7,573	273	-	273
11	23	276	2,203	2,502	-	-	-
12	44	528	2,516	3,088	-	-	-
13	-	189	2,696	2,885	-	-	-
14	659	424	2,379	3,462	-	-	-
15	143	527	3,542	4,212	84	333	417
16	69	1,388	4,584	6,041	21	833	854
17	-	-	601	601	-	-	-
18	287	4	282	573	-	-	-
Total	**41,032**	**13,911**	**44,786**	**99,729**	**4,527**	**1,298**	**5,825**

Source: JDS (2018)

16.8 Underground Mine Production

16.8.1 Mine Operations

Longhole stoping would be the main mining method at Macmillan Pass. Individual stope tonnages range from 3,800 t up to 116,000 t.

Longhole drilling of mainly down holes with 89 mm diameter is planned at sub-level spacing of 20 m to 30 m. Some stoping would include drilling of up-holes in retreat mining. Slots would be developed by drop raising. Emulsion would be used for longhole blasting.

The stopes would be mucked with remote controlled 14 t (7.5-yard) LHDs. Mineralized material would be transported by the LHDs to level remucks and/or directly to the muck pass. Muck would be loaded from the level remuck and/or muck pass into haul trucks and transported to the surface stockpile. Backfill would be dumped by truck into the stope.

Backfill would consist of cemented rock fill (CRF) for primary stopes and unconsolidated waste rock for secondary stopes. All of the development waste generated from underground mining activities would be used as mine backfill. No development waste would remain on surface at the end of the mine life. The remaining backfill deficit is planned to be sourced from the open pit waste stockpiled during operations.

Mined rock, rather than paste or sand tailings was selected to minimize the amount of waste rock stored on surface at the end of the mine life.

CRF consists of waste rock mixed with cement slurry. Cement slurry would be produced by a portable CRF batch plant, consisting of a colloidal batch mixer and spray bar equipment. The slurry would be sprayed onto the waste material in the truck before being dumped into the empty stope.

A cement binder content of 5% was assumed for primary longhole stoping. The quantities of cement required in the backfill mixes are estimates only, since no test work has been done to date. Backfill testing is recommended for future studies to define the optimum mix recipe.

Table 16-13 and Table 16-14 outline the schedules for mine production and backfill placement.

16.9 Mine Services

16.9.1 Mine Ventilation

Airflow requirements were estimated based on Canadian diesel ventilation regulations. Where the engine model could not be sourced in the regulations, the ventilation rate of 100 cfm/hp (0.06 m^3/s per kW) was used. The ventilation requirement was then multiplied by the overall equipment utilization and the estimated diesel engine utilization. Airflow requirements of the underground equipment fleet are shown in Table 16-15.

Table 16-15: Ventilation Requirements

Item	Max Quantity	Engine Power (kW)	Engine Utilization (%)	Total Power (kW)	CANMET Ventilation (CFM)	Total Ventilation (CFM)
Truck (50t/20.0m^3)	10	515	85%	4,374	32	276
LHD (6.7t/3.0m^3)	2	150	85%	255	9	16
LHD (14t/5.4m^3)	10	256	85%	2,174	16	137
Jumbo - 2 Boom	4	52	30%	63	6	8
Bolter	2	52	30%	31	6	4
Explosives Truck	2	110	30%	66	7	4
Longhole Drill	4	120	30%	144	8	9
DTH Drill	3	120	30%	108	8	7
Jackleg / Stoper	6	-	30%	-	-	-
Scissor Lift	1	96	30%	29	6	2
Shotcrete + Transmixer	1	110	30%	33	7	2
Grout Pump	1	-	30%	-	-	-
Personnel Carrier	1	110	30%	33	7	2
Fuel/Lube Truck	2	110	30%	66	7	4
Boom Truck	2	110	30%	66	7	4
Electrician Truck	2	118	30%	71	9	5
Grader	2	292	30%	175	19	11
Utility Vehicle	2	16	30%	10	1	1
Backhoe	1	38	30%	11	3	1
Telehandler	1	38	30%	11	3	1
Supervisor Truck	3	118	30%	106	9	8
Mine Wide Ventilation Required (m^3/s)						494
Misc. Additional Headings (m^3/s)						38
Ventilation Leakage @ 10% (m^3/s)						53
Total Mine Ventilation (m^3/s)						586

Source: JDS (2018)

The above ventilation requirements is the maximum required between both Tom and Jason zones. As each zone will be serviced by separate ramps and ventilation raises, the peak requirements for each are estimated to be 390 m^3/s and 310 m^3/s respectively.

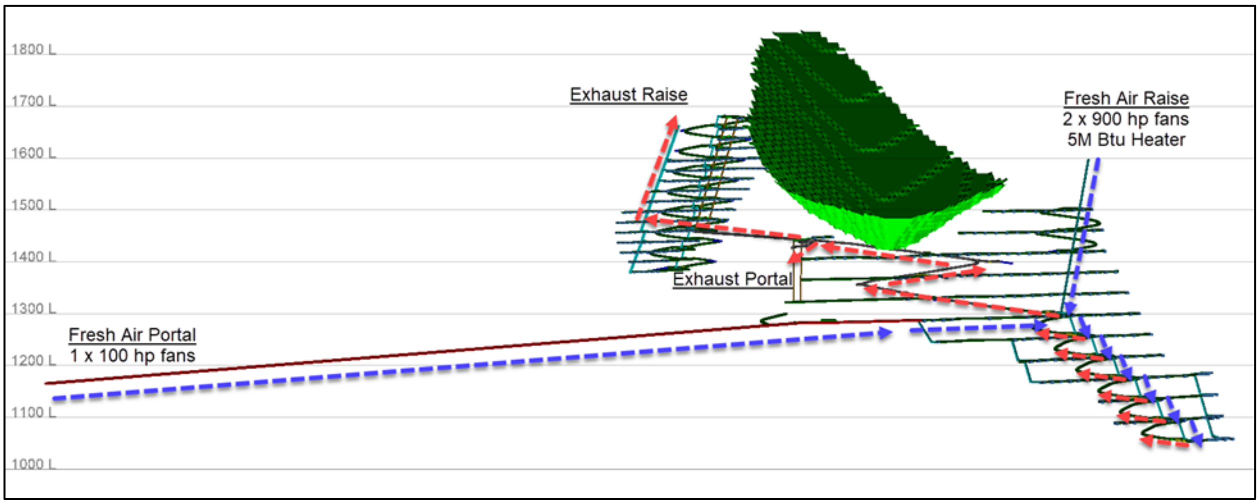
First principles ventilation calculations were used to estimate power requirements for the ventilation network. Where possible fans were planned with multiples in parallel rather than single larger fans to aid in future servicing and replacement.

Tom underground workings will receive fresh air from a raise with 2 x 670 kW fans mounted in parallel, providing 390 m^3/s at peak operation, supplemented by 1 x 75 kW fan installed on the conveyor drive to maintain positive pressure to the mine underground workings. Exhaust air will exit the Tom workings through the existing portal and return air raise driven in Tom East. As the mine is developed it may be possible to change direction of flow seasonally to work with natural ventilation offered by ambient temperature differences between portal elevations.

Jason underground workings will receive fresh air from a raise with 2 x 620 kW fans mounted in parallel, providing 310 m^3/s, supplemented by 100 m^3/s of fresh air naturally drawn down the main ramp. Two 700 kW fans mounted in parallel on a return air raise will draw 410 m^3/s of exhaust air from the mine at peak operation.

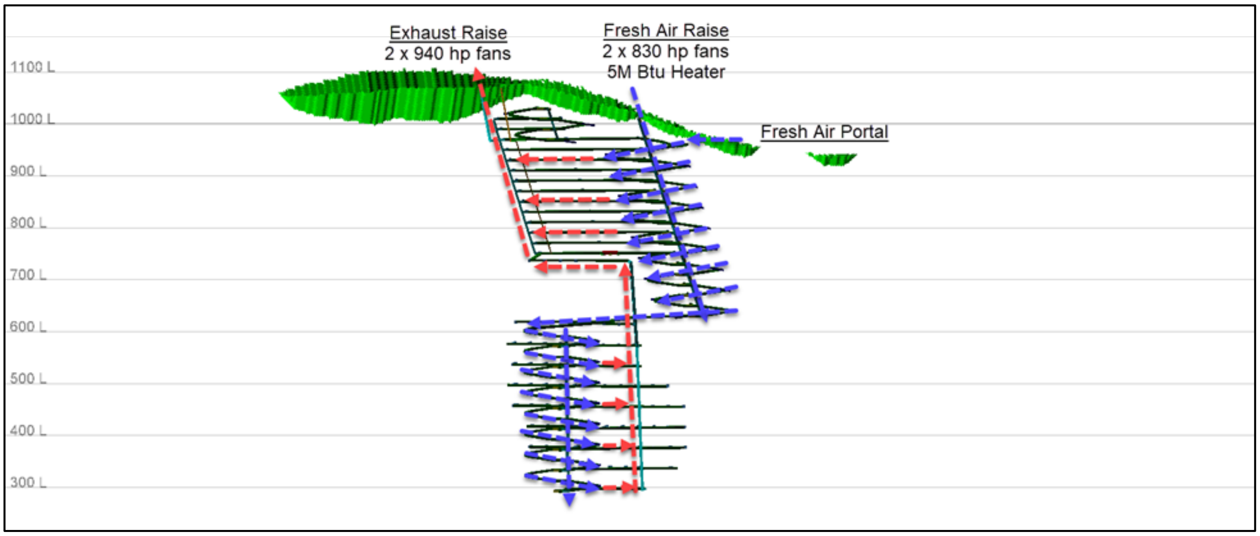
Several auxiliary ventilation raises are planned to distribute fresh air through the mine workings with 100 hp and 37 kW booster fans installed on each working level. Development fans will be used in dead headings such as ramp advancement to ensure all working faces are ventilated.

Figure 16-22: Tom Ventilation Schematic



Source: JDS (2018)

Figure 16-23: Jason Ventilation Schematic



Source: JDS (2018)

16.9.2 Mine Air Heating

Mine air would be heated to a minimum +1.5°C by a direct-fired propane heater located at the fresh air raises of both Tom and Jason. The air would be pulled into the heater drift by the main ventilation fan.

Intake air would require heating to prevent water from freezing underground and to provide acceptable working conditions while mining is ongoing. A 5 M Btu heater would be required at Tom and a 4 M Btu heater at Jason.

Heating calculations were based on average monthly temperatures collected at the site weather station. It was estimated that an average 0.5 M m^3 of propane would be required throughout the year.

16.9.3 Electrical Power

The majority of electrical power consumption at the mine would arise from:

- Main and auxiliary ventilation fans;
- Mine air compressors;
- Batch plant;
- Drilling, explosives loading and ground support equipment;
- Dewatering pumps; and
- Refuge stations.

High-voltage cables would enter the mine via the two portals and would be distributed to electrical sub-stations near the mining zones. High-voltage power would be delivered at 4160 V and reduced to 600 V at electrical sub-stations.

Total electrical power consumption for underground mining was estimated at 41,894,000 kWh/a during production.

16.9.4 Compressed Air

Compressed air would be required for longhole drills and face pumps. Compressed air would be provided by two stationary 150 kW compressors at each Tom and Jason, with one operating and one on standby at each location. Reticulation of compressed air through the mine would be in 150 mm diameter pipes.

16.9.5 Service Water Supply

Service water for drilling, dust control, washing and fire suppression would be sourced from a sump at the top of the underground workings and distributed in 50 mm diameter steel piping.

16.9.6 Dewatering

Groundwater inflows into the mine will vary throughout the year. Increased flow rates can be expected during the snowmelt in spring. An average groundwater inflow of 500 gpm (31 L/s) has been assumed for each Tom and Jason zone for mine water management exercises.

Tom Zone

Tom zone contains over 3 km of flooded mine workings that would be dewatered to commence mine development. Old remuck bays would be converted to small sumps during access development. Small portable 15 kW submersible pumps would be installed in these sumps and would discharge into steel pipes with 150 mm diameter while a 75 kW barge pump is pushed down ramp to dewater the existing workings. Once the conveyor drive is completed it would naturally dewater all working levels above 1,282 masl. For workings below 1,282 masl sump stations would be established on each level to direct water to 1,282 level during development, and a permanent sump on the lowest level would be established with three x 37 kW pumps working in parallel.

Jason Zone

Jason zone does not have any existing workings. Three permanent sump stations are envisioned throughout the Jason mine to leapfrog 31 L/s to surface. Sump stations contain two to three x 37 kW pumps working in parallel each. During ramp development temporary sumps would be established on the down ramp side of each level to catch inflow on the working level before flowing down ramp. These temporary sumps would direct water to the nearest permanent sump until the level is fully mined and dewatered.

Face drilling equipment will carry portable pumps to keep the face clear during development. Other small pumps will be installed in underground infrastructure such as shops and crusher chambers.

In pit dewatering pumps would be used during open pit operation, and prior to the removal of any crown pillars between open pit and underground mining areas.

16.9.7 Explosives Storage and Handling

Primary explosives storage magazines would be located on surface for open pit mining and staging to secondary underground magazines. Secondary magazines would be located underground to provide explosives storage for up to seven days. Bulk explosives and detonators would be stored in two separate facilities.

Bulk emulsion would be used as the major explosive for mine development and production.

Explosives handling, loading, and detonation would be carried out by trained and authorized personnel.

Typically, underground operations of this rock type require powder factors of approximately 1.0 kg/t for successful blasting with good fragmentation in development drifting, and 0.5 to 0.8 kg/t in longhole stoping.

16.9.8 Fuel Storage and Distribution

Initially, haul trucks, LHDs and auxiliary mobile equipment would be re-fueled on surface at a fuel station from a 15,000 L Enviro-Tank located close to the lower portal. Drilling equipment would be re-fueled underground with a fuel / lube truck. Once the mine is developed a secondary fuel station would be established near the underground work shop at both Tom and Jason. Open pit and other surface equipment will be fueled on surface, either at the fuel station located at the mill facility, or the satellite fuel stations at each Tom and Jason portal.

16.10 Mine Equipment

The required underground mobile equipment was determined based on the selected mining methods, mine production rate and geometry of the mine workings. Scheduled quantities of work in combination with cycle times, productivities, availabilities, and efficiencies formed the basis to determine the fleet size.

Table 16-16 summarizes the underground mobile fleet. Open pit mining equipment is assumed contractor operated and equipment fleet table excludes contractor mining equipment.

Table 16-16: Mobile Mine Equipment Requirements

Description	Maximum Required
Underground Equipment	
Truck (50 t/20.0 m^3)	10
LHD (6.7 t/3.0 m^3)	2
LHD (14 t/5. 4m^3)	10
Jumbo - 2 Boom	4
Bolter	2
Longhole Drill	4
DTH Drill	3
Large Explosives Truck	2
Scissor Lift	1
Shotcrete + Transmixer	1
Jackleg / Stoper	6
Grout Pump	1
Personnel Carrier	1
Fuel / Lube Truck	2
Boom Truck	2
Electrician Truck	2
Grader	2
Utility Vehicle	2
Backhoe	1
Telehandler	1
Mechanics Truck	2
Supervisor Truck	3
Surface Equipment	
FEL (11.5 m^3, WA900)	1
Surface Truck (90.0t , 777)	3

Source: JDS (2018)

Haulage requirements for LHDs and trucks were estimated for mineralized material, waste and backfill. At Jason, mineralized material is hauled to a stockpile near the portal, where it is re-handled and loaded into surface haul trucks for transportation to the processing plant. At Tom, mineralized material is crushed underground and conveyed to surface directly to the mill stockpile. Open pit trucks and loaders would be utilized for any surface material movement beyond the underground mine stockpiles at the portal, which includes mineral and waste stockpiling, as well as managing screen plant products for backfill preparation.

16.11 Mine Personnel

The Macmillan Pass underground mine department would employ 181 people during mine development and ramp up. Once in full production there would be a peak requirement of 327 mine employees rotating on two different rosters. All labour and staff would be working on a two weeks on two weeks off basis.

Mine personnel will reside in camp on site and be transported to and from the Tom and Jason mine by bus on a daily basis. The mine would operate on two 12-hour shifts per day, seven days per week

Table 16-17 below outlines the anticipated mine labour force quantities, and rotation schedules.

Table 16-17: Mine Personnel Summary

Position	Rotation	Year 1	Year 5	Year 10	Peak
Mine General					
Mining Manager	2x2	1	1	1	1
Mining Superintendent	2x2	1	1	1	1
Technical Services Superintendent	2x2	1	1	1	1
Mine Shift Foreman	2x2	4	4	4	4
Maintenance Superintendent	2x2	1	1	1	1
Maintenance Shift Foreman	2x2	4	4	4	4
Training Officer	2x2	2	2	2	2
Subtotal - Mining Management		***14***	***14***	***14***	***14***
Drill and Blast					
Jumbo Driller	2x2	16	16	12	16
Production Driller	2x2	4	16	4	16
Blaster	2x2	4	8	8	8
Blasting Helper	2x2	4	8	8	8
Utility Vehicle Operator / Nipper	2x2	4	16	8	16
Subtotal - Drill and Blast		***32***	***64***	***40***	***64***
Load and Haul					
LHD Operator	2x2	12	36	48	48
Truck Driver	2x2	8	28	40	40
Rock Truck Driver	2x2	4	8	12	12
UG Crusher Operator	2x2	0	4	4	4
Subtotal - Load and Haul		***24***	***76***	***104***	***104***
Support Services					
Ground Support / Bolter / Shotcrete	2x2	12	12	12	12
Development Service	2x2	4	4	4	4
Construction Miner	2x2	4	4	4	4
Utility Vehicle Operator / Nipper	2x2	14	8	8	14
UG Labourer	2x2	4	12	12	12
Electrician	2x2	8	8	8	8
Subtotal - Support Services		***46***	***48***	***48***	***48***
Quarry / Backfill Operations					
Screen / CRF Operator	2x2	0	8	8	8
Subtotal - Quarry / Backfill		***0***	***8***	***8***	***8***
Mine Maintenance					
Heavy Equipment Mechanic	2x2	4	12	16	16
Drill Mechanic	2x2	4	6	4	6
Welder / Mechanic	2x2	2	6	8	8
Electrician / Instrument	2x2	4	4	4	4
UG Crusher / Conveyor Mechanic	2x2	4	4	4	4
Tireman	2x2	4	4	4	4
Apprentice	2x2	8	8	8	8
Dry / Lampman / Bitman	2x2	4	4	4	4
Labourer / Trainee	2x2	8	8	8	8
Subtotal - Mine Maintenance		***42***	***56***	***60***	***62***
Technical Services					
Maintenance Planner	2x2	1	1	1	1
Chief Mining Engineer	2x2	1	1	1	1
Senior Mine Engineer	2x2	1	1	1	1
Mine Engineer	2x2	2	2	2	2
Ventilation Engineer	2x2	1	1	1	1
Geotechnical Engineer	2x2	1	1	1	1
Mine Technician	2x2	2	2	2	2
Surveyor	2x2	4	4	4	4
Surveyor Assistant	2x2	4	4	4	4
Clerk	2x2	2	2	2	2
Senior Geologist	2x2	1	1	1	1
Mine Geologist	2x2	2	2	2	2
Technician / Geo Control	2x2	0	4	4	4
Chief Geologist	2x2	1	1	1	1
Subtotal - Technical Services		***23***	***27***	***27***	***27***
Grand Total		**181**	**293**	**301**	**327**

Source: JDS (2018)

16.12 Mine Development Plan

The underground mine at Tom and Jason are envisioned to start development one year after operations commence in the open pits at Tom and Jason respectively. Development priority is given to areas which offer quickest access to high value mineral, and to capital infrastructure required for safe and efficient mineral production.

Table 16-18 lists the underground development schedule.

Table 16-18: Underground Development Schedule

Underground Development	Units	Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18
Metres - Rehab	m	1,781	-	1,563	-	-	-	-	-	-	9	-	-	-	-	-	-	-	-	-	-
Metres - Ramp	m	14,591	-	1,722	3,919	1,292	1,637	821	1,449	1,464	101	1,886	300	-	-	-	-	-	-	-	-
Metres - Access	m	4,913	-	536	732	553	501	360	679	525	192	345	340	-	-	-	48	48	54	-	-
Metres - Electrical	m	456	-	38	30	12	18	44	26	-	8	52	228	-	-	-	-	-	-	-	-
Metres - Sump	m	1,392	-	25	81	75	100	130	115	-	-	535	308	23	-	-	-	-	-	-	-
Metres - Remuck	m	740	-	40	49	45	90	81	55	-	10	310	60	-	-	-	-	-	-	-	-
Metres - Refuge	m	70	-	10	10	-	-	-	-	-	-	30	20	-	-	-	-	-	-	-	-
Metres - Conveyor Drive	m	140	-	-	140	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Metres - Shop	m	225	-	-	110	-	-	115	-	-	-	-	-	-	-	-	-	-	-	-	-
Metres - Muck Pass Drive	m	764	-	-	83	57	25	120	45	45	-	257	87	-	-	-	15	15	15	-	-
Metres - Vent Drive	m	2,529	-	75	633	171	366	146	260	218	168	186	306	-	-	-	-	-	-	-	-
Metres - Footwall Drive	m	13,431	-	1,101	1,649	2,064	802	1,197	2,021	1,064	1,015	533	1,187	-	44	-	596	80	-	-	78
Metres - Auxiliary	m	1,761	-	171	328	188	182	151	194	126	56	191	134	-	2	-	30	4	-	-	4
Metres - Crosscut Waste	m	12,150	-	922	15	767	934	869	831	911	1,675	630	1,300	276	526	189	394	523	1,388	-	-
Metres - Cross Cut Mineral	m	20,624	-	1,438	-	1,344	1,786	2,735	845	1,306	1,702	788	636	757	1,699	851	684	2,231	1,639	155	28
Metres - Wall Slashing	m	24,162	-	-	-	1,101	1,182	832	1,155	1,967	2,621	1,878	2,667	1,446	817	1,845	1,695	1,311	2,945	446	254
Total Lateral Development	m	99,729	-	7,641	7,779	7,669	7,623	7,601	7,675	7,626	7,557	7,621	7,573	2,502	3,088	2,885	3,462	4,212	6,041	601	573
Lateral Development Rate	m/day	19	-	21	21	21	21	21	21	21	21	21	21	7	8	8	9	12	17	2	2
Jumbo Productivity	m/mth	455	-	628	639	630	627	625	631	627	621	626	622	206	254	237	285	346	497	49	47
	m/mth/jumbo	146	-	157	160	158	157	156	158	157	155	157	207	103	127	119	142	173	248	49	47
Metres - Alimak Vent Raise	m	875	-	-	291	-	-	-	-	290	84	42	168	-	-	-	-	-	-	-	-
Metres - Alimak Muck Pass	m	650	-	-	238	112	-	69	189	-	-	42	-	-	-	-	-	-	-	-	-
Metres - Alimak Escape Way	m	1,160	-	263	402	88	-	-	-	160	40	165	42	-	-	-	-	-	-	-	-
Metres - Alimak Stope Raise	m	1,298	-	-	-	-	-	-	-	132	-	-	-	-	-	-	-	333	833	-	-
Metres - Longhole Muck Pass	m	807	-	-	-	-	43	147	-	63	-	428	21	-	-	-	-	84	21	-	-
Metres - Longhole Vent Raise	m	614	-	-	64	190	128	0	232	-	-	-	-	-	-	-	-	-	-	-	-
Metres - Longhole Escape Way	m	421	-	-	86	21	84	126	62	-	-	-	42	-	-	-	-	-	-	-	-
Metres - Longhole Stope Raise	m	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Grand Total Vertical Development	m	5,825	-	263	1,081	411	255	342	483	645	124	677	273	-	-	-	-	417	854	-	-
Vertical Advance Rate	m/day	2	-	1	3	1	1	1	1	2	-	2	1	-	-	-	-	1	2	-	-

Source: JDS (2018)

16.13 Mine Production Plan

Jason open pit is short lived and offers quick access to mill feed and will be mined first while Tom open pit is developed. Once Jason pit is depleted the bulk of mill feed will be supplied by Tom pit while the underground mines at Tom and Jason are developed.

Jason underground utilizes a top down mine method and does not require underground crushing, so Jason Main is the first zone to commence production. Quickly followed is production from both Tom East and Tom West, which offer the highest available grades. Mineral production at Tom does not commence until the underground crusher, conveyor, and muck passes are commissioned in Year 3. As the underground mines are developed, highest grade with lowest cost of access is prioritized.

The combined underground and open pit production schedule and material balances are shown in Table 16-19 and Table 16-20.

Table 16-19: Mine Production Schedule

Mine Production	Units	Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18
Underground Production																					
Mineral	kt	28,427	-	114	-	1,412	1,760	1,760	1,765	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,485
Waste	kt	3,746	-	320	608	350	307	278	393	322	224	350	294	19	38	13	74	48	102	0	5
Ag	g/t	45.8	-	60.9	-	64.8	67.2	58.8	49.9	58.1	59.2	77.9	88.9	65.0	21.0	26.6	23.5	20.8	11.4	14.8	26.6
Pb	%	3.6	-	5.8	-	5.3	5.6	4.9	3.6	4.6	4.3	5.1	6.2	4.8	1.9	2.9	2.2	1.8	1.4	1.4	2.5
Zn	%	5.2	-	6.5	-	6.1	7.2	6.9	6.3	5.7	4.8	5.0	4.7	4.7	5.2	4.5	4.7	4.7	4.1	3.8	4.9
NSR	$/t	162.1	-	224.0	-	210.0	235.0	215.0	183.0	189.0	169.0	192.0	206.0	176.0	128.0	133.0	125.0	117.0	97.0	93.0	133.4
Open Pit Production																					
Mineral	kt	4,229	150	1,677	1,824	579	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste	kt	20,934	3,764	11,297	5,447	427	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag	g/t	6.1	9.2	5.7	6.3	5.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb	%	2.9	7.5	2.0	3.8	1.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn	%	27.1	89.4	14.8	40.1	5.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NSR	$/t	152.4	275.9	133.7	169.3	121.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Combined Production																					
Mineral	kt	32,656	150	1,791	1,824	1,991	1,760	1,760	1,765	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,485
Waste	kt	24,680	3,764	11,617	6,055	777	307	278	393	322	224	350	294	19	38	13	74	48	102	0	5
Ag	g/t	40.7	9.2	9.3	6.3	47.6	67.2	58.8	49.9	58.1	59.2	77.9	88.9	65.0	21.0	26.6	23.5	20.8	11.4	14.8	26.6
Pb	%	3.6	7.5	2.3	3.8	4.2	5.6	4.9	3.6	4.6	4.3	5.1	6.2	4.8	1.9	2.9	2.2	1.8	1.4	1.4	2.5
Zn	%	8.0	89.4	14.3	40.1	6.0	7.2	6.9	6.3	5.7	4.8	5.0	4.7	4.7	5.2	4.5	4.7	4.7	4.1	3.8	4.9
NSR	$/t	160.9	275.9	139.4	169.3	184.3	235.0	215.0	183.0	189.0	169.0	192.0	206.0	176.0	128.0	133.0	125.0	117.0	97.0	93.0	133.4
Mill Schedule																					
Mineral	kt	32,656	-	1,825	1,825	1,826	1,825	1,825	1,825	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,574
Ag	g/t	43.4	-	24.7	40.0	51.5	65.0	56.9	48.4	58.1	59.2	77.9	88.9	65.0	21.0	26.6	23.5	20.8	11.4	14.8	25.3
Pb	%	3.6	-	2.8	3.8	4.5	5.4	4.8	3.5	4.6	4.3	5.1	6.2	4.8	1.9	2.9	2.2	1.8	1.4	1.4	2.4
Zn	%	5.3	-	6.2	6.3	6.1	7.1	6.8	6.2	5.7	4.8	5.0	4.7	4.7	5.2	4.5	4.7	4.7	4.1	3.8	4.8
NSR	$/t	160.9	-	154.7	169.3	191.8	230.1	210.8	180.1	189.0	169.0	192.0	206.0	176.0	128.0	133.0	125.0	117.0	97.0	93.0	130.1

Source: JDS (2018)

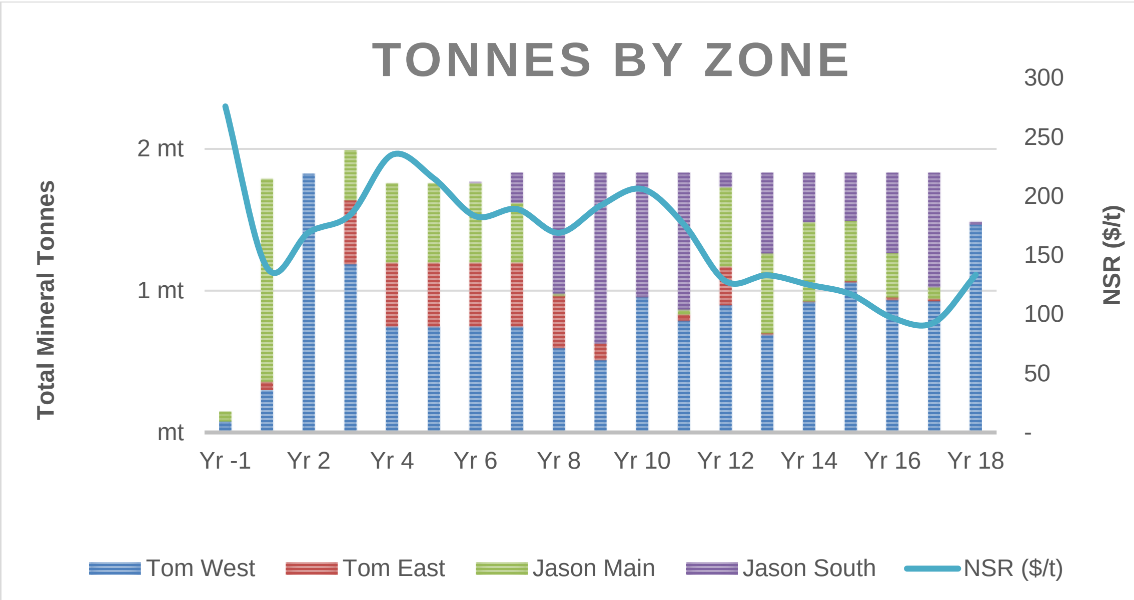
Table 16-20: Material Balance

Item	Units	Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18
Material Balance																					
Mineral - Tom	kt	19,333	79	360	1,824	1,639	1,200	1,200	1,200	1,200	968	630	959	837	1,172	703	928	1,067	953	946	1,467
Mineral - Jason	kt	13,323	71	1,431	0	352	560	560	565	630	862	1,200	871	993	658	1,127	902	763	877	884	18
Mineral - Total	kt	32,656	150	1,791	1,824	1,991	1,760	1,760	1,765	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,485
Waste - Tom	kt	17,947	2,709	7,495	5,998	635	181	84	93	35	89	234	191	8	31	7	59	30	68	-	-
Waste - Jason	kt	6,734	1,054	4,122	57	142	125	195	301	287	135	116	102	11	7	6	16	18	34	-	5
Waste - Total	kt	24,680	3,764	11,617	6,055	777	307	278	393	322	224	350	294	19	38	13	74	48	102	-	5
Waste Allocation																					
Waste to Backfill	kt	16,521	-	69	-	827	1,042	1,067	1,081	1,112	1,106	1,066	1,031	1,052	1,139	1,108	1,108	1,108	1,060	737	809
Waste to Stockpile	kt	21,375	3,764	11,548	6,055	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste from Stockpile	kt	-13,215	-	-	-	-59	-735	-788	-687	-790	-882	-716	-737	-1,032	-1,101	-1,095	-1,034	-1,060	-958	-737	-804
Stockpile Summary																					
Mineral - Tom	kt	-	79	-	16	182	117	52	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste - Tom	kt	8,159	2,709	10,167	16,165	16,173	15,656	15,019	14,392	13,698	13,183	13,036	12,665	12,169	11,483	11,052	10,574	9,979	9,513	8,963	8,159
Mineral - Jason	kt	-	71	134	97	97	97	97	88	88	88	88	88	88	88	88	88	88	88	88	-
Waste - Jason	kt	-	1,054	5,145	5,202	5,143	4,925	4,774	4,713	4,617	4,249	3,681	3,314	2,778	2,363	1,699	1,143	678	187	-	-
TRSF Facility	kt	27,956	-	1,562	3,125	4,688	6,251	7,813	9,376	10,942	12,509	14,076	15,642	17,209	18,776	20,342	21,909	23,476	25,042	26,609	27,956

Source: JDS (2018)

Figure 16-24 illustrates the breakdown of mine plan tonnes by zone, annually.

