NI 43-101 TECHNICAL REPORT
PALMER PROJECT
ALASKA, USA

JDS Energy & Mining Inc.

CONSTANTINE metal resources ltd.

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Constantine Metal Resources 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.

Constantine Metal Resources 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.

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1 Executive Summary

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Constantine Metal Resources Ltd. (Constantine or the Company) to manage and compile a Preliminary Economic Assessment (PEA) for the Palmer Project (Palmer or the Project), the results of which are summarized in this Technical Report as per the guidelines of the Canadian Securities Administrator's National Instrument 43-101 (NI 43-101) Form 31-101F1. Other contributors to this PEA include Klohn Crippen Berger Ltd., Core Geoscience Services Inc., and Advantage Geoservices Ltd.

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

1.2 Project Description and Ownership

The Palmer Project is located 60 km northwest from Haines, Alaska in the Alaska panhandle. It lies 2 km from the Haines Highway, which links the deep-sea port of Haines, Alaska, USA with Haines Junction, Yukon, Canada on the Alaska Highway.

The Project consists of a contiguous block of land consisting of 340 federal unpatented lode mining claims, which cover an area of approximately 6,765 acres (~2,738 hectares or 27 km^2) and 63 state mineral claims that cover an area of approximately 9,200 acres (~3,680 hectares or 37 km^2). These core claims are surrounded by land leased by the Company from the Alaska Mental Health Trust which total 65,772 acres, giving a Project total of 81,737 acres (~ 33,078 hectares or 330 km^2). Constantine, through its wholly owned US subsidiary Constantine North Inc., has a 99-year Mineral Lease Agreement on the 340 federal unpatented lode mining claims with Alyu Mining, Inc. and Haines Mining-Exploration Inc. (collectively the "Owners"), both of Haines, Alaska.

The host rock at Palmer is the same late Triassic volcanogenic massive sulfide (VMS) belt as the high-grade producing Greens Creek Mine and the Windy Craggy copper deposit.

Constantine signed an Option and Joint Venture Agreement (Agreement) with Dowa Metals & Mining Co., Ltd. of Japan (Dowa) on February 1st, 2013. Under the terms of the Agreement, Dowa had the option to earn a 49% interest in the Project by making aggregate expenditures of US$22,000,000 over a four-year period. On January 5th, 2017, the Company announced that Dowa had completed its US$22 million earn-in to the Project and had exercised its option to participate as a partner in the Project. A Joint Venture was formed (the Dowa JV) for the purpose of further exploring and developing Project, with Constantine owning a 51% participating interest and Dowa owning a 49% participating interest.

Total expenditures on the Project to the end of 2018 have been US$46.1 million (as of January 31st, 2019).

1.3 Geology and Mineralization

The Project lies within a mafic-dominated, bimodal sequence of submarine volcanic rocks host to VMS mineralization. These rocks are part of a ~600 km-long, discontinuously exposed belt of Late Triassic, rift-related volcanic and sedimentary rocks belonging to the Alexander Terrane. Throughout southeast Alaska and northwest British Columbia, the Alexander Terrane hosts numerous VMS occurrences, prospects and deposits, including the giant Windy Craggy Cu-Co-Au deposit in British Columbia, and the precious metals-rich Ag-Zn-Pb-Au Greens Creek Mine in southeast Alaska (Taylor, 1997). The Project area itself is underlain by Paleozoic and lower Mesozoic metasedimentary and metavolcanic rocks that have been intruded locally by Cretaceous and Tertiary granitic plutons.

The Project hosts two known VMS deposits, the Palmer deposit, which consists of the South Wall and RW Zones, and the newly discovered AG Zone deposit, located three km to the southwest (At the South Wall and RW Zones, six mineralization styles have been identified and are grouped according to dominant mineral assemblages and texture and include: Barite mineralization (Zn-rich), Massive Pyrite Mineralization (Cu-rich), Semi-massive and Stringer-style Mineralization, Massive Pyrrhotite Mineralization, Carbonate Mineralization and Barite-Carbonate Mineralization. At the AG Zone deposit, mineralization consists of Massive and Semi-massive sulfide and barite, and feeder-style stringers and replacement. Four alteration facies are associated with the known mineralized zones and include: Quartz-Pyrite, Muscovite, Carbonate-Chlorite and Epidote.

The South Wall Zones (SWZI, SWZII-III, SWEMZ) are located on the south-facing, steeply dipping limb of megascopic, deposit-scale anticline, disrupted by recognized thrust faulting, normal faulting and strike-slip faulting. The RW Zones (RW East, RW West, RW Oxide) are located on the north-facing, gently dipping upper limb of the same anticline. The RW Oxide Zone is the near surface equivalent of the RW East Zone where sulfide minerals of massive barite-sulfide mineralization have been oxidized and leached, depleting the zone of copper and zinc and enriching the silver and gold grades. The AG Zone deposit (including the AG Main Lens and AG Footwall Zone) is located three km m to the southwest, on a steep Nunatak between the Saksaia and South Saksaia Glaciers.

Drilling to date has defined a total plunge length of near-continuous South Wall mineralization of 700 m, and a total strike length to 550 m, with exhalative mineralization occurring at more than one stratigraphic level. The RW Zones have been defined over a dip length of 325 m, and a total strike length of 800 m. The new AG Zone deposit has a strike length of 550 m and a vertical extent of 250 m. Reconstruction of the primary depositional environment of the Palmer Deposit (via unfolding and restoration of post-mineralization fault offset) yields a single continuous mineralized system that is over 1.5 km in length. The known zones are open to expansion in multiple directions, and most notably, the thickest mineralized intersection is located at the lower limit of the South Wall drilling done to date.

Figure 1-1). Numerous other mineralized prospects are also present throughout the property and share similar alteration and mineralogical characteristics to the known zones, suggesting a large-scale, property-wide Late Triassic mineralizing event.

At the South Wall and RW Zones, six mineralization styles have been identified and are grouped according to dominant mineral assemblages and texture and include: Barite mineralization (Zn-rich), Massive Pyrite Mineralization (Cu-rich), Semi-massive and Stringer-style Mineralization, Massive Pyrrhotite Mineralization, Carbonate Mineralization and Barite-Carbonate Mineralization. At the AG Zone deposit, mineralization consists of Massive and Semi-massive sulfide and barite, and feeder-style stringers and replacement. Four alteration facies are associated with the known mineralized zones and include: Quartz-Pyrite, Muscovite, Carbonate-Chlorite and Epidote.

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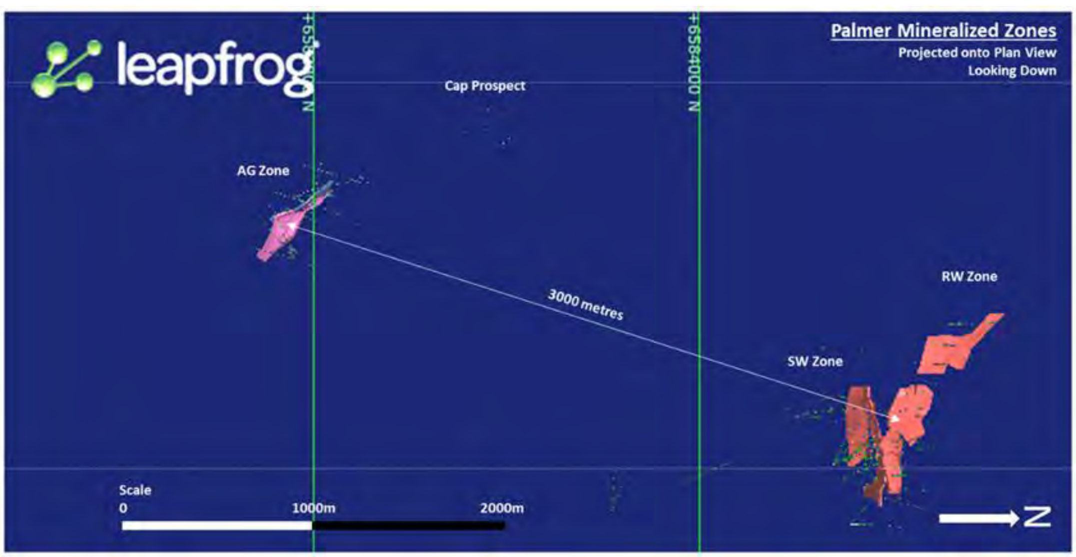
Figure 1-1: 3D Leapfrog Plan View of RW/SW Zones and AG Zones



Source: Constantine Metal Resources Ltd. (2018)

1.4 Metallurgical Testing and Mineral Processing

Three test programs were completed by SGS Mineral Services (SGS) in 2013, 2018 and 2018 (AG Zone mineralogy).

The most recent test program to evaluate the Palmer deposit was completed by SGS Canada Inc. in 2018. The program included mineralogy, comminution, and rougher/cleaner Cu, Zn, Py and Ba sequential flotation. One composite, the High Ba composite, representing the Palmer Deposit was tested to confirm a preliminary recovery flowsheet and associated flotation conditions for Cu and Zn using the criteria from the SGS 2013 program. The tailings generated from the Cu/Zn flotation were used to develop the Ba flowsheet and flotation parameters. The optimized conditions from the program were applied to locked cycle tests for all three saleable concentrates.

QEMSCAN analysis of the High Ba composite, representing the Palmer deposit and prepared at a grind size of 80% passing (P_{80}) of 70 µm, shows that the sulfide material content is mainly barite, pyrite, sphalerite and chalcopyrite with the remaining material mostly gangue. Mineralogy indicates chalcopyrite and sphalerite liberation ranges from 79% to 90%. Barium is predominately associated with barite (99.1%) and 95.4% liberated. A second sample representing the AG Zone deposit was analyzed using QEMSCAN by SGS. The results indicate the sample has similar mineral content and liberation to the Palmer deposit except for higher galena and very low chalcopyrite (0.01%) content. Copper in the AG Zone is mainly associated with tennantite-tetrahedrite. Liberation was similar for both samples.

Comminution test work found that the A x b was 89.6, the bond ball mill work index (BWi) performed at a grind of 106 µm produced results of 6.3 kWh/t and 6.9 kWh/t and the abrasion index (Ai) was 0.119 g. The Palmer Deposit can be classified as soft to very soft and mildly abrasive.

Based on the results from SGS (2018), saleable Cu, Zn and Ba concentrates can be produced at a primary grind size of 80% passing (P_{80}) 72 µm, and rougher concentrate regrind sizes of P_{80} 35 µm for Cu and 50 µm for Zn. For the High Ba composite, locked cycle flotation test results achieved recoveries of 88.9% Cu, 93.1% Zn and 91.1% Ba at concentrate grades of 24.5% Cu, 61.3% Zn and 52.4% Ba. The calculated head grades for the High Ba samples were 1.66% Cu, 10.3% Zn and 29.2% Ba. A summary of the SGS-14063-002 locked cycle test (LCT) results is shown in Table 1-1.

Table 1-1: Locked Cycle Testing Results BL0148-LCT21

Product	Weight (%)	Assay (% or g/t)					Distribution (%)				
		Cu	Zn	Au	Ag	Ba	Cu	Zn	Au	Ag	Ba
Cu Con	6.0	**24.5**	8.21	3.17	521	**-**	**88.9**	4.8	49.5	70.8	2.2
Zn Con	15.7	0.67	**61.3**	0.49	56.7	-	6.3	**93.1**	20.1	20.1	6.0
Ba Con	50.9	-	-	-	-	**52.3**	-	**-**	-	-	**91.1**

*Note: Cu and Zn results are a weighted average from Locked Cycle Tests 14063-002 LCT1-F10-8 cycles D&E and Ba from Locked Cycle Test 14063-002 LCT1-Ba-CF8 cycles E&F
Source: SGS (2018)

Test work to provide flotation results have not been completed on the AG Zone. The results from the Palmer deposit and mineralogical analysis of the AG Zone deposit were used to predict the estimated Cu, Zn and Ba concentrate grades and recoveries for the economic model.

1.5 History, Exploration and Drilling

Base-metal sulfides and barite were first discovered in the Glacier Creek prospect area in 1969 by local prospector Merrill Palmer. Exploration work by historic operators from 1969-1999 at Palmer included a variety of property-wide geological, geochemical, and geophysical surveys and diamond drilling. Total drilling by all historical operators was 7,545 m in 37 holes.

Constantine was formed out of Rubicon Minerals Corporation in 2006 with the primary purpose of exploring the Palmer Exploration Project. Constantine has completed a variety of exploration surveys and approximately 60,200 m of drilling in 156 holes to the end of 2018. This work has led to the discovery of massive sulfide deposits at the Palmer deposit (including the South Wall and RW Zones) in the Glacier Creek Prospect area, and the AG Zone deposit at the Nunatak Prospect Area. Total cumulative diamond drilling by the Company since 2006 is 60,200 m in 156 drill holes with total cumulative diamond drilling by all operators at 67,745 m in 193 completed holes.

1.6 Mineral Resource Estimates

1.6.1 Palmer Deposit Resource Estimate

Following completion of the 2017 drilling campaign, an independent mineral resource estimate for the RW and SW Zones in the Palmer main area (Palmer deposit) was prepared by James N. Gray, P. Geo., of Advantage Geoservices Ltd. in accordance with Canadian Securities Administrators' NI 43-101 and conforms to the Canadian Institute of Mining "Estimation of Mineral Resources and Mineral Reserves Best Practices" guidelines. The resource incorporates all exploration drilling in the Palmer deposit area completed to the end of 2017. One hundred and eight exploration (108) diamond drill holes for 44,900 m and geological surface mapping were used to generate the geological and structural model for the South Wall and RW zones. Sixty (60) of the holes intersect the interpreted mineralized solids. Outlier assays were capped and all assays within the mineralized zones composited to 1.5-m lengths. Metal grades were estimated using inverse distance cubed interpolation into a 3D block model with block dimensions of 6 x 6 x 6 m. Density was estimated by inverse distance squared interpolation, with unique density values determined by conventional analytical methods for virtually all assay samples. Three dimensional geologic solids were constructed by Darwin Green, Vice President of Exploration and reviewed by Ian Cunningham-Dunlop, P.Eng., Vice President, Advanced Projects, and, in general, were limited to material grading > 0.5% Cu or > 2% Zn that could be demonstrated to be correlative with definable stratabound zones. As a general rule, solids were extended no more than 50 m up-dip, down-dip and along strike from a drill hole except where geology supports extension in the plunge direction of mineralization. A total of four solids were constructed for sulfide mineralization: South Wall Zone 1, South Wall Zone 2-3-EM, RW West, and RW East.

Indicated Resources include only a portion of the upper part of the South Wall Zone, where drill density and confidence in the geological model are highest. Indicated Mineral Resource blocks meet the criteria of being a minimum 25-m distance away from the outer edge of the mineralized geological solid, estimated by a minimum of three holes, and have an average distance to three holes of less than or equal to 50 m; remaining estimated blocks are classified as Inferred Mineral Resource.

The Indicated and Inferred Mineral Resource for the RW and South Wall Zones is tabulated below in Table 1-2 for a range of NSR (Net Smelter Return) cut-off values. Based on assumed underground mining and milling costs, the resource utilizes a base case cut-off of $75 per tonne. The resource has an effective date of September 27th, 2018 based on a data cut-off of May 1st, 2018.

Table 1-2: 2018 Palmer Deposit Mineral Resource Estimate at a $75/t NSR Cut-off

INDICATED AND INFERRED MINERAL RESOURCE ESTIMATE (effective date September 27, 2018)								
Category	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$ %)	ZnEq* (%)	CuEq* (%)
Indicated	4,677	1.49	5.23	30.8	0.30	23.9	11.67	3.84
Inferred	5,338	0.96	5.20	29.2	0.28	22.0	9.90	3.26
Contained Metal								
Category		Cu (M lbs)	Zn (M lbs)	Ag (M oz)	Au (K oz)	Barite (K tonnes)	ZnEq (M lbs)	CuEq (M lbs)
Indicated		154	539	4.6	45.1	1,118	1,203	396
Inferred		113	612	5.0	48.1	1,174	1,166	383

Notes

1. The cut-off date for drill data included in the resource is May 1st, 2018.
2. Net Smelter Return (NSR) equals (US$16.01 x Zn% + US$48.67 x Cu% + US$23.45 x Au g/t + US$0.32 x Ag g/t). NSR formula is based on estimated metallurgical recoveries, assumed metal prices, and assumed offsite costs that include transportation of concentrate, smelter treatment charges, and refining charges.
3. Assumed metal prices are US$1.15/lb for zinc (Zn), US$3.00/lb for copper (Cu), US$1250/oz for gold (Au), US$16/oz for silver (Ag).
4. Estimated metal recoveries are 93.1% for zinc, 89.6% for copper, 90.9% for silver (70.8% to the Cu concentrate and 20.1% to the Zn concentrate) and 69.6% for gold (49.5% to the Cu concentrate and 20.1% to the Zn concentrate) as determined from metallurgical locked cycle flotation tests completed in 2018.
5. Barite is not included in the NSR value.
6. Zinc equivalent (ZnEq%) and Copper equivalent (CuEq%) values calculated based on the NSR formula above plus an assumed net-value for barite as described below (e.g. CuEq = (total NSR value + $BaSO_4$ net-value)/US$48.67.
7. $BaSO_4$ net-value equals US$0.566 x $BaSO_4$% (e.g. a resource grade of 24% $BaSO_4$ x $0.566 = US$13.6 per tonne or 0.85% ZnEq). Formula based on barite recovery of 91.1% from metallurgical tests, assumed wholesale drilling-grade barite price in nearest North American markets of US$227/metric tonne, and assumed all-in transportation cost of US$150/tonne.
8. Mineral resources as reported are undiluted.
9. Mineral resource tonnages have been rounded to reflect the precision of the estimate.

Source: Constantine (2019)

Palmer Deposit mineral resource highlights include:

- Indicated Resource of 4,677,000 tonnes grading 11.67% zinc equivalent (3.84% CuEq). This represents the first Indicated Resource for Palmer, and accounts for 47% of the total resource.
- Inferred Resource of 5,338,000 tonnes grading 9.90% zinc equivalent (3.26% CuEq). This includes the addition of new areas of Inferred resource totaling 1.89 million tonnes, for a total tonnage increase of 23%*.
- First resource to report barite mineralization for the Palmer deposit, highlighting the opportunity for barite to contribute value as an industrial mineral co-product.

**Previous resource estimate of 8.125 million tonne Inferred grading 1.41% copper, 5.25% zinc, 0.32 g/t gold and 31.7 g/t silver (Gray and Cunningham-Dunlop, 2015). 2015 resource estimate utilizes an NSR cut-off of US$75/t with assumed metal prices of US$1200/oz for gold, US$18/oz* *for silver, US$2.75/lb for copper, and US$1.00/lb for zinc, and estimated metal recoveries determined from metallurgical locked cycle flotation tests.*

1.6.2 AG Zone Deposit Resource Estimate

Following completion of the 2018 summer drilling campaign, an independent mineral resource estimate for the AG Zone deposit was prepared by James N. Gray, P. Geo., of Advantage Geoservices Ltd. in accordance with Canadian Securities Administrators' NI 43-101 and conforms to the Canadian Institute of Mining "Estimation of Mineral Resources and Mineral Reserves Best Practices" guidelines. The AG Zone deposit mineral resource incorporates all exploration drilling in the AG Zone deposit area completed since initial discovery in 2017. Twenty-nine (29) exploration diamond drill holes for 10,766 metres and geological surface mapping were used to generate the geological and structural model for the AG Zone deposit. Twenty (20) holes intersected the interpreted mineralized solids. Outlier assays were capped and all assays within the mineralized zones were composited to 1.5-metre lengths. Metal grades were estimated using inverse distance cubed interpolation into a three-dimensional (3D) block model with block dimensions of 6 x 6 x 6 metres, which is consistent with the main Palmer deposit. Density was estimated by inverse distance squared interpolation with unique density values determined by conventional analytical methods for all assay samples. Three dimensional geologic solids were constructed by Darwin Green, Vice President of Exploration, and reviewed by Ian Cunningham-Dunlop, P.Eng., Vice President, Advanced Projects, and, in general, were limited to material grading > 2% Zn or > 60 grams per tonne (g/t) Ag that could be correlated with definable stratabound zones. As a general rule, solids were extended no more than 50 metres up-dip, down-dip and along strike from a drill hole except where geology supported extension between holes in the trend of mineralization. Two (2) solids for sulfide mineralization were included in the Inferred Mineral Resource: the AG main lens, and the AG footwall zinc zone.

The Inferred Mineral Resource for the AG Zone deposit is tabulated below in Table 1-3 for a range of % ZnEq cut-off values. Based on assumed underground mining and milling costs, the resource utilizes a base case cut-off of 5.0% ZnEq. The AG Zone deposit mineral resource has an effective date of December 18th, 2018 based on a data cut-off of November 15th, 2018.

Table 1-3: AG Zone Deposit Only: Sensitivity by Cut-off Grade

INFERRED MINERAL RESOURCE ESTIMATE (effective date December 18, 2018)								
Cut-off Grade	**Tonnes**	**Zn**	**Cu**	**Pb**	**Ag**	**Au**	**Barite**	**ZnEq**
(% ZnEq)	**(1,000s)**	**(%)**	**(%)**	**(%)**	**(g/t)**	**(g/t)**	**($BaSO_4$%)**	**(%)**
4.5	4,648	4.48	0.12	0.90	114.2	0.50	34.1	8.68
5.0	**4,256**	**4.64**	**0.12**	**0.96**	**119.5**	**0.53**	**34.8**	**9.04**
5.5	3,975	4.78	0.13	1.00	122.2	0.54	34.7	9.31
Contained Metal								
Cut-off Grade		**Zn**	**Cu**	**Pb**	**Ag**	**Au**	**Barite**	**ZnEq**
(% ZnEq)		**(M lb)**	**(M lb)**	**(M lb)**	**(M oz)**	**(K oz)**	**(K tonnes)**	**(M lbs)**
4.5		459	12	92	17.1	74.7	1,583	889
5.0		**435**	**11**	**90**	**16.4**	**72.5**	**1,480**	**848**
5.5		419	11	88	15.6	69.0	1,379	816

Notes

1. Includes all drill holes completed at AG Zone; drilling completed between June 2017 and September 2018.
2. Zinc Equivalent (ZnEq) based on assumed metal prices and 100% recovery and payable for Cu, Zn, Pb, Ag and Au.
3. ZnEq equals = ($66 x Cu% + $25.3 x Zn% + $22 x Pb% + $0.51 x Ag g/t + $40.19 x Au g/t) / 25.3.
4. Assumed metal prices are US$3.00/lb for copper (Cu), US$1.15/lb for zinc (Zn), US$ $1.00/lb for lead, US$1250/oz for gold (Au), US$16/oz for silver (Ag).
5. Barite ($BaSO_4$) not included in the Cut-off determination or reported ZnEq.
6. Mineral resources as reported are undiluted.
7. Mineral resource tonnages have been rounded to reflect the precision of the estimate.

Source: Constantine (2019)

1.6.3 Total Palmer Project Mineral Resources

The Palmer Project Indicated and Inferred Mineral Resource includes the RW, South Wall and AG Zones as presented in Table 1-4. Copper equivalent grade is also included in the table due to the importance of copper in the SW Zone.

Table 1-4: 2019 Palmer Project Mineral Resource Statement

INDICATED AND INFERRED MINERAL RESOURCE ESTIMATE (effective date December 18, 2018)

Zone	Cut-off	Resource Category	Tonnes (1,000s)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$%)	ZnEq (%)	CuEq (%)
RW and	$75/t NSR	Indicated	4,677	5.23	1.49	-	30.8	0.30	23.9	10.21	3.92
South Wall	$75/t NSR	Inferred	5,338	5.20	0.96	-	29.2	0.28	22.0	8.74	3.35
AG Zone	5.0% ZnEq	Inferred	4,256	4.64	0.12	0.96	119.5	0.53	34.8	9.04	3.46
Total:		Indicated	4,677	5.23	1.49	-	30.8	0.30	23.9	10.21	3.92
		Inferred	9,594	4.95	0.59	0.43	69.3	0.39	27.7	8.87	3.40

CONTAINED METAL

	Resource Category	Zn (M lb)	Cu (M lb)	Pb (M lb)	Ag (M oz)	Au (K oz)	Barite (K tonnes)	ZnEq (M lbs)	CuEq (M lbs)
Total:	Indicated	539	154	-	4.6	45.1	1,116	1,053	404
	Inferred	1,047	124	90	21.4	120.6	2,654	1,876	719

Notes:

1. Zinc Equivalent (ZnEq) and Copper Equivalent (CuEq) based on assumed metal prices and 100% recovery and payable for Cu, Zn, Pb, Ag and Au.
2. CuEq = (25.3 x Zn% + 66 x Cu% + 22 x Pb% + 0.51 x Ag g/t + 40.19 x Au g/t)/66.
3. ZnEq equals = ($66 x Cu% + $25.3 x Zn% + $22 x Pb% + $0.51 x Ag g/t + $40.19 x Au g/t) / 25.3.
4. Assumed metal prices are US$3.00/lb for copper (Cu), US$1.15/lb for zinc (Zn), US$ $1.00/lb for lead

Source:

Total Palmer Project mineral resource highlights include:

- Indicated Resource of 4,677,000 tonnes grading 10.21% zinc equivalent (3.92% copper equivalent).
- Inferred Resource of 9,594,000 tonnes grading 8.87% zinc equivalent (3.4% copper equivalent). This includes the addition of new areas of Inferred Resource totaling 4.256 million tonnes, for a total tonnage increase of 80%*.
- SW, RW and AG Zones confirm the multi-deposit district potential of the Palmer Project.
- Opportunity to discover additional deposits and expand the two known key resource areas is considered excellent.