Figure 16-24: Tonnes by Zone



Source: JDS (2018)

Backfill for Tom and Jason will be provided by development waste and open pit waste generated during mine operations. Waste rock will be stockpiled on surface, screened, and returned underground as needed. Jason open pit was sized such that no waste rock would remain on surface at the end of the mine life. In the event that waste does remain it may be deposited back into the open pit to avoid permanent waste management facility closure costs. Likewise, if the Jason zones extend and additional waste is required for backfill the Jason pit walls may be slashed to provide material, as to save cost of a separate fill quarry.

Tom open pit is larger and will incur a permanent waste rock surplus at the end of the mine life. Like Jason, the Tom pit will provide waste rock for backfill requirements underground.

Tom and Jason open pit and underground mines would be operated in parallel to achieve a nominal 5,000 tonnes per day mine rate. Multiple faces and stopes in various stages of development and production will be required. Tom and Jason mines both offer flexible mine designs which permit the extraction of mineral in various locations at once.

17 Recovery Methods

The Fireweed Zinc Macmillan Pass Project has identified two similar Pb/Zn/Ag deposits, Tom and Jason. The recent metallurgical test program BL0236 completed at Base Metallurgical Labs in Kamloops, BC, summarized in Section 13, has demonstrated that standard Pb and Zn sequential flotation will yield an overall Pb recovery of 75%, at a concentrate grade of 62% Pb, and a Zn recovery of 89%, at a concentrate grade of 58% Zn (BL0236-LCT45). Results from this test program were used to develop the corresponding process design criteria, mass balance, mechanical equipment list, flowsheet and operating costs.

The process plant will include:

- Primary crushing;
- Semi-Autogenous Grinding (SAG) mill operating in open circuit;
- Ball mill grinding in reverse closed-circuit with cyclones;
- Sequential Pb and Zn flotation circuits, each incorporating conventional and column flotation, regrind and three cleaning stages;
- Concentrate dewatering circuits using thickeners and pressure filters;
- Concentrate storage and load-out facilities; and
- Pumping and storage of slurry tailings.

The mining schedule dictates the requirement for two crushing circuits. One circuit will be installed above ground to process Jason material; while the second circuit will be installed underground to process Tom material. Approximately 65% Tom and 35% Jason material with average LOM head grades of 3.6% Pb and 5.2% Zn will provide a total throughput of 5,000 t/d to the process plant. The crushing circuits will operate at an availability of 75% or 18 hours per day. The process plant will operate 24 hours per day, 365 days per year at an availability of 92%.

Primary crushing circuits will reduce the material down to a product size of 80% passing (P_{80}) 110 mm. The subsequent two stage grinding circuit will target a P_{80} grind size of 50 µm, before Pb and Zn are recovered into concentrates using sequential flotation. Zn rougher and Zn 1st cleaner tailings, designated as final tailings, will be pumped to the tailings facility.

The process plant will consist of grinding as well as Pb and Zn flotation circuits, each consisting of rougher flotation, cleaner column flotation, concentrate regrind and three stages of conventional tank cell flotation. Both concentrates will be dewatered in concentrate thickeners and pressure filters to produce a target moisture content of 8%.

17.1 Introduction

The processing facilities will consist of the following unit operations:

- Two jaw crusher circuits - A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product P_{80} of 110 mm;

- Primary Grinding – A SAG mill in open circuit, producing a T_{80} transfer size of approximately 1,000 µm;
- Secondary Grinding – A ball mill in reverse closed circuit with a cluster of hydrocyclones, producing a final target product size P_{80} of 50 µm;
- Pb Flotation – Rougher and cleaner flotation to produce a saleable Pb concentrate;
- Pb Rougher Concentrate Regrind – A stirred regrind mill in open circuit, reducing Pb rougher concentrate to a P_{80} of 15 µm;
- Pb Concentrate Dewatering – A 7 m diameter high-rate thickener to achieve an underflow solids density of 55%, and a pressure filter to reduce the concentrate to a final moisture content of 8%;
- Zn Flotation – Rougher and cleaner flotation to produce a saleable Zn concentrate;
- Zn Rougher Concentrate Regrind – A stirred regrind mill in open circuit, reducing Zn rougher concentrate to a P_{80} of 25 µm;
- Zn Concentrate Dewatering – A 12 m diameter high-rate thickener to achieve an underflow solids density of 55%, and a pressure filter to reduce the concentrate to a final moisture content of 8%; and
- Final Tailings – pumping and slurry storage in the tailings facility.

17.2 Plant Design Criteria

The Process Design Criteria and Mass Balance detail the annual production capabilities, major mass flows and capacities, and availability for the process plant. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 22. Key process design criteria from Section 13 are summarized in Table 17-1.

Table 17-1: Process Design Criteria

Criteria	Unit	Nominal Value	Source
General			
Crushing and Process Plant Throughput	t/d	**5,000**	2018 mine plan
Process Plant Availability	%	**92**	Industry Standard
Process Plant Throughput	t/h	**226**	Engineering Calculation
LOM Average Pb Head Grade	%	**3.6**	2018 mine plan
LOM Average Zn Head Grade	%	**5.2**	2018 mine plan
LOM Average Ag Head Grade	g/t	**45.8**	2018 mine plan
Overall Pb Recovery	%	**75**	Base Met (2018): BL0236 LCT-45
Pb Concentrate Grade	% Pb	**62**	Base Met (2018) LCT-45
Ag Recovery	%Ag	**59**	Base Met (2018) LCT-45
Overall Zn Recovery	%	**88.9**	Base Met (2018): BL0236 LCT-45
Zn Concentrate Grade	% Zn	**58.4**	Base Met (2018) LCT-45
Crushing			
Availability / Utilization	%	**75**	Industry Standard
Number of Crushing Stages	-	**1**	Vendor Recommended
Crushing System Product Size (P_{80})	mm	**110**	Vendor Simulation – estimated based on CSS of 125 mm
Crushed Material Stockpile			
Stockpile Capacity (live)	t	**5,000**	Design Consideration
Stockpile Capacity (live)	h	**24**	Engineering Calculation
Grinding			
SMC – Comps 1 and 3A	Mia – kWh/t	**10.7/14.5**	Base Met (2018): BL0236
	Axb	**55.8/80.8**	Base Met (2018):BL0236
Bond Ball Mill Work Index	kWh/t	**14**	Base Met (2018): BL0236
Primary Grinding Mill Type	-	**SAG Mill**	Industry Standard for primary grinding to target transfer size
Mill Diameter	m	**6.4**	Vendor Recommended
Mill Length	m	**3.7**	Vendor Recommended
Installed Power	kW	**1,865**	Vendor Recommended
Circuit Configuration	-	**Open**	Design Consideration
Primary Grinding Transfer Size (T_{80})	µm	**1,000**	Design Consideration
Secondary Grinding Mill Type	-	**Ball Mill**	Selected to achieve target product size
Mill Diameter	m	**5.5**	Vendor Recommended
Mill Length	m	**9.3**	Vendor Recommended
Installed Power	kW	**4,476**	Vendor Recommended
Circuit Configuration	-	**Reverse Closed**	Industry Standard
Circulating Load	%	**300**	Industry Standard
Final Product Size (P_{80})	µm	**50**	Base Met (2018): BL0236 Blend Composite
Flotation			
Rougher Flotation Time Scale-up	-	**2.0**	Industry Standard
Cleaner Flotation Time Scale-up	-	**4.0**	Industry Standard
Pb Rougher Flotation			
Laboratory Retention Time	min	**6**	Base Met (2018): BL0236 LCT-45
Design Retention Time	min	**12**	Engineering Calculation based on 2.0x scale-up factor
Number of Rougher Flotation Cells	#	**6**	Designed to achieve retention time – conc. from first two cells to column
Rougher Flotation Cell Size	m³	**20**	Designed to achieve retention time
Pb Regrind Circuit			
Rougher Concentrate Mass Pull	%	**2.6/5.1**	Base Met (2018): BL0236 LCT-45
Regrind Mill Type	-	**Stirred Mill**	Industry Standard
Final Product Size (P_{80})	µm	**15**	Base Met (2018): BL0236 LCT-45
Pb Cleaner Flotation			
Column Flotation	mxm	**1 – 1.5x8.0**	Vendor Recommended
Number of Stages	#	**3**	Base Met (2018): BL0236 LCT-45
Laboratory Retention Time	min	**6 / 3 / 2**	Base Met (2018): BL0236 LCT-45
Design Retention Time	min	**24 / 12 / 8**	Engineering Calculation based on 4.0x scale-up factor
Number of Cleaner Flotation Cells	#	**6 / 3 / 2**	Designed to achieve retention time
Cleaner Flotation Cell Sizes	m³	**5 / 3 / 3**	Designed to achieve retention time
Zn Rougher Flotation			
Laboratory Retention Time	min	**6**	Base Met (2018): BL0236 LCT-45
Design Retention Time	min	**12**	Engineering Calculation based on 2.0x scale-up factor
Number of Rougher Flotation Cells	#	**7**	Designed to achieve retention time - conc. from first two cells to column
Rougher Flotation Cell Size	m³	**20**	Designed to achieve retention time
Zn Regrind Circuit			
Rougher Concentrate Mass Pull	%	**5.4/13.2**	Base Met (2018): BL0236 LCT-45
Regrind Mill Type	-	**Stirred Mill**	Industry Standard

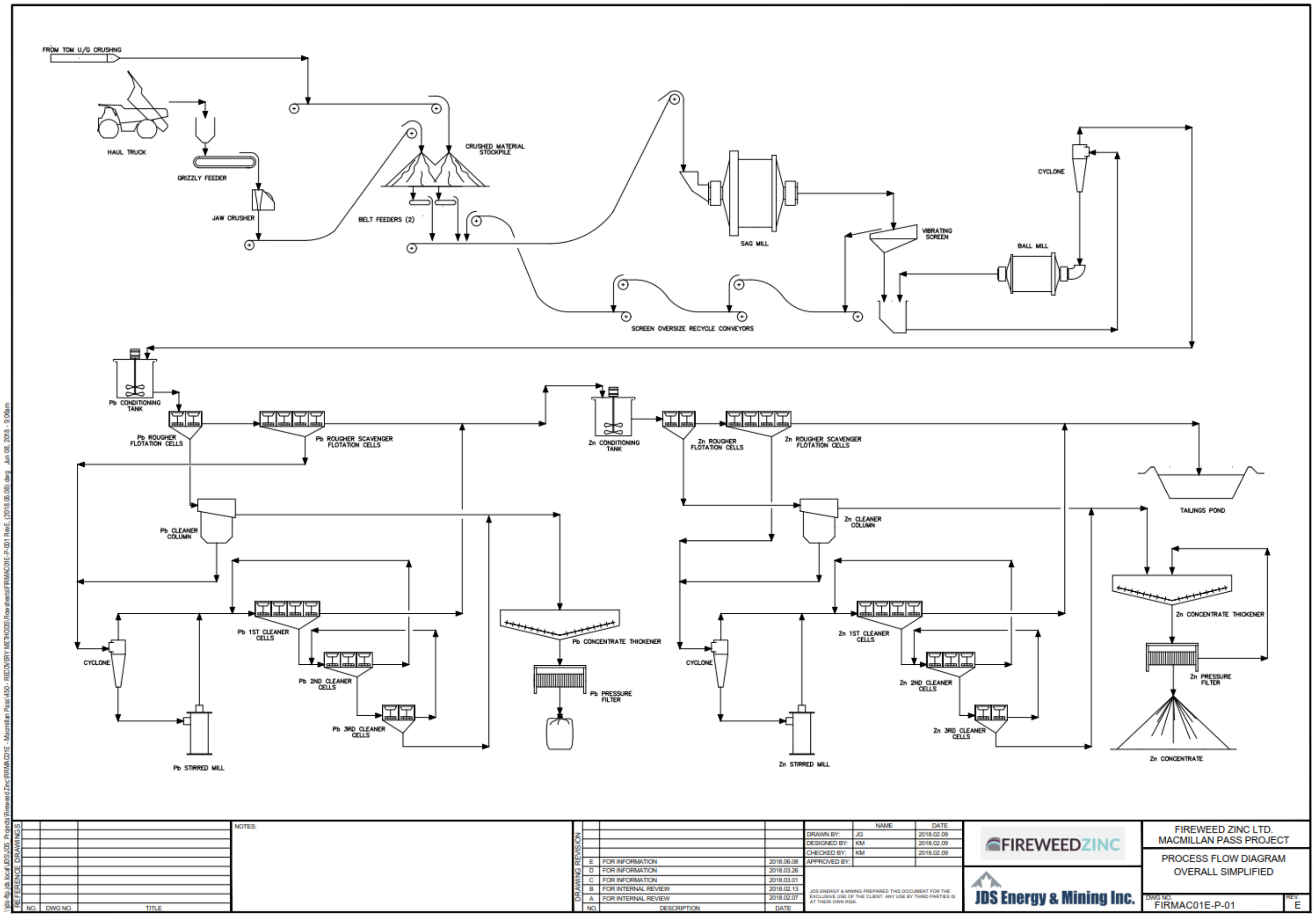
Criteria	Unit	Nominal Value	Source
Final Product Size (P_{80})	µm	**25**	Base Met (2018): BL0236 LCT-45
Zn Cleaner Flotation			
Column Flotation	mxm	**1 – 3.0x8.0**	Vendor Recommended
Number of Stages	#	**3**	Base Met (2018): BL0236 LCT-45
Laboratory Retention Time	min	**7 / 4 / 3**	Base Met (2018): BL0236 LCT-45
Design Retention Time	min	**28 / 16 / 12**	Engineering Calculation based on 4.0x scale-up factor
Number of Cleaner Flotation Cells	#	**8 / 6 / 3**	Designed to achieve retention time
Cleaner Flotation Cell Size	m^3	**10**	Designed to achieve retention time
Concentrate Dewatering			
Thickener Type	-	**High Rate**	Industry Standard
Pb Thickener Loading Rate	$t/h/m^2$	**0.26**	Design Consideration
Pb Thickener Diameter	m	**7**	Vendor Recommended
Zn Thickener Loading Rate	$t/h/m^2$	**0.27**	Design Consideration
Zn Thickener Diameter	m	**12**	Vendor Recommended
Filtration Type	-	**Pressure**	Industry Standard
Final Zn and Pb Concentrate Moisture Content	%	**8**	Design Consideration

Source: JDS (2018)

17.3 Plant Design

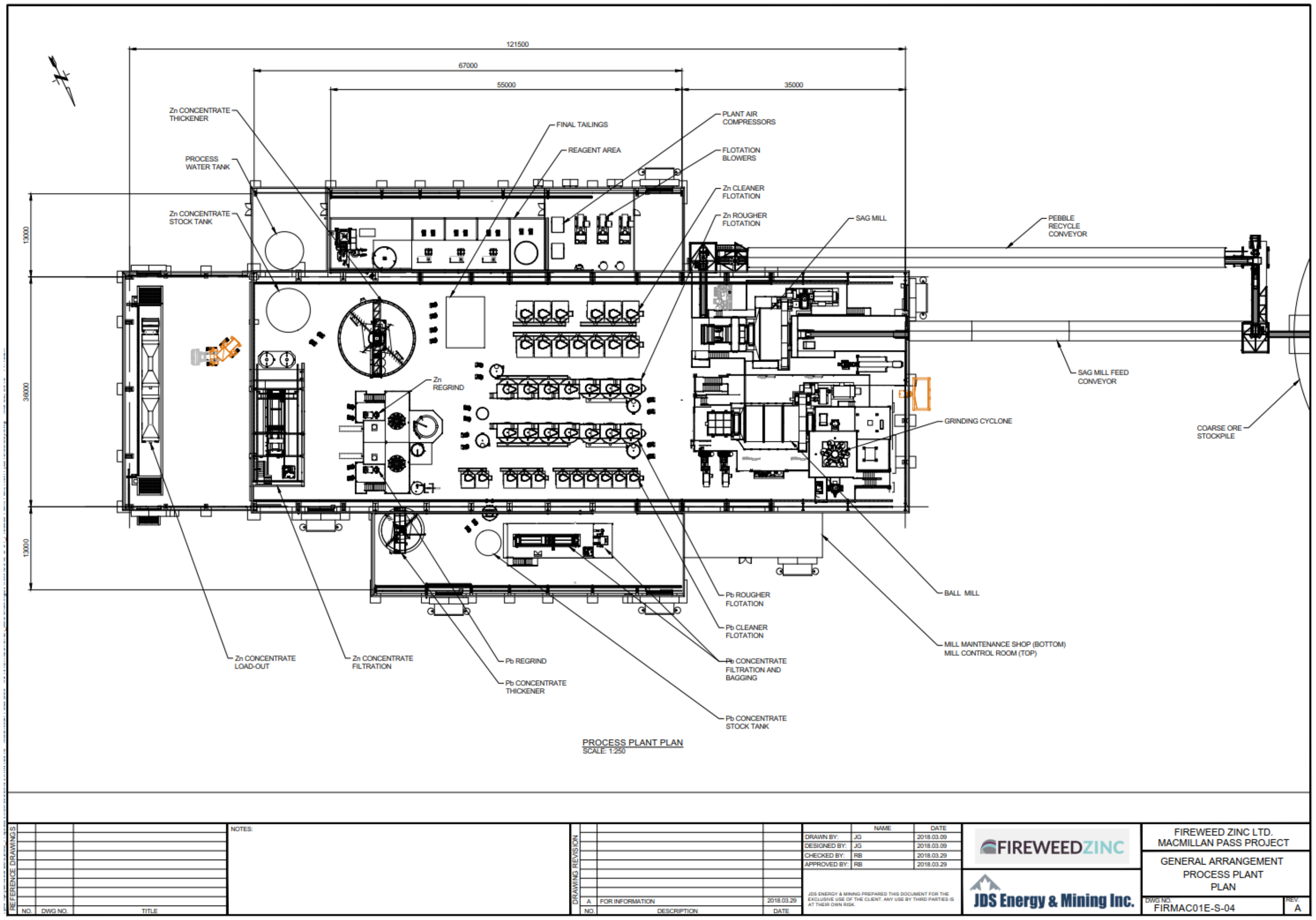
The process flowsheet and plant layout are shown in Figure 17-1 and Figure 17-2.

Figure 17-1: Overall Process Flowsheet



Source: JDS (2018)

Figure 17-2: Process Plant Layout



Source: JDS (2018)

17.4 Process Plant Description

17.4.1 Crushing

Two crushing circuits will operate to process the mined material from the Tom and Jason deposits. The crushing plants for Tom will be located underground and will include a 42” x 48” jaw crusher with an installed power of 160 kW. Jason material will be stockpiled near the above ground jaw crusher or direct dumped through an 800 mm static grizzly into a dump pocket. Stockpiled ROM material will be re-handled by a front-end loader and fed into the crusher. The material will discharge through the static grizzly into a feed hopper. Oversize material from the static grizzly will be removed for later size reduction using a rock breaker. The Jason jaw crusher 636 mm x 1,016 mm (25” x 40”) with an installed power of 90 kW, will process 150 t/h. The crusher, with a closed side setting (CSS) of 125 mm, will produce a final product P_{80} of approximately 110 mm.

Product from the two crushers will be conveyed to the 5,000 tonne or 24 hour live crushed material stockpile. Two belt feeders, located in a corrugated tunnel under the stockpile, will be installed with variable frequency drives (VFD) to control the reclaim rate feeding the grinding circuit. Each belt feeder will be capable of providing the total throughput of 226 t/h.

17.4.2 Grinding

The grinding circuit will consist of a primary SAG mill followed by a secondary ball mill. The primary SAG mill will operate in open circuit, while the secondary ball mill will operate in reverse closed circuit with a cluster of hydrocyclones. The grinding circuit will be able to process a nominal throughput of 226 t/h (fresh feed), and produce a final product P_{80} of 50 µm.

Product from the crushing circuits will be conveyed to a 6.4 m diameter by 3.7 m long SAG mill with an installed power of 1,865 kW motor. A belt-scale on the feed conveyor will monitor feed rate. Water will be added to the SAG mill to maintain the slurry charge in the mill at a constant density of 70%. Slurry will overflow from the SAG mill onto a screen. The screen oversize will discharge onto a series of three recycle conveyors and returned to the feed end of SAG mill. The screen undersize at a transfer size (T_{80}) of approximately 1,000 µm will flow into the cyclone feed pump box.

Product from the primary SAG mill screen undersize will flow into the cyclone feed pump box and combine with the secondary ball mill discharge before being pumped up to a cluster of ten (eight operating / two standby) 375 mm hydrocyclones for size classification. The coarse underflow will flow by gravity to the secondary ball mill, 5.5 m diameter by 9.3 m long ball mill with an installed power of 4,476 kW, for additional grinding. The fine cyclone overflow, at a final product P_{80} of 50 µm, will report to the Pb Conditioning Tank. The hydrocyclones have been designed for a 300% circulating load.

17.4.3 Flotation

17.4.3.1 Lead Circuit

Cyclone overflow will flow by gravity to a 19 m^3 Pb conditioning tank, which will provide 2 minutes of conditioning time prior to Pb flotation. Frother methyl isobutyl carbinol (MIBC), sulphide collector sodium xanthate (PAX), Zn depressant sodium cyanide (NaCN), pH modifier soda ash, zinc sulphate ($ZnSO_4$) and carbon depressant PE26 will be added to the conditioning tank. The slurry will then gravitate to the rougher flotation circuit, which consists of six 20 m^3 flotation tanks cells operating in series.

The Pb rougher concentrate from cells 1 and 2 will be pumped to a 1.5 m diameter x 8.0 m high flotation column. The column concentrate will report to the Pb concentrate thickener for dewatering, while the tailings will be pumped to the Pb regrind circuit.

Pb rougher concentrate from cells 3 through 6 will be collected in a common launder and pumped to the regrind circuit. The rougher concentrate and column tailings will be pumped to a cluster of three (two operating / one spare) 150 mm densifying cyclones to achieve a cyclone underflow density of 50%. The cyclone underflow will then flow by gravity to a pump box where density control water will be added to ensure an adequate feed density to the regrind mill. The slurry will then be pumped to a 400 kW stirred mill where high-intensity grinding with 3 mm ceramic grinding media will reduce the bulk concentrate to a P_{80} of 15 µm. The product will then combine with the cyclone overflow and be pumped to the Pb first cleaner flotation circuit.

Regrind product and the Pb second cleaner tailings will feed six 5.0 m^3 Pb first cleaner tank cells. The Pb first cleaner concentrate will be collected in a common launder and fed to the Pb second cleaner flotation circuit. The Pb first cleaner tailings will combine with the Pb rougher tailings and be pumped to the Zn conditioning tank.

The Pb first cleaner concentrate will combine with the Pb third cleaner tailings and flow into the first of three 3.0 m^3 Pb second cleaner flotation tank cells. The Pb second cleaner concentrate will be collected in a common launder and pumped to the third cleaner flotation cells, while the Pb second cleaner tailings will flow back to the Pb first cleaner flotation feed box.

The Pb second cleaner concentrate will flow into the first of two 3.0 m^3 Pb third cleaner flotation tank cells. The Pb third cleaner concentrate will be collected in a common launder and pumped to the Pb concentrate thickener, while the Pb third cleaner tailings will flow back to the Pb second cleaner flotation feed box.

Pb concentrate from the column and third cleaners will report to a 7 m diameter Pb thickener. The thickener overflow will be sent to the process water tank. Thickened Pb concentrate will be pumped to an 8 hour stock tank that feeds a pressure filter for further dewatering. Pb final concentrate, at approximately 8% moisture, will be bagged and loaded onto trucks for transportation to Skagway, Alaska.

17.4.3.2 Zinc Circuit

Tailings from the rougher and first cleaner flotation circuits will feed a 60 m^3 Zn conditioning tank, which will provide 5 minutes of conditioning time prior to Zn flotation. Frother methyl isobutyl carbinol (MIBC), Polyfroth H57, SIPX, pH modifier lime and copper sulphate ($CuSO_4$) will be added to the conditioning tank. The slurry will then gravitate to the rougher flotation circuit, which consists of seven 20 m^3 flotation tanks cells operating in series.

The Zn rougher concentrate from cells 1 and 2 will be pumped to a 3.0 m diameter x 8.0 m high flotation column. The column concentrate will report to the Zn concentrate thickener for dewatering, while the tailings will be pumped to the Zn regrind circuit.

Zn rougher concentrate from cells 3 through 7 will be collected in a common launder and pumped to the regrind circuit. The rougher concentrate and column tailings will be pumped to a cluster of five (4 operating / 1 spare) 150 mm densifying cyclones to achieve a cyclone underflow density of 50%. The cyclone underflow will then flow by gravity to a pump box where density control water will be added to ensure an adequate feed density to the regrind mill. The slurry will then be pumped to a 400 kW stirred mill where high-intensity grinding with 3 mm ceramic grinding media will reduce the bulk concentrate to a P_{80} of 25 µm. The product will then combine with the cyclone overflow and be pumped to the Zn first cleaner flotation circuit.

Regrind product and the Zn second cleaner tailings will feed seven 10 m^3 Zn first cleaner tank cells. The Zn first cleaner concentrate will be collected in a common launder and fed to the Zn second cleaner flotation circuit. The Zn first cleaner tailings will combine with the Zn rougher tailings and be pumped to the final tailings pumpbox.

The Zn first cleaner concentrate will combine with the Zn third cleaner tailings and flow into the first of three 10 m^3 Zn second cleaner flotation tank cells. The Zn second cleaner concentrate will be collected in a common launder and pumped to the third cleaner flotation cells, while the Zn second cleaner tailings will flow back to the Zn first cleaner flotation feed box.

The Zn second cleaner concentrate will flow into the first of two 10 m^3 Zn third cleaner flotation tank cells. The Zn third cleaner concentrate will be collected in a common launder and pumped to the Zn concentrate thickener, while the Zn third cleaner tailings will flow back to the Zn second cleaner flotation feed box.

Zn concentrate from the column and third cleaners will report to a 12 m diameter Zn thickener. The thickener overflow will be sent to the process water tank. Thickened Zn concentrate will be pumped to a pressure filter for further dewatering. Zn final concentrate, at approximately 8% moisture, will be loaded onto trucks for transportation to Skagway, Alaska.

17.4.4 Tailings Management Facility

Zn rougher tailings and Zn first cleaner tailings will combine in the final tailings pump box and be pumped to the tailings management facility. Water will be reclaimed from the facility as make-up water in the plant.