  **Previous Inferred mineral resource estimate of 5.338 million tonnes grading 0.96% copper, 5.20% zinc, 0.28 g/t gold and 29.2 g/t silver (Gray and Cunningham-Dunlop, 2018). The 2018 Inferred mineral resource estimate utilizes an NSR cut-off of US$75/t with assumed metal prices of US$1250/oz for gold, US$16/oz for silver, US$3.00/lb for copper, and US$1.15/lb for zinc, and estimated metal recoveries determined from metallurgical locked cycle flotation tests.*

1.7 Mining Methods

The Palmer Project consists of two deposits, the Palmer and the AG Zone, and is proposed to be mined with transverse and longitudinal longhole (LH) stoping.

The mine will be accessed with an exploration drift, driven at 5 m wide (W) x 5 m high (H) from the 680 Exploration Portal, which is located at 680 m above sea level (masl). This drift will act as a haulage route from the AG Zone deposit to the underground crusher located near the Palmer deposit. A second portal, the 510 Conveyor Drift will transport material from the underground crusher to the process plant. The 510 Conveyor Drift is 6 m W x 5 m H and will act as secondary egress for the Palmer deposit. The third portal required is the 1000 portal and will act has the secondary egress for the AG Zone deposit. All accesses have been sized to accommodate the necessary ventilation ducting and services.

Vertical development will include a raise to transport mineralized material and ventilation raises in both the Palmer and AG Zone deposits.

Stope sub-levels are 5 m W x 5 m H and spaced at 20 m vertical increments. Transverse stopes will be 16 m W x 24 m long (L), while longitudinal stopes will be 12 m W x 18 m L. This produces a typical hanging wall (HW) and footwall (FW) exposure of 16 m W x 20 m H and 18 m W x 20 m H for transverse and longitudinal stopes respectively.

Stopes have been categorized as either primary or secondary. Primary stopes will be mined first and the void will be filled with cemented paste backfill before the adjacent secondary stope is mined.

The mine plan is shown by resource classification in Table 1-5. This does not constitute a mining reserve, as the table contains inferred resources.

Table 1-5: Palmer Mine Plan by Resource Classification (Including Mine Dilution)

Palmer Deposit	Tonnes (kt)	Cu (%)	Zn (%)	Ba (%)	Au (g/t)	Ag (g/t)
Indicated	4,798	1.33	5.02	12.42	0.27	27.25
Inferred	3,553	0.91	4.05	10.72	0.25	20.71
AG Zone Deposit	**Tonnes (kt)**	**Cu (%)**	**Zn (%)**	**Ba (%)**	**Au (g/t)**	**Ag (g/t)**
Indicated	-	-	-	-	-	-
Inferred	4,130	0.11	4.03	16.77	0.46	101.05
Total Mine Plan	**12,481**	**0.81**	**4.41**	**13.38**	**0.33**	**49.81**

Notes:

1. Mine Plan Mineral Resources are estimated at a cut-off of US$80.10 US$/t net smelter return (NSR) for both the Palmer Deposit and AG Zone Deposit.
2. Metal prices used for the NSR cut-off were: Copper US$ 3.00/lb; Zinc US$ 1.15/lb; Gold US$ 1,250/oz; Silver US$ 16/oz
3. Totals may not add up correctly due to rounding.

Source: JDS (2019)

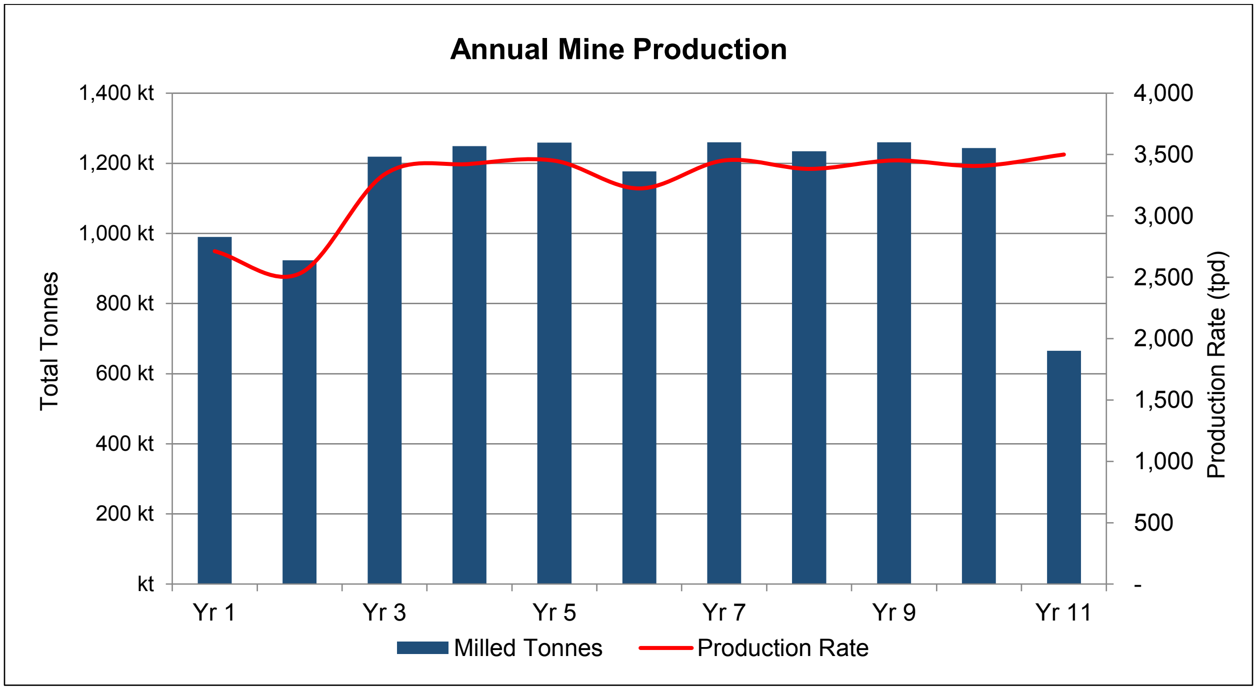
Diesel trackless equipment will be used throughout the mine. The total haulage fleet will consist of seven (7) 30 tonne (t) haul trucks, four (4) 10 t load haul dump machines (LHD), and one (1) 4.5 t LHD. The four 10 t LHDs will have remote capability.

The mine will require a full-time work force of mining, maintenance, services, technical and administrative personnel. Mine operations will run 365 d/a at 22 h/d through two – 11-hour shifts, allowing one hour for smoke clearing at each shift change.

Surface facilities will be located at the 680 Exploration Portal and/or at the mill site. Underground facilities will include a shop and booster pump stations for the paste backfill.

Mine production is expected to commence in year 1 at approximately 2,700 tonnes per day (t/d), with full production in years 3 to 11 at an average rate of approximately 3,400 t/d. The mine will be in production for 11 years, with production ending in Q2 of Year 11. A summary of annual mine production is presented in Figure 1-2.

Figure 1-2: Annual Mine Production



Source: JDS (2019)

1.8 Recovery Methods

Material from the mine will feed a single-stage crushing plant located underground. The crushed product will be conveyed from underground to feed the process plant at a rate of 3,500 t/d producing saleable Cu, Zn and Ba concentrates. The process plant will operate 24 h/d, 365 days per year, with an estimated availability of 92%.

The primary grinding circuit will consist of a semi-autogenous grinding (SAG) mill operating in open circuit followed by a ball mill operating in reverse closed circuit with cyclones to achieve a target grind size of 80% passing (P_{80}) 72 µm. The material will then be fed to sequential Cu and Zn rougher / cleaner flotation circuits. The Cu and Zn regrind circuits will further liberate the rougher concentrates, with target P_{80} grind sizes of 35 µm and 50 µm, respectively. The tailings from the Zn rougher and first cleaner flotation will feed the pyrite (Py) rougher flotation followed by Ba rougher / cleaner flotation circuits.

The Cu, and Zn flotation circuits will consist of rougher flotation followed by rougher concentrate regrind and three stages of cleaning. The Ba circuit will include rougher flotation and three stages of cleaning with no regrind. The final concentrates will be thickened then filtered to the targeted moisture content of 8% for Cu and Zn and loaded into trucks as a bulk concentrate for transport to a local port and sent to smelters. Ba will be further dried to 1% moisture and bagged before being transported to Haines for barging to railhead at Prince Rupert, BC. The Py rougher concentrate will be filtered and mixed as paste for deposition underground. The tailings from the process will be dewatered and either filtered for stacking in the filtered tailings facility or mixed as part of the paste and pumped underground.

1.9 Project Infrastructure

The Project infrastructure is designed to support the operation of a 3,500 t/d mine and processing plant, operating on a 24 hour per day, seven day per week basis. The Project envisions the upgrading or construction of the following key infrastructure items:

- Crushed mineralized rock bin, and mill feed conveyor
- Process plant and re-agent storage warehouse
- Liquid natural gas (LNG) fuel power generating plant and LNG storage facility
- On-site power distribution with overhead power lines
- Tailings filtering and paste backfill plant
- Tailings management facility / waste rock storage facility (TMF/WRSF)
- Temporary mine rock stockpile (TMRS)
- Water treatment plant (WTP)
- Administration and mine dry building
- Warehouse
- 120,000 L of on-site fuel storage and distribution
- Industrial waste management facilities such as the incinerator, and
- Site water management facilities.

1.9.1 Tailings and Waste Rock Management

The majority (78%) of tailings will be utilized as underground backfill. The pyrite tailings separated by flotation and potentially acid generating (PAG) waste rock will be placed underground as components of backfill.

A filtered tailings management facility / waste rock storage facility (TMF/WRSF) has been designed at a site approximately 6 km from the Conveyor Portal to store the remaining portion of tailings and non-potentially acid generating (NPAG) waste rock not going underground. Tailings, and the portion of NPAG rock required for the construction of the TMF/WRSF, will be hauled and placed by truck.

Due to desulfurization and removal of deleterious minerals by flotation, tailings stored on surface are NPAG. The TMF/WRSF is therefore not expected to require water treatment.

1.9.2 Water Management

Drainage from the Exploration Portal constructed within NPAG rock is not expected to require water treatment and will be directed to a settling pond prior to discharge to a land application disposal (LAD) system. If treatment is required, flows can be directed to the mill site Water Treatment Pond (MWTP) by gravity pipeline and then to the Water Treatment Plant (WTP). Drainage from the Conveyor Portal and mill site will be conveyed to the MWTP and directed to the WTP for treatment. Treated water will be discharged to a LAD.

Runoff from the lined TMF/WRSF will be collected and directed to a Contact Water Collection Pond (CWCP) and discharged to a LAD system if water quality is suitable or directed to treatment if required.

1.9.3 Mill Foundations

For initial estimation of potential costs for mill foundations, it was determined that the mill building may require a concrete raft foundation and compacted structural fill. Although a final design for mill foundation may also include piers and a locally thicker slab under heavy equipment and thinner sections in non-load bearing regions, for PEA costing an average thickness of concrete of 1 m is assumed with 100 kg of steel reinforcing per m^3.

1.10 Environment and Permitting

The Company has carried out ongoing environmental baselines studies to support permitting, exploration and engineering activities. Such studies include hydrology, hydrogeology; acid rock drainage potential, vegetation and wildlife, cultural resources, environmental liabilities, and annual environmental monitoring.

Constantine is currently exploring the Project under an approved Federal Mine Plan of Operations and Environmental Assessment (DOI BLM-AK-A020-2016-006-EA) granted on August 23rd, 2016, as amended under the Constantine Mine Plan 2017 Modification and Environmental Assessment on September 21st, 2017 (DOI-BLM-AK-010-2017-025-EA). The Company also holds various permits and licenses from the State of Alaska including: Plan of Operations for Surface Exploration (Uplands Lease 9100759), Plan of Operations for Surface Construction (Uplands Lease 9100759), Multi-Year (2019-2023) Land Use Permit for Hardrock Exploration and Reclamation and three Temporary Water Use Authorizations for supplying water to drills. Constantine elected to utilize the Statewide Bond Pool and is currently bonded for 40.0 acres of disturbance.

Constantine has conducted community relations activities since 2006. As part of their ongoing efforts, the Company conducts regular stakeholder meetings, maintains community outreach materials, hosts project site tours, attends and supports local programs and events, supports local hire and procurement, and participates in local community organizations.

1.11 Capital and Operating Cost Estimates

1.11.1 Capital Costs

The initial, sustaining and closure capital costs with no escalation are shown in Table 1-6. All capital costs are in 2019 US Dollars and are considered Class 4 estimates (-20%/30%).

Table 1-6: Capital Cost Summary

Area	Pre-production ($M)	Sustaining ($M)	Closure ($M)	Total ($M)
Mining	55	108	-	163
Site Development	12	1	-	13
Mineral Processing	75	3	-	78
Tailings Management	2	3	-	5
On-Site Infrastructure	34	1	-	35
Off-Site Infrastructure	-	-	-	-
Project Indirects	26	-	-	26
EPCM	32	-	-	32
Owner Costs	8	-	-	8
Closure	-	-	31	31
Salvage Value	-	-	-6	-6
Subtotal	245	115	25	385
Contingency	33	-	-	33
Total CAPEX	**278**	**115**	**25**	**418**

Source: JDS (2019)

The capital cost estimate was compiled using a combination of quotations, database costs, and scaling factors.

The capital cost estimate includes the costs required to develop, sustain and close the property for a planned 11 Year mine life. Initial capital costs are expensed over a 24-month pre-production construction and commissioning period. The sustaining capital is carried over operating Years 1 through 11, and closure costs are incurred in Year 12.

1.11.2 Operating Costs

The operating cost estimates are based on a combination of experiential judgment, reference to similar operating 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.

The LOM costs are summarized in Table 1-7.

Table 1-7: LOM Total Operating Cost Estimates

Description	Total Estimate ($M)	Average Unit Cost ($/t)
UG Mining	362.7	29.06
Processing	210.0	16.83
G&A	103.3	8.28
Total Operating Costs	**676.0**	**54.17**

Source: JDS (2019)

In addition to the OPEX, the sustaining cost averages $11.23/tonne over the LOM. The total operating cost-plus sustaining cost totals $65.40/tonne over the LOM.

1.12 Economic Analysis

An 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 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.

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.

1.12.1 Main Assumptions

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 April 15th, 2019. These have been compared to the spot prices at the close of London Metal Exchange on April 15th, 2019. The Barite price used in this PEA study was selected based on an average price of competitive current wholesale prices of Barite Concentrate.

Table 1-8 outlines the metal prices and exchange rate used in the economic analysis.

Table 1-8: Metal Price and Exchange Rates Used in Economic Analysis

Parameter	Unit	Base Price Value	Spot Price Value
Copper Price	US$/lb	2.82	2.93
Zinc Price	US$/lb	1.22	1.36
Silver Price	US$/oz	16.26	15.00
Gold Price	US$/oz	1,296	1,289
Barite Price	US$/tonne	220	220
Exchange Rate	US$:C$	0.75	0.75

Source: JDS (2019)

No market studies have been completed for the project at this time. Except for barite, all commodities considered in this Study are regularly sold commercially on vast international markets. The terms selected are in-line with current market conditions.

This PEA has assumed that all zinc and copper concentrates are transported and smelted in Asia. CMR has an agreement with Dowa that includes the right of first refusal on the zinc concentrate.

The barite concentrate will be dried and bagged on site and trucked to Haines, where they will be loaded onto a barge to Prince Rupert. It will then be distributed to markets across North America and Canada via rail.

Table 1-9 outlines the recoveries, payable terms, treatment charges and transportation costs used in the economic analysis.

Table 1-9: Concentrate Terms

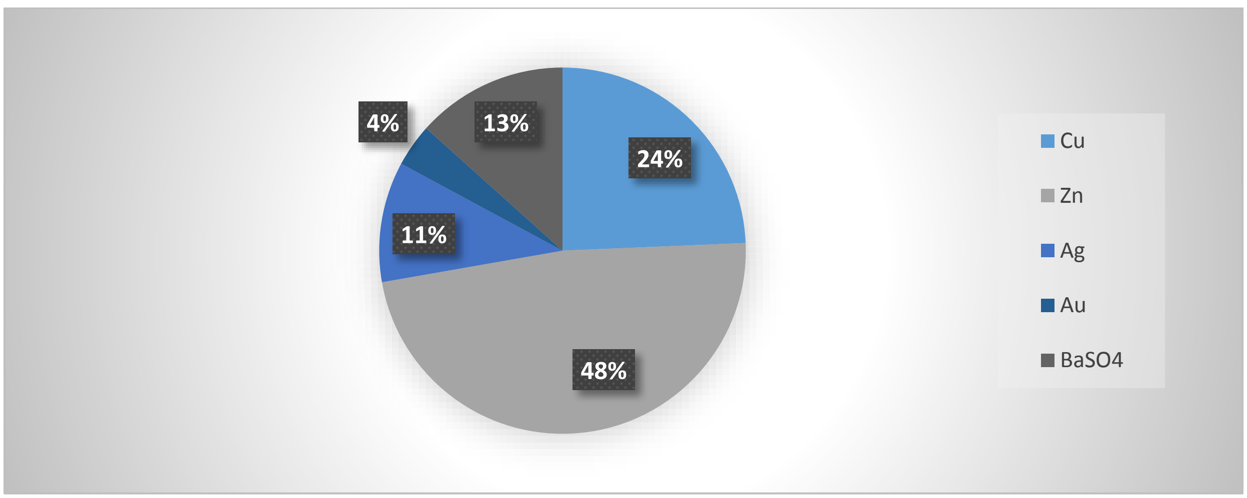
Assumptions and Inputs	Unit	Value
Copper Concentrate		
Metal Recovery to Concentrate	% Cu	88.5%
	% Zn	4.8%
	% Ag	70.8%
	% Au	49.5%
Cu Concentrate Grade Produced	% Cu	25%
Moisture Content	%	8%
Minimum Deduction	%Cu/tonne	1%
	g/t Ag	31.10
	g/t Au	1.24
Metal Payable	% Cu	97%
	% Ag	90%
	% Au	90%
Cu Treatment Charge	US$/dmt con	86
Cu Refining Charge	US$/lb	0.086
Ag Refining Charge	US$/oz	0.75
Au Refining Charge	US$/oz	6
Zn Penalties per % over 4%	US$/tonne	2
Cu Concentrate Transport Cost	US$/wmt con	91
Zinc Concentrate		
Metal Recovery to Concentrate	% Zn	91.5%
	% Cu	6.3%
	% Ag	20.1%
	% Au	20.1%
Zn Concentrate Grade Produced	% Zn	61%
Moisture Content	%	8%
Minimum Deduction	% Zn/tonne	8%
	g/t Ag	93.3
	g/t Au	0.311
Metal Payable	% Zn	85%
	% Ag	70%
	% Au	70%
Zn Treatment Charge	US$/dmt con	172
Ag Refining Charge	US$/oz	0.75

Assumptions and Inputs	Unit	Value
Au Refining Charge	US$/oz	6
Zn Concentrate Transport Cost	US$/wmt con	91
Barite Concentrate		
Metal Recovery to Concentrate	% Barite	91%
Barium Concentrate Grade	% Ba	52%
Moisture Content	%	1%
Barite Treatment Charge	US$/t con	0
Barite Concentrate Transport Cost	US$/tonne	132

Source: JDS (2019)

The distribution of NSR revenues by commodity is shown in Figure 1-3. As can be seen, Zinc is the most significant commodity, accounting for 48% of NSR revenue.

Figure 1-3: NSR Revenue Distribution by Commodity



Source: JDS (2019)

1.12.2 Results

The economic results for the Project based on the assumptions outlined in Section 1.12.1 are shown in Table 1-10.

Table 1-10: Summary of Results

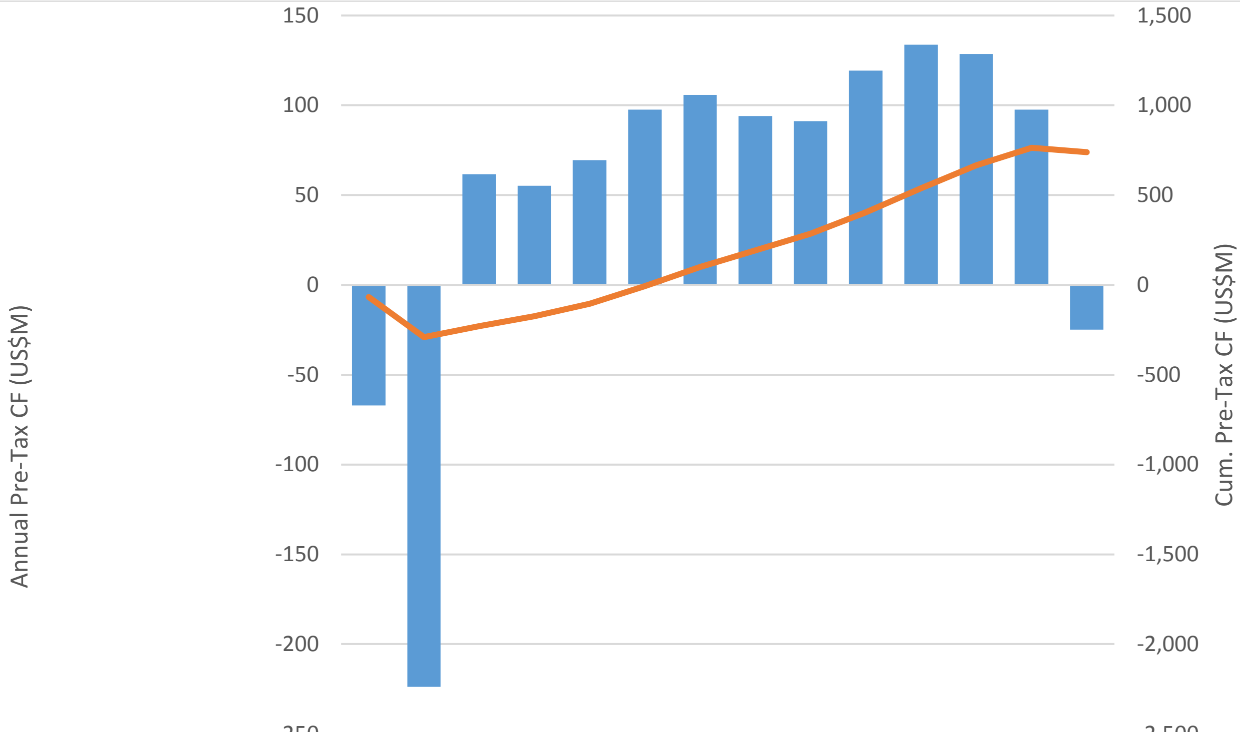
Parameter	Unit	Base Value Price	Spot Value Price
Capital Cost			
Pre-production Capital	US$M	245	245
Pre-production Contingency	US$M	33	33
Total pre-production Capital	US$M	278	278

Parameter	Unit	Base Value Price	Spot Value Price
Sustaining and Closure Capital	US$M	115	115
Total Capital Costs	**US$M**	**418**	**418**
Cash Flows			
Working Capital	US$M	13	13
Pre-Tax Cash Flow	US$M	738	861
	US$/a	69	80
Taxes	US$M	158	187
Post-Tax Cash Flow	US$M	581	675
	US$/a	54	63
Economic Result			
Pre-Tax $NPV_{7\%}$	US$M	354	433
Pre-Tax IRR	%	24%	28%
Pre-Tax Payback	years	3.1	2.6
Post-Tax $NPV_{7\%}$	US$M	266	328
Post-Tax IRR	%	21%	24%
Post-Tax Payback	years	3.3	2.9

Source: JDS (2019)

Figure 1-4 shows the project pre-tax cash flow of the Palmer Project.