17.4.5 Reagents Handling and Storage

Reagents added to the Pb and Zn flotation circuits will be prepared and distributed from the reagent handling facility. This area includes various mixing and storage tank units. All reagent areas will be bermed with sump pumps, which can transfer spills to the final tailings pump box or back to the corresponding mix tank. The one exception will be the Flocculant preparation area. Flocculant spillage will be returned to the storage tank. The reagents will be mixed, stored and then delivered through individual supply loops with dosage controlled by flow meters and manual control valves. The reagent storage tanks have been sized with capacity to handle one day of production. The reagents will be delivered to the mine site either in powder form or as solutions.

Table 17-2 summarizes the reagents used in the plant and their estimated daily consumption rates. The table also includes other major process consumables.

Table 17-2: Reagent and Process Consumables

Description	Delivered Form	Average Daily Usage
Sodium Ethyl Xanthate (SEX)	1 tonne bags (dry)	1.11 t/d
Sodium Cyanide (NaCN)	500 kg bags (dry)	675 kg/d
Zinc Sulphate ($ZnSO_4$)	1 tonne bags (dry)	2.4 t/d
PE-26	50 kg bags (dry)	125 kg/d
Soda Ash (Na_2CO_3)	1 tonne bags (dry)	5 t/d
Sodium Isopropyl Xanthate (SIPX)	50 kg bags (dry)	170 kg/d
Copper Sulphate ($CuSO_4$)	1 tonne bags (dry)	5.25 t/d
Lime ($Ca(OH)_2$)	2 tonne bags (dry)	7.75 t/d
Methyl Isobutyl Carbinol (MIBC)	1 tonne totes (liquid)	935 kg/d
Polyfroth H57	1 tonne totes (liquid)	750 kg/d
Flocculant	25 kg bags (dry)	11 kg/d
Antiscalent	1 tonne totes (liquid), or 50 kg barrels	25 kg/d
SAG Mill Grinding Media – 125 mm chrome steel	1 tonne bags	2.7 t/d
Ball Mill Grinding Media – 75 mm chrome steel	1 tonne bags	4.3 t/d
Pb Stirred Mill Grinding Media – 3 mm ceramic	500 kg bags	41 kg/d
Zn Stirred Mill Grinding Media – 3 mm ceramic	500 kg bags	40 kg/d

Source: JDS (2018)

17.4.6 Plant Air

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

Blowers will be used to supply air to the flotation cells.

17.4.7 Water

Fresh water will be supplied from nearby streams and/or the underground mines. The water will be stored in the firewater tank with the top portion flowing by gravity into the plant for gland services, reagent mixing and spray water.

The source of process water will be reclaimed tailings pond water and concentrate thickener overflows. This will be used as make-up water throughout the plant.

17.4.8 Assay Laboratory

The Assay Laboratory will consist of a sample preparation/metallurgical module and a wet laboratory module. The two containers will be located outside the process plant.

The Laboratory will perform testwork for the underground mine workings, the mill, and the environmental group. Atomic absorption (AA) machines will be used to measure the grade of Pb, Zn and Fe. Samples may also be analyzed for C, SiO_2, S, and SO_4. The concentrates will be tested for Pb, Zn, As, Sb, Hg, Fe, and Cd using the AA machine, and SiO_2 and C with be measured with other methods. The high grade concentrates will be assayed for Pb and Zn by titration or X-ray florescence.

Water samples will be tested to Yukon Water Board requirements prior to discharge into the surrounding streams. Two main tests will be performed, water quality and acid rock drainage (ARD) potential. Water samples will be analyzed for sulphates, ammonia, nitrates, nitrites, cyanide, thiocyanide, pH, and hardness. One sample will be collected for ADR testing bi-weekly. Samples will also be prepared to be sent for analysis by third party laboratories that meet regulatory standards.

18 Project Infrastructure and Services

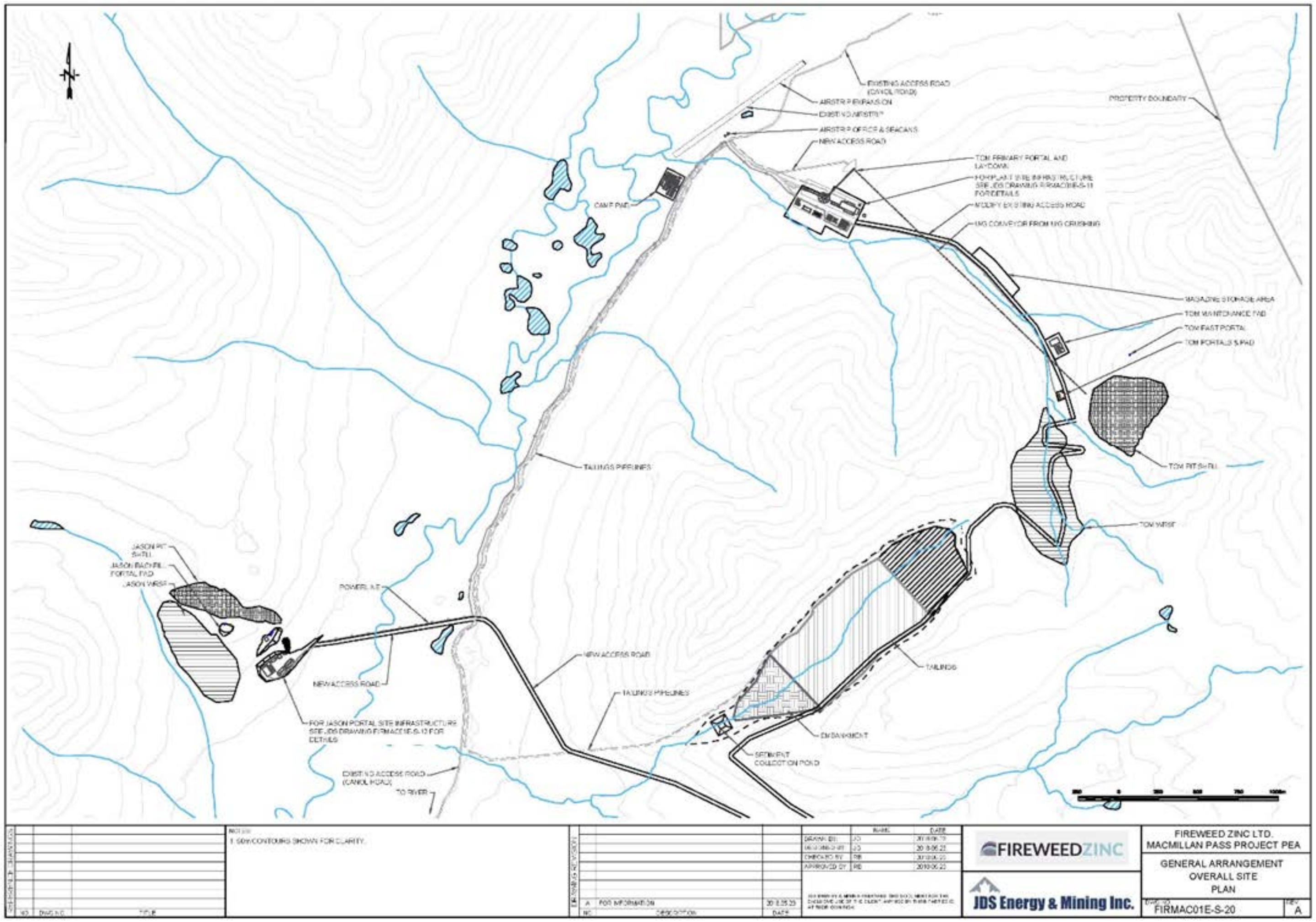
The project envisions the upgrading or construction of the following key infrastructure items:

- Upgrading approximately 230 km of the Canol Road, an all-seasonal access road from Ross River to the project site location;
- Primary crusher and coarse ore stockpile;
- Process plant facilities;
- LNG power plant and LNG storage facilities;
- On-site power distribution with overhead power lines;
- TMF and WRSF;
- Permanent camp (established for the construction stage);
- Administration and mine dry buildings;
- Truck shop and warehouse;
- 300,000 L of on-site fuel storage and distribution;
- Airstrip;
- Industrial waste management facilities such as the incinerator;
- Site sewage treatment facilities; and
- Site storm water management facilities.

18.1 General Site Layout

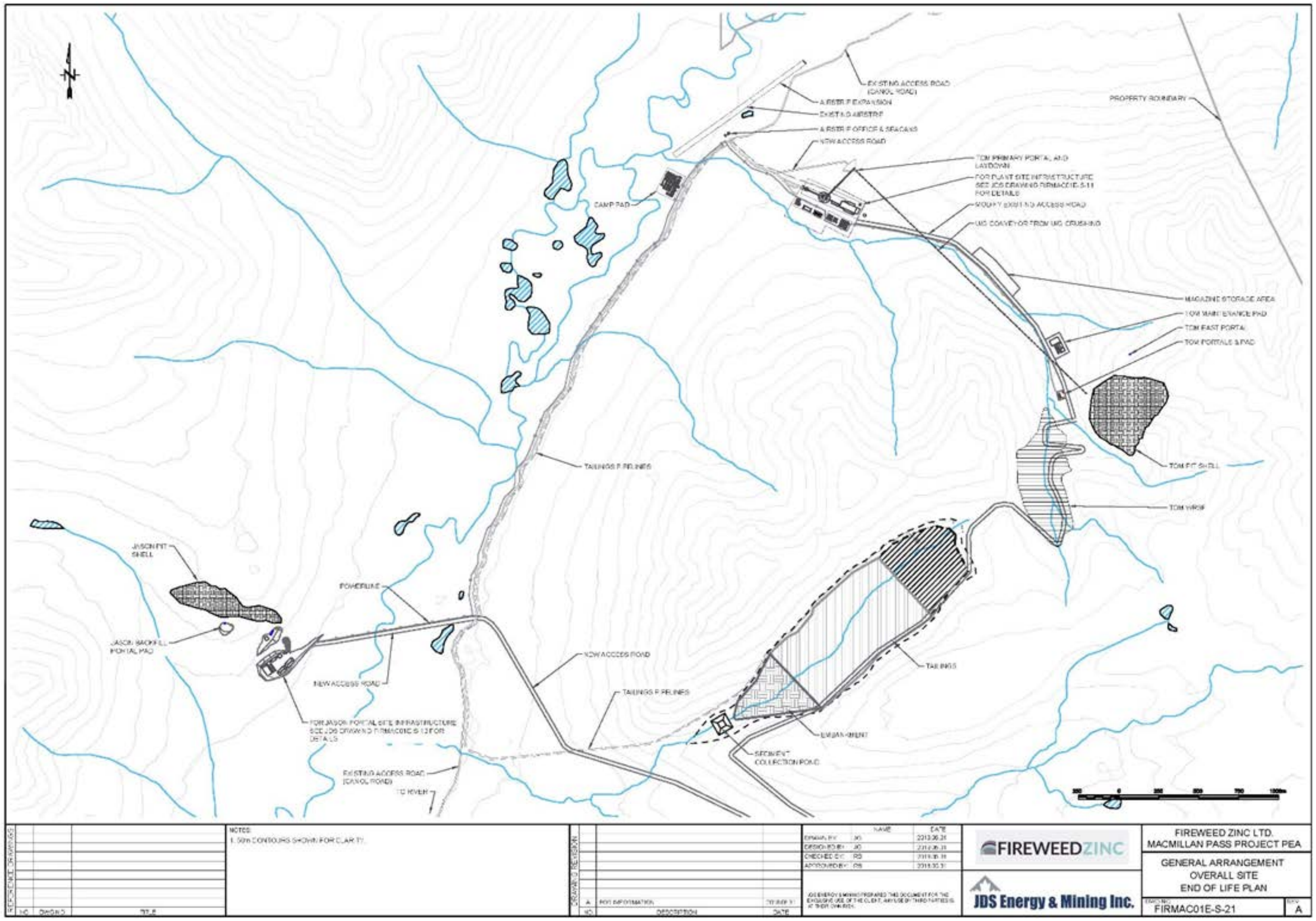
The overall project site layout site is shown in Figure 18-1, with maximum extent of the waste rock piles, and the overall project site in the final year of operation is shown in Figure 18-2. A layout for the envisioned plant site area is shown in Figure 18-3. The plant facilities are planned to be located on elevated ground, where historical geo-technical investigation shows the bedrock is shallow, and there is no permafrost. Foundation conditions and drainage requirements are expected to be better in this location as opposed to a lower lying area. The Jason Portal area supporting infrastructure is shown in Figure 18-4. The TMF would be located south of the plant site, in a valley. The proposed location minimizes the TMF footprint, construction earthwork volume and catchment area, while maximizing the storage capacity of the impoundment.

Figure 18-1: Overall Site Layout



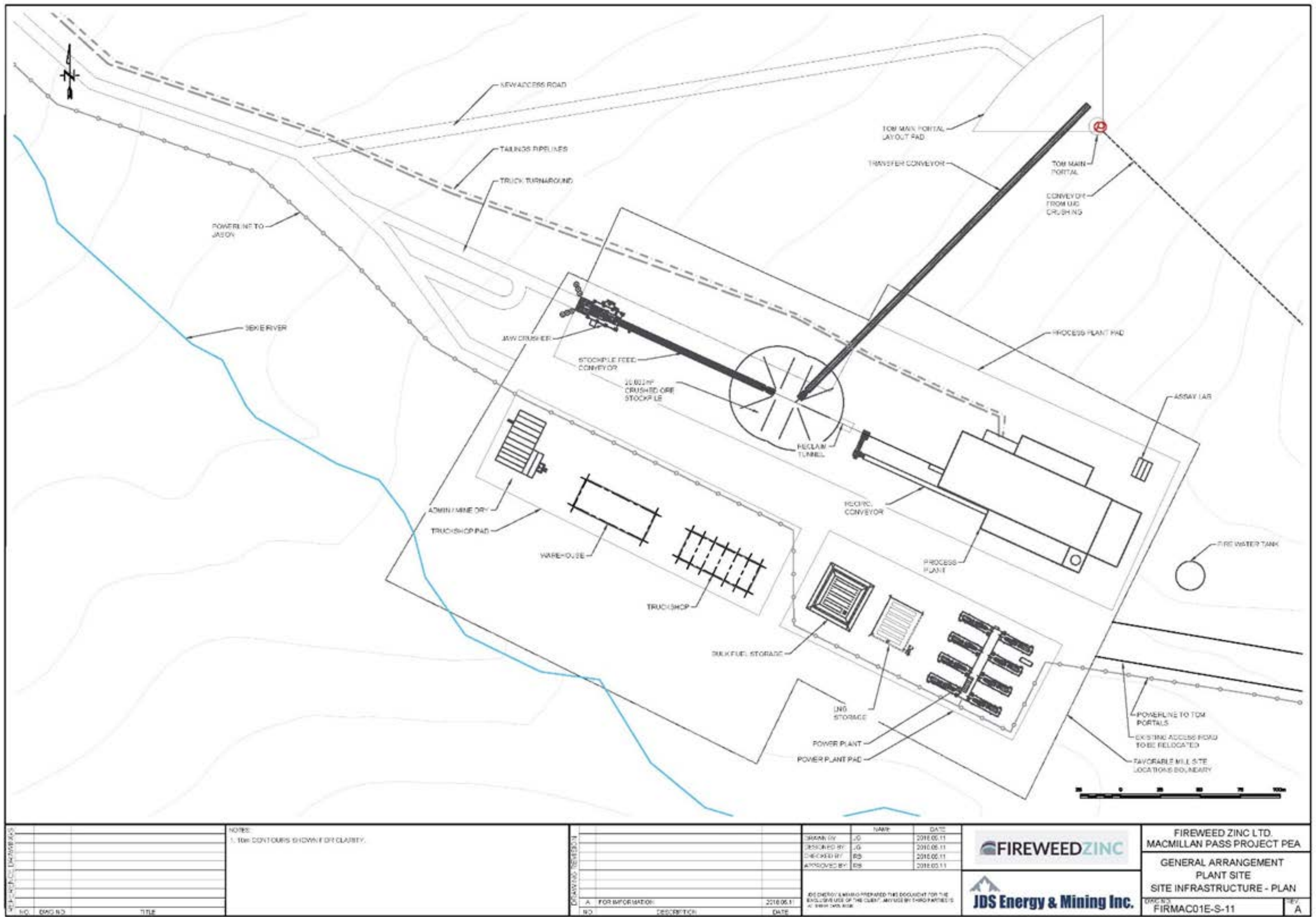
Source: JDS (2018)

Figure 18-2: Overall Site Layout in Final Year



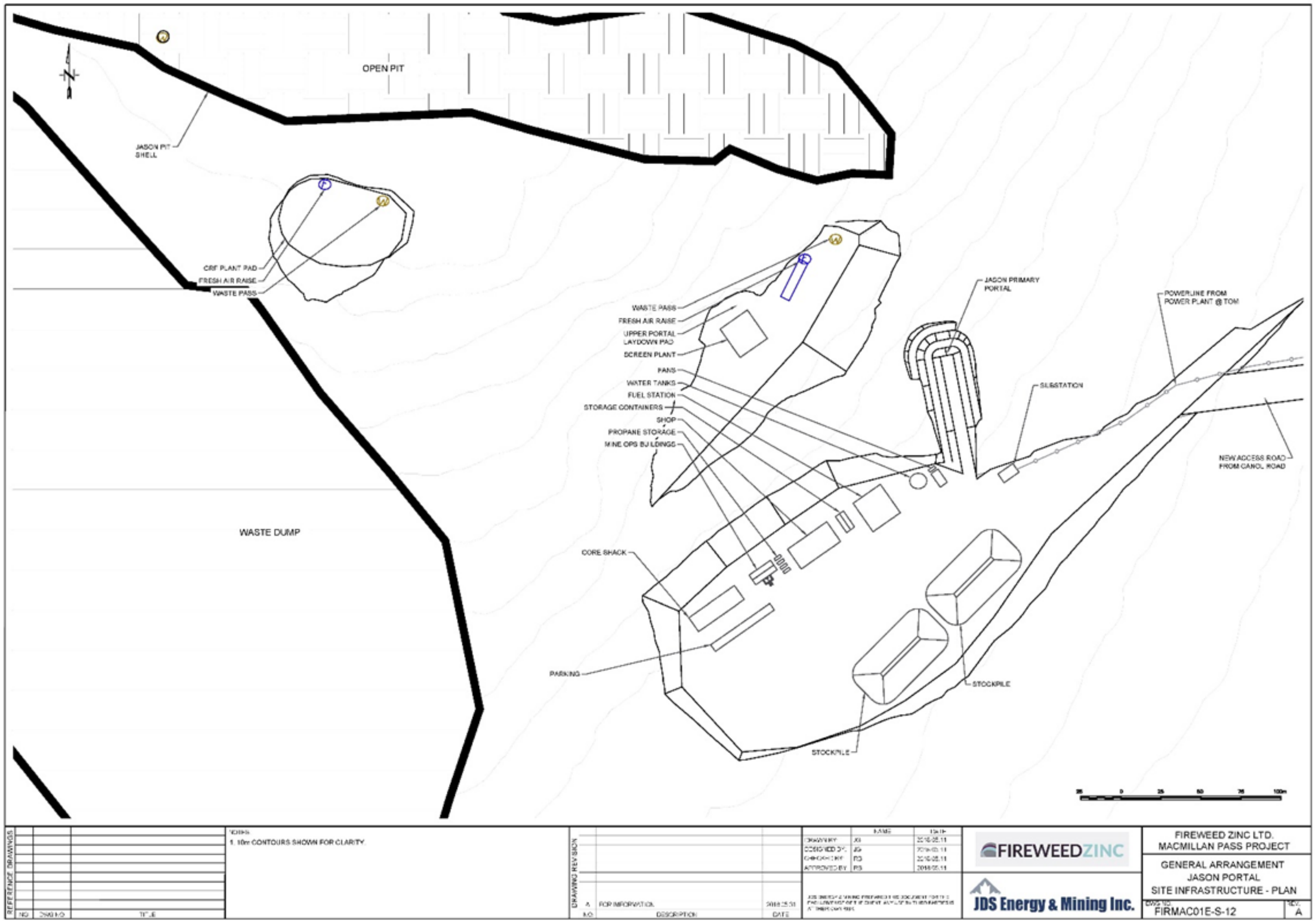
Source: JDS (2018)

Figure 18-3: Plant and Site Infrastructure



Source: JDS (2018)

Figure 18-4: Jason Portal Infrastructure

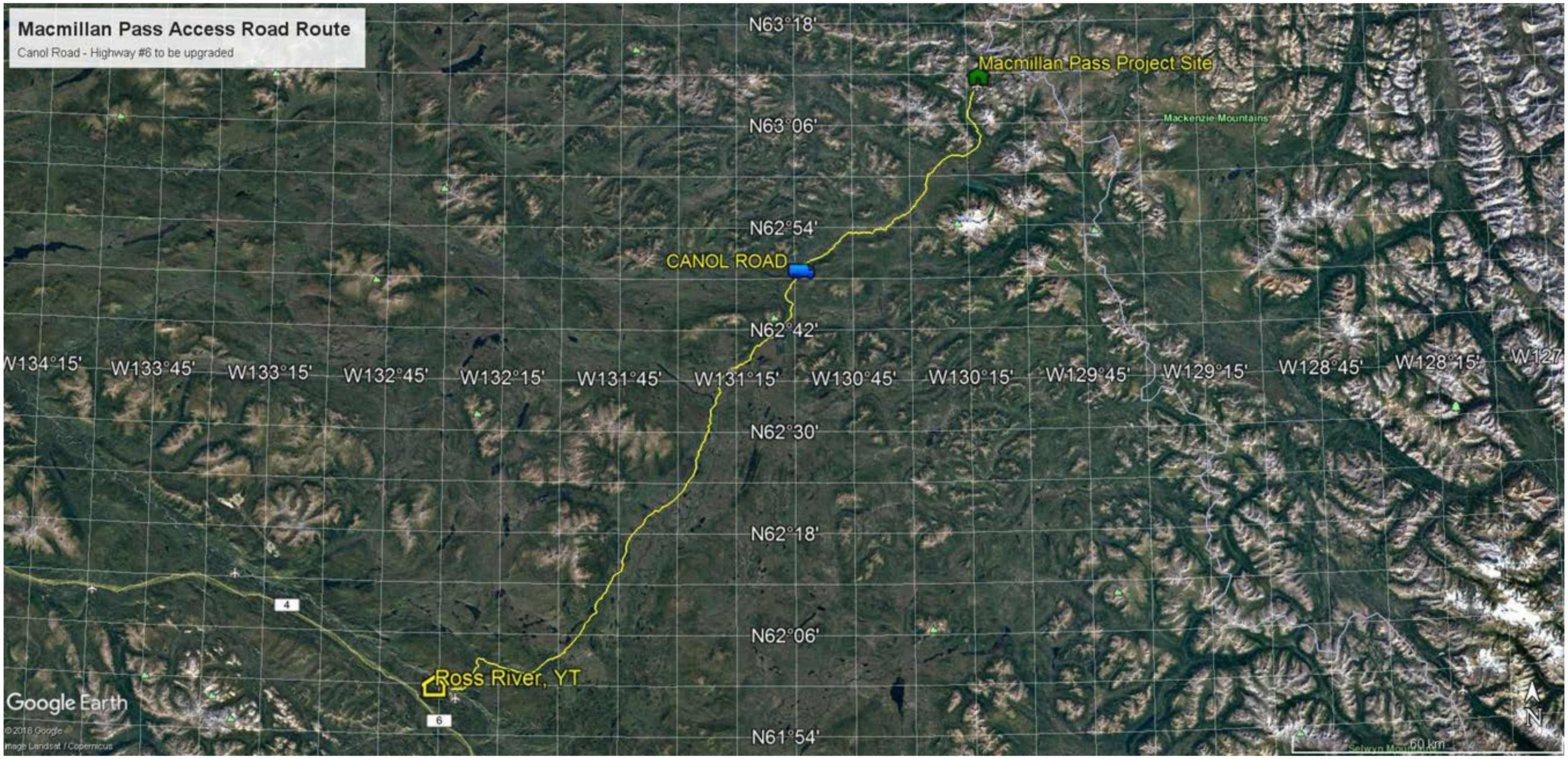


Source: JDS (2018)

18.2 Site Access Road

Approximately 230 km of an all season access road will have to be upgraded to support construction and mine operations from the project site to the ferry crossing at Ross River, BC. The Canol Road, or Highway #6, was built during World War II to access and maintain an oil pipeline route. It is currently used to transport supplies for exploration work at the Project site but is not in good condition and will need to be improved to service the Project for construction and operations. The final construction is planned to be a radio assist, single lane, gravel road with inter-visible turn-outs. The road will be approximately 5 m wide road with a maximum designed grade of 10%. The alignment of the road is shown in Figure 18-5.

Figure 18-5: Access Road Route



Source: Google Maps & JDS (2018)

18.3 Power Supply and Distribution

Power necessary to support the Macmillan Pass operation will be supplied by on-site generator sets. A single power plant set up comprising eight natural gas-fired reciprocating engine generator sets (gensets) in a N+2 (6+2) arrangement will provide electricity to operate the mine, processing plant and site infrastructure. Each genset will be driven by a 2,500 kW cat engine G3520H (or equivalent) operating at 1,500 rpm, and generating power at 13.8 kV. The plant will be initially set up with seven gensets to begin operation, with additional gensets added in year 1. Provision has been allowed for to set up an additional smaller gensets to operate the camp through years 4 to 10, when the underground mine draws peak demand.

To maximize the overall efficiency, this power plant will operate as a combined heat and power plant (CHP Plant), providing heat to the process plant and site infrastructure buildings at the plant site.

The power plant will be modular with all gensets interconnected. Each genset will be packaged in a walk-in, sound-attenuated enclosure that is constructed, assembled and tested prior to shipment to site.

A LNG storage facility with sufficient capacity for five to seven days of operation, with vaporizer and a bermed containment area will provide fuel for the power plant.

18.4 Process Plant

The process plant is planned to be located in a pre-engineered structural steel building with dimensions of 121.5 m long by 36 m wide. Additional lean-to areas will house the re-agent area, lead concentrate filtering and load out, and the control room and the plant maintenance shop. Overhead cranes will be provided for equipment maintenance. The building will be heated to 5°C by glycol air handlers and unit heaters.

18.5 Ancillary Facilities

18.5.1 Camp

The camp will comprise single-occupancy rooms with central washrooms. It will be used during the construction stage and throughout the operations stage. There will be seven dormitory wings, each capable of housing 42 people for a total of 294 beds.

The kitchen / dining / recreation complex will include the following:

- Kitchen complete with cooking, preparation and baking areas, dry food storage and walk-in freezer / cooler. The kitchen will be provided with appropriate specialized fire detection and suppression systems;
- Dining room with serving and lunch preparation areas;
- First aid room;
- Mudroom complete with coat and boot racks, benches and male-female washrooms;
- Housekeeping facilities;
- Reception desk and lobby; and
- Recreation area.

The camp will be constructed from modular units manufactured off-site in compliance with highway transportation size restrictions. Camp modules will rest on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will connect the main camp complex and dormitory wings.

18.5.2 Truck Shop and Warehousing

A main truck shop will service the Tom UG mine at the process plant location, and a smaller satellite facility will service the Jason UG mine. The Tom truck shop will be a 48 m long by 18 m wide structural steel, pre-engineered building designed to accommodate facilities for repair and maintenance of mining equipment and light vehicles. The Jason facility will be a similar building, 32m long by 18 m wide. These buildings will also provide warehouse storage space for the mine operations.

A warehouse will be located at the plant site, consisting of an insulated sprung structure with overhead doors. Covered cold storage will also be provided at each portal with 40-ft. sea containers.

18.5.3 Mine Dry and Office Complex

The main site office complex, and the Tom portal and process plant mine dry will be located at the process plant site. A small satellite facility will be located at the Jason portal. Each will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will rest on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout.

The mine dry at the plant site facility will service construction and operations staff during the life of the project. It will contain the following:

- Male and female clean and dirty lockers;
- Showers and washroom facilities with separate male and female sections;
- The site office facility will contain the following items:
    - Private offices;
    - Main boardroom; and
    - Mine operations line-up area.

18.5.4 Fuel Storage

On-site diesel fuel storage is designed with a one week supply capacity. A total of four 75,000 L tanks will be installed within a lined containment berm. Fuel dispensing equipment for mining, plant services, and freight vehicles will be located adjacent to the fuel tank bund and the fueling area will drain into the bund. A fuel transfer module will provide fuel to the power plant day tank and diesel consumers in the process plant. One station with three tanks will support the site surface equipment and Tom portal, and a single tank station will support the Jason portal.

LNG and diesel will be transported by contractor to the Tom and Jason site on a daily basis via the main access road.

18.5.5 Explosives Storage

Explosive storage at the Tom and Jason Project would consist of the following components:

- Bulk ammonium nitrate (AN) storage and loading facility; and
- Explosive storage magazines.

Bulk AN prill will be shipped to site in one-tonne tote bags within 20-ft ISO containers. The AN storage area will allow for a one week supply of AN.

Packaged explosives and explosive detonators will be stored in approved explosive magazines located on separate pads. The powder magazine will be a 40-ft container magazine holding explosives, and the cap magazine will be a 20-ft container magazine holding detonators.

The design of all storage facilities will meet government regulations and will be located according to required separation distances as regulated by the Explosives Regulatory Division (ERD) of Natural Resources Canada (NRC).

18.5.6 Air Strip

An existing air strip at the project site will be upgraded to serve personnel transport for the construction and operating periods. The strip will be lengthened to 1,000 m with additional turn around extensions at each end. Navigation aids and full lighting will be installed for year round use. The strip will be capable of handling 40 passenger aircraft, such as a Dash 8-100, and the project will be served by charter aircraft flying out of Whitehorse, YT.