Figure 1-4: Annual Pre-Tax Cash Flow



	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Annual Pre-Tax CF US$M	-67	-224	62	55	70	98	106	94	91	119	134	129	98	-25
Cum. Pre-Tax CF US$M	-67	-291	-229	-174	-104	-7	99	193	284	403	537	666	763	738

Source: JDS (2019)

1.12.3 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 -20% to +20%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM.

Table 1-11 and Figure 1-5 show the results of the sensitivity tests.

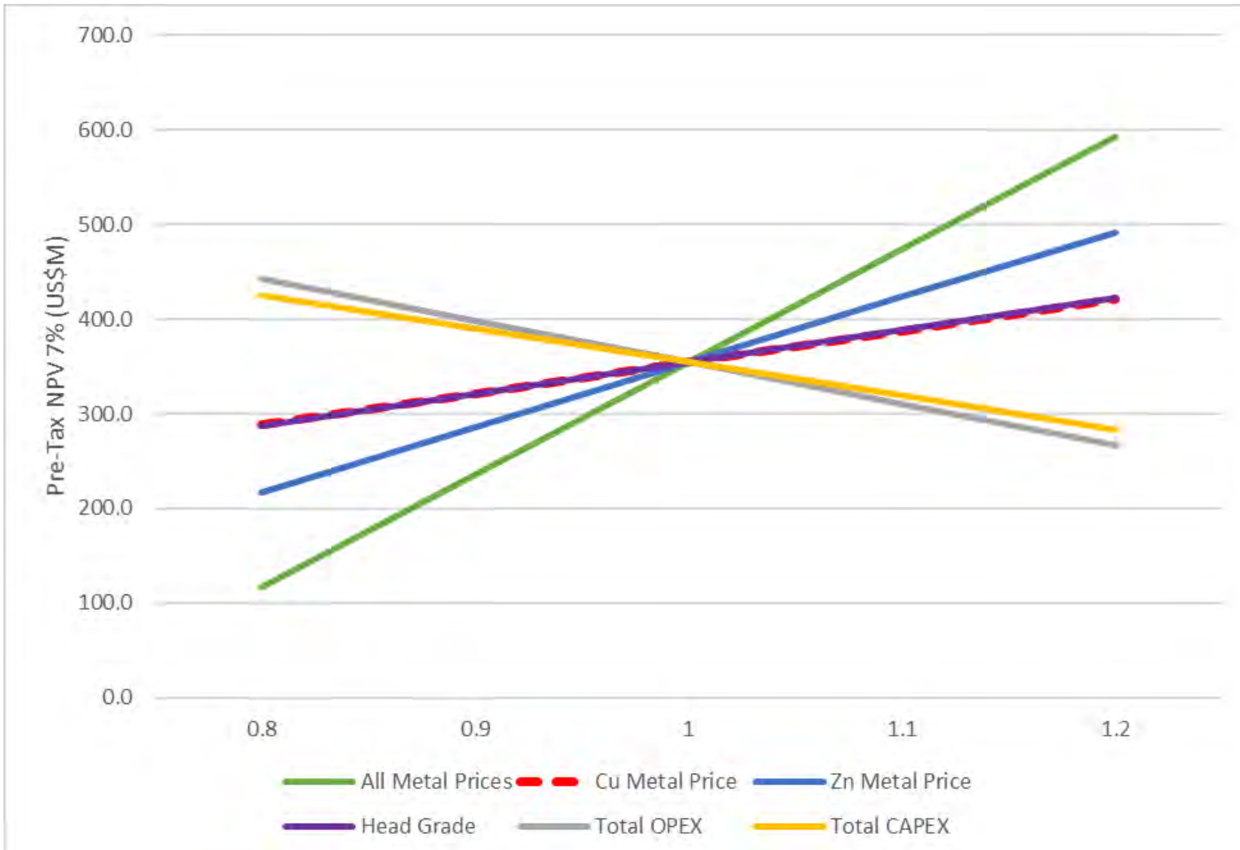
Table 1-11: Sensitivity Results (Pre-Tax NPV_7%)

Parameter	-20%	-10%	Base	+10%	+20%
Zinc Metal Price	217	286	354	423	492
Copper Metal Price	288	321	354	388	421
Metal Prices	117	236	354	473	592
Head Grade	287	321	354	388	423
OPEX	442	398	354	311	267

Parameter	-20%	-10%	Base	+10%	+20%
CAPEX	425	390	354	319	284

Source: JDS (2019)

Figure 1-5: Sensitivity, Pre-Tax NPV @ 7% Discount Rate



Source: JDS (2019)

The analysis revealed that the project is most sensitive to overall metal prices, followed by zinc metal price specifically, operating costs, head grade and capital costs. Of those elements tested, the Project showed the least sensitivity to copper metal price.

1.13 Interpretation and Conclusions

1.13.1 Risks

There are several risks associated with the Project that should be considered. Some are generic and shared by nearly all mining projects, including:

- Sensitivity to metal pricing (as discussed in Section 22.6

- Cost escalation
- Permitting difficulties, costs, and delays, and
- Efficiency of construction management: a project can be properly estimated and improperly constructed, resulting in significant construction cost overruns.

There are also several site-specific risks that are identified and discussed in detail in the following sections.

It should be noted that the project is not sensitive to any exchange rates, as all revenues and nearly all capital and operating cost items were sourced in $US rather than converted from other currencies.

1.13.1.1 Avalanche

Glacier Creek valley is subject to snow avalanches between October and June with the most active periods between November and April, owing to high snowfall and steep terrain. Constantine has been studying the local avalanche cycles since 2010 in order to understand and mitigate avalanche risk. The results of that monitoring program suggest that the access road to the 680 Exploration Portal site is subject to periodic avalanches that could restrict access both during periods of high avalanche danger and during snow clearing operations after avalanches. The monitoring program has also informed design and placement of proposed mine infrastructure.

The mill site, backfill plant and LNG power plant have been located in areas mapped as low hazard zones. Metal shed coverings were assumed for both the 680 Exploration Portal and the 510 Conveyor Portal for the protection of workers and equipment. Secondary egress and/or ventilation adits for both mines were located in low risk rock outcrop areas. 300 Kt of NPAG rock in the development schedule was assumed to be used for construction activities during the pre-production period, including the construction of avalanche deflection berms as required.

An Operational Avalanche Safety Plan will be required during winter operation, which will include site-specific weather and avalanche forecasting, road closures and artificial triggering and cleanup. No operational downtime or special equipment other than the usual snow removal fleet was included for avalanche control measures, as the mines can be accessed from separate portals. It is assumed that mine entry via the 510 Conveyor Portal will be possible during periods of time when the road to the 680 Exploration Portal is blocked from the clean-up activities after intentionally triggered avalanches.

1.13.1.2 510 Conveyor Portal Construction

The 510 Conveyor Portal is determined by the location of the processing plant and the centroid of the orebody. The talus slope it is collared into is less than ideal for the construction of a mine adit and could prove to be more costly and time-consuming than currently considered in the PEA. This was mitigated by using a ground penetrating radar analysis, supported by one visual outcrop, to locate the portal in the minimum depth of talus based on existing information.

1.13.1.3 AG Metallurgical Response

The test work completed on the AG Zone deposit is limited to mineralogy. Recoveries for the deposit were based on a comparison of the AG Zone deposit mineralogy to the Palmer deposit and existing flotation test work performed on Palmer deposit mineralization. The copper and zinc recoveries were discounted for the AG Zone deposit because the copper is lower grade and present as tennantite rather than chalcopyrite.

The actual recovery of copper in the AG Zone deposit will only be confirmed through testwork. This impacts the recovery of copper, but also the loss of zinc to the copper concentrate. This risk is further complicated by the high lead grade in the AG Zone deposit, as compared to the Palmer deposit, which may require a combined lead-copper flotation or separate lead flotation circuit.

1.13.1.4 Site Surface Geotechnical Conditions

No site-specific subsurface information is currently available in the vicinity of the mill site or TMF/WRSF except for a single line of geophysical surveying near the mill site.

Characterization of foundations is a key step during design to identify potential critical conditions. In the PEA design, foundation conditions are largely assumed based on surface observations. Key design considerations for foundation characterization include:

- Potential presence of weak layers that could govern stability of the TMF/WRSF, pond embankments and other structures on site
- Potential presence of soft or weak layers that could result in differential or excess settlement of building foundations
- Potential for foundation strength loss or liquefaction (static or dynamic) which could govern stability of all structures, and
- Hydrogeologic conditions, particularly beneath the TMF/WRSF pile and collection ponds (i.e. influencing uplift of liner during construction).

The processing plant is located on deep till. Rafting foundations have been assumed for the larger equipment and adequate allowances have been included in the design and costs estimate to allow for this requirement.

1.13.1.5 Water Management

The risks associated with water management on site are as follows:

- Basic assumptions were made for surface and underground water flows based on preliminary drilling and hydro-geologic information.
- Flows from the AG Zone workings are not yet defined and have not been addressed in the water balance. The impact of AG Zone inflows on water management facility design and water treatment requirements are not known.
- Little information is available on the hydraulic conductivity of deeper sections of the ore deposits. Consequently, estimates of potential inflows are restricted to approximate assessments of recharge based sustained average flows and little information is available on peak flow rates.
- The underground water collection and treatment strategy includes identification of water requiring treatment and separation from water that does not. Inability to differentiate would increase the amount of water requiring treatment and associated cost.
- Geochemical test work to date indicates runoff from tailings will be relatively benign and operational controls will be in place for geochemical monitoring and management to confirm this. Therefore,

the PEA design includes discharge of contact water collected at the CWCP after settlement to the TMF/WRSF LAD without additional treatment either during operations or upon closure. Additional tailings and waste rock characterization and more detailed operational planning will be required in future design stages to confirm indications of initial test work. Future results may indicate that water treatment at the TMF/WRSF may be necessary.

1.13.1.6 Seismicity

Stability analysis indicates that under maximum design earthquake (MDE) loadings, seismic displacements of the TMF/WRSF are possible. Deformations could impact pile integrity, pile infrastructure (roads, pipelines, etc.) and liner and drainage systems. Other site structures may also be impacted by seismic loading.

1.13.1.7 Geochemical Management (Waste Materials)

Geochemical testing to date on tailings is limited to initial pilot plant samples, and geochemical characterization of all waste rock has not yet been carried out. The limited testing represents uncertainty related to geochemical management. However, preliminary results from tailings testing indicate that the mill circuit is successful at removing the majority of sulfides from the tailings and the material is relatively benign. Pyrite tailings will be stored underground. A portion of PAG waste rock will be temporarily stored at the mill site before being placed underground later in operations.

1.13.1.8 TMF/WRSF Dust Management

Wind-blown tailings could impact and exceed air quality standards if areas of the TMF/WRSF pile are left unmitigated. Although the TMF/WRSF design includes progressive reclamation and waste rock armoring of pile slopes, filtered tailings can be susceptible to dusting if left exposed. Temporary dust management alternatives prior to placement of a reclamation cover include: synthetic dust suppressants, wind fences and temporary sand and gravel erosion protection layers.

1.13.1.9 TMF/WRSF Post-Closure

Long-term closure goals for the TMF/WRSF include providing a stable pile and limiting long-term risk to the downstream environment (safety, water and air quality). The PEA design addresses these goals by eliminating ponded water on the TMF/WRSF pile surfaces, routing water around the pile and progressively covering the pile with a low-permeability cover to limit water and oxygen ingress. Pile drain down water will be greatly reduced upon closure due to the completion of the low-permeability cover. Long-term risks to water quality are not fully defined by short-term geochemical testing and the need for additional long-term measures is uncertain. Differential settlement of tailings has the potential to modify drainage paths resulting in formation of localized ponds.

1.13.2 Opportunities

The Project has several opportunities to improve the current results that should be investigated further as part of the ongoing development of the Project.

1.13.2.1 Expansion and Definition of Resources

The simplest and most obvious opportunity is to continue to explore the property to expand and better define the resources. The current level of drilling is sufficient to define a resource but has certainly not closed off either deposit and there is ample opportunity to either or both. An expansion of mineral resources would allow the mine plan to contemplate either a higher mining rate, longer mine life, or some combination of both. Increased resource definition through tighter drill spacing would allow the resource to form the basis of a reserve for a PFS or FS.

1.13.2.2 Sale of Pyrite

The current study treats pyrite as a waste product that must be returned underground as paste backfill because of its acid-generating potential. Pyrite is a saleable commodity, however, as it is used to generate sulfuric acid. As the pyrite is already ground, floated, and filtered there would be no incremental processing costs associated with making the product saleable. If a contract could be arranged with an adequate margin to ship the product to a smelter, this could be a significant opportunity.

There will be 3.0 Mt of pyritic tails generated over the LOM. This total exceeds the quantity of material (NPAG rock and de-sulfide tails) sent to the TSF. As such, it is possible that pyrite sales could result in the elimination of a TSF for surface waste storage. As such, it may even be beneficial to the Project to sell the pyrite at a loss.

1.13.2.3 AG Zone Deposit Metallurgy

While the limited testwork for the AG Zone deposit has been identified as a risk in Section 25.1.3, it must also be considered an opportunity. AG Zone deposit recoveries were estimated by applying a 10% deduction recovery to the copper (from 89% to 79%) and a 5% to the zinc from the AG zone deposit (from 93% to 88%). Should testwork prove these deductions to be conservative, the economics of the AG Zone deposit will improve. Similarly, the increased presence of lead may impose a complication on the copper recovery, but it may also provide an additional revenue stream from the sale of a lead concentrate or a combined copper-lead concentrate.

1.13.2.4 Good Used Equipment

The purchase and employment of good used equipment is always a consideration to be made to reduce capital costs. In general, this should not be considered for the mobile mining, as the savings do not warrant the reduction of availability, making production targets hard to achieve. However, much of the major equipment list for the processing plant, backfill plant, the surface fleet, and power generation and distribution could be comprised of used units should suitable unit be available at the time of procurement. Realistic assumptions and expectations should be applied to the purchase of used equipment, as there are usually significant costs for rebuilding, retrofitting, or modernizing componentry.

1.13.2.5 Conveyor for AG Zone Deposit

Haulage costs for the AG Zone deposit are significant due to the long haulage distance. A trade-off study should be performed to determine if a second conveyor should be installed to replace the truck haulage. This would require that a second crusher installation be installed for the AG Zone deposit to size rock for the conveyor.

1.13.2.6 Dust

Wind-blown tailings could impact and exceed air quality standards if areas of the TMF/WRSF pile are left unmitigated. Although the Project design includes progressive reclamation and waste rock armoring of pile slopes, filtered tailings can be susceptible to dusting if left exposed. Temporary dust management alternatives prior to placement of a reclamation cover include synthetic dust suppressants, wind fences and temporary sand and gravel erosion protection layers. Pile staging during operations should be optimized to advance waste rock and progressive reclamation as early as possible and minimize areas with exposed tailings.

1.13.2.7 Post Closure

Long-term closure goals for the TMF/WRSF include providing a stable pile and limiting long-term risk to the downstream environment (safety, water and air quality). The PEA design addresses these goals by eliminating ponded water on the TMF/WRSF pile surfaces, routing water around the pile and progressively covering the pile with a low-permeability cover to limit water and oxygen ingress. Pile drain down water will be greatly reduced upon closure due to the completion of the low-permeability cover. Long-term risks to water quality are not fully defined by short-term geochemical testing and the need for additional long-term measures is uncertain. Differential settlement of tailings has the potential to modify drainage paths resulting in formation of localized ponds.

Resiliency must be included in closure design and construction to accommodate tailings consolidation (cover design and material specifications). Additional geochemical characterization of tailings is required to confirm that long term water quality criteria will be met for discharge to the environment.

1.14 Recommendations

1.14.1 General

The primary finding of this PEA is that the Project has the potential to be economic based on the data acquired to-date and the mine model that was produced for this evaluation. Accordingly, it is recommended that exploration continue on the Project and that the Company continue to advance its permits and baseline environmental data collection with the ultimate goal of constructing the mine and putting it into production.

To this end, the Authors believe continued delineation drilling is warranted and has the potential to expand the known resource and increase its confidence. Once this is done, it would be worthwhile to replace this PEA with a pre-feasibility study (PFS) or feasibility study (FS) that includes a mining reserve based on a mine model applied to the expanded resource.

Given the mountainous terrain, this drilling can only be effectively achieved from underground development. There is also significant potential to discover additional mineralized zones within the greater Palmer Project. Accordingly, the planned exploration adit, drift, and underground drill program is recommended.

1.14.2 Recommended Work Plan – Exploration Program

The following activities are recommended as part of the next exploration program, currently anticipated for summer 2019:

- Review the option of a lateral underground exploration adit to provide access to the mineral resource area for delineation drilling, hydrological and geotechnical studies, and metallurgical testing. This may be more cost effective for drilling on close-spaced centers for conversion from Inferred to Indicated mineral resource categories than drilling from surface and would also facilitate year-round drilling, which is currently impractical during the winter months.
- Geotechnical, hydrogeological, engineering, environmental, avalanche risk studies and permitting work to aid in the assessment of a conceptual underground exploration development.
- Prepare and submit a Plan of Operations permit application in support of the conceptual underground exploration program.
- 10,0000 m of resource-scale definition and exploration drilling on 100 m and 50 m nominally spaced centers to test the limits of the known mineralized zones. Priority drill areas would include the on-strike and down-dip extensions of the collective South Wall and RW Zones¸ with emphasis on the potential 200 m down-dropped faulted offset of the Zone II-III-EM.
- Drill test existing regional exploration targets.
- Development of new regional exploration targets within the Federal and State mining claims, and within the greater Mental Health Trust Lands that are under lease.

1.14.3 Recommended Work Plan – Pre-feasibility or Feasibility Study

The logical next step for the project is to produce a pre-feasibility study (PFS) or Feasibility Study (FS). This effort would require the following field programs and engineering evaluations:

- Complete geotechnical characterization program for the PEA underground mine and infrastructure including geotechnical core drilling and oriented core and/or televiewer;
- Investigation of the talus slope to finalize the location of the 510 Conveyor Portal.
- Ongoing metallurgical testwork to confirm the flotation characteristics of the AG Zone deposit through lock-cycle test.
- Ongoing metallurgical testwork on blended samples of Palmer deposit and AG Zone deposit mineralization, matching the production forecast to simulate the predicted scheduled mill feed.
- A new avalanche hazard analysis should be performed now that site facilities have been generally located to identify specific concerns and countermeasures that should be applied to the next level of study. This would include design and location of avalanche deflection berms to protect the site roads, industrial complex, water treatment ponds and the two portals.
- The 510 Portal location should be finalized, and a detailed engineering design should be completed for a snow cover shed and avalanche deflection structure(s).
- Hydrogeological studies of AG Zone to assess flow rates and treatment requirements will be required. Potential measures to reduce inflows during operation such as construction of bulkheads between operational and depleted zones should be investigated.
- Continue to advance understanding of geochemistry and waste management strategies. Include TMF/WRSF water treatment as a contingency in project costing estimates if required. 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.

- It is recommended that a site investigation program to characterize the foundations and potential borrow materials be conducted. A detailed site investigation plan has not been prepared as part of the PEA, but would likely include the following:
    - Drilling to perform penetration tests (correlated to soil density), visually describe the soils, collect samples for laboratory testing, measure in-situ hydrogeologic properties of the soils and install geotechnical instrumentation (inclinometers and vibrating wire piezometers) or groundwater monitoring wells. Shear wave velocity measurements could be collected in standpipe piezometers to inform SSHA and liquefaction triggering assessments.
    - Cone penetration testing (CPT) to provide continuous soil classification that can be used in part of liquefaction triggering assessments. Shear wave velocities may also be measured.
    - Geophysics to infill data between drill holes and CPT soundings.
    - Test pits to characterize borrow sources.
- Mitigation measures for foundation improvement could be developed (weight drop or other foundation densification, stone columns, etc.), or if impractical, re-siting of the mill site, TMF/WRSF, CWCP and other structures could be required as a result of the investigation findings.
- Site-specific seismic hazard assessment (SSHA) to characterize seismic hazard to assess if MDE adopted is appropriate. Perform deformation analysis to quantify the magnitude of seismic deformations. If required, revise designs to reduce deformation to acceptable levels. Conduct a site-specific seismic hazard assessment to establish seismic loadings for structures and site facilities. Perform stability assessments to confirm designs comply with design criteria for static and seismic stability.
- Develop temporary storage contingencies outside of the TMF/WRSF for periods when pyrite tailings cannot be placed immediately underground. Options could include use of the lined temporary PAG rock / ore stockpile adjacent to the mill, a storage shed or a separate, contained area or section of the TMF/WRSF.
- Install instruments to establish baseline water quality and hydrogeological conditions.
- Optimize site-wide water balance to evaluate interactions between surface water storages and how the water management system will perform under prolonged dry or wet conditions, flood events, at different stages of the mine life, and following an operational upset of the WTP.
- Conduct additional testing on tailings and waste rock to define geotechnical properties. The next design stages for the TMF/WRSF should also anticipate settling and provide long term positive drainage to prevent ponding.
- Conduct additional testing on tailings and waste rock to further define geochemical properties. Assess tailings effluent quality to inform water management designs and water treatment needs.
- Conduct wind tunnel trials to assess dusting potential and appropriate mitigation strategies for the filtered tailings.

- Complete a constructability review, including material requirements and preparation of an execution plan, for the TMF/WRSF design to define risks to cost, schedule and identify areas for potential optimization.
- Undertake an operations review of the TMF/WRSF to assess whether adequate flexibility and management of risk have been incorporated into the design.
- Review the effect of tailings consolidation and differential settlement between structural and non-structural zones on closure cover design,
- Undertake a Failure, Modes, and Effects Analysis (FMEA) of the TMF/WRSF and other project components specific to technical assessments of risks, consequence, design resilience and potential operational failure modes. Results of the FMEA can be used (1) to provide guidance for instrumentation monitoring during operations; (2) to establish a surveillance program; and (3) to screen out failure modes that can be effectively managed by the Observational Method during operations.

The following trade-off evaluations should be performed as part of the next phase of work on this Project:

- The construction of a loading terminal in Haines to replace the Skagway facility.
- The optimal end-product form and packaging of the barite to maximize the sales margins.
- The best processing option for the AG Zone deposit should be evaluated, including preparation of a lead concentrate, suppression of the lead in the floatation circuits to maintain the value of the copper concentrate, or the generation of a cooper-lead concentrate.
- The economic viability of selling pyrite for the production of sulfuric acid off-site instead of returning it to the mine as backfill. Environmental baseline studies to include water quality sampling, species of interest studies, environmental rock geochemistry studies, and meteorological data collection.
- Continued ABA testwork to chemically characterize the TSF.
- Ongoing engagement with community, local stakeholders and governments with continued local hiring practices.

1.14.4 Recommended Budget

A proposed budget of US$30.0 million for the PFS recommendations is shown below in Table 1-13.

Table 1-12: Proposed Budget for PFS

Component	Estimated Cost ($US M)	Comment
Resource and Exploration Drilling (Surface and Underground. All-in Cost with assays, helicopter, salaries, supplies)	**8.0**	Conversion of inferred to indicated & measured resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	**0.5**	Comminution, DMS, flotation optimization, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Underground Development	**18.0**	Access for underground drilling and possible bulk sample. Based on actual quotes.
Geochemistry	**0.4**	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	**0.6**	Site investigation drilling, sampling and lab testing.
Geotechnical, Hydrology & Hydrogeology	**0.8**	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	**1.1**	PFS-level mine, infrastructure and process design, cost estimation, scheduling & economic analysis.
Environment	**0.8**	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology.
Total	**30.0**	**Excludes corporate overheads and future permitting activities.**

Source: JDS (2019)

2 Introduction

2.1 Basis of Technical Report

JDS Energy & Mining Inc. (JDS) was commissioned by Constantine Metal Resources Ltd. (Constantine) to manage and compile a Preliminary Economic Assessment (PEA) for the Palmer Project, the results of which are summarized in this Technical Report as per the guidelines of the Canadian Securities Administrator's National Instrument 43-101 (NI 43-101) Form 31-101F1. Other contributors to this PEA include Klohn Crippen Berger Ltd. and Advantage Geoservices Ltd.