18.6 Waste Management

The waste rock management strategy takes advantage of topographic conditions at the deposits. The waste rock storage sites are adjacent to the proposed Open Pit locations and are identified as the Tom and Jason Waste Rock Management Facilities (TWRMF and JWRMF).

Waste rock will be hauled along contour and pushed out across the basin in increments. The TWRMF layout provides capacity for approximately 16.2 Mt of waste rock stacked at an overall slope of 3H:1V (for closure). The JWRMF provides capacity for approximately 5.2 Mt of waste rock storage stacked at an overall slope of 2.5H:1V. The facilities will be developed in stages depending on the mining sequence.

The TWRMF will be covered with a geomembrane liner and capped to prevent infiltration. Contact water from the seepage collection system will be treated and shallow foundation seepage in the bedrock will be directed to the Tom Open Pit as part of the long-term closure plan. The JWRMF is a temporary stockpile as waste rock from the JWRMF is used for underground backfill and will not be a component for the closure and reclamation plan.

18.6.1 Tom Waste Rock Management Facility (TWRMF)

The TWRMF has the following specific design features:

- Potential seepage from the TWRMF will be controlled by incorporating an underdrain system and seepage collection pond located downstream of the facility;

- Shallow foundation seepage in the bedrock will be directed to the open pit using a concrete cut-off wall at the downstream toe of the facility;
- Contact water will be collected in the underdrain system during operations for use in the process or treated and released; and
- Non-contact water diversion ditches will divert non-contact water around the facility.

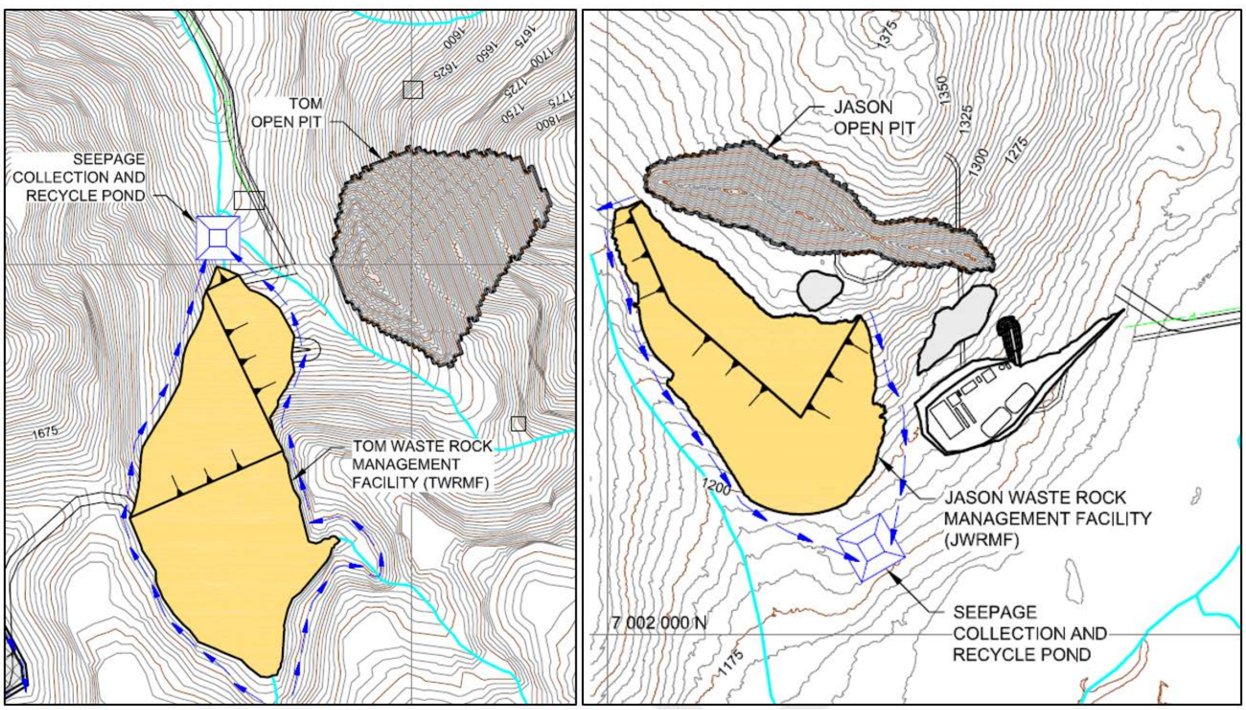
18.6.2 Jason Waste Rock Management Facility (JWRMF)

The JWRMF has the following specific design features:

- Potential seepage from the JWRMF will be controlled by incorporating an underdrain system and collection pond located downstream of the facility;
- Contact water will be used in the milling process or treated during the operational life of the facility;
- Non-contact water diversion ditches to divert non-contact water; and
- All waste rock will be used as underground backfill.

General arrangements for the Tom and Jason WRMFs are shown on Figure 18-6.

Figure 18-6: Tom and Jason WRMF General Arrangements



Source: Knight Piesold (2018)

18.7 Tailings Management

18.7.1 Tailings Management BAT Alternatives Assessment

The Tailings Management Facility (TMF) location and tailings technology selected for the PEA was identified in the Tailings Management BAT Alternatives Assessment completed in 2018 (KP, 2018). The BAT assessment considered two main tailings technologies and management strategies; conventional slurry tailings and filtered tailings. The weighted BAT assessment identified Candidate 6S, conventional slurry tailings management, as the preferred technology. The main factors for this conclusion are as follows:

- A greater ability to mitigate ARD/ML generation potential with continuous tailings deposition, wetting of the beach surface and maintenance of a pond within the facility;
- The tailings deposition and water management strategy is operationally simpler than the other candidates;
- Process and runoff water is contained within the same facility. Water for mill reclaim and surplus water treatment and release is sourced from the supernatant pond in the TMF;
- No additional mill processes are required; and
- There is a lower risk of operational problems (complications due to climatic conditions, etc.).

18.7.2 Tailings Management Facility Staging and Filling Schedule

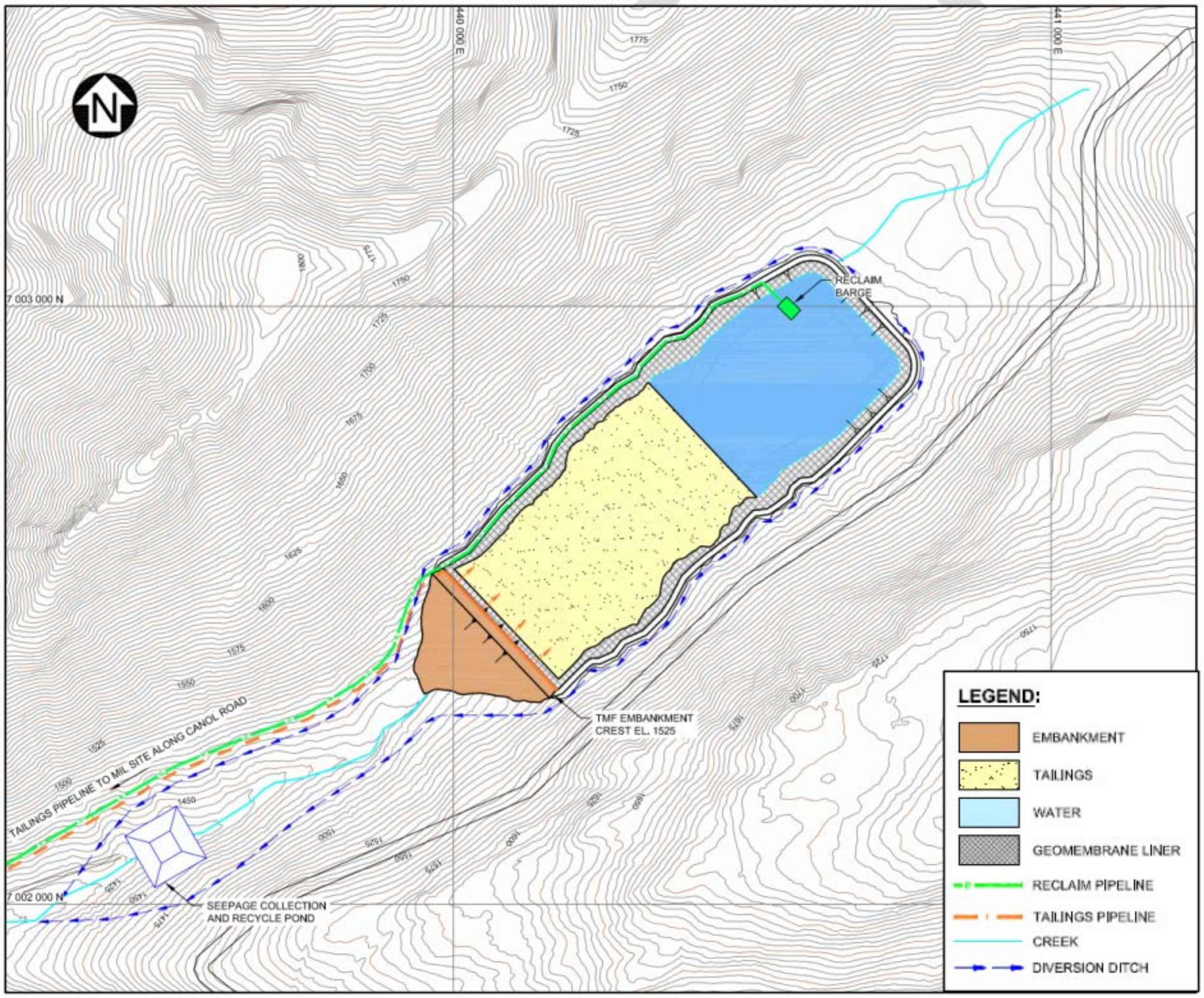
The preliminary design includes the development of the TMF in six stages over the 18 year mine life. This staged approach offers the following advantages:

- The ability to refine design, construction, and operating methodologies as experience is gained with local conditions and constraints;
- The ability to adjust plans at a future date to remain current with evolving best practices (engineering and environmental);
- To allow the observational approach to be utilized in the ongoing design, construction and operation of the facility. The observational approach can deliver substantial cost savings and a higher level of safety. It also enhances knowledge and understanding of site-specific conditions; and
- The potential to reduce initial capital costs and defer capital expenditure relating to TMF construction until the mine is operating.

The TMF has been designed to store 27.8 Mt of tailings, process water and an allowance for storm storage and freeboard. The initial settled dry density of the tailings was estimated to be 1.1 tonnes per cubic meter (t/m^3), consolidating to 1.3 t/m^3 by Year 3. An initial starter dam will be constructed to contain the first two years of tailings and associated water management. The dam would be raised over the mine life to increase the storage capacity while maintaining a minimum freeboard at all times.

The Stage 1 general arrangement is shown on Figure 18-7: TMF Stage 1 General Arrangement The facility would provide sufficient freeboard to manage contact water run-off, storm storage and process water. Reclaim water would be recirculated back to the mill and used as process water.

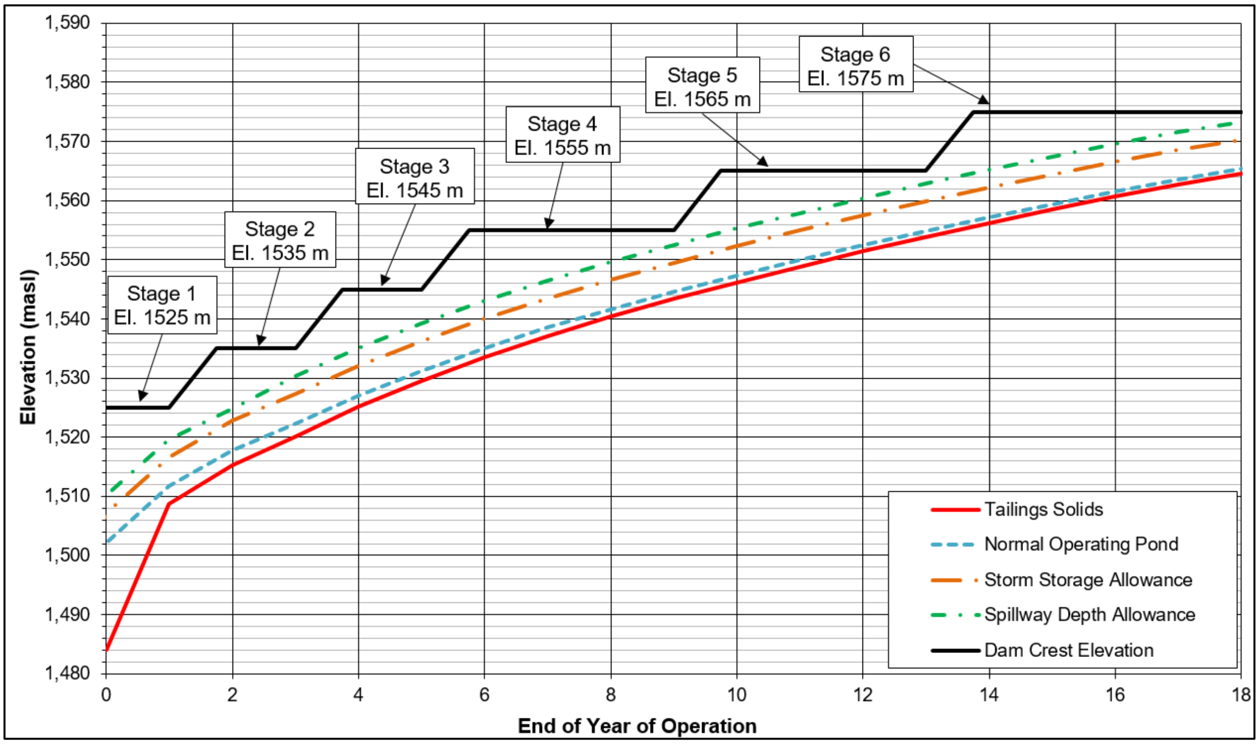
Figure 18-7: TMF Stage 1 General Arrangement



Source: Knight Piesold (2018)

The embankment stages are shown on the TMF filling schedule on Figure 18-8: TMF Filling Schedule. The embankment will be raised in 10 m lifts to provide the required storage capacity in the facility at each stage. The filling schedule and timing for staged expansions will be reviewed on an on-going basis during operations. The actual rate of filling may vary, depending on a variety of operating factors.

Figure 18-8: TMF Filling Schedule



Notes:

1. Tailings tonnages and rampup schedule provided by JDS (JDSFWZ – Material Balance – R10B-20180514)
2. Allowance for a minimum of 3 metres of operational freeboard included.
3. Allowance for a minimum of 5 metres included for storm storage allowance.
4. Average settled tailings dry density assumed to be 1.1 t/m^3 in Year 1 and consolidating to 1.30 t/m^3 by Year 3.

Source: JDS & Knight Piesold (2018)

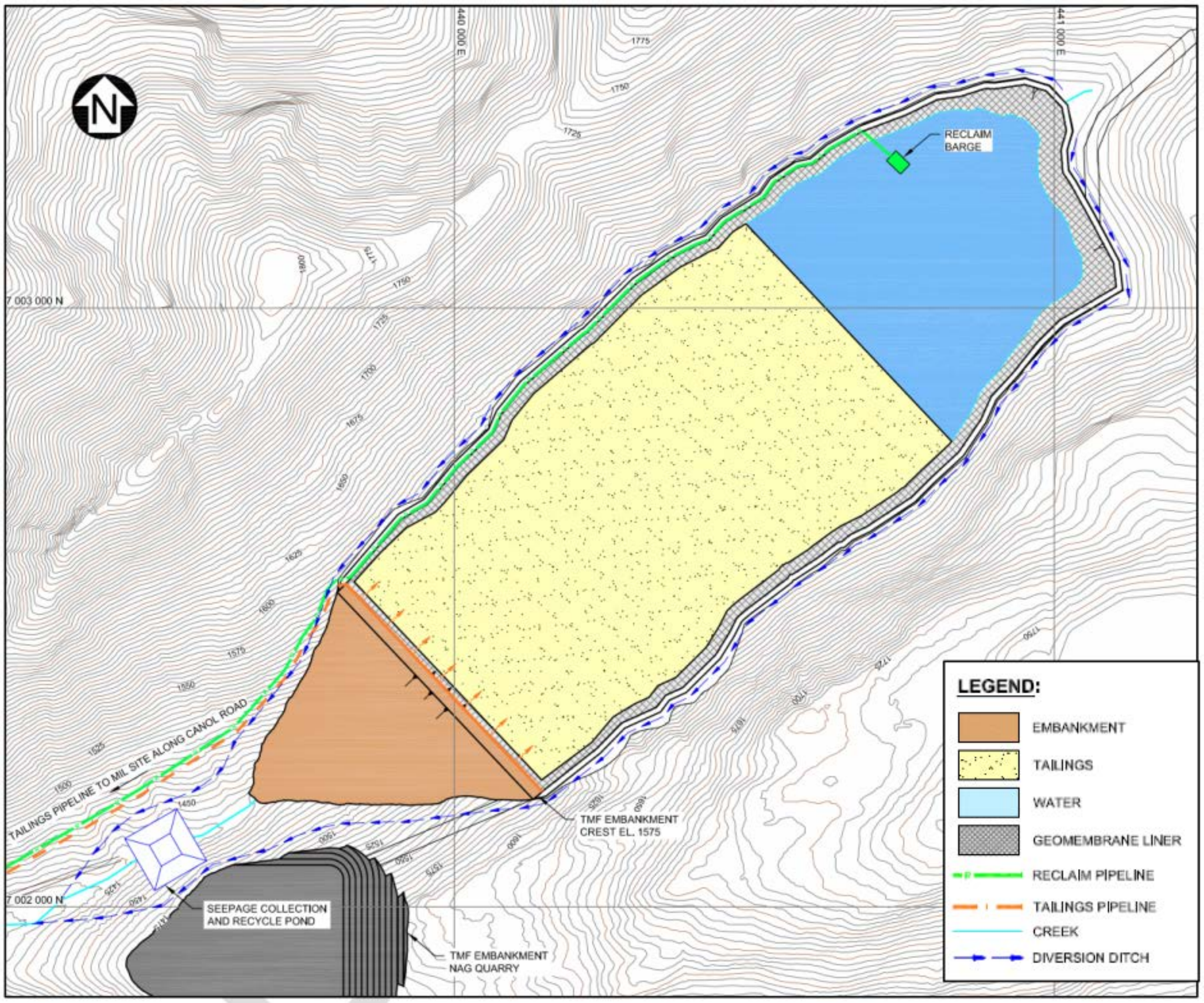
18.7.3 Tailings Management Facility Design

The TMF impoundment is assumed to be fully lined with a geomembrane liner at this stage of design. The impoundment basin and upstream face of the embankment will be lined to minimize seepage from the facility. The filter zones provide a bedding surface for the liner and will aid in preventing the migration of fines in the event of liner damage. The TMF will be expanded through the downstream method of construction using locally borrowed materials. The TMF site has capacity for future expansion if required. The TMF final arrangement is shown on Figure 18-9: TMF Final General Arrangement. The TMF has the following specific features for tailings and water management:

- Zoned embankment with processed filter / drainage zones;
- Borrow sources will be located inside the impoundment where possible;
- Fully lined impoundment to minimize seepage losses;
- Basin underdrain system;
- Foundation drainage system;

- Tailings beach;
- Tailings distribution system;
- Reclaim water system; and
- Non-contact water diversion ditches.

Figure 18-9: TMF Final General Arrangement



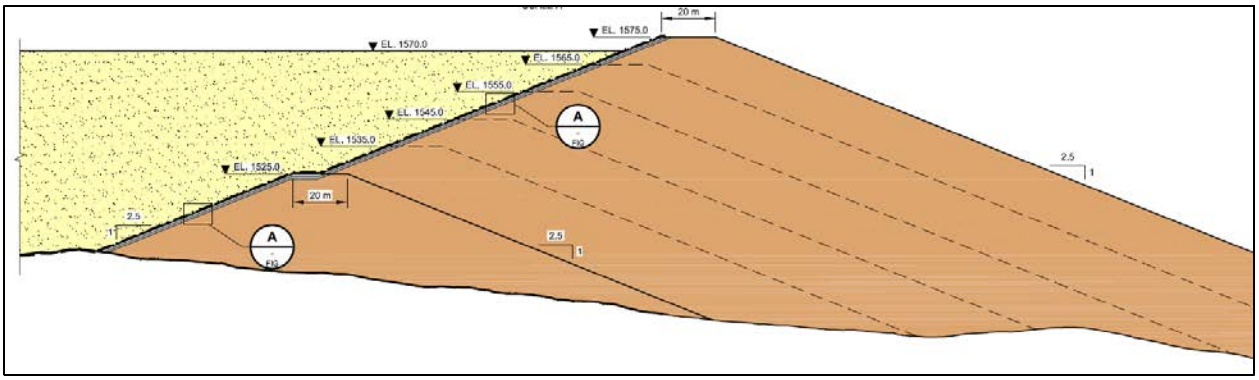
Source: Knight Piesold (2018)

The TMF is created by constructing one cross-valley embankment using material from borrow sources within the TMF basin or from other local borrow sources. Potential borrow areas will need to be investigated to determine their suitability for construction material.

The tailings embankment is designed to be a rockfill structure with granular filter zones on the upstream face. The embankment construction material will be borrowed from the TMF impoundment where possible and the entire facility will be lined with a HDPE geomembrane liner.

The TMF embankment is raised with the downstream method of construction to facilitate staged lining of the upstream embankment face. The upstream slope is 2.5H:1V to facilitate geomembrane placement. The downstream slope is 2.5H:1V to facilitate reclamation on the embankment slope. The minimum embankment crest width is 20 m to allow working space for liner anchoring, haul traffic, and tailings pipelines. The maximum embankment height is approximately 120 m, while the maximum elevation difference between the dam crest and the lowest ground elevation at the toe of the embankment is 140 m. The starter embankment is constructed to El. 1,525 m and has an approximate fill volume of 0.9 Mm^3. The final embankment is constructed to El. 1,575 m and has a total fill volume of approximately 6.4 Mm^3. The embankment cross section is shown on Figure 18-10.

Figure 18-10: TMF Embankment Section



Source: Knight Piesold (2018)

The majority of fill for the Stage 1 embankment will be general rockfill sourced from TMF basin shaping activities for liner installation in the impoundment. Subsequent embankment expansions will incorporate construction materials from inside the impoundment where possible and local borrow pits. External borrows would be located downstream of the embankment to facilitate a relatively simple and close haul to the downstream toe. The upstream face of the embankments includes a layer of filter sand, which will function as a geomembrane liner bedding. The geomembrane liner will be installed on the filter sand material. Instrumentation is included for ongoing monitoring of the performance of the TMF embankment. The instrumentation will include vibrating wire piezometers installed in the foundation and embankment fill, in addition to inclinometers and survey monuments.

The TMF will be designed to safely manage and store the Inflow Design Flood. An emergency spillway will be incorporated into the embankment abutment at each stage, as a contingency to convey excess water safely from the TMF should the water in the facility be managed incorrectly. The primary objective of the spillway is to protect the integrity of the TMF embankment during an emergency and is not intended to be used at any stage during operations.

18.7.4 Seepage Control Measures

Potential seepage from the TMF will be controlled by incorporating the following measures:

- Foundation Drain;
- Geomembrane Liner; and
- Basin Underdrain.

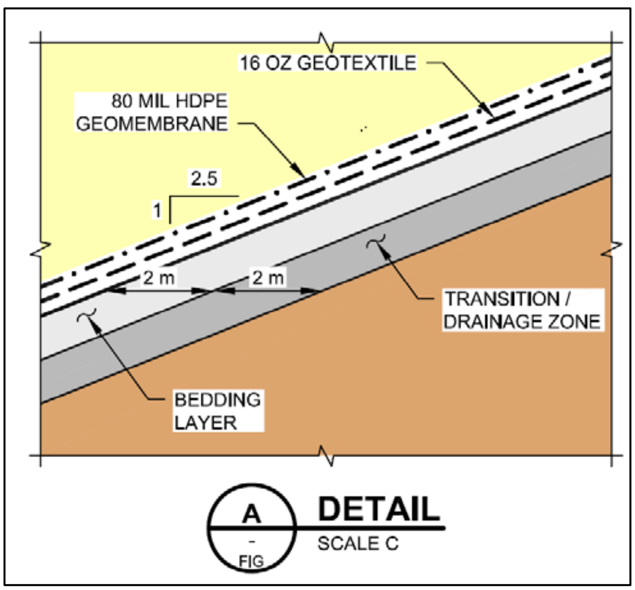
Foundation Drain

A foundation drains will be installed below the geomembrane liner to collect groundwater flows and potential seepage. Collected water will drain to the seepage collection and recycle pond downstream of the TMF embankment. The foundation drain comprises interconnected perforated pipes surrounded by drain gravel. The foundation drain will be constructed using processed material generated during construction of the TMF, or from local borrow sources. The foundation drain will be constructed beneath the TMF liner bedding layer.

Geomembrane Liner

The entire TMF basin, including the upstream embankment face, will be lined with an 80-mil HDPE geomembrane. The liner system includes a layer of 16 oz/yd^2 non-woven geotextile below the geomembrane, for protection from the adjacent materials. The liner system also incorporates a prepared subgrade comprising processed bedding material. The geomembrane is effectively impermeable, with seepage only possible through defects that may occur during fabrication and/or installation. The liner system will need be anchored into the foundation and embankment. The geomembrane liner detail is shown on Figure 18-11 below:

Figure 18-11: Embankment Geomembrane Liner Detail



Source: Knight Piesold (2018)

Basin Underdrain

An internal basin underdrain will be installed above the geomembrane on the basin floor to promote tailings consolidation while maintaining a low head on the geomembrane. The basin underdrain will connect to an internal wet well sump and recycle pump system. Collected water will be recycled to the TMF supernatant pond. The basin underdrain will be constructed using processed material generated during construction of the TMF, or from local borrow sources. The underdrain includes perforated drain pipes within a free draining surround. A 300 mm thick layer of filter sand will be placed on the basin floor above the geomembrane liner and surrounding the drain pipes to assist in providing drainage while preventing the migration of fine tailings and protecting the geomembrane liner.

Seepage Collection and Recycle

A seepage collection and recycle pond will be located downstream of the embankment and will collect seepage from the TMF basin, runoff from the downstream slope of the TMF embankment slope and flow from the foundation drain. Water collected in this pond will be recycled to the TMF supernatant pond using submersible pumps and HDPE pipelines.

18.7.5 Tailings Management Facility Operations

Tailings will be delivered to the TMF from the mill via a tailings pumping system in a single tailings pipeline. The pipeline will be installed with a heat tracing system to prevent freezing during winter operations. Tailings will be discharged from the delivery pipelines into the TMF from a series of valved off-takes located along the TMF embankment crest. The sandy coarse fraction of the tailings will settle rapidly after discharge and will accumulate close to the discharge points, forming a gentle beach. Finer tailings particles will travel farther and settle at a steeper slope adjacent to and beneath the supernatant pond. The tailings beaches will be developed with the intent to maximize storage volume and to control the location of the supernatant pond. Selective tailings deposition will be used to maintain the supernatant pond away from the embankment.

Water will be reclaimed from the tailings pond using a floating reclaim barge and reused as process water. In order to float the barge and to provide adequate process water requirements at start up, the TMF will first be partially filled with water from surface runoff. The reclaim water pipeline will extend from the barge to the mill along Canol Road.

18.8 Water Management

The water management strategy for the Project utilizes water from within the project area to the maximum practical extent. The plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of process water. Site runoff water will be stored on site within the TMF. The primary sources of contact water for the Project are as follows:

- Tom and Jason underground dewatering flows;
- Tom and Jason open pit dewatering flows;
- Precipitation runoff from site facilities; and
- Water recycle from the tailings supernatant ponds.

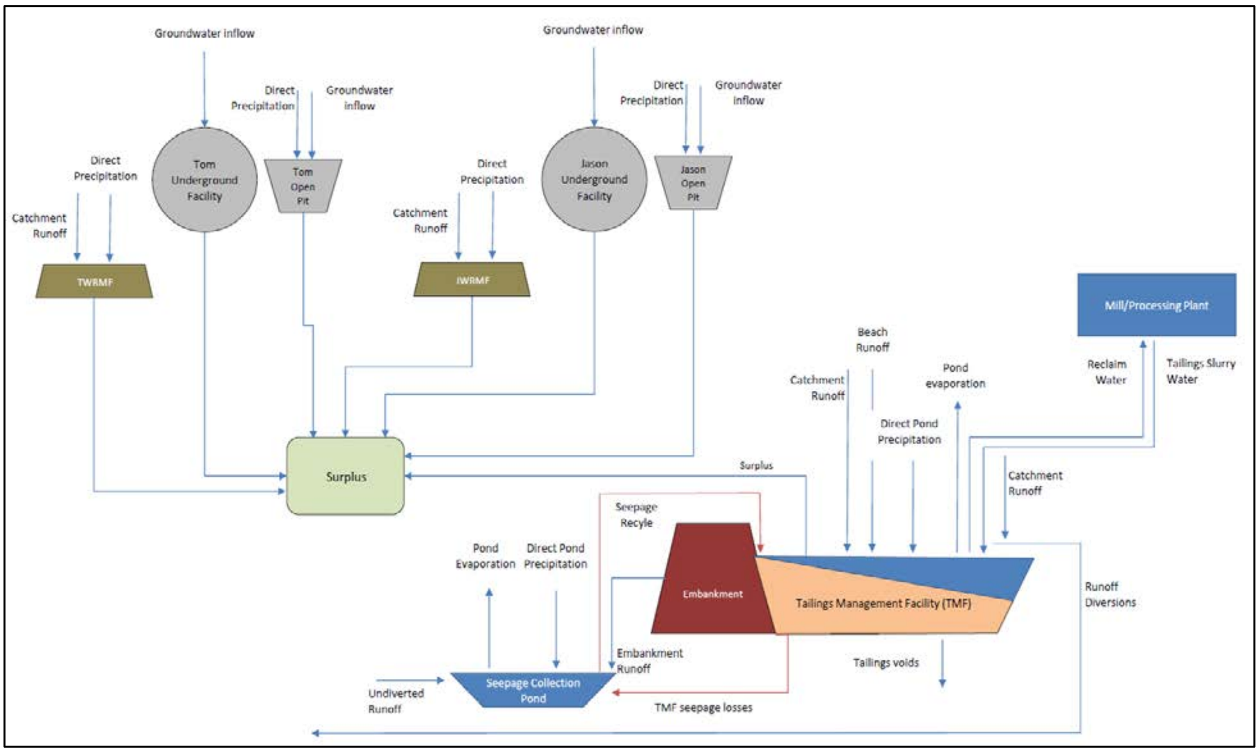
Diversion channels will be constructed to route non-contact water from the upstream catchment around site components including the TMF, the two WRMFs (Tom and Jason), the two Open Pits (Tom and Jason), and the mill to minimize the amount of contact water to be managed. All contact water from these facilities or their associated construction areas will be managed in either the TMF supernatant pond, water management ponds, or treated using a Water Treatment Plant and discharged.

Dewatering flows from the Tom and Jason deposits (both underground and Open Pit dewatering) will be managed close to each deposit before being discharged, with treatment (if required). Contact water from the mill site, stockpiles and WRMFs will be collected and recycled to the TMF supernatant pond.

A preliminary water balance model was prepared to estimate the magnitude of annual surplus or deficit conditions for the Project. The model was developed using a monthly time step and included inputs and losses from the mill, mine development (open pits and undergrounds), stockpiles, Waste Rock Management Facilities (WRMFs) and the Tailings Management Facility (TMF). Non-contact water was assumed to be diverted around the mine facilities to the downstream waterways wherever possible. A schematic of the water management plan used in the water balance model is shown on Figure 18-12.

The preliminary water balance indicates that there is sufficient water to satisfy the mill requirements without additional make-up water under average climate conditions. The TMF is estimated to be in an annual surplus of approximately 0.2 Mm^3/year.

Figure 18-12: Site Wide Water Balance Flow Schematic



Source: KP (2018)

18.9 Closure and Reclamation

TMF closure and reclamation activities will be carried out concurrently during operations (where possible) and primarily at the end of economically viable mining. Closure and reclamation activities will be in line with international closure standards. Measures must be taken to ensure that:

- Dust is not emitted from the facility as a result of the loss of moisture from the surface of the TMF;

- Runoff does not affect surface or groundwater;
- The TMF embankments remain stable; and
- The stored tailings remain physically and chemically stable.

The primary objective of the closure and reclamation initiatives will be to eventually return the TMF site to a self-sustaining facility with pre-mining land capability. The TMF will be required to maintain long-term geochemical and physical stability, protect the downstream environment and manage surface water. Activities that will be carried out during operations and at closure to achieve these objectives are discussed below.

Surface facilities will be removed in stages and full reclamation of the TMF will be initiated upon mine closure. General aspects of the closure will include:

- Selective discharge of tailings around the facility prior to closure to establish a final tailing beach that will facilitate surface water management and reclamation;
- Removal of surface water ponds and capping;
- Geomembrane cover over the TMF surface and placement of a soil cover over the geomembrane;
- Dismantling and removal of the tailings and reclaim delivery systems and all pipelines, structures and equipment not required beyond mine closure;
- Establishment of a permanent spillway at the TMF;
- Removal of the seepage collection pump-back systems at such time that suitable water quality for direct release is achieved;
- Removal and re-grading of all access roads, ponds, ditches and borrow areas not required beyond mine closure; and
- Long-term stabilization and vegetation of all exposed erodible materials.

TWRMF closure and rehabilitation activities will be carried out concurrently during operations (where possible) and primarily at the end of economically viable mining. Closure and rehabilitation activities will be in line with international closure standards, so that:

- Runoff does not affect surface or groundwater; and
- The facility remains physically and chemically stable.

General aspects of the closure activities will include:

- Placement of an engineered closure cover to minimize infiltration to the facility;
- Establishment of closure water management ditches;
- Contact water collected in seepage collection system and treated as part of the long term closure plan; and
- Shallow foundation seepage in the bedrock directed to the Tom open pit as part of the long-term closure plan.

The JWRMF is a temporary stockpile and will not be a component for the closure and reclamation plan.

Groundwater monitoring wells and all other geotechnical instrumentation will be retained for use as long-term dam safety monitoring devices. Post-closure requirements will also include annual inspection of the TMF and an ongoing evaluation of water quality, flow rates and instrumentation records to confirm design assumptions for closure.

Industry standard reclamation methods will be employed to close out the remainder of the Project sites. Hazardous materials will be collected for offsite disposal including hazardous components of vehicles and equipment (i.e., fuel tanks, gear boxes and glycol-based coolant). Buildings and equipment stripped of hazardous components will be demolished and disposed in an approved landfill, offsite. Culverts will be removed from roads and the natural drainage restored, but the roads will otherwise remain intact.

Once all buildings, facilities and equipment have been removed, the footprints (whether bedrock or pads) will be re-contoured to allow for restoration of natural drainage to the receiving environment.

19 Market Studies and Contracts

19.1 Market Studies

No market studies have been completed for the project at this time, but the concentrates are very clean and likely to be attractive to smelters.

19.2 Contracts

No contractual arrangements for smelting exist at this time. Furthermore, no contractual arrangements have been made for the sale of zinc or lead concentrate at this time.

19.3 Metal Prices

The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical lead, zinc and silver prices are shown in Figure 19-1, Figure 19-2 and Figure 19-3. Historical exchange rate trends are plotted in Figure 19-4.

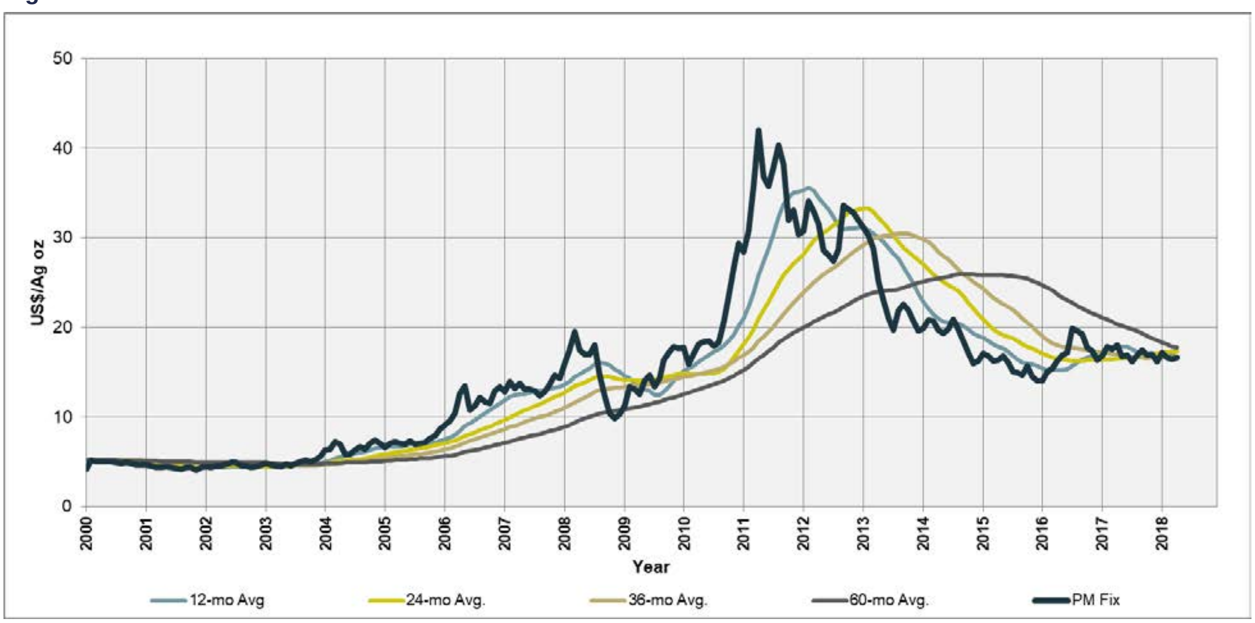
Figure 19-1: Historical Lead Price

Source: London Metals Exchange (2018)

Figure 19-2: Historical Zinc Price

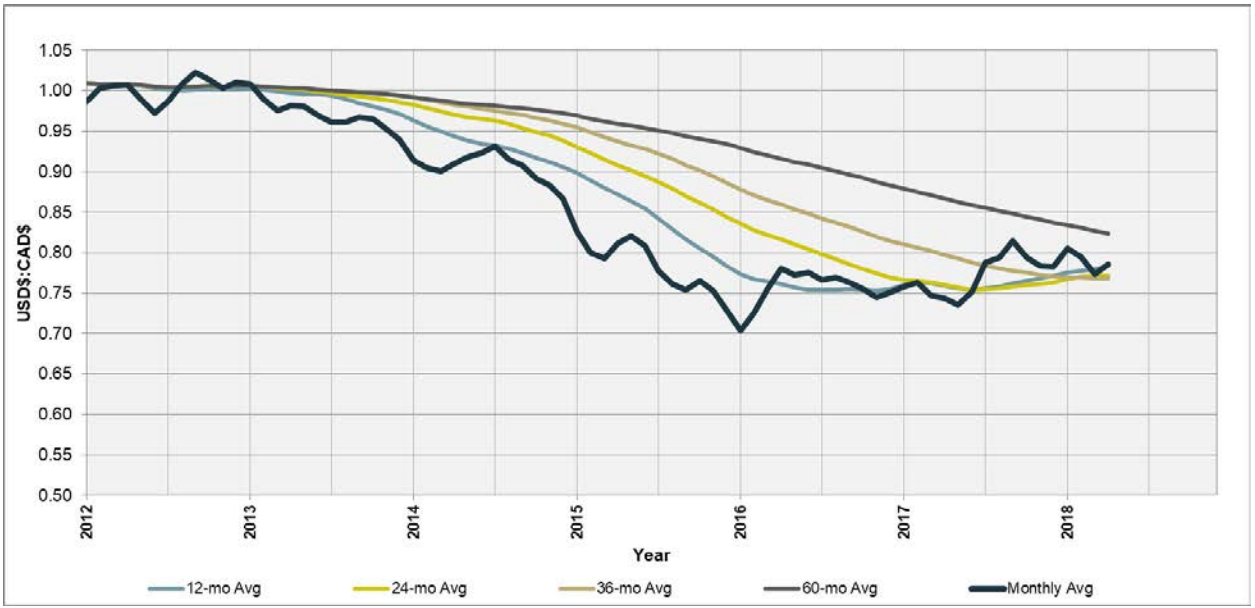
Source: London Metals Exchange (2018)

Figure 19-3: Historical Silver Price



Source: London Metals Exchange (2018)

Figure 19-4: Historical US$:C$ F/X Rates



Source: London Metals Exchange (2018)

The lead price, zinc price, and silver price used in this PEA study were selected based on the average of three years past and projected two years forward by analysis of London Metal Exchange futures as of 30 April 2018. These parameters are in line with other recently released comparable Technical Reports.

A sensitivity analysis on metal prices and exchange rates was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Figure 19-1 outlines the metal prices used in the PEA economic analysis.

It must be noted that metal prices are highly variable and are driven by complex market forces and are extremely difficult to predict.

Table 19-1: Metal Price and Exchange Rate

Parameter	Unit	Value
Lead Price	US$/lb	0.98
Zinc Price	US$/lb	1.21
Silver Price	US$/oz	16.80
Exchange Rate	US$:C$	0.77

Source: JDS (2018)

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Overview

In order to advance development of the Macmillan Pass project, Fireweed will require an assessment under the Yukon Environmental and Socio-economic Assessment Act (YESAA) prior to licensing and permitting.

In early 2018 upon exercise of the Hudbay option (see Section 4.3.1), Fireweed continued and expanded the collection of baseline environmental data, including social and heritage information. The collection of baseline information, which is ongoing, is centred on those elements which are important for assessment and regulatory processes and includes soils, surficial geology, geochemistry, wildlife, water, climate, vegetation, fish, aquatic resources, heritage resources, and socio-economic setting. The data collected to date is to inform both assessment and regulatory processes as well as establish baseline conditions for future monitoring.

Fireweed Zinc is committed to engaging with local communities and First Nations toward building respectful relationships though dialogue and collaborative processes.

The Macmillan Pass property lies within an area of unsettled overlapping territorial claims by the Kaska First Nations and Na-Cho Nyak Dun First Nation. Negotiations between the First Nations, and the federal and territorial governments are continuing but are not affecting Fireweed's ability to carry out work on the project.

The nearest community to the project site is the First Nations community of Ross River (population 350) located 200 kilometers to the southwest. There are no permanent settlements at or near the project site. Fireweed Zinc believes in preferential hiring of Ross River based businesses and personnel whenever practical, for work on the Macmillan Pass Project.

First Nation input and participation will be central to the planning and implementation of baseline data collection, re-vegetation trials, and closure and reclamation planning. Through ongoing dialogue, First Nations environmental and socio-economic values are being identified and will be incorporated into and reflected in the assessment.

20.2 Regulatory Licences, Permits and/or Authorizations

20.2.1 Overview

Concurrent with the assessment of the project proposal, Fireweed will be applying for an upgraded water licence from the Yukon Water Board, a quartz mining licence from the Government of Yukon, Department of Energy, Mines and Resources, and other authorizations. Although the environmental assessment process and permitting processes can run concurrently, authorizations cannot be issued in advance of the completion of the ESA process and the issuance of the Decision Document.

Table 20-1 provides a listing of some of the federal and territorial acts, regulations, guidelines applicable to the Macmillan Pass Project.

Table 20-1: Applicable Acts, Regulations and Guidelines Relevant to the Macmillan Pass Project

Acts	Regulations	Guidelines / Permits / Licences
Federal		
Aeronautics Act	Canadian Aviation Regulations, General Operating and Flight Rules	
Canadian Environmental Protection Act (1999 c.33)	Storage Tank Systems for Petroleum Products and Allied Petroleum Products Regulations	Canadian Council of Ministers for the Environment (CCME) - Environmental Code of Practice for Above-ground and Underground Storage Tank Systems Containing Petroleum and Allied Petroleum Products
		Notice with respect to substances in the National Pollutant Release Inventory
	Environmental Emergency Regulations [SOR/2003-307]	
	Interprovincial Movement of Hazardous Waste and Hazardous Recyclable Material Regulations	Environment Canada Technical Document of Batch Waste Incineration (January 2010)
	Pollutant Discharge Reporting Regulations, SOR-95-351	
Canada Water Act (1985 c.11)		
Canada Wildlife Act (1985 w9)		
Species at Risk Act (2002 c.29)		
Migratory Birds Convention Act (1994 c.22)	Migratory Birds Regulations (C.R.C., c. 1035)	
Fisheries Act [R.S.C. c. F-14]	Metal Mining Effluent Regulations [SOR/ 2002-2222]	
		Fisheries Productivity Investment Policy: A Proponent's Guide to Offsetting
		Freshwater Intake End-of-Pipe Fish Screen Guideline
Explosives Act (1985 c.E-17)	Ammonium Nitrate and Fuel Oil Order	Blasting Permit, Purchase and Possession Permit, Permit to Transport Explosives
	Explosives Regulations	
Navigation Protection Act (R.S. 1985 c. N-22)	Navigable Waters Works Regulations	Section 5(2) approval
National Fire Code of Canada		

Acts	Regulations	Guidelines / Permits / Licences
National Building Code of Canada		
Transport of Dangerous Goods Act [1992, c.34]	Transportation of Dangerous Goods Regulations [SOR/2001-286]	
Yukon First Nations Land Claims Settlement Act		
Yukon Environmental and Socio-economic Assessment Act	Assessable Activities, Exceptions and Executive Committee Projects Regulations	Decision Document
	Decision Body Time Periods and Consultation	
	Regulations	
First Nation of Nacho Nyak Dun Final Agreement	Heritage	
Territorial – Yukon		
Boiler and Pressure Vessel Act		Pressure Vessel Boiler Permit
Building Standards Act		Building Permit, Plumbing Permit
Environment Act	Air Emissions Regulations	Air Emission Permit
	Contaminated Sites Regulation	Land Treatment Facility Permit
	Storage Tank Regulations	Above-Ground Storage Tank Permit
	Solid Waste Regulations	Waste Management Permit
	Special Waste Regulations	Special Waste Permit
	Spills Regulations	Contaminated Material Relocation Permit
Forest Protection Act	Forest Protection Regulation	Open Burn Permit
Gas Burning Devices Act		Gas Installation Permit
		Gas Burning Devices Permit
Highways Act	Bulk Commodity Haul Regulations	Access Permit
	Highways Regulations	Work in Highway Right-of-way Permit
Occupational Health and Safety Act	Blasting Regulation	Blasters Permit
	Occupational Health and Safety Regulations	
	Workplace Hazardous Materials Regulation	
	Public Health Regulations	Sewage Disposal System Permit

Acts	Regulations	Guidelines / Permits / Licences
Public Health and Safety Act	Camp Sanitation Regulations	
	Sewage Disposal Systems Regulation	
	Drinking Water Regulation	
Quartz Mining Act	Quartz Mining Land Use Regulation	Quartz Mine Licence
	Security Regulation	
	Quartz Mining Fees and Forms Regulation	
Territorial Lands (Yukon) Act	Territorial Lands Regulation	Commercial Timber Permit
	Land Use Regulation	Land Use Permit
		Quarry Permit
		Aerodrome Licence
Waters Act	Waters Regulation	Type A Water Licence, Type B Water Licence
Wildlife Act	Wildlife Regulation	
Workers Compensation Act	Multiple Requirements	
Yukon Land Claim Final Agreements, An Act Approving		
Yukon Historic Resources Act	Archaeological Sites Regulations	Archaeological Sites Permit

Ross River Dena Council is the lead First Nations group for the Macmillan Pass Project but they have not signed a settlement agreement with federal or territorial governments so must be consulted in that undefined context.

Source: JDS (2018)

The four main authorizations that will be required before construction can begin are briefly described below.

20.2.2 YESAB Project Proposal

Fireweed will need to submit a Project Proposal to the Yukon Environmental and socio-Economic Assessment Board (YESAB) Executive Committee for consideration during their ESA. The proposal will describe the Project, the environmental and socio-economic conditions, the potential effects of the Project on these conditions, mitigations for adverse effects, and monitoring proposed to inform environmental management decisions. The proposal will contain a record of consultation with potentially affected communities and First Nations.

20.2.3 Water Licence

The Yukon Water Board regulates the use of water and/or the discharge of waste to water. It issues water licences that specify in part the quantity of water that can be used, conditions specific to the discharge of waste to water, monitoring requirements, and environmental management plans to be prepared and implemented. Fireweed's Macmillan Pass Project currently operates under a "Type B" water licence with regard to the Tom adit but will require a Type "A" Water Licence for future mine development.

20.2.4 Quartz Mining Licence

The Yukon Quartz Mining Act, administered by the Government of Yukon's Department of Energy, Mines and Resources, regulates hard rock mineral exploration and mine development in the Yukon. A quartz licence serves as a regulatory and decision-making framework that delineates how a company will develop and manage the mine over the life of the Project. The Macmillan Pass Project will require a Quartz Mining Licence.

20.2.5 Environmental and Mine Operations Plans

Environmental management plans will be assembled under an Environmental Management Program, which provides overarching direction for environmental and development management at the Macmillan Pass Project. It will be supported by a suite of project-specific mitigation, monitoring and/or management plans that set out the Project's standards and requirements under the Quartz Mining Licence and/or Water Licence for particular areas of environmental management.

20.3 Environmental Studies

Baseline environmental studies were initiated in 2001 and expanded in 2008. Current baseline environmental work in the Project area consists of the following:

- 12 surface water monitoring stations;
- 3 ground water monitoring stations;
- 1 Fireweed weather station near the Tom deposit and one larger federal government weather station at the airstrip; and
- In 2018 Fireweed initiated new wildlife and heritage studies for the Project area.

Baseline and associated research studies are currently ongoing. All the information collected will be used in preparing an ESA, and applying for permits, authorizations and/or licences required to construct, operate, and ultimately close the mine.

20.4 Permitting Considerations

Exploration work is subject to the Mining Land Use Regulations of the Yukon QMA and to the Yukon Environmental and Socio-Economic Assessment Act (YESAA). A land use permit must be obtained and YESAA Board approval issued before large-scale exploration is conducted.

Since the exercise of the property option on 7 February 2018, all title and project permits have been transferred into Fireweed's name. Fireweed currently holds a Class 3 land use permit for exploration activities on the Tom and Jason properties (LQ00325) under the QMA and Quartz Mining Land Use Regulations with a renewal date of 21 September 2021. However Fireweed has applied for a new upgraded land use permit to allow for larger exploration programs on the project and in the meantime continues to operate under the old permit. A waste management permit issued in 2011 (81-029) has been extended to 31 December 2021.

Currently water use and discharge of water from the Tom adit are governed by a Type B water use licence (QZ15-060-01) granted on 24 July 2015 and extended until 31 December 2020. The discharge from the lower Tom adit has elevated metals levels as do natural drainages in the area. The adit discharge as well as local natural drainages have been the subject of water quality monitoring and water sampling a minimum of four times per year and reporting since 2001. Continued efforts will be required to monitor compliance with the water licence.

Any potential future development of the Tom and Jason deposits will require an environmental assessment under YESAA and a Yukon Mining Licence and Lease issued by the Yukon Government. A preliminary environmental investigation was undertaken on the Jason deposit by Gartner Lee Limited (Pearson, 2006). Additional permits will be required from the territorial and federal governments to further develop the deposits. For example, development of mining activities in the Yukon requires the issuance of a Type A water licence by the Yukon Water Board.

20.5 First Nations Consultations

The Macmillan Pass Project lies within an area of overlapping territorial claims by the Kaska First Nations and Na-Cho Nyak Dun First Nation. This area has been withdrawn from staking (Ross River Area OIC 2013/224 and OIC 2013/60) pending settlement of land claims. The First Nations have not reached a land claim settlement with the Yukon government, and so the terms of any future development of the Tom and Jason deposits remain uncertain and will require First Nations consultations. Also, to obtain and renew permits for the project Fireweed is required, under Yukon permitting procedures, to consult with the affected First Nations. However, the current staking moratorium does not prevent exploration or development work to be carried out on existing claims and Fireweed reports good relations with local First Nations during the 2017 and 2018 work programs in which they hired local First Nations workers and purchased supplies and fuel from local First Nations businesses.

20.6 Other Significant Factors and Risks

20.6.1 Acid Rock Drainage

Surface and ground water monitoring has been ongoing on the Property since 2001 but to date, no formal acid rock drainage (ARD) studies or management plans have been undertaken aside from a preliminary investigation undertaken by Lee Gartner Limited in 2006. This study suggested that waste and country rock from the Tom deposits was potentially acid generating (PAG). Fireweed indicates that further studies will begin in 2018 with a program of geochemical sampling to define ARD potential based on lithological domains.

Seventeen years of continuous surface water monitoring of drainages in the vicinity of the Tom deposits and nearby sections of the South Macmillan River indicate that all streams in the Tom vicinity have naturally high ARD characteristics and elevated metal values thought to be mainly sourced from weathering of pyrite and other minerals in the host rocks to the deposits. Studies show that the Tom adit's groundwater contribution to acidity and metal loading at downstream locations since the adit was plugged is low relative to the naturally high acidity and metal values in local streams (Macphail et al., 2018). Never the less, ARD potential from any potential future mining operation is probably high (to be confirmed by detailed ABA and other studies still to be done) and despite discharge into streams with natural high acidity and elevated metal values, the Project will likely require appropriate ARD mitigation measures during and after potential future mining operations.

It is recommended that ARD studies including planned ABA studies, be carried out to understand the natural and potential industrial sources of ARD in the area. Then, as the Project moves forward to potential mine development, it is recommended that an ARD management plan be developed using the information collected to develop appropriate measures to mitigate potential adverse effects on the receiving environment from industrial sources of ARD and elevated metal values in surface and ground waters. These measures should include those described in Sections 16.3.1, 18.6 and 18.7 of this Report which include the use of geomembrane liners to minimize water infiltration into potential ARD waste rock piles and tailings. Such a liner was installed in 2010 over Tom adit waste rock pile and underlies the Tom adit water channel and is assumed to continue to work to specification with no required maintenance beyond periodic inspections (Macphail et al., 2018).

20.6.2 Other

The lower adit on the Tom property was partially plugged in 2010 to flood the mine workings and reduce the flow of acid mine drainage (AMD) from oxidation of sulphides in the mine workings. A waste pile from underground development at Tom West has also been covered with an impermeable barrier to reduce AMD from the site. The lower adit continues to make water as designed and metal contents and other parameters of the discharge water are monitored and have been within standards set in the current Type B water use licence (G. Gorzynski, personal communication, February 2018).

A preliminary environmental investigation of the Jason property in 2006 by Gartner Lee Limited noted that several exploration boreholes below an elevation of 1,250 m were discharging water. Water samples from one of these boreholes and four samples of surface water exceeded the Canadian Council of Ministers of the Environment (CCME) Aquatic Life guidelines for several metals, including Cd and Zn. Elevated metal concentrations and lowered pH levels reflect natural groundwater discharge from the site, as the Earn Group sediments are regionally elevated with respect to several metals, including Zn, Cd, Pb and Ag (Mackie et al., 2015). In 2015, a number of drill pads and collars at the Jason property were rehabilitated and holes plugged with cement when ground conditions allowed it. Water still flows from some holes where proper cementing has not yet been completed (G. Gorzynski, personal communication, 2018).

21 Capital Cost Estimate

21.1 Capital Cost Summary

LOM project capital costs total C$1,053.6M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to a 5,000 t/d production. Initial capital costs total $404.3M and are expended over a 24-month pre-production construction and commissioning period;
- Sustaining Capital Costs – includes all costs related to the development, acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $574.5M and are expended in operating years 1 through 18;
- Closure Costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $74.7M (net of equipment salvage values) and are incurred in Year 19.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors. Once compiled, the overall cost estimate was top-down benchmarked against similar operations.

Table 21-1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q2 2018 dollars with no escalation.

Table 21-1: Capital Cost Summary

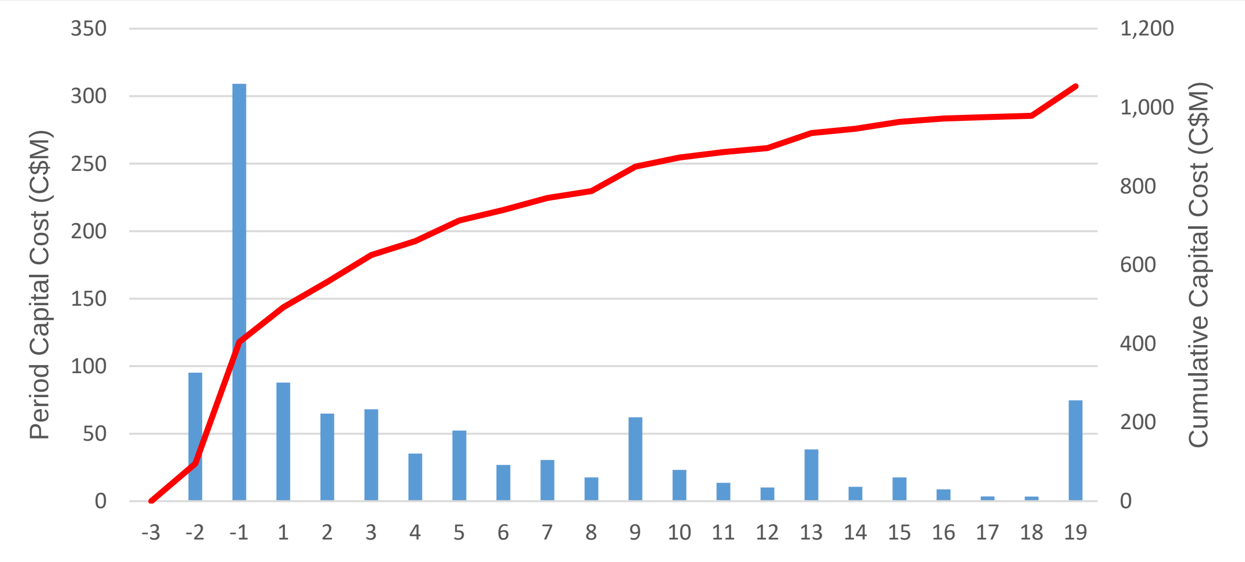
Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
Mining	30.3	378.4	-	408.6
Site Development	12.0	1.1	-	13.1
Mineral Processing	70.6	5.5	-	76.1
Tailings Management	32.7	113.9	-	146.6
Infrastructure	129.7	21.4	-	151.1
Indirect Costs Incl. EPCM	63.5	-	-	63.5
Owners Costs	7.0	-	-	7.0
Closure Costs	-	-	56.7	56.7
Subtotal Pre-Contingency	**345.8**	**520.3**	**56.7**	**922.7**
Contingency	58.6	54.2	18.1	130.9
Total Capital Costs	**404.3**	**574.5**	**74.7**	**1,053.6**

Source: JDS (2018)

21.2 Capital Cost Profile

All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21-1 presents an annual life of mine (LOM) capital cost profile.