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

2.2 Scope of Work

JDS was commissioned to act as lead author for the PEA, including the following scope of work:

- Mine design and costing
- Metallurgical and process design and costing
- Infrastructure design and costing
- Preparation of an economic model

Other contributors to the report include the following:

- Klohn Crippen Berger Ltd. was commissioned to provide the following:
    - Tailings/waste rock disposal options, and locations and site recommendation
    - Engineering criteria and bills of quantities for waste disposal facilities
    - Basic design of water management facilities
    - Geotechnical evaluation of mill foundations.
- Advantage Geoservices Ltd. was commissioned to provide current resource models for the Palmer deposit and AG Zone deposit.
- Core Geoscience Services Inc. was commissioned to provide advice on permitting requirements

2.3 Qualification Person Responsibilities and Site Inspections

Table 2-1 shows the responsibility matrix for the QPs of the Study.

Table 2-1: QP Responsibility Matrix

Qualified Person	Company	Site Visit	Report Sections
Richard Goodwin, P.Eng.	JDS Energy & Mining Inc.	September 12-13, 2018	1 (except 1.3, 1.4, 1.6, 1.8, 1.10), 2, 3, 4, 5, 6, 15, 16 (except 16.2), 18 (except 18.7, 18.8), 19, 21, 22, 24, 25 26, 27
Kelly McLeod, P.Eng.	JDS Energy & Mining Inc.		1.4, 1.8, 13, 17
Michael Levy, P.Eng.	JDS Energy & Mining Inc.		16.2
Jim Casey, P.E., P.Eng.	Klohn Crippen Berger Ltd.	September 21, 2018	18.7, 18.8
James Gray, P.Geo.	Advantage Geoservices Ltd.		1.3, 1.6, 7, 8, 9, 10, 11, 12, 14, 23
Jack DiMarchi, CPG	Core Geoscience LLC		1.10, 20

Source: JDS (2019)

2.4 Sources of Information

Sections 4 through 12 were extracted from the Technical Report "*NI 43-101 Technical Report and Updated Resource Estimate to include the AG Zone Deposit for the Palmer Exploration Project, Porcupine Mining District, Southeast Alaska, USA*" prepared for Constantine Metal Resources Ltd. by James N. Gray, P. Geo., Advantage Geoservices Ltd. and Ian R. Cunningham-Dunlop, P. Eng., Constantine Metal Resources Ltd., 31 January 2019.

Metallurgical assumptions were derived from the following test work reports:

- SGS Canada Inc, "*The Recovery of Copper, Zinc, Silver and Gold from the Palmer Samples*", Project No. 14063-001, (Issued: 28 October 2013)
- SGS Canada Inc, "*Barite Metallurgical Testwork on the Palmer VMS Project*", Project No. 14063-002, (Issued: 30 July 2018)
- SGS Canada Inc, "*Comminution and Mineralogy on the Palmer VMS Deposit*", Project No. 14063-03 Revision 1, (Issued: 14 December 2018)

Jurisdictional wetlands within the study area were defined by HDR, Inc. in their report: *Jurisdictional Determination Report, Palmer Exploration Project, Constantine North Inc., Haines Alaska, January 2018.*

All waste and water management information was extracted from a larger report prepared by Klohn Crippen Berger in support of this PEA: *Constantine Mining, LLC Palmer Preliminary Economic Assessment Study, Waste Management, Water Management and Mill site Foundation Design",* July 2019.

2.5 Units, Currency, Abbreviations and Acronyms

Metric units are used throughout the Report unless specifically stated otherwise.

All monies are expressed in 2019 $US unless specifically stated otherwise.

A table of common units and abbreviations used throughout the report is shown as Table 2-2.

Table 2-2: Units and Abbreviations Used throughout the Study

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
ABA	acid base accounting
ADEC	Alaska Department of Environmental Conservation
ADNR	Alaska Department of Natural Resources
ARD	acid rock drainage
Au	gold
B	billion
BD	bulk density
BLM	Bureau of Land Management
BoQ	bill of quantities
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
Cu	copper
CWA	clean water act
CWCP	contact water collection pond
d	day
d/a	days per year (annum)
d/wk	days per week
DBM	design basis memorandum
dmt	dry metric ton
DWT	dead weight tonnes
DQO	data quality objectives
EASP	exploration adit settling pond
EDF	environmental design flood
EIS	environmental impact statement
ELOS	equivalent linear overbreak/sloughing
FOS	factor of safety
ft	foot
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/t	grams per tonne

Symbol / Abbreviation	Description
gal	gallon (us)
GARD	Global Acid Rock Drainage
gpm	Gallons (US) per minute
GJ	gigajoule
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)
HDS	high density sludge water treatment
hp	horsepower
HQ	drill core diameter of 63.5 mm
Hz	hertz
IDF	inflow design flood
INAP	International Network for Acid Prevention
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
km	kilometre
km/h	kilometres per hour
kPa	kilopascal
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	mine level
L/min	litres per minute
L/s	litres per second
LAD	land application disposal
LADPA	the least environmental damaging practicable alternative
LOM	life of mine
m	metre
m H	drift height in metres
m W	drift width in metres
M	million
m/min	metres per minute

Symbol / Abbreviation	Description
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
MAP	mean annual precipitation
masl	metres above mean sea level
MCA	multiple criteria assessment
MDE	maximum design earthquake
MEND	mine environmental neutral discharge program
mg	milligram
ML	metals leaching
Mm^3	million cubic metres
MPa	megapascal
MSGP	multi-sector general permit
MSP	mill site sediment pond
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
Mw	moment magnitude
MWTP	mill site water treatment pond
NAD	North American datum
NEPA	national environmental policies act
NI 43-101	national instrument 43-101
NPAG	Non-potentially acid-generating
NPR	neutralization potential ratio (ABA)
NQ	drill core diameter of 47.6 mm
ORP	oxygen-reducing potential
oz	troy ounce
P. Eng.	Professional engineer
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
Pb	lead
PEA	preliminary economic assessment
PGA	peak ground acceleration
PMF	probable maximum flood
PMP	probable maximum precipitation
PMSA	probable maximum snow accumulation
ppb	parts per billion
ppm	parts per million
QA/QC	quality assurance/quality control

Symbol / Abbreviation	Description
QAPP	quality assurance protection plan
QP	qualified person
RMR	rock mass rating
ROM	run of mine
ROD	record of decision
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
S.G.	specific gravity
SG	specific gravity
SOP	standard operating procedure
SSHA	site-specific hazard assessment
SWE	snow water equivalent
SW Zone	South Wall
t	tonne (1,000 kg) (metric ton)
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
TDS	total dissolved solids
TMF/WRSF	tailings management facility/ waste rock storage facility
TSS	total suspended solids
US	United States
USACE	United States army corps of engineers
USGS	United States geologic survey
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VMS	volcanic massive sulfide
wk	week
wmt	wet metric ton
WMP	water management pond
WOTUS	waters of the United States
WRSF	waste rock storage facility
WRSP	waste rock sediment pond
WTP	water treatment plant
µm	microns
Zn	zinc

3 Reliance on Other Experts

This report has been prepared by the Authors for Constantine. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the Authors at the time of preparation of this report
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by Constantine and other third-party sources.

For the purpose of this report, the Authors have relied on ownership information provided by Constantine. The Authors have not researched Property title or mineral rights for the Palmer Exploration Project and express no opinion as to the ownership status of the Project. Effort was made to review the information provided for obvious errors and omissions; however, the Authors are not responsible for any errors or omissions relating the legal status of claims described within this report.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 Property Description and Location

4.1 Property Location

The Project is in the Porcupine Mining District, 55 km northwest of the town of Haines, in Southeast Alaska, USA. The western boundary of the Project is coincident with the international border and the Province of British Columbia, Canada (Figure 4-1). The Project lies 2 km from the Haines Highway, which links the deep-sea port of Haines, a terminal of the Alaska Marine Highway system, with British Columbia, Yukon, and the Alaska Highway. The geographic co-ordinates of the center of the Project are approximately 136°25'N and 59°20'W.

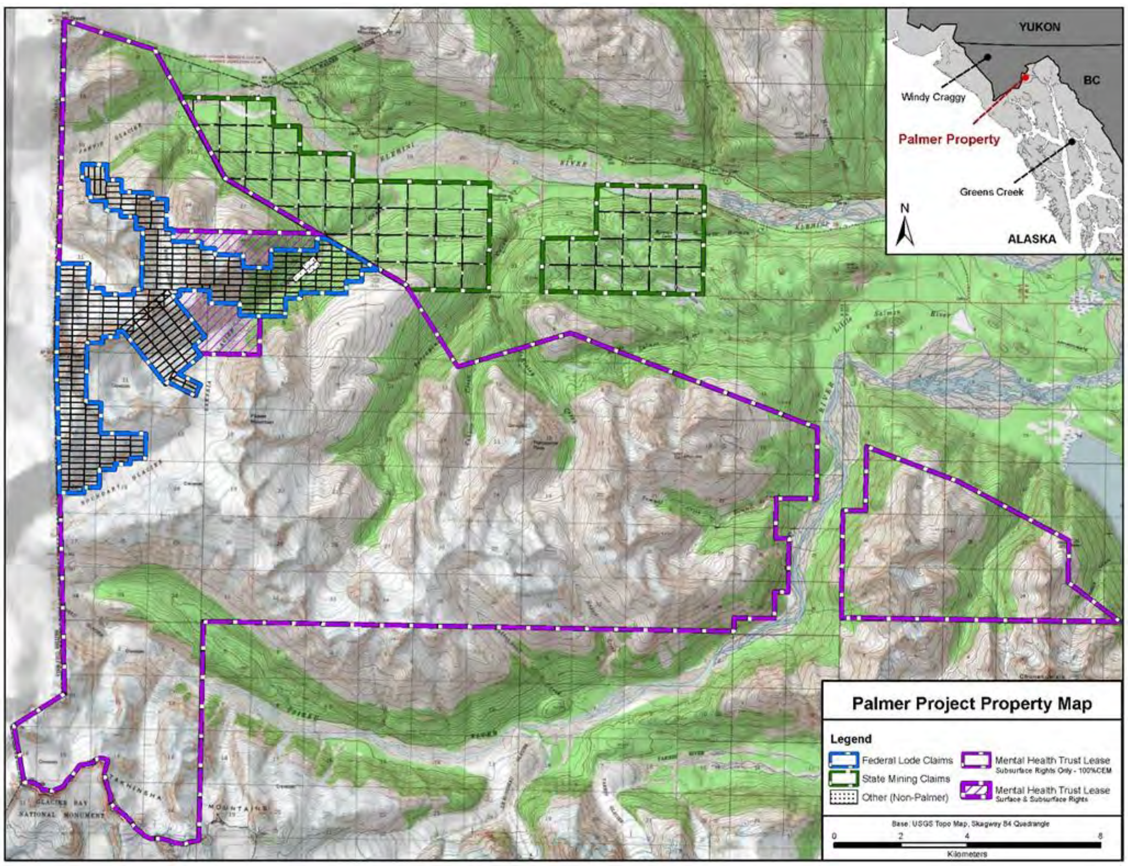
Figure 4-1: Project Location Map

Source: Constantine Metal Resources Ltd. (2018)

4.2 Project Description

The Project consists of a contiguous block of land (Figure 4-2) consisting of 63 state mineral claims that cover an area of approximately 9,200 acres (~3,680 hectares or 37 km^2) (Table 4-1) and 340 federal unpatented lode mining claims, which cover an area of approximately 6,765 acres (~2,738 hectares or 27 km^2) (Table 4-2). These core claims are surrounded land leased by the Company from the Alaska Mental Health Trust which total 65,772 acres, giving a Project total of 81,737 acres (~ 33,078 hectares or 330 km^2) (Table 4-3).