Figure 21-1: Capital Cost Distribution



Source: JDS (2018)

21.3 Key Estimate Assumptions

The following key assumptions were made during development of the capital estimate:

- Open pit mining will be performed by contractor;
- Underground mine development activities will be performed by the Owners forces; and
- All surface construction (civil, structural, architectural, mechanical, piping, electrical, and instrumentation) will be performed by contractors.

21.4 Key Estimate Parameters

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-20%/+30%). The overall Project definition is estimated to be 10%;
- Estimate Base Date: The base date of the estimate is February 2018. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate; and
- Currency: All capital costs are expressed in Canadian Dollars (CAD$). Portions of the estimate were estimated in US Dollars (US$) and converted to Canadian Dollars at a rate of CA$1.00: US$0.78.

21.5 Basis of Estimate

21.5.1 Mine Capital Costs

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases and similar mines in western Canada. Table 21-2 summarizes the underground mine capital cost estimate.

Given that open pit mining is contracted, and underground mining commences during commercial production, the pre-production costs associated with the mine are mostly capitalized operating costs associated with pre-stripping the open pits.

Table 21-2: Mine Capital Costs

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
Underground Mobile Equipment	-	159.6	159.6
Underground Infrastructure	-	35.6	35.6
Capital Development	-	174.8	174.8
Mine Ventilation	-	5.2	5.2
Capitalized Production	-	-	0
OP Pre-Stripping	28.0	-	28.0
OP Mobilization	2.3	2.3	4.6
OP Other Equipment & Spares	-	0.8	0.8
Total Mining (excl. Contingency)	**30.2**	**378.4**	**408.6**

Source: JDS (2018)

21.5.1.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. Open pit mobile equipment are assumed contracted to the site and were not priced.

21.5.1.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.5.1.3 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes, and underground infrastructure.

21.5.1.4 Mine Ventilation

Mine ventilation includes the labour, equipment, and materials to install all primary ventilation equipment to supply ventilation requirements during mine development.

21.5.1.5 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, mineralized material extraction, mine maintenance, and mine general costs) incurred prior to and during commissioning and ceasing at commencement of commercial operations and generation of project revenues. They are included as a pre-production capital cost. Once plant feed is processed, these costs transition to operating expenses.

The basis of these costs is described in Section 22, Operating Costs, as they are estimated in the same manner. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

Given that underground mining commences during commercial production, there are no pre-production capital costs.

21.5.1.6 Open Pit Pre-Stripping

Open Pit pre-stripping includes all contracted labour, equipment, and material to remove overburden and waste material from the open pits. Small amounts of mineral may be extracted and stockpiled, the cost of which is included in the pre-stripping capital.

21.5.1.7 Open Pit Mobilization

Open pit mobilization accounts for all costs to fully mobilize and demobilize the open pit contractor’s labour and equipment fleet.

21.5.2 Surface Construction Costs

Surface construction costs include site development, crushing plant, mineral processing plant, tailings management facility, on-site and off-site infrastructure. These cost estimates are primarily based on database or recently quoted costs, with factors applied for minor cost elements. Table 21-3 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Table 21-3: Surface Construction Basis of Estimate

Commodity	Basis
Contractor Labour Rates	Database values based on similar Northern Canadian projects.
Access Road	Database costs per kilometre for road upgrades or construction, based on assessed difficulty of construction for each section. Database costs for bridges.
Bulk Earthworks, Including On-Site Roads	Estimate volumes from preliminary site layout model. Database unit rates for bulk excavation and fill, grading and surfacing. Allowances for surface drainage and site water management.
Concrete	Quantities developed based on building sizes outlined in general arrangements and cross checked against similar projects. Database unit rates in Yukon from recent contractor quotations in the region.
Structural Steel	Quantities developed based on equipment types and sizes and cross checked against similar projects. Database unit rates in Canada.
Pre-Engineered Buildings	Database unit rates ($/m^2) applied against the building sizes outlined in the general arrangements. Database allowances for lighting, small power, electrical/control rooms, and fire detection.
Modular Buildings & Warehouses	Database costs from similar Arctic projects for the mine dry, administration offices, mine maintenance building, mine warehouse, and camp structures.
Mechanical Equipment	A combination of quoted costs and database costs from recent quotations on similar projects. Over 90% of the mechanical process equipment was quoted for this project. A combination of actual install hours based on equipment size and database factors applied against mechanical equipment costs for installation.
Piping	Database factors applied against mechanical equipment costs.
Electrical and Instrumentation	Database factors applied against mechanical equipment costs.
On-site Power Transmission Lines	Database costs from similar projects. Quantities developed based on general arrangements and site layouts.

Source: JDS (2018)

21.5.2.1 Surface Construction Sustaining Capital

Sustaining capital costs are included in the estimate for continued construction of the Tailings Management Facility. The balance of the facility is constructed in years 1, 3, 5, 9 and 13.

The sustaining capital cost estimate also includes additional generators in year 1 and 4 as UG mining power demand increases.

Allowances are provided for the processing plant, on-site infrastructure and the access road for major equipment overhauls, minor capital projects and road upgrades such as bridge replacements.

21.5.3 Indirect Costs

Indirect costs are those that not directly accountable to a specific cost object. Table 21-4 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-4: Indirect Cost Basis of Estimate

Commodity	Basis
Heavy Equipment	Factor (1.5%) of on-site direct costs for heavy equipment rental (i.e. 100 t + crane), and factor (1%) of off-site infrastructure direct costs
Contractor Field Indirect Costs	Factor (6.0%) for the following items:
	Time based cost allowance for general construction site services (temporary power, heating & hoarding, contractor support, etc.) applied against the surface construction schedule
	Construction offices and ablution facilities
	Combination of diesel and transmission line construction power
	Contractor mobilization
Freight & Logistics	Factor (8%) for freight and logistics related to the materials and equipment required for the crushing plant, mineral processing plant, on-site and off-site infrastructure. Factor excludes mining equipment as prices are FOB site
Vendor Representatives	Factor (1%) of direct costs for the provision of vendor services for commissioning equipment
Capital Spares	Factor (5%) of direct costs for spare parts
Start-up and Commissioning	Factored (2%) of direct costs for start-up and commissioning support
Detailed Engineering & Procurement	Factor (7%) applied against direct and indirect hours for engineering management, detailed design, drawings, and major equipment procurement
Project & Construction Management	Staffing plan built up against the development schedule for Project management, health and safety, construction management, field engineering, Project controls, and contract administration
	Database unit (hourly) rates

Source: JDS (2018)

21.5.4 Owners Costs

Owner’s costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- Pre-production General & Administration: Costs of the Owner's labour and expenses (safety, finance, security, purchasing, management, etc.) incurred prior to commercial production;
- Surface Support: Costs of the Owner's surface support labour, maintenance, and equipment usage; costs for contract water supply and waste removal prior to commercial production; and
- Pre-production Milling: Costs of the Owner's processing labour, power, first fills and consumables incurred before declaration of commercial production.

21.5.5 Closure Cost Estimates

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for a surface mine in Canada's Arctic. Activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF; and
- Re-vegetation and seeding allowances.

The majority of closure costs are incurred immediately following completion of operations (Year 19).

21.5.6 Cost Contingency

An overall contingency of 12% was applied to the LOM capital costs of the project. LOM project contingency amounts to $127.4 M. The overall contingency is a blend of separate factors that were applied different areas as follows:

- Mobile mining equipment and capital development – 0%;
- Underground infrastructure – 20%;
- Process Plant, Site Infrastructure and Project Indirect Costs – 20%;
- Tailings Management – 35%; and
- Off-site Infrastructure (Access Road) – 10%.

21.6 Processing Capital Costs

The processing plant capital costs are provided in Table 21-5:

Table 21-5: Process Plant CAPEX

Processing Plant CAPEX	Unit	Initial	Sustaining	Total
Crushing & Ore Handling	$M CAD	1.91	0.15	2.06
Fine Ore Storage & Reclaim	$M CAD	2.37	0.19	2.56
Grinding	$M CAD	18.63	1.49	20.12
Lead Circuit				
Pb Rougher Flotation	$M CAD	2.28	0.18	2.47
Pb Regrind	$M CAD	2.91	0.23	3.14
Pb Cleaner	$M CAD	2.49	0.20	2.69
Pb Dewatering - Concentrate	$M CAD	2.59	0.21	2.79
Zinc Circuit				
Zn Rougher Flotation	$M CAD	2.41	0.19	2.60
Zn Regrind	$M CAD	3.66	0.29	3.95
Zn Cleaner	$M CAD	4.24	0.34	4.58
Zn Dewatering - Concentrate	$M CAD	3.44	0.28	3.72
Tailings	$M CAD	2.60	-	2.60
Reagents	$M CAD	1.92	-	1.92
Plant Utilities, Building, & General				
Plant Building	$M CAD	16.23	1.30	17.53
Plant Water Systems	$M CAD	0.46	0.07	0.53
Plant Air Systems	$M CAD	0.71	0.11	0.83
Assay Lab	$M CAD	1.70	0.27	1.97
TOTAL		**70.56**	**5.51**	**76.07**

Source: JDS (2018)

21.7 Infrastructure Capital Cost Estimate

The infrastructure capital cost is provided in Table 21-6.

Table 21-6: Infrastructure CAPEX

Infrastructure CAPEX	Unit	Initial	Sustaining	Total
Tailings Management Facility	$M CAD	28.09	111.77	139.85
Tom Waste Rock Storage Facility	$M CAD	2.78	1.48	4.26
Jason Waste Rock Storage Facility	$M CAD	1.84	0.67	2.51
Camp Complex and Accommodations	$M CAD	9.60	0.77	10.37
Power Supply & Distribution				
LNG Generators and Fuel Storage	$M CAD	28.20	8.03	36.24
On-Site Power Distribution	$M CAD	2.13	0.13	2.25
Water Supply & Distribution	$M CAD	0.84	5.44	6.27
Waste Management	$M CAD	1.30	0.10	1.40
Ancillary Buildings				
Mine Dry	$M CAD	0.40	0.03	0.43
Mine Office	$M CAD	0.35	0.03	0.38
Mine Maintenance Shop / Truck Shop	$M CAD	1.91	0.15	2.06
Mine/Plant Warehouse	$M CAD	0.85	0.07	0.92
Emergency Response Facility	$M CAD	0.10	0.01	0.11
Explosives Storage Magazines	$M CAD	0.09	0.01	0.10
Surface Mobile Equipment	$M CAD	4.33	-	4.33
Bulk Fuel Storage & Distribution	$M CAD	0.73	-	0.73
IT & Communications	$M CAD	0.61	-	0.61
Main Access Road	$M CAD	78.28	6.65	84.93
TOTAL		**162.41**	**135.33**	**297.74**

Source: JDS (2018)

21.8 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any Project sunk costs (studies, exploration programs, etc.);

- Provincial sales tax;
- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

22.1 Operating Cost Summary

The operating cost estimate (OPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a PEA study.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven Project execution strategies.

Total LOM operating costs amount to C$2,677.6 M or an average unit cost of C$82.00/tonne processed. The LOM costs are summarized in Table 22-1. OP mining costs average C$4.45 per OP tonne moved while UG mining costs average C$52.02 per UG tonne mined.

Table 22-1: LOM Total Operating Cost Estimate

Description	Total Estimate (C$ M)	Average Unit Cost (C$/t Processed)
OP Mining	111.9	3.43
UG Mining	1,478.7	45.28
Processing	748.5	22.92
G&A	338.6	10.37
Total Operating Costs	**2,677.6**	**82.00**

Source: JDS (2018)

22.2 Mine Operating Cost Estimate

Mine operating costs to support the development and extraction of mineralized material from both Tom and Jason deposits for a nominal 5,000 tonnes per day operation is summarized below in Table 22-2.

Table 22-2: Breakdown of Mine Operating Costs

Mine Operating Cost	$/t Mined (CAD)	$M/a (CAD)
Lateral Waste Development	1.47	2.3
Production	24.58	38.8
Backfill	16.43	25.9
Mine Maintenance	3.23	5.1
Mine General	6.21	9.8
Total Mine Operating Costs	**52.02**	**82.0**

Source: JDS (2018)

22.3 Process Operating Costs

Process operating costs were estimated to include all lead and zinc recovery steps required to produce saleable concentrates. The crushing and process plants were designed to process 5,000 t/d at availabilities of 75% and 92%, respectively. Labour rates and benefit packages were based on industry information compiled by JDS. Power costs were calculated from the total installed power assuming $0.171/kWh. Liner pricing and Vendor recommended spare parts for one year of operation were used to estimate mill and crusher wear costs. Costs for media were determined using engineering calculations based on mill power draw, abrasion index and vendor quotes for media as a cost per tonne. Reagent costs were developed using the metallurgical test results summarized in Section 13 and pricing supplied by Vendors. Equipment maintenance was calculated by applying a factor of 4% to major process equipment cost. A breakdown of the process operating costs is summarized in Table 22-3.

Table 22-3: Breakdown of Process Operating Costs

Process Operating Costs (LOM Total)	$/t processed (CAD)	$M/a (CAD)
Labour	2.98	5.4
Power	7.09	12.9
Maintenance, Consumables & Tailings Facility	12.38	22.6
Total Processing OPEX	**22.45**	**41.0**

Source: JDS (2018)

22.4 General and Administration Costs

General and administrative costs comprise the following categories:

- Administration, site services and power plant labour;
- On-site items as such camp catering, health and safety, environmental, human resources, legal, external consulting, communications and office supplies, site service equipment operation and maintenance; and
- Employee travel via air charter from Whitehorse.

The total G&A unit operating cost is estimated at $10.45/t of plant feed processed. Table 22-4 summarizes the annual G&A operating costs.

Table 22-4: G&A OPEX Estimate by Area

Parameter	$/t processed	LOM ($M)
G&A Labour	3.43	112
G&A Items - On-site	5.19	169
Employee Travel	1.84	60
Total Operating Cost – G&A	**10.45**	**342**

Source: JDS (2018)

23 Economic Analysis

This PEA is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the results of this PEA will be realized.

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, US$:C$ exchange rates, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

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

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 and Section 22 of this report (presented in 2018 dollars). The economic analysis has been run with no inflation (constant dollar basis).

23.1 Assumptions

The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 23-1 outlines the metal prices and exchange rate assumptions used in the economic analysis. The base case metal prices were selected based on the average of three years past and projected two years forward by analysis of London Metal Exchange futures as of 30 April 2018. The spot prices were at the close of London Metal Exchange on 30 April 2018. These parameters are in line with other recently released comparable Technical Reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 23-1: Metal Price and Exchange Rates Used in Economic Analysis

Parameter	Unit	Base Price Value	Spot Price Value
Lead Price	US$/lb	0.98	1.05
Zinc Price	US$/lb	1.21	1.42
Silver Price	US$/oz	16.80	16.38
Exchange Rate	US$:C$	0.77	0.77

Source: JDS (2018)

Other economic factors include the following:

- Discount rate of 8% (sensitivities using other discount rates have been calculated);
- Closure cost of $56.7 M (pre-contingency);
- Nominal 2018 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- Working capital calculated as two months of operating costs (mining, processing, and G&A) in Year 1;
- Results are presented on 100% ownership; and
- No management fees or financing costs (equity fund-raising was assumed).

23.2 Processing and Concentrate Terms

Mine revenue is derived from the sale of zinc concentrate and lead concentrate into the international marketplace. No contractual arrangements for refining exist at this time. Details regarding the terms used for the economic analysis can be found in the market studies (Section 19) of this report.

Table 23-2 outlines the recoveries, payable terms, treatment charges and transportation costs used in the economic analysis.

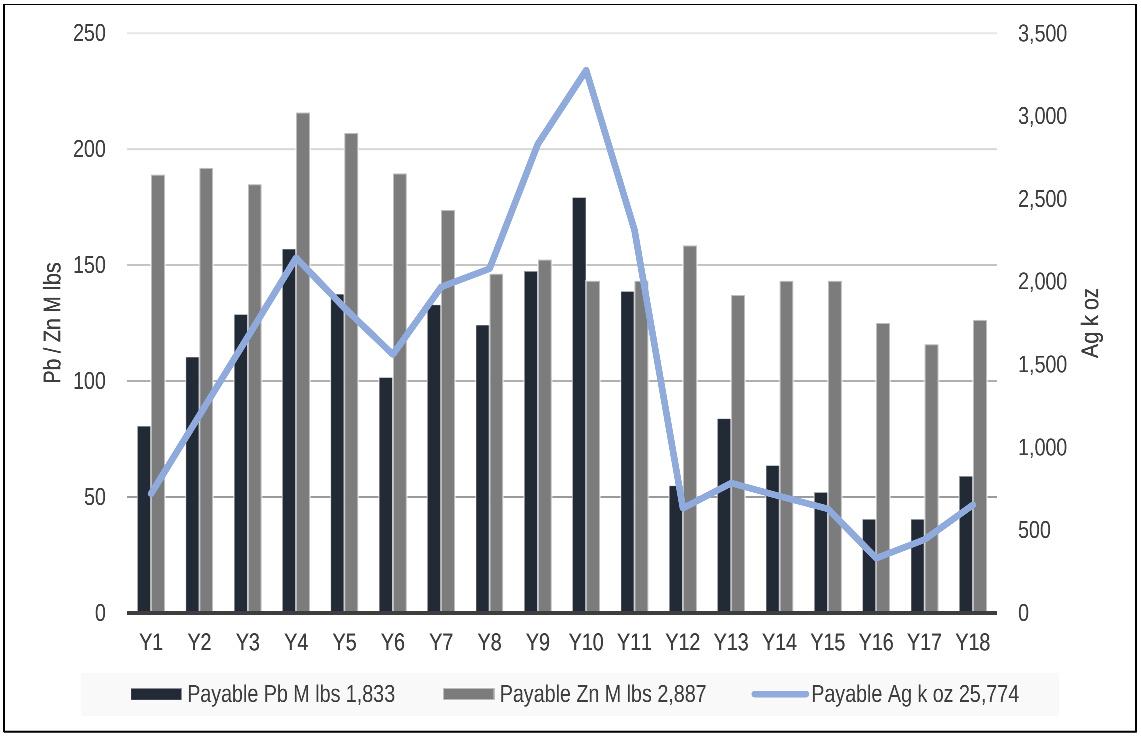
Table 23-2: Concentrate Terms

Assumptions & Inputs	Unit	Value
Lead Concentrate		
Metal Recovery to Concentrate	% Pb	75.4
	% Zn	4.8
	% Ag	59.4
Pb Concentrate Grade Produced	% Pb	61.5
Minimum Deduction	% Pb/t	3.0
	g/t Ag	50.0
Metal Payable	% Pb	95.0
	% Ag	95.0
Pb Treatment Charge	US$/dmt conc.	170
Ag Refining Charge	US$/oz	1.50
Moisture Content	%	8.0
Pb Concentrate Transportation Cost	C$/wmt	211.85
Zinc Concentrate		
Metal Recovery to Concentrate	% Pb	7.5
	% Zn	88.9
	% Ag	22.2
Zn Concentrate Grade Produced	% Zn	58.4
Minimum Deduction	% Pb/t	0.0
	%Zn/t	8.0
	g/t Ag	93.31
Metal Payable	% Pb	0.0
	% Zn	85.0
	% Ag	70.0
Zn Treatment Charge	US$/dmt conc.	190
Ag Refining Charge	US$/oz	1.50
Moisture Content	%	8.0
Zn Concentrate Transportation Cost	C$/wmt	211.85
Hg Content	%	0.0155
Base Hg content	%	0.01
Penalty per 0.01% Hg above base	0.01%	1.75
Hg Content Penalty	US$/dmt conc.	0.96
SiO_2 Penalty	US$/dmt conc.	2.00

Source: JDS (2018)

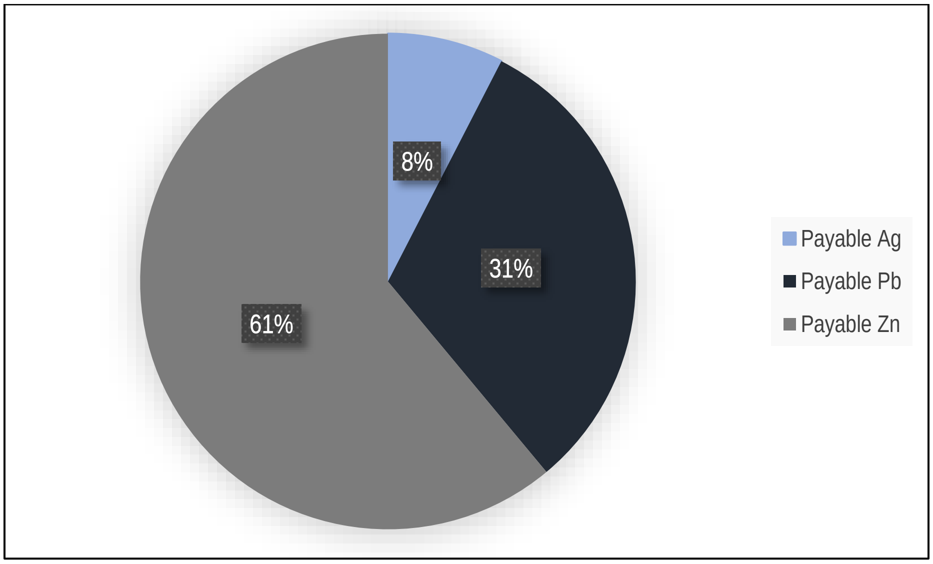
Figure 23-1 shows a breakdown of the payable lead, zinc, and silver recovered during the mine life. A total of 1,833 Mlbs of lead, 2,887 Mlbs of zinc, and 25,774 koz of silver are projected to be produced during the mine life. Zinc accounts for about 61% of project revenues, lead for about 31%, and silver for about 8%, this is illustrated in Figure 23-2.

Figure 23-1: Payable Metal Production by Year



Source: JDS (2018)

Figure 23-2: Revenue Distribution



Source: JDS (2018)

23.3 Summary of Capital Cost Estimates

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors. Once compiled, the overall cost estimate was top-down benchmarked against similar operations.

Table 23-3 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q2 2018 dollars with no escalation.

Table 23-3: Capital Cost Summary

Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
Mining	30.3	378.4	-	408.6
Site Development	12.0	1.1	-	13.1
Mineral Processing	70.6	5.5	-	76.1
Tailings Management	32.7	113.9	-	146.6
Infrastructure	129.7	21.4	-	151.1
Indirect Costs Incl. EPCM	63.5	-	-	63.5
Owners Costs	7.0	-	-	7.0
Closure Costs	-	-	56.7	56.7
Subtotal Pre-Contingency	**345.8**	**520.3**	**56.7**	**922.7**
Contingency	58.6	54.2	18.1	130.9
Total Capital Costs	**404.3**	**574.5**	**74.7**	**1,053.6**

Source: JDS (2018)

23.4 Summary of Operating Cost Estimates

Total LOM operating costs amount to C$2,677.6 M or an average unit cost of C$82.00/tonne processed. The LOM costs are summarized in Table 23-4. OP mining costs average C$4.45 per OP tonne moved while UG mining costs average C$52.02 per UG tonne mined.

Table 23-4: LOM Total Operating Cost Estimate

Description	Total Estimate (C$ M)	Average Unit Cost (C$/t Processed)
OP Mining	111.9	3.43
UG Mining	1,478.7	45.28
Processing	748.5	22.92
G&A	338.6	10.37
Total Operating Costs	**2,677.6**	**82.00**

Source: JDS (2018)

23.5 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate, value of the potential project economics. A tax model was prepared by a tax consultant with applicable Yukon mineral tax regime experience. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- 15% federal income tax rate;
- 12% Yukon tax rate; and
- Graduated Yukon Quartz Mining Royalty rate based on taxable income.

Total taxes for the project amount to $615.7 M.

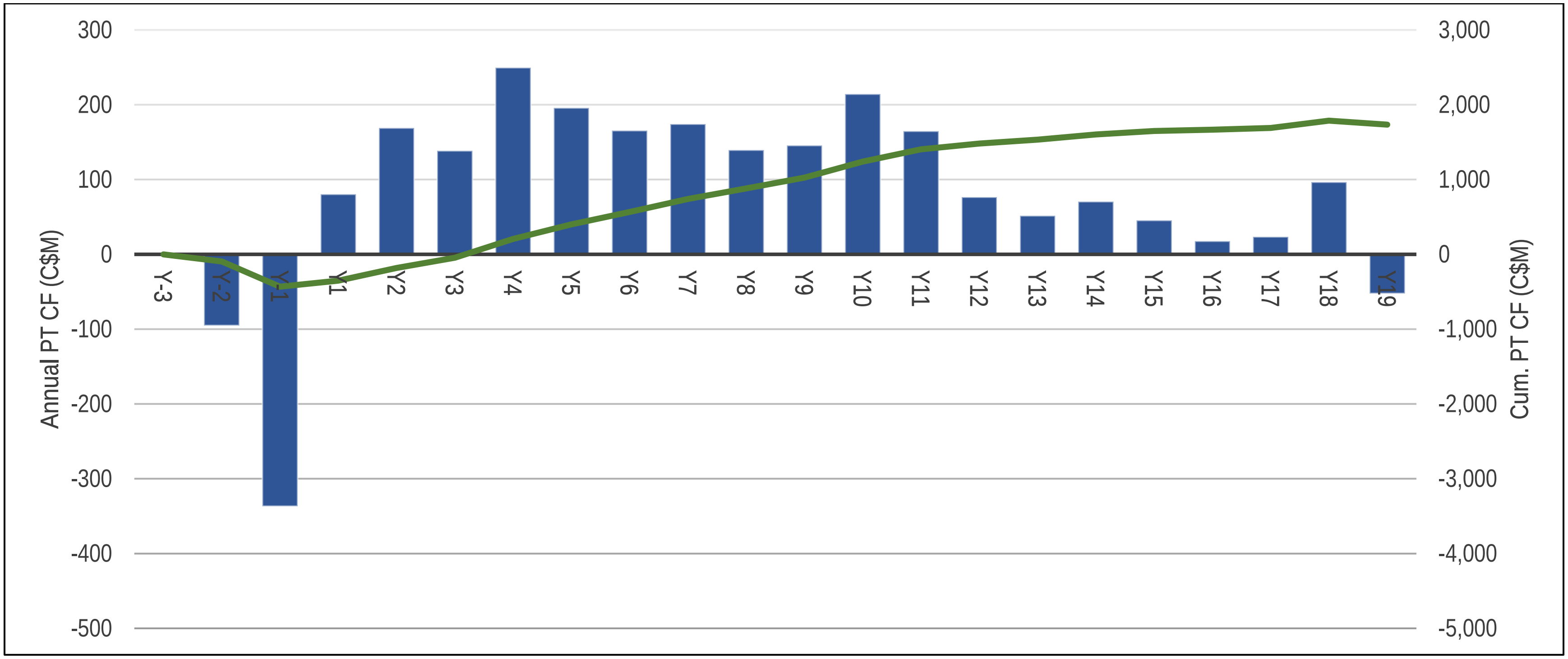
23.6 Economic Results

At this preliminary stage, the project has an after-tax IRR of 24% and a net present value using an 8% discount rate (NPV8%) of $448 M using the metal prices described in Section 19.

Figure 23-3 shows the projected pre-tax cash flows, and Table 23-5 summarizes the economic results of the Macmillan Pass Project.

The pre-tax break-even zinc price for the project is approximately US$0.80/lb, based on the LOM plan presented herein, a lead price of US$0.98/lb, silver price of US$16.80/oz, and an FX rate of 0.77 US$:C$.

Figure 23-3: Annual Pre-Tax Cash Flow



Source: JDS (2018)

Table 23-5: Summary of Results

Parameter	Unit	Base Price Value	Spot Price Value
Capital Costs			
Pre-Production Capital	C$M	345.8	345.8
Pre-Production Contingency	C$M	58.6	58.6
Total Pre-Production Capital	**C$M**	**404.3**	**404.3**
Sustaining & Closure Capital	C$M	577.0	577.0
Sustaining & Closure Contingency	C$M	72.3	72.3
Total Sustaining & Closure Capital	C$M	649.3	649.3
Total Capital Costs Incl. Contingency	**C$M**	**1,053.6**	**1,053.6**
Cash Flows			
Working Capital	C$M	22.4	22.4
Pre-Tax Cash Flow	**LOM C$M**	**1,734.8**	**2,580.6**
	C$M/a	**96**	**142**
Taxes	LOM C$M	615.7	911.4
After-Tax Cash Flow	**LOM C$M**	**1,119.1**	**1,669.2**
	C$M/a	**62**	**92**
Economic Results			
Pre-Tax $NPV_{8\%}$	**C$M**	**779**	**1,214**
Pre-Tax IRR	**%**	**31.9**	**42.4**
Pre-Tax Payback	**Years**	**3.2**	**2.5**
After-Tax $NPV_{8\%}$	**C$M**	**448**	**729**
After-Tax IRR	**%**	**23.5**	**31.3**
After-Tax Payback	**Years**	**4.0**	**3.2**

Source: JDS (2018)

23.7 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -15% to +15%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the metal prices were evaluated at a +/- 10% range to the base case, while the mill feed grade and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

Notwithstanding the above noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the project is most sensitive to metal prices, followed by mill feed grade, exchange rate, and operating costs. The Project showed the least sensitivity to capital costs. Table 23-6 and Figure 23-4 show the results of the sensitivity tests.

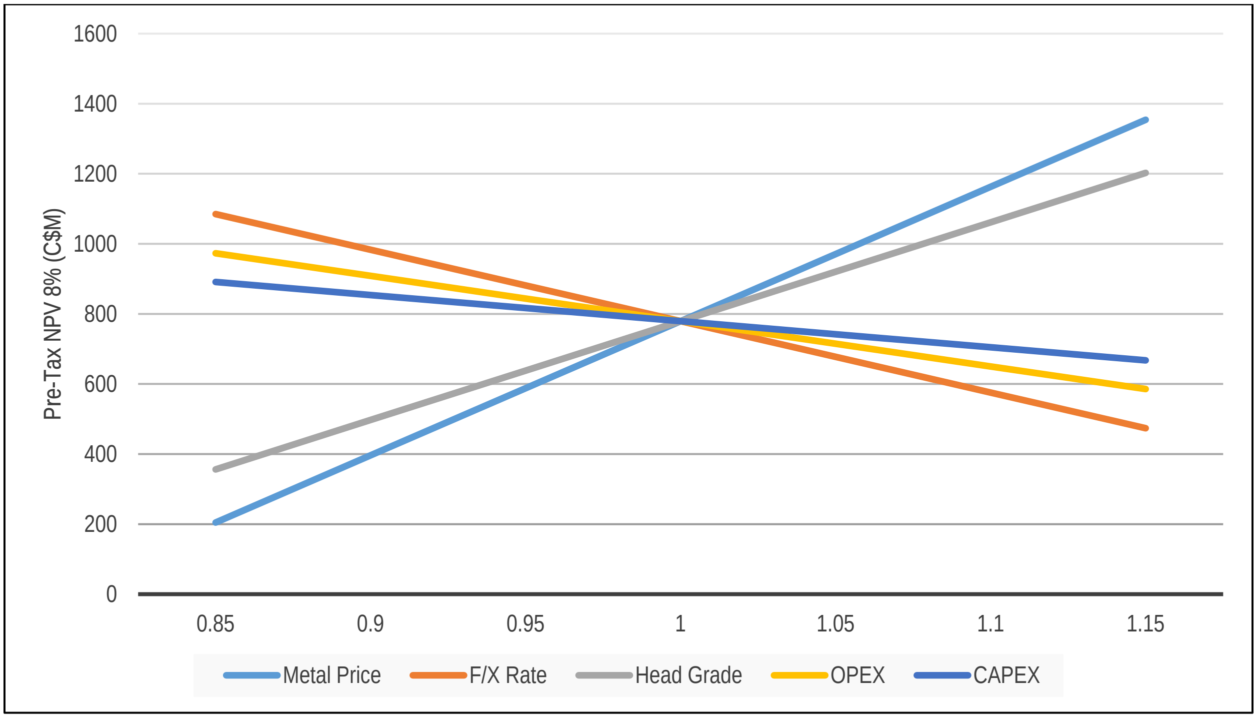
The economic cash flow model for the project is illustrated in Table 23-7.

Table 23-6: Sensitivity Results (Pre-Tax NPV$_{8\%}$)

Parameter	-15%	-10%	-5%	Base	+5%	+10%	+15%
Metal Price	204	396	588	779	971	1,163	1,354
C$:US$ FX	1,085	983	881	779	677	576	474
Mill Feed Grade	356	497	638	779	920	1,061	1,203
OPEX	973	908	844	779	715	650	586
CAPEX	891	854	817	779	742	705	667

Source: JDS (2018)

Figure 23-4: Sensitivity, Pre-Tax NPV @ 8% Discount Rate



Source: JDS (2018)

Table 23-7: Economic Cash Flow Model

	Unit	LOM Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
METAL PRICES & F/X RATE																									
Zn	US$/lb	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21
Pb	US$/lb	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98
Ag	US$/oz	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80
F/X	USD:CAD	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77
PRODUCTION SCHEDULE																									
OPEN PIT																									
Ore Mined	ktonnes	4,229			150	1,677	1,824	579	0	0	0	0	0	0	0	0	0								
Zn	%	6.1%			9.2%	5.7%	6.3%	5.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%								
Pb	%	2.9%			7.5%	2.0%	3.8%	1.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%								
Ag	g/t	27.1			89.4	14.8	40.1	5.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0								
Waste Mined	ktonnes	20,934			3,764	11,297	5,447	427	0	0	0	0	0	0	0	0	0								
Total Mined	ktonnes	25,163	-	-	3,914	12,973	7,270	1,006	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Strip Ratio	w:o	5.0	-	-	25.1	6.7	3.0	0.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Rate	*ktpd*	4.2	-	-	*0.4*	*4.6*	*5.0*	*1.6*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Metal																									
Zn	ktonnes	258	-	-	14	96	115	32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb	ktonnes	124	-	-	11	34	70	9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag	kg	114,700	-	-	13,410	24,864	73,045	3,380	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
UNDERGROUND																									
Ore Mined	ktonnes	28,427			0	114	0	1,412	1,760	1,760	1,765	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,485	0	0
Zn	%	5.2%			0.0%	6.5%	0.0%	6.1%	7.2%	6.9%	6.3%	5.7%	4.8%	5.0%	4.7%	4.7%	5.2%	4.5%	4.7%	4.7%	4.1%	3.8%	4.9%	0.0%	0.0%
Pb	%	3.6%			0.0%	5.8%	0.0%	5.3%	5.6%	4.9%	3.6%	4.6%	4.3%	5.1%	6.2%	4.8%	1.9%	2.9%	2.2%	1.8%	1.4%	1.4%	2.5%	0.0%	0.0%
Ag	g/t	45.8			0.0	60.9	0.0	64.8	67.2	58.8	49.9	58.1	59.2	77.9	88.9	65.0	21.0	26.6	23.5	20.8	11.4	14.8	26.6	0.0	0.0
Mining Rate	*ktpd*	4.9	-	-	-	*0.3*	-	*3.9*	*4.8*	*4.8*	*4.8*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*4.1*	-	-
Contained Metal																									
Zn	ktonnes	1,476	-	-	-	7	-	86	127	121	111	104	88	92	86	86	95	82	86	86	75	70	73	-	-
Pb	ktonnes	1,037	-	-	-	7	-	75	99	86	64	84	79	93	113	88	35	53	40	33	26	26	37	-	-
Ag	kg	1,302,859	-	-	-	6,946	-	91,514	118,272	103,488	88,083	106,323	108,336	142,557	162,687	118,950	38,430	48,678	43,005	38,064	20,862	27,084	39,580	-	-
TOTAL																									
Ore Mined	ktonnes	32,656			150	1,791	1,824	1,991	1,760	1,760	1,765	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,485	0	0
Zn	%	5.3%			9.2%	5.8%	6.3%	5.9%	7.2%	6.9%	6.3%	5.7%	4.8%	5.0%	4.7%	4.7%	5.2%	4.5%	4.7%	4.7%	4.1%	3.8%	4.9%	0.0%	0.0%
Pb	%	3.6%			7.5%	2.3%	3.8%	4.2%	5.6%	4.9%	3.6%	4.6%	4.3%	5.1%	6.2%	4.8%	1.9%	2.9%	2.2%	1.8%	1.4%	1.4%	2.5%	0.0%	0.0%
Ag	g/t	43.4			89.4	17.8	40.1	47.7	67.2	58.8	49.9	58.1	59.2	77.9	88.9	65.0	21.0	26.6	23.5	20.8	11.4	14.8	26.6	0.0	0.0
Waste Mined	ktonnes	20,934			3,764	11,297	5,447	427	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Mined	ktonnes	53,590			3,914	13,087	7,270	2,418	1,760	1,760	1,765	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,485	-	-
Mining Rate	*ktpd*	4.9	-	-	*0.4*	*4.9*	*5.0*	*5.5*	*4.8*	*4.8*	*4.8*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*4.1*	-	-
Contained Metal																									
Zn	ktonnes	1,734	-	-	14	104	115	118	127	121	111	104	88	92	86	86	95	82	86	86	75	70	73	-	-
Pb	ktonnes	1,161	-	-	11	41	70	84	99	86	64	84	79	93	113	88	35	53	40	33	26	26	37	-	-
Ag	kg	1,417,559	-	-	13,410	31,810	73,045	94,895	118,272	103,488	88,083	106,323	108,336	142,557	162,687	118,950	38,430	48,678	43,005	38,064	20,862	27,084	39,580	-	-
MILL SCHEDULE																									
Ore Milled	ktonnes	32,656				1,825	1,825	1,826	1,825	1,825	1,825	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,830	1,574	-	-
Zn	%	5.3%				6.2%	6.3%	6.1%	7.1%	6.8%	6.2%	5.7%	4.8%	5.0%	4.7%	4.7%	5.2%	4.5%	4.7%	4.7%	4.1%	3.8%	4.8%	0.0%	0.0%
Pb	%	3.6%				2.8%	3.8%	4.5%	5.4%	4.8%	3.5%	4.6%	4.3%	5.1%	6.2%	4.8%	1.9%	2.9%	2.2%	1.8%	1.4%	1.4%	2.4%	0.0%	0.0%
Ag	g/t	43.4				24.7	40.0	51.5	65.0	56.9	48.4	58.1	59.2	77.9	88.9	65.0	21.0	26.6	23.5	20.8	11.4	14.8	25.3	-	-
Milling Rate	*ktpd*	5.0	-	-	-	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*5.0*	*4.3*	-	-
Contained Metal																									
Zn	ktonnes	1,733	-	-	-	113	115	111	130	124	114	104	88	92	86	86	95	82	86	86	75	70	76	-	-
Pb	ktonnes	1,161	-	-	-	51	70	82	99	87	64	84	79	93	113	88	35	53	40	33	26	26	37	-	-
Ag	kg	1,417,628	-	-	-	44,996	73,003	93,999	118,622	103,838	88,373	106,323	108,336	142,557	162,687	118,950	38,430	48,678	43,005	38,064	20,862	27,084	39,821	-	-
Zn CONCENTRATE																									
Recovery to Zn Concentrate	% Zn	88.9%	-	-	-	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	0.0%	0.0%
	% Pb	7.5%	-	-	-	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	7.5%	0.0%	0.0%
	% Ag	22.2%	-	-	-	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	22.2%	0.0%	0.0%
Metal in Zn Concentrate	Zn ktonnes	1,541	-	-	-	101	102	99	115	111	101	93	78	81	76	76	85	73	76	76	67	62	67	-	-
	Zn Mlbs	3,397	-	-	-	222	226	217	254	244	223	204	172	179	169	169	187	161	169	169	147	136	149	-	-
	Pb ktonnes	87	-	-	-	4	5	6	7	7	5	6	6	7	8	7	3	4	3	2	2	2	3	-	-
	Pb Mlbs	192	-	-	-	8	12	13	16	14	11	14	13	15	19	15	6	9	7	5	4	4	6	-	-
	Ag kg	314,714	-	-	-	9,989	16,207	20,868	26,334	23,052	19,619	23,604	24,051	31,648	36,117	26,407	8,531	10,807	9,547	8,450	4,631	6,013	8,840	-	-
	Ag koz	10,118	-	-	-	321	521	671	847	741	631	759	773	1,017	1,161	849	274	347	307	272	149	193	284	-	-
Pull Factor		12	-	-	-	11	10	11	9	10	11	12	14	13	14	14	13	15	14	14	16	17	14	-	-
Zn Concentrate Grade	% Zn	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%	58.4%
	% Pb	3.30%	0%	0%	0%	2%	3%	4%	4%	3%	3%	4%	4%	5%	6%	5%	2%	3%	2%	2%	2%	2%	2%	0%	0%
	g/t Ag	119.3	-	-	-	57.8	92.4	123.5	133.5	121.8	113.3	148.7	179.9	227.2	275.8	201.7	58.9	86.2	72.9	64.5	40.5	56.8	76.5	-	-
	Ag oz	3.8	-	-	-	1.9	3.0	4.0	4.3	3.9	3.6	4.8	5.8	7.3	8.9	6.5	1.9	2.8	2.3	2.1	1.3	1.8	2.5	-	-
Zn Concentrate Produced	dmt	2,638,164	-	-	-	172,739	175,461	168,921	197,260	189,223	173,200	158,787	133,715	139,287	130,930	130,930	144,858	125,358	130,930	130,930	114,215	105,858	115,563	-	-
	wmt	2,867,569	-	-	-	187,759	190,718	183,610	214,413	205,677	188,261	172,595	145,343	151,399	142,315	142,315	157,455	136,259	142,315	142,315	124,147	115,063	125,612	-	-
Moisture Content	%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Minimum Deduction	Units	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Zn Payable	% Payable	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Zn Payable based on Min. deduction	%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%
Zn Payable based on %	%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%
Payable Zn in Zn Concentrate	ktonnes	1,310	-	-	-	86	87	84	98	94	86	79	66	69	65	65	72	62	65	65	57	53	57	-	-
	Mlbs	2,887	-	-	-	189	192	185	216	207	190	174	146	152	143	143	159	137	143	143	125	116	126	-	-
Revenues Zn in Zn Concentrate	**US$M**	**3,493.4**	-	-	-	**228.7**	**232.3**	**223.7**	**261.2**	**250.6**	**229.3**	**210.3**	**177.1**	**184.4**	**173.4**	**173.4**	**191.8**	**166.0**	**173.4**	**173.4**	**151.2**	**140.2**	**153.0**	-	-
	C$M	**4,536.9**	-	-	-	**297.1**	**301.7**	**290.5**	**339.2**	**325.4**	**297.9**	**273.1**	**230.0**	**239.5**	**225.2**	**225.2**	**249.1**	**215.6**	**225.2**	**225.2**	**196.4**	**182.0**	**198.7**	-	-
Minimum Deduction	Units	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
	% Payable	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
	%	3.3%	0%	0%	0%	2%	3%	4%	4%	3%	3%	4%	4%	5%	6%	5%	2%	3%	2%	2%	2%	2%	2%	0%	0%
	%	0.0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Payable Pb in Zn Concentrate	ktonnes	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mlbs	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Revenues Pb in Zn Concentrate	**US$M**	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Min. Deduction Ag in Zn Conc	g/t Ag	93	-	-	-	93	93	93	93	93	93	93	93	93	93	93	93	93	93	93	93	93	93	-	-
Ag Payable	% Payable	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
Payable Ag in Zn Concentrate	Ag koz	2,228	-	-	-	-	-	115	178	121	78	198	260	420	538	319	-	-	-	-	-	-	-	-	-
Revenues Ag in Zn Concentrate	**US$M**	**37.4**	-	-	-	-	-	**1.9**	**3.0**	**2.0**	**1.3**	**3.3**	**4.4**	**7.1**	**9.0**	**5.4**	-	-	-	-	-	-	-	-	-
	C$M	**48.6**	-	-	-	-	-	**2.5**	**3.9**	**2.6**	**1.7**	**4.3**	**5.7**	**9.2**	**11.7**	**7.0**	-	-	-	-	-	-	-	-	-
Zn Treatment Charge	US$/dmt conc	190.00	-	-	-	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	-	-
	US$M	501.3	-	-	-	32.8	33.3	32.1	37.5	36.0	32.9	30.2	25.4	26.5	24.9	24.9	27.5	23.8	24.9	24.9	21.7	20.1	22.0	-	-
	C$M	651.0	-	-	-	42.6	43.3	41.7	48.7	46.7	42.7	39.2	33.0	34.4	32.3	32.3	35.7	30.9	32.3	32.3	28.2	26.1	28.5	-	-
Ag Refining Charge	US$/Ag oz	1.50	-	-	-	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	-	-
	US$M	3.3	-	-	-	-	-	0.2	0.3	0.2	0.1	0.3	0.4	0.6	0.8	0.5	-	-	-	-	-	-	-	-	-
	C$M	4.3	-	-	-	-	-	0.2	0.3	0.2	0.2	0.4	0.5	0.8	1.0	0.6	-	-	-	-	-	-	-	-	-

Total Treatment + Refining Charges	**US$M**	**504.6**	-	-	-	**32.8**	**33.3**	**32.3**	**37.7**	**36.1**	**33.0**	**30.5**	**25.8**	**27.1**	**25.7**	**25.4**	**27.5**	**23.8**	**24.9**	**24.9**	**21.7**	**20.1**	**22.0**	-	-
	C$M	**655.3**	-	-	-	**42.6**	**43.3**	**41.9**	**49.0**	**46.9**	**42.9**	**39.6**	**33.5**	**35.2**	**33.4**	**32.9**	**35.7**	**30.9**	**32.3**	**32.3**	**28.2**	**26.1**	**28.5**	-	-
Zn Concentrate Transport Cost	US$/wmt conc	**163.12**	-	-	-	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	-	-
	US$M	**467.8**	-	-	-	30.6	31.1	30.0	35.0	33.6	30.7	28.2	23.7	24.7	23.2	23.2	25.7	22.2	23.2	23.2	20.3	18.8	20.5	-	-
	C$M	**607.5**	-	-	-	39.8	40.4	38.9	45.4	43.6	39.9	36.6	30.8	32.1	30.1	30.1	33.4	28.9	30.1	30.1	26.3	24.4	26.6	-	-
Hg Penalties	US$/dmt conc	**0.96**	-	-	-	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	-	-
	US$M	**2.5**	-	-	-	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	-	-
	C$M	**3.3**	-	-	-	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.1	-	-
SiO2 Penalties	US$/dmt conc	**2.00**	-	-	-	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	-	-
	US$M	**5.3**	-	-	-	0.3	0.4	0.3	0.4	0.4	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.2	0.2	0.2	-	-
	C$M	**6.9**	-	-	-	0.4	0.5	0.4	0.5	0.5	0.4	0.4	0.3	0.4	0.3	0.3	0.4	0.3	0.3	0.3	0.3	0.3	0.3	-	-
Zn Price Participation	US$/dmt conc	**0.00**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NSR Zn in Zn Concentrate	US$M	**2,516.6**	-	-	-	164.8	167.4	161.1	188.2	180.5	165.2	151.5	127.6	132.9	124.9	124.9	138.2	119.6	124.9	124.9	109.0	101.0	110.2	-	-
	C$M	**3,268.3**	-	-	-	214.0	217.4	209.3	244.4	234.4	214.6	196.7	165.7	172.6	162.2	162.2	179.5	155.3	162.2	162.2	141.5	131.1	143.2	-	-
NSR Pb in Zn Concentrate	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NSR Ag in Zn Concentrate	US$M	**34.1**	-	-	-	-	-	1.8	2.7	1.9	1.2	3.0	4.0	6.4	8.2	4.9	-	-	-	-	-	-	-	-	-
	C$M	**44.3**	-	-	-	-	-	2.3	3.5	2.4	1.5	3.9	5.2	8.3	10.7	6.3	-	-	-	-	-	-	-	-	-
Total Zn Concentrate NSR	**US$M**	**2,550.7**	-	-	-	**164.8**	**167.4**	**162.9**	**190.9**	**182.4**	**166.4**	**154.5**	**131.5**	**139.3**	**133.1**	**129.8**	**138.2**	**119.6**	**124.9**	**124.9**	**109.0**	**101.0**	**110.2**	-	-
	C$M	**3,312.5**	-	-	-	**214.0**	**217.4**	**211.5**	**247.9**	**236.8**	**216.1**	**200.6**	**170.8**	**180.9**	**172.9**	**168.5**	**179.5**	**155.3**	**162.2**	**162.2**	**141.5**	**131.1**	**143.2**	-	-
Pb CONCENTRATE																									
Recovery to Pb Concentrate	% Pb	**75.4%**	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%	75%
	% Zn	**4.8%**	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
	% Ag	**59.4%**	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%
Metal in Pb Concentrate	Pb ktonnes	**875**	-	-	-	39	53	61	75	66	48	63	59	70	86	66	26	40	30	25	19	19	28	-	-
	Pb Mlbs	**1,929**	-	-	-	85	116	136	165	145	107	140	131	155	189	146	58	88	67	55	43	43	62	-	-
	Zn ktonnes	**83**	-	-	-	5	6	5	6	6	5	5	4	4	4	4	5	4	4	4	4	3	4	-	-
	Zn Mlbs	**183**	-	-	-	12	12	12	14	13	12	11	9	10	9	9	10	9	9	9	8	7	8	-	-
	Ag kg	**842,071**	-	-	-	26,728	43,364	55,835	70,461	61,680	52,494	63,156	64,352	84,679	96,636	70,656	22,827	28,915	25,545	22,610	12,392	16,088	23,654	-	-
	Ag koz	**27,073**	-	-	-	859	1,394	1,795	2,265	1,983	1,688	2,031	2,069	2,722	3,107	2,272	734	930	821	727	398	517	760	-	-
Pull Factor		**23**	-	-	-	29	21	18	15	17	23	18	19	16	13	17	43	28	37	45	58	58	34	-	-
Pb Concentrate Grade	% Pb	**61.5%**	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%
	% Zn	**5.8%**	0.0%	0.0%	0.0%	8.7%	6.5%	5.3%	5.1%	5.6%	6.9%	4.9%	4.4%	3.8%	3.0%	3.8%	10.7%	6.1%	8.4%	10.2%	11.5%	10.6%	7.9%	0.0%	0.0%
	g/t Ag	**592**	-	-	-	427	506	558	578	577	666	612	667	740	695	656	535	444	518	560	395	512	516	-	-
	Ag koz	**19**	-	-	-	14	16	18	19	19	21	20	21	24	22	21	17	14	17	18	13	16	17	-	-
Pb Concentrate Produced	dmt	**1,422,916**	-	-	-	62,603	85,765	99,984	121,911	106,806	78,834	103,206	96,475	114,424	139,104	107,693	42,629	65,065	49,359	40,385	31,411	31,411	45,853	-	-
	wmt	**1,546,648**	-	-	-	68,046	93,223	108,678	132,512	116,093	85,689	112,180	104,864	124,374	151,200	117,058	46,335	70,722	53,652	43,897	34,142	34,142	49,840	-	-
Moisture Content	%	**8%**	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Minimum Deduction	%Pb/tonne	**3%**	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%
Pb Payable	% Payable	**95%**	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Pb Payable based on Min. deduction	%	**59%**	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%	59%
Pb Payable based on %	%	**58%**	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%	58%
Payable Pb in Pb Concentrate	ktonnes	**831**	-	-	-	37	50	58	71	62	46	60	56	67	81	63	25	38	29	24	18	18	27	-	-
	Mlbs	**1,833**	-	-	-	81	110	129	157	138	102	133	124	147	179	139	55	84	64	52	40	40	59	-	-
Revenues Pb in Pb Concentrate	**US$M**	**1,796.1**	-	-	-	**79.0**	**108.3**	**126.2**	**153.9**	**134.8**	**99.5**	**130.3**	**121.8**	**144.4**	**175.6**	**135.9**	**53.8**	**82.1**	**62.3**	**51.0**	**39.6**	**39.6**	**57.9**	-	-
	C$M	**2,332.6**	-	-	-	**102.6**	**140.6**	**163.9**	**199.9**	**175.1**	**129.2**	**169.2**	**158.2**	**187.6**	**228.0**	**176.5**	**69.9**	**106.7**	**80.9**	**66.2**	**51.5**	**51.5**	**75.2**	-	-
Min. Deduction Ag in Pb Conc	g/t Ag	**50**	-	-	-	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	-	-
Ag Payable	% Payable	**95%**	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Payable Ag in Pb Concentrate	Ag koz	**23,547**	-	-	-	721	1,193	1,553	1,966	1,721	1,483	1,771	1,818	2,412	2,739	1,994	632	784	705	629	331	443	652	-	-
Revenues Ag in Pb Concentrate	**US$M**	**395.6**	-	-	-	**12.1**	**20.1**	**26.1**	**33.0**	**28.9**	**24.9**	**29.8**	**30.5**	**40.5**	**46.0**	**33.5**	**10.6**	**13.2**	**11.8**	**10.6**	**5.6**	**7.4**	**11.0**	-	-
	C$M	**513.7**	-	-	-	**15.7**	**26.0**	**33.9**	**42.9**	**37.5**	**32.4**	**38.6**	**39.7**	**52.6**	**59.8**	**43.5**	**13.8**	**17.1**	**15.4**	**13.7**	**7.2**	**9.7**	**14.2**	-	-
Pb Treatment Charge	US$/dmt conc	**170.00**	-	-	-	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	170.00	-	-
	US$M	**241.9**	-	-	-	10.6	14.6	17.0	20.7	18.2	13.4	17.5	16.4	19.5	23.6	18.3	7.2	11.1	8.4	6.9	5.3	5.3	7.8	-	-
	C$M	**314.2**	-	-	-	13.8	18.9	22.1	26.9	23.6	17.4	22.8	21.3	25.3	30.7	23.8	9.4	14.4	10.9	8.9	6.9	6.9	10.1	-	-
Ag Refining Charge	US$/Ag oz	**1.50**	-	-	-	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	-	-
	US$M	**35.3**	-	-	-	1.1	1.8	2.3	2.9	2.6	2.2	2.7	2.7	3.6	4.1	3.0	0.9	1.2	1.1	0.9	0.5	0.7	1.0	-	-
	C$M	**45.9**	-	-	-	1.4	2.3	3.0	3.8	3.4	2.9	3.5	3.5	4.7	5.3	3.9	1.2	1.5	1.4	1.2	0.6	0.9	1.3	-	-
Total Treatment + Refining Charges	**US$M**	**277.2**	-	-	-	**11.7**	**16.4**	**19.3**	**23.7**	**20.7**	**15.6**	**20.2**	**19.1**	**23.1**	**27.8**	**21.3**	**8.2**	**12.2**	**9.4**	**7.8**	**5.8**	**6.0**	**8.8**	-	-
	C$M	**360.0**	-	-	-	**15.2**	**21.3**	**25.1**	**30.7**	**26.9**	**20.3**	**26.2**	**24.8**	**30.0**	**36.0**	**27.7**	**10.6**	**15.9**	**12.3**	**10.1**	**7.6**	**7.8**	**11.4**	-	-
Pb Concentrate Transport Cost	US$/dmt conc	**163.12**	-	-	-	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	163.12	-	-
	US$M	**252.3**	-	-	-	11.1	15.2	17.7	21.6	18.9	14.0	18.3	17.1	20.3	24.7	19.1	7.6	11.5	8.8	7.2	5.6	5.6	8.1	-	-
	C$M	**327.7**	-	-	-	14.4	19.7	23.0	28.1	24.6	18.2	23.8	22.2	26.3	32.0	24.8	9.8	15.0	11.4	9.3	7.2	7.2	10.6	-	-
NSR Pb in Pb Concentrate	US$M	**1,301.9**	-	-	-	57.3	78.5	91.5	111.5	97.7	72.1	94.4	88.3	104.7	127.3	98.5	39.0	59.5	45.2	37.0	28.7	28.7	42.0	-	-
	C$M	**1,690.8**	-	-	-	74.4	101.9	118.8	144.9	126.9	93.7	122.6	114.6	136.0	165.3	128.0	50.7	77.3	58.7	48.0	37.3	37.3	54.5	-	-
NSR Ag in Pb Concentrate	US$M	**360.3**	-	-	-	11.0	18.3	23.8	30.1	26.3	22.7	27.1	27.8	36.9	41.9	30.5	9.7	12.0	10.8	9.6	5.1	6.8	10.0	-	-
	C$M	**467.9**	-	-	-	14.3	23.7	30.9	39.1	34.2	29.5	35.2	36.1	47.9	54.4	39.6	12.6	15.6	14.0	12.5	6.6	8.8	13.0	-	-
Total Pb Concentrate NSR	**US$M**	**1,662.2**	-	-	-	**68.3**	**96.7**	**115.2**	**141.6**	**124.1**	**94.8**	**121.5**	**116.1**	**141.6**	**169.2**	**129.0**	**48.7**	**71.5**	**55.9**	**46.6**	**33.8**	**35.5**	**51.9**	-	-
	C$M	**2,158.7**	-	-	-	**88.7**	**125.6**	**149.7**	**183.9**	**161.1**	**123.1**	**157.8**	**150.8**	**183.9**	**219.7**	**167.6**	**63.2**	**92.9**	**72.7**	**60.5**	**43.9**	**46.1**	**67.4**	-	-
NSR Royalty	US$M	**4.0**	-	-	4.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	**5.2**	-	-	5.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total NSR	**US$M**	**4,208.8**	-	-	4.0	233.1	264.1	278.1	332.5	306.4	261.2	276.0	247.6	280.9	302.3	258.8	186.9	191.1	180.8	171.5	142.7	136.5	162.2	-	-
	C$M	**5,466.0**	-	-	5.2	302.7	343.0	361.2	431.8	397.9	339.3	358.5	321.6	364.8	392.6	336.1	242.7	248.2	234.9	222.7	185.4	177.3	210.6	-	-
	C$/tonne	**167.38**	-	-	-	165.87	187.94	197.77	236.63	218.05	185.85	195.89	175.73	199.33	214.54	183.68	132.61	135.62	128.34	121.69	101.30	96.87	133.80	-	-
OPEX																									
OP Mining	C$M	**111.9**			-	53.9	36.8	20.8	0.1	0.1	0.1	-	-	-	-	-	-	-	-	-	-	-	0.1		
	C$/tonne	**4.45**	-	-	-	4.16	5.07	20.67	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
UG Mining	C$M	**1,478.7**			-	19.1	10.8	71.7	86.2	89.2	86.1	92.9	103.3	96.2	94.2	97.3	95.4	97.6	93.2	98.9	98.5	90.5	57.6	-	-
	C$/tonne	**52.02**	-	-	-	167.12	-	50.75	48.99	50.68	48.78	50.74	56.47	52.54	51.46	53.15	52.13	53.35	50.95	54.06	53.82	49.46	38.77	-	-
Processing	C$M	**748.5**	-	-	-	41.8	41.8	41.9	41.8	41.8	41.8	41.9	41.9	41.9	41.9	41.9	41.9	41.9	41.9	41.9	41.9	41.9	36.1	-	-
	C$/tonne	**22.92**	-	-	-	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	22.92	-	-
G&A	C$M	**338.6**				19.7	19.7	20.4	18.8	18.8	18.9	19.1	19.2	19.0	18.9	18.6	18.5	18.5	18.5	18.7	18.5	17.7	16.9	0.1	-
	C$/tonne	**10.37**				10.78	10.79	11.18	10.30	10.30	10.33	10.43	10.47	10.41	10.35	10.17	10.13	10.13	10.13	10.21	10.09	9.70	10.72	-	-
Total OPEX	**C$M**	**2,677.6**	-	-	-	**134.5**	**109.2**	**154.7**	**146.9**	**149.9**	**146.9**	**153.9**	**164.4**	**157.1**	**155.1**	**157.8**	**155.9**	**158.1**	**153.7**	**159.6**	**158.9**	**150.2**	**110.6**	**0.1**	-
	C$/tonne	**82.00**	-	-	-	**73.70**	**59.83**	**84.72**	**80.51**	**82.14**	**80.47**	**84.09**	**89.86**	**85.87**	**84.73**	**86.24**	**85.18**	**86.41**	**84.00**	**87.19**	**86.83**	**82.08**	**70.29**	-	-
Net Operating Income	**C$M**	**2,788.4**	-	-	5.2	168.2	233.8	206.5	284.9	248.0	192.4	204.6	157.1	207.6	237.6	178.3	86.8	90.1	81.1	63.1	26.5	27.1	100.0	0.1	-
	C$/tonne	**85.39**	-	-	-	92.16	128.11	113.05	156.12	135.91	105.38	111.80	85.87	113.46	129.81	97.44	47.43	49.22	44.34	34.49	14.47	14.80	63.51	-	-
CAPEX																									
Mining	C$M	**408.6**		-	30.3	39.9	60.5	35.7	29.0	21.1	25.7	29.6	16.1	31.1	22.0	12.3	8.7	7.4	9.4	16.3	7.8	2.5	3.3	-	-
Site Development	C$M	**13.1**		3.6	8.4	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	-	-
Mineral Processing	C$M	**76.1**		11.8	58.8	-	0.4	0.2	0.4	0.2	0.4	0.2	0.4	0.2	0.4	0.2	0.4	0.2	0.4	0.2	0.4	0.2	-	-	-
Tailings Mangement	C$M	**146.6**		8.7	24.0	24.5	-	22.3	-	22.3	-	-	-	22.3	-	-	0.2	22.3	-	-	-	-	-	-	-
On-Site Infrastructure	C$M	**66.2**		14.9	36.5	9.5	0.1	0.0	3.2	0.0	0.1	0.0	0.4	0.0	0.1	0.3	0.1	-	0.1	0.3	-	0.1	-	-	-
Off-Site Infrastructure	C$M	**84.9**		39.1	39.1	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	-	-	-
Project Indirects	C$M	**43.0**		4.3	38.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EPCM	C$M	**20.5**		6.6	13.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Owner Costs	C$M	7.0		0.3	6.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure	C$M	56.7		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	56.7	-
Subtotal	C$M	922.7	-	89.3	256.5	74.4	61.5	58.8	33.1	44.2	26.7	30.3	17.4	54.2	23.0	13.3	9.9	30.5	10.4	17.4	8.6	3.4	3.3	56.7	-
Contingency	C$M	130.9		5.9	52.7	13.5	3.4	9.2	2.2	8.1	0.2	0.1	0.3	7.9	0.2	0.2	0.3	7.9	0.2	0.2	0.1	0.1	0.0	18.1	-
Total CAPEX	**C$M**	**1,053.6**	**-**	**95.2**	**309.2**	**87.9**	**64.9**	**68.0**	**35.3**	**52.3**	**26.9**	**30.5**	**17.6**	**62.1**	**23.2**	**13.6**	**10.2**	**38.4**	**10.6**	**17.6**	**8.7**	**3.5**	**3.4**	**74.7**	**-**
	C$/tonne	**32.26**																							
Pre-Production	**C$M**	**404.3**	**-**	**95.2**	**309.2**																				
Sustaining	**C$M**	**649.3**				**87.9**	**64.9**	**68.0**	**35.3**	**52.3**	**26.9**	**30.5**	**17.6**	**62.1**	**23.2**	**13.6**	**10.2**	**38.4**	**10.6**	**17.6**	**8.7**	**3.5**	**3.4**	**74.7**	**-**
Working Capital	C$M	0.0			22.4																			-22.4	
Net Pre-Tax Cash Flow	**US$M**	**1,335.8**	0.0	-73.3	-259.3	61.8	130.1	106.6	192.2	150.7	127.4	134.1	107.4	112.1	165.1	126.8	59.0	39.8	54.3	35.1	13.7	18.1	74.4	-40.4	0.0
	C$M	**1,734.8**	0.0	-95.2	-336.8	80.3	168.9	138.5	249.6	195.8	165.5	174.1	139.5	145.5	214.4	164.7	76.6	51.7	70.6	45.6	17.7	23.5	96.6	-52.4	0.0
Cumulative Net Pre-Tax Cash Flow	**C$M**		0.0	-95.2	-431.9	-351.6	-182.7	-44.3	205.4	401.2	566.6	740.8	880.3	1,025.8	1,240.2	1,404.9	1,481.5	1,533.2	1,603.8	1,649.3	1,667.1	1,690.6	1,787.2	1,734.8	1,734.8
Taxes	**C$M**	**615.7**	-	-	-	**7.1**	**41.3**	**40.6**	**71.4**	**60.8**	**42.2**	**48.8**	**39.0**	**58.0**	**67.6**	**52.2**	**21.0**	**22.5**	**17.4**	**14.3**	**2.0**	**3.6**	**30.5**	**-24.7**	-
Income Tax	*C$M*	*422.2*	-	-	-	-	*29.4*	*28.8*	*50.1*	*44.0*	*32.8*	*38.0*	*27.4*	*38.8*	*46.9*	*33.8*	*13.8*	*15.0*	*13.7*	*9.5*	*1.5*	*2.5*	*20.7*	*-24.7*	-
Yukon Mineral Royalty	*C$M*	*193.5*	-	-	-	*7.1*	*11.9*	*11.8*	*21.2*	*16.8*	*9.4*	*10.8*	*11.5*	*19.2*	*20.8*	*18.4*	*7.2*	*7.5*	*3.7*	*4.8*	*0.5*	*1.1*	*9.8*	-	-
Net Post-Tax Cash Flow	**US$M**	861.7	-	-73.3	-259.3	56.4	98.3	75.3	137.3	103.9	94.9	96.5	77.4	67.4	113.0	86.7	42.8	22.5	40.9	24.1	12.1	15.3	50.9	-21.3	-
	C$M	1,119.1	-	-95.2	-336.8	73.2	127.7	97.8	178.3	134.9	123.2	125.3	100.5	87.5	146.7	112.5	55.6	29.2	53.2	31.2	15.7	19.9	66.2	-27.7	-
Cumulative Net Pre-Tax Cash Flow	**C$M**		-	-95.2	-431.9	-358.7	-231.1	-133.3	45.0	180.0	303.2	428.5	529.0	616.6	763.3	875.8	931.4	960.6	1,013.8	1,045.0	1,060.8	1,080.7	1,146.8	1,119.1	1,119.1

ECONOMIC INDICATORS		
Pre-Tax Results		
Pre-Tax NPV	**US$M**	**600.1**
	C$M	**779.3**
Pre-Tax NPV	**US$M**	**1,335.8**
	C$M	**1,734.8**
IRR	**%**	**31.9%**
Payback	**Years**	**3.2**
Post-Tax Results		
Post-Tax NPV	**US$M**	**345.2**
	C$M	**448.3**
Post-Tax NPV	**US$M**	**861.7**
	C$M	**1,119.1**
IRR	**%**	**23.5%**
Payback	**Years**	**4.0**

Source: JDS (2018)

24 Adjacent Properties

Adjacent claims are owned by major and junior mining and mineral exploration companies. These claims cover known precious and base metal prospects and anomalies, none of which are at the advanced stage of the Macmillan Pass Project.

25 Other Relevant Data and Information

There are no additional relevant data, information or explanation necessary to make this report understandable and not misleading.

26 Interpretations and Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used in the PEA.

The Macmillan Pass project contains a substantial zinc, lead and silver resource that can be mined by open pit and underground methods and recovered with conventional flotation processing.

Based on the assumptions used for this preliminary evaluation, the project is economic and should proceed to the pre-feasibility stage.

There is a likelihood of improving the project economics by identifying additional mineral resources within the development area that may justify increased mine production or extend the mine life.

To date, the QPs are not aware of any fatal flaws for the Project.

26.1 Risks

As with most mining Projects, there are many risks that could affect the economic viability of the Project. Many of these risks are based on lack of detailed knowledge and can be managed as more sampling, testing, design, and detailed engineering are conducted. Table 26-1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches.

The most significant potential risks associated with the project are uncontrolled dilution, uncontrolled groundwater inflow in the mines, lower metal recoveries than those projected, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource and reserve estimates.

Table 26-1: Main Project Risks

Risk	Explanation / Potential Impact	Possible Risk Mitigation
Dilution	Higher than expected dilution can have a severe impact on project economics. The mine must ensure accurate drilling and blasting practices are implemented to minimize dilution from wall rock, backfill and other low grade mineralized zones.	A well planned and executed grade control plan is necessary immediately upon commencement of mining.
Water Inflow	The management of water on-site is a critical component of the project design. Basic assumptions were made for surface and underground water flows based on preliminary drilling and hydro-geologic information.	Continued collection and analysis of data relating to underground and surface water needs to be continued on-site over the near-term to enhance the local hydrological knowledge.
Metallurgical Recoveries	While it is believed that the various programs of sampling and metallurgical test work conducted to date are appropriate to support a PEA, factors other than process conditions, such as dilution, plant ramp-up that could lead to reduced metal recovery and / or increased processing OPEX costs. If LOM, metal recoveries is lower, or costs higher, than estimated, the Project economics would be negatively impacted.	Additional sampling and test work should be conducted in the next project phase. Early process team recruitment and training, implementation of good quality instrumentation and process control.
Resource Modeling	All Mineral Resource estimates carry some risk and are one of the most common issues with Project success. 74% of the resources in the PEA mine plan are classified as Inferred.	Infill drilling and increased sampling is recommended in order to provide a greater level of confidence in certain areas. Infill drilling required with Inferred resources to convert them to reserves.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of Project success. If OPEX increases then the mining cut-off grade would increase and, all else being equal, the size of the optimized pit would reduce yielding fewer mineable tonnes.	Active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Timely Approval of Project Authorizations	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.
Development Schedule	The Project development could be delayed for a number of reasons and could impact Project economics. A change in schedule would alter the Project economics.	Select EPCM firm and develop detailed construction schedule
Acid Rock Drainage	Acid Rock Drainage at the Project site could pose problems during permitting due to it's adverse environmental effects.	Continue with rigorous monitoring program and highlight the fact that there is naturally acidic waters in un-mined areas in the valley during the permitting process.
Materials Balance	The TMF embankment and many pads, roads, and foundations are constructed with mined material (overburden and mine rock), that could be potentially acid generating (PAG) and the production of mine rock according to the mine plan may not be sufficient to provide the capacity needed for all uses.	Early production/excavation of mine rock (non-mineralized) from the pit to assure an adequate supply of construction material
Mine Geotech	The geotechnical nature of the open pit and underground stope wall rock, including the nature and orientation of faults and secondary geological structures could impact pit slopes and stope sizes.. Pit slopes and stope sizes could be increased or decreased and thus alter the pit and stope designs, mineable tonnes, and strip ratio.	Improved geotechnical knowledge and modelling if necessary
Availability of Experienced and Skilled Operating and Maintenance Personnel	Providing employment opportunities to the local and Indigenous communities is an objective of the Company. However, during the key early operating years there may be a need to acquire skilled and seasoned employees outside of the regional area.	Use of sophisticated screening techniques to ensure those recruited have the necessary attitude and aptitude to succeed and provide a comprehensive training program for those new to the industry.

Source: JDS (2018)

26.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the project. The major opportunities that have been identified at this time are summarized in Table 26-2, excluding those typical to all mining projects, such as changes in metal prices, exchange rates, and etc. Further information and assessments are needed before these opportunities should be included in the project economics.

Table 26-2: Major Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of Mineral Resources	The mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource as well as discover new mineralized zones.	Increased mine life.
Pit Slope Steepening	Pit slope angles could potentially be improved which may increase slope angles (conversely it could also make them shallower).	An increase in overall pit slopes for all domains in all pits would reduce the strip ratio and increase the tonnes of metals mined.
Project Strategy and Optimization	With additional detailed planning and a series of strategic option reviews the Project may be able to add value.	Planning and executing the Project with the optimum mine design/schedule and processing systems would result in the maximum possible value to shareholders and other economic stakeholders.
Potential to Purchase Good Used Equipment	There is considerable used equipment on the market that could be utilized.	Capital cost reduction
Pre-sorting Mineralized Material	Test work could indicate pre-sorting viability.	Increased processed grade and reduced OPEX.
Improve Concentrate Haul	Evaluate opportunities for optimization of concentrate haul by use of pipelines or rail systems	Decrease operating costs
Reduce tailings embankment borrow cost	Existence of non-acid-generating material within the pre-strip prism will reduce cost of borrow	Reduce Capital Cost

Source: JDS (2018)

27 Recommendations

27.1 Recommended Work Programs

27.1.1 Waste and Water Management

Recommendations for the next phase of engineering for the Project are summarized below:

- Collection of site-specific meteorological and hydrology data. This data will be used to confirm seasonal runoff values and design storms;
- Complete a detailed BAT assessment for waste and water management in future studies. The assessment will confirm the preferred location, tailings management technology and water management strategy;
- Evaluation of thickened tailings and the use of tailings for backfill;
- Complete site investigations programs at the TMF, Waste Rock Management Facilities and Mill Site to support the next phase of design and to comply with updated regulatory requirements;
- Confirm the geotechnical characteristics of the tailings and construction materials;
- Complete characterization of tailings and construction materials to access potential acid rock drainage and other potential chemical releases (metal leaching);
- Complete hydrogeological site investigation programs to estimate the inflows to the open pit and underground mines and determine the dewatering requirements;
- Optimize the water balance to incorporate updated runoff and process flow estimates;
- Tailings materials and properties should be reviewed during the next phase of design to be sure they are representative, especially if any changes to the process occur. Representative tailings samples should be provided and tested when they become available; and
- Develop a full closure plan for the waste and water management facilities based on the final design configuration.

27.1.2 Infrastructure and Logistics Management

Recommendations for the next phase of engineering for the Project are summarized below:

- Investigation of Ross River barge crossing. Capacity, scheduling, availability and seasonality;
- Site access road review to scope out the current condition of the road and update plans for re-construction and updating, building upon historical reports prepared for the Yukon Government;
- Review existing port facilities at Prince Rupert and Stewart, BC, and at Skagway, Alaska, and determine suitability, future availability, and the CAPEX and OPEX of bulk ore and container loading;
- Evaluate feasibility of slurry pipeline for concentrate transport; and

- Investigate feasibility of transfer to rail haulage from Carcross to Skagway.

27.1.3 Metallurgical Test Program

Recommendations for additional metallurgical test work are listed below:

- Phase I – three global composites representing the proposed mine plan for Year 0 to 1, 1 to 3 and 3 to end of mine, be prepared and used for flowsheet optimization. The composites will be subjected to mineralogical analysis, comminution test work, flotation tests including locked cycle tests and settling and filtering assessments.
- Phase II – 25 composites representing discrete continuous intervals of mineralization to be used to assess variability in the deposit. The samples will undergo mineralogical analysis as well as comminution test work. The optimized flowsheet and parameters established in the Phase I program will be used as the basis for the flotation test work in Phase II to establish metallurgical performance.
- A test program on Jason South including mineralogical analysis, comminution test work, and flotation tests including locked cycle tests should be completed as part of the next stage of engineering.

27.1.4 Mining and Mine Geotechnical

Recommendations for the PFS phase of engineering for the Project are summarized below:

- Review upcoming resource drilling plan and select approximately five to ten resource drillholes at each deposit for detailed geotechnical logging;
- Orient all natural fractures as well as open foliation and bedding planes for the resource drillholes selected for detailed geotechnical logging;
- Develop structural geology (faults ,shear, bedding, foliation) model of mineralized zones;
- Conduct cemented rock fill laboratory testing to confirm backfill design and binder requirements;
- Investigate the use of paste backfill and conduct laboratory testing to confirm design parameters if selected as a fill method;
- Collect approximately 10 core samples a minimum length of three times the core diameter from each major geologic unit for laboratory strength testing. Emphasis should be made on the immediate HW, FW and mineralized zone as well as critical infrastructure areas;
- Carry out basic laboratory strength testing program consisting of UCS, Brazilian indirect tensile strength and triaxial compressive strength; and
- Conduct geotechnical mapping of drifts in open section of Tom exploration adit.

27.2 Costs

It is estimated that a pre-feasibility study and supporting field work would cost approximately $10.3 million. A breakdown of the key components of the next study phase is as follows in Table 27-1.

Table 27-1: Cost Estimate to Advance to Pre-feasibility Phase

Component	Estimated Cost ($C M)	Comment
Resource Drilling	5.0	Conversion of inferred to indicated resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.5	Comminution, flotation optimization, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Geochemistry	0.5	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	0.8	Site investigation drilling, sampling and lab testing
Geotechnical, Hydrology & Hydrogeology	1.0	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	1.5	PFS-level mine, infrastructure and process design, cost estimation, scheduling & economic analysis.
Environment	1.0	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**10.3**	Excludes corporate overheads and future permitting activities

28 Units of Measure, Calculations and Abbreviations

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ac	acre
Acfm	actual cubic feet per minute
ACK	apparent coherent kimberlite
ALT	active layer thickness
ALT	active layer thickness
amsl	above mean sea level
AN	Qilaq mineral tenure
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
BHPB	BHP Billiton limited
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
cfm	cubic feet per minute
CHI	Chidliak mineral tenure
CHP	combined heat and power plant
CIM	Canadian institute of mining and metallurgy
CK	coherent kimberlite
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre

Symbol / Abbreviation	Description
cP	centipoise
c/s	carats per stone
c/t	carat per tonne
Cr	chromium
ct	carat
Cu	copper
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DBCEI	De Beers Canada exploration Inc.
DGPS	differential global positioning system
DICAN	Diamonds International Canada
DMS	dense media separation
dmt	dry metric ton
DNLUP	draft Nunavut land use plan
DTC	diamond trading company
DWT	dead weight tonnes
EA	environmental assessment
EIS	environmental impact statement
ELC	ecological land classification
ERD	explosives regulatory division
EWR	enhanced winter road
FEL	front-end loader
FOC	fisheries and oceans Canada
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
g/t	grams per tonne
Ga	billion years
gal	gallon (us)
GJ	gigajoule

Symbol / Abbreviation	Description
GPa	gigapascal
gpm	gallons per minute (US)
GSC	geological survey of Canada
GTZ	glacial terrain zone
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
ha	hectare
HG	high grade
HK	hypabyssal kimberlite
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
INAC	Indigenous and Northern Affairs Canada
IOL	Inuit owned land
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
kg	kilogram
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
KIM	kimberlitic indicator mineral
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
kPa	kilopascal

Symbol / Abbreviation	Description
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LG	low grade
LGM	last glacial maximum
LOM	life of mine
m	metre
M	million
m/d	metres per day
m/min	metres per minute
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
m^3/s	cubic metres per second
Ma	million years
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
mamsl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
MBP	melt-bearing pyroclasts
mbs	metres below surface
mbsl	metres below sea level
Mct	million carats

Symbol / Abbreviation	Description
mg	milligram
mg/L	milligrams per litre
MIDA	microdiamond
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations
MMSIM	metamorphosed massive sulphide indicator minerals
mo	month
MPa	megapascal
MSC	Mineral Services Canada Inc.
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
NAD	North American datum
NG	normal grade
Ni	nickel
NI 43-101	national instrument 43-101
NIRB	Nunavut impact review board
NLCA	Nunavut lands claim agreement
Nm^3/h	normal cubic metres per hour
NMR	Nunavut mining regulations
NPC	Nunavut planning commission
NQ	drill core diameter of 47.6 mm
NRC	natural resources Canada
NSA	Nunavut settlement area
NTI	Nunavut Tunngavik Incorporated
NU	Nunavut
NUPPA	Nunavut planning and project assessment act
NWB	Nunavut water board
OP	open pit
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
PEA	preliminary economic assessment

Symbol / Abbreviation	Description
PFK	processed fine kimberlite
PFS	preliminary feasibility study
PGE	platinum group elements
PK	pyroclastic kimberlite
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QIA	Qikiqtani Inuit Association
QP	qualified person
RC	reverse circulation
RIA	regional Inuit associations
RMR	rock mass rating
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
RVK	resedimented volcaniclastic kimberlite
s	second (time)
S.G.	specific gravity
Scfm	standard cubic feet per minute
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SFD	size frequency distribution
SG	specific gravity
SRC	Saskatchewan Research Council
SRK	SRK consulting services Inc.
st/kg	stones per kilogram
st/t	stones per metric tonne
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
TCR	total core recovery
TFFE	target for further exploration
TMF	tailings management facility
tph	tonnes per hour

Symbol / Abbreviation	Description
ts/hm^3	tonnes seconds per hour metre cubed
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VEC	valued ecosystem components
VK	volcaniclastic kimberlite
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight
wk	week
wmt	wet metric ton
WRSF	waste rock storage facility
WRSF	waste rock storage facility
µm	microns
µm	micrometre

Scientific Notation	Number Equivalent
1.0E+00	1
1.0E+01	10
1.0E+02	100
1.0E+03	1,000
1.0E+04	10,000
1.0E+05	100,000
1.0E+06	1,000,000
1.0E+07	10,000,000
1.0E+09	1,000,000,000
1.0E+10	10,000,000,000

Rock Type	Description
ACK	Apparent Coherent Kimberlite
CK	Coherent Kimberlite
CRX	Country rock xenolith
HK	Hypabyssal Kimberlite
LSTX	Paleozoic carbonate xenolith
PK	Pyroclastic Kimberlite
RVK	Resedimented Volcaniclastic Kimberlite
VK	Volcaniclastic Kimberlite

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Qualified Persons Certificates

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CERTIFICATE OF AUTHOR

I, Michael Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Macmillan Pass Project, Yukon Territory, Canada" with an effective date of May 23, 2018, (the "Technical Report") prepared for Fireweed Zinc Ltd.;

2. I am currently employed as Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

   I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over ten years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and technical report writing for mining projects worldwide;

   I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

4. I visited the Macmillan Pass Project site September 5, 2017;

5. I am responsible for Sections 1.1, 1.2, 1.7, 1.9 to 1.13, 2, 3, 15, 16 (except 16.4), 18 (except 18.6, 18.7, 18.8, 18.9), 19, 20, 21, 22, 23, 24, 25, 26, 27, 28, 29 of this Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no involvement with the property that is the subject of this Technical Report;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: May 23, 2018
Signing Date: July 6, 2018

(Original signed and sealed) "Michael Makarenko, P. Eng."

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I, Kelly McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Technical Report, Macmillan Pass Project, Yukon Territory, Canada” with an effective date of May 23, 2018, (the “Technical Report”) prepared for Fireweed Zinc Ltd.;

2. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 10 years consulting in the mining industry in metallurgy and process design engineering;

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I have not personally visited the Macmillan Pass Project site;

6. I am responsible for Sections 1.5, 1.8, 13 & 17 of this Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have not had prior involvement with the property that is the subject of this Technical Report;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: May 23, 2018
Signing Date: July 6, 2018

(Original signed and sealed) “Kelly McLeod, P. Eng.”

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I, Michael Levy, P. Eng., do hereby certify that:

1. I am currently employed as Geotechnical Engineering Manager with JDS Energy & Mining Inc. with an office at Suite 100 – 14143 Denver West Parkway, Golden, Colorado, 80401;
2. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Macmillan Pass Project, Yukon Territory, Canada" with an effective date of May 23, 2018, (the "Technical Report") prepared for Fireweed Zinc Ltd.;
3. I am a Professional Civil Engineer (P.Eng. #2692) registered with the Association of Professional Engineers Yukon and Colorado (P.E. #40268). I am a current member of the International Society for Rock Mechanics (ISRM) and the American Society of Civil Engineers (ASCE). I hold a bachelor's degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004. I have practiced my profession continuously since 1999 and have been involved in a numerous mining and civil geotechnical projects across the Americas;
4. I visited the Macmillan Pass Project site September 5, 2017;
5. I am responsible for Section 16.4 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. I have read the definition of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association, and past relevant experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1; and,
10. As of the effective date of the Report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: May 23, 2018
Signing Date: July 6, 2018

(Original signed and sealed) "Michael Levy, P. Eng."

Michael Levy, P. Eng.

Certificate of Qualified Person – Dr Dennis Arne

As a Qualified Person of this Technical Report covering the Project named as the NI 43-101 Technical Report on the Macmillan Pass Project, Yukon Territory, Canada, I, Dennis Arne do hereby certify that:

I am a Principal Consultant of CSA Global Geosciences Canada Ltd, and carried out this assignment for CSA Global Geosciences Canada Ltd of Suite 610, 1155 West Pender Street, Vancouver, British Colombia, Canada (dennis.arne@csaglobal.com).

The Technical Report to which this certificate applies is titled "NI43-101 Technical Report on the Macmillan Pass Project, Yukon Territory, Canada" and is dated 23 May 2018 ("the effective date").

I hold a BSc (Hons), MSc, PhD and Graduate Diploma, and am a registered Professional Geologist in good standing of the Engineers and Geoscientists British Colombia (#34686) and a Registered Professional Geoscientist of the Australian Institute of Geoscientists (#10064). I am familiar with NI 43-101 and, by reason of education, experience in exploration and evaluation of hydrothermal deposits, including sediment-hosted base metal deposits, and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. My experience includes more than 35 years in geology.

I personally visited the project that is the subject of this Technical Report between 31 August 2011 and 2 September 2011, between 24 July 2017 and 28 July 2017, and between 25 June 2018 and 29 June 2018, for a total of 13 days.

I am responsible for Sections 1.3, 4, 5, 6, 7, 8, 9, 11, and 12 of the Report.

I am independent of the issuer and the Property as described in Section 1.5 of NI 43-101.

I co-authored the Issuer's previous NI 43-101 technical report for the MacMillan Pass Project dated February 24, 2018.

I have read NI 43-101 and this Technical Report for which I am responsible has been prepared in compliance with N1 43-101.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2018.

"Signed and Stamped"

Dr. Dennis Arne, PGeo (BC), MAIG (RPGeo)
Principal Consultant - Geochemistry
CSA Global Geosciences Canada Ltd

Certificate of Qualified Person – Mr Leon McGarry

As a Qualified Person of this Technical Report covering the Project named as the NI 43-101 Technical Report, Macmillan Pass Project, Yukon Territory, Canada, I, Leon McGarry do hereby certify that:

4. I am a Senior Consultant of CSA Global Geosciences Canada Ltd, and carried out this assignment for CSA Global Geosciences Canada Ltd of Suite 501, 365 Bay Street, Toronto, Ontario, Canada (leon.mcgarry@csaglobal.com).
5. The Technical Report to which this certificate applies is titled "NI 43-101 Technical Report Macmillan Pass Project, Yukon Territory, Canada" and is dated 23 May 2018 ("the effective date").
6. I hold a BSc (Hons) and am a registered member of the Association of Professional Geoscientists of Ontario (2348) and the Association of Professional Engineers and Geoscientists of Saskatchewan (34929). I am familiar with NI 43-101 and, by reason of education, experience in exploration and evaluation of hydrothermal deposits, including sediment-hosted base metal deposits, and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. My experience includes more than 10 years in geology.
7. I have not personally visited the project that is the subject of this Technical Report.
8. I am responsible for Sections 1.4, 1.6, 10 and 14 of the Report.
9. I am independent of the issuer, and the Property applying all the tests in section 1.5 of National Instrument 43-101.
10. I co-authored the Issuer's previous NI 43-101 technical report for the MacMillan Pass Project dated February 24, 2018. I prepared the current Mineral Resource Estimate for the Project which has an effective date of 10 January 2018.
11. I have read NI 43-101 and this Technical Report for which I am responsible has been prepared in compliance with N1 43-101.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2018

"Signed and Stamped"

Leon McGarry, PGeo (ON, SK)
Senior Consultant – Resources
CSA Global Geosciences Canada Ltd

CERTIFICATE OF QUALIFIED PERSON

I, Kenneth Embree, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Technical Report, Macmillan Pass Project, Yukon Territory, Canada” with an effective date of May 23, 2018, (the “Technical Report”) prepared for Fireweed Zinc Ltd.;
2. I am employed as Managing Principal of Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
3. I am a graduate of the University of Saskatchewan with a B.Sc. in Geological Engineering (1986). I have practiced my profession continuously since 1986. My experience includes tailings and waste and water management for mine developments in Canada, the US and South America.
4. I am a Professional Engineer in good standing with Engineers and Geoscientists of British Columbia in the area of geological engineering (No. 17439). I am also registered as a Professional Engineer in Ontario (No. 100040332), Yukon (No. 2694) and the Northwest Territories and Nunavut (No. L3766).
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Macmillan Pass Project site September 5, 2017;
7. I am responsible for Sections 18.6, 18.7, 18.8 and 18.9 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have had no involvement with the property that is the subject of this Technical Report;
10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: May 23, 2018
Signing Date: July 6, 2018

(Original signed and sealed) “Kenneth Embree, P.Eng.”

Kenneth Embree, P. Eng.