Figure 4-2: Project Claim Map



Source: Constantine (2018)

Table 4-1: List of 63 State Lode Mining Claims

CLAIM #	SECTION	TOWNSHIP	RANGE	CLAIM #	SECTION	TOWNSHIP	RANGE
661267	16SW	T28S	53E	662069	26NE	T28S	54E
661268	16SE	T28S	53E	662070	25NW	T28S	54E
661269	15SW	T28S	53E	662071	25NE	T28S	54E
661270	15SE	T28S	53E	662072	25SE	T28S	54E
661271	21NE	T28S	53E	662073	25SW	T28S	54E
661272	22NW	T28S	53E	662074	26SE	T28S	54E
661273	22NE	T28S	53E	662075	26SW	T28S	54E
661274	23NW	T28S	53E	662078	29SE	T28S	54E
661275	21SE	T28S	53E	662079	29SW	T28S	54E
661276	22SW	T28S	53E	662080	30SE	T28S	54E
661277	22SE	T28S	53E	662081	30SW	T28S	54E
661278	23SW	T28S	53E	662082	31NW	T28S	54E
661279	23SE	T28S	53E	662083	31NE	T28S	54E
661280	24SW	T28S	53E	662084	32NW	T28S	54E
661281	27NW	T28S	53E	662085	32NE	T28S	54E
661282	27NE	T28S	53E	662088	34NW	T28S	54E
661283	26NW	T28S	53E	662089	34NE	T28S	54E
661284	26NE	T28S	53E	662090	35NW	T28S	54E
661285	25NW	T28S	53E	662091	35NE	T28S	54E
661286	25NE	T28S	53E	662092	36NW	T28S	54E
661287	26SW	T28S	53E	662093	36NE	T28S	54E
661288	26SE	T28S	53E	662094	36SE	T28S	54E
661289	25SW	T28S	53E	662095	36SW	T28S	54E
661290	25SE	T28S	53E	662096	35SE	T28S	54E
661291	35NE	T28S	53E	662097	35SW	T28S	54E
661292	36NW	T28S	53E	662098	34SE	T28S	54E
661293	36NE	T28S	53E	662099	34SW	T28S	54E
662062	30NW	T28S	54E	662102	32SE	T28S	54E
662063	30NE	T28S	54E	662103	32SW	T28S	54E
662064	29NW	T28S	54E	662104	31SE	T28S	54E
662065	29NE	T28S	54E	662105	31SW	T28S	54E
662068	26NW	T28S	54E				

Source: Constantine Ltd. (2018)

Table 4-2: List of 340 Federal Unpatented Lode Mining Claims

Claim Name	BLM No.	Claim Name	BLM No.	Claim Name	BLM No.
#1 of Marmot Mine	AA 27186	Jarvis 3	AA 51513	Clay #53	AA 52687
#2 of Marmot Mine	AA 27187	Jarvis 4	AA 51514	Clay #54	AA 52688
#3 of Marmot Mine	AA 27188	Jarvis 5	AA 51515	Clay #55	AA 52689
#4 of Marmot Mine	AA 27189	Jarvis 6	AA 51516	Clay #56	AA 52690
M.V.P. Mining Claims #1	AA 27190	Jarvis 7	AA 51517	Clay #57	AA 52691
M.V.P. Mining Claims #2	AA 27191	Jarvis 8	AA 51518	Clay #58	AA 52692
Marmot #5	AA 27192	"Ice" #43	AA 51519	Clay #59	AA 52693
Marmot #6	AA 27193	"Ice" #44	AA 51520	Clay #60	AA 52694
Marmot #7	AA 27194	"Ice" #45	AA 51521	Marmot Hole #1	AA 52945
Marmot #8	AA 27195	"Ice" #46	AA 51522	Marmot Hole #2	AA 52946
Marmot #9	AA 27196	"Ice" #47	AA 51523	Marmot Hole #3	AA 52947
Marmot #10	AA 27197	"Ice" #48	AA 51524	Marmot Hole #4	AA 52948
Marmot Claim #20	AA 27198	"Ice" #49	AA 51525	Marmot Hole #5	AA 52949
Marmot Claim #21	AA 27199	"Ice" #50	AA 51526	Marmot Hole #6	AA 52950
Marmot Claim #22	AA 27200	"Ice" #51	AA 51527	Marmot Hole #7	AA 52951
Marmot Claim #23	AA 27201	"Ice" #54	AA 51528	Marmot Hole #8	AA 52952
Marmot Claim #24	AA 27202	"Ice" #55	AA 51529	Fey #1	AA 52953
Marmot Claim #25	AA 27203	"Ice" #56	AA 51530	Fey #2	AA 52954
Marmot Claim #26	AA 27204	"Ice" #57	AA 51531	Fey #3	AA 52955
Marmot Claim #27	AA 27205	"Ice" #60	AA 51532	Fey #4	AA 52956
Marmot Claim #28	AA 27206	"Ice" #61	AA 51533	Fey #5	AA 52957
Marmot Claim #29	AA 27207	"Ice" #62	AA 51534	Fey #6	AA 52958
Marmot Claim #30	AA 27208	"Ice" #63	AA 51535	Fey #7	AA 52959
Marmot Claim #31	AA 27209	"Ice" #64	AA 51536	Fey #8	AA 52960
Marmot #32	AA 27210	"Ice" #65	AA 51537	Fey #9	AA 52961
Marmot #33	AA 27211	"Ice" #66	AA 51538	Fey #10	AA 52962
Marmot #101	AA 27213	"Ice" #67	AA 51539	Fey #11	AA 52963
Marmot #102	AA 27214	"Ice" #68	AA 51540	Fey #12	AA 52964
Marmot #103	AA 27215	"Ice" #69	AA 51541	Fey #13	AA 52965
Marmot #104	AA 27216	"Ice" #70	AA 51542	Fey #14	AA 52966
Marmot #105	AA 27217	"Ice" #71	AA 51543	Fey #15	AA 52967
Marmot #106	AA 27218	"Ice" #72	AA 51544	Fey #16	AA 52968
Marmot #107	AA 27219	"Ice" #73	AA 51545	Fey #17	AA 52969
Marmot #108	AA 27220	"Ice" #74	AA 51546	Fey #18	AA 52970
Marmot #109	AA 27221	Kic #1	AA 51558	Fey #19	AA 52971
Marmot #110	AA 27222	Kic #2	AA 51559	Fey #20	AA 52972

Claim Name	BLM No.	Claim Name	BLM No.	Claim Name	BLM No.
Marmot 111	AA 27223	Kic #3	AA 51560	Boundless #1	AA 52973
Marmot #112	AA 27224	Kic #4	AA 51561	Boundless #2	AA 52974
Marmot 113	AA 27225	Kic #5	AA 51562	Boundless #3	AA 52975
Marmot #114	AA 27226	Kic #6	AA 51563	Boundless #4	AA 52976
Marmot #115	AA 27227	Kic #7	AA 51564	Boundless #5	AA 52977
Marmot #116	AA 27228	Kic #8	AA 51565	Boundless #6	AA 52978
Marmot #117	AA 27229	Kic #9	AA 51566	Boundless #7	AA 52979
Marmot 118	AA 27230	Kic #10	AA 51567	Boundless #8	AA 52980
Marmot 119	AA 27231	Kic #11	AA 51568	Boundless #9	AA 52981
Marmot #120	AA 27232	Kic #12	AA 51569	Boundless #10	AA 52982
Marmot #121	AA 27233	Kic #13	AA 51570	Boundless #11	AA 52983
Marmot 122	AA 27234	Kic #14	AA 51571	Boundless #12	AA 52984
Marmot #123	AA 27235	Kic #15	AA 51572	Boundless #13	AA 52985
Marmot 124	AA 27236	Kic #16	AA 51573	Boundless #14	AA 52986
Marmot #125	AA 27237	"Hot Dawg" #1	AA 51574	Boundless #15	AA 52987
Marmot #126	AA 27238	"Hot Dawg" #2	AA 51575	Boundless #16	AA 52988
Marmot #127	AA 27239	"Hot Dawg" #3	AA 51576	Boundless #17	AA 52989
Marmot #128	AA 27240	"Hot Dawg" #4	AA 51577	Boundless #18	AA 52990
Marmot #129	AA 27241	"Hot Dawg" #5	AA 51578	Boundless #19	AA 52991
Marmot #130	AA 27242	"Hot Dawg" #6	AA 51579	Boundless #20	AA 52992
Marmot #131	AA 27243	"Hot Dawg" #7	AA 51580	Boundless #21	AA 52993
Marmot #132	AA 27244	"Hot Dawg" #8	AA 51581	Boundless #22	AA 52994
Marmot #134	AA 27246	"Hot Dawg" #9	AA 51582	Boundless #23	AA 52995
Marmot #135	AA 27247	"Hot Dawg" #10	AA 51583	Boundless #24	AA 52996
Marmot #136	AA 27248	"Hot Dawg" #11	AA 51584	Boundless #25	AA 52997
Marmot #137	AA 27249	"Hot Dawg" #12	AA 51585	Boundless #26	AA 52998
Marmot #138	AA 27250	"Hot Dawg" #13	AA 51586	Boundless #27	AA 52999
Marmot #139	AA 27251	"Hot Dawg" #14	AA 51587	Boundless #28	AA 53000
Marmot #140	AA 27252	"Hot Dawg" #15	AA 51588	Boundless #29	AA 53001
Marmot #141	AA 27253	"Hot Dawg" #16	AA 51589	Boundless #30	AA 53002
Marmot #142	AA 27254	"Hot Dawg" #17	AA 51590	Boundless #31	AA 53003
Marmot #143	AA 27255	"Hot Dawg" #18	AA 51591	Boundless #32	AA 53004
Marmot #144	AA 27256	"Hot Dawg" #19	AA 51592	Boundless #33	AA 53005
Marmot #145	AA 27257	"Hot Dawg" #20	AA 51593	Boundless #34	AA 53006
Marmot #146	AA 27258	"Hot Dawg" #21	AA 51594	Boundless #35	AA 53007
Marmot #147	AA 27259	"Hot Dawg" #22	AA 51595	Boundless #36	AA 53008
Marmot #148	AA 27260	"Hot Dawg" #23	AA 51596	Boundless #37	AA 53009

Claim Name	BLM No.	Claim Name	BLM No.	Claim Name	BLM No.
Marmot #149	AA 27261	"Hot Dawg" #24	AA 51597	Boundless #38	AA 53010
Marmot #150	AA 27262	"Hot Dawg" #25	AA 51598	Boundless #39	AA 53011
Marmot #151	AA 27263	"Hot Dawg" #26	AA 51599	Boundless #40	AA 53012
Marmot #152	AA 27264	"Hot Dawg" #27	AA 51600	Boundless #41	AA 53013
Marmot #153	AA 27265	"Hot Dawg" #28	AA 51601	Boundless #42	AA 53014
Marmot #154	AA 27266	Clay #17	AA 52651	Boundless #43	AA 53015
Marmot #155	AA 27267	Clay #18	AA 52652	Boundless #44	AA 53016
Marmot #156	AA 27268	Clay #19	AA 52653	Boundless #45	AA 53017
Marmot #157	AA 27269	Clay #20	AA 52654	Connexion #1	AA 53018
Marmot #158	AA 27270	Clay #21	AA 52655	Connexion #2	AA 53019
Marmot #159	AA 27271	Clay #22	AA 52656	Connexion #3	AA 53020
Marmot #160	AA 27272	Clay #23	AA 52657	Connexion #4	AA 53021
Marmot #161	AA 27273	Clay #24	AA 52658	Connexion #5	AA 53022
Marmot #162	AA 27274	Clay #25	AA 52659	Connexion #6	AA 53023
Marmot #163	AA 27275	Clay #26	AA 52660	Connexion #7	AA 53024
Marmot #164	AA 27276	Clay #27	AA 52661	Connexion #8	AA 53025
Marmot #166	AA 27277	Clay #28	AA 52662	Connexion #9	AA 53026
Marmot #167	AA 27278	Clay #29	AA 52663	Connexion #10	AA 53027
Marmot #171	AA 27279	Clay #30	AA 52664	Connexion #11	AA 53028
Marmot #172	AA 27280	Clay #31	AA 52665	Connexion #12	AA 53029
Rat Dawg 43	AA 29575	Clay #32	AA 52666	Connexion #13	AA 53030
Rat Dawg 44	AA 29576	Clay #33	AA 52667	Connexion #14	AA 53031
Rat Dawg 53	AA 29577	Clay #34	AA 52668	Connexion #15	AA 53032
Rat Dawg 54	AA 29578	Clay #35	AA 52669	Connexion #16	AA 53033
Rat Dawg #55	AA 29579	Clay #36	AA 52670	Connexion #17	AA 53034
Rat Dawg 56	AA 29580	Clay #37	AA 52671	Connexion #18	AA 53035
Rat Dawg #57	AA 29581	Clay #38	AA 52672	Connexion #19	AA 53036
Rat Dawg 58	AA 29582	Clay #39	AA 52673	Connexion #20	AA 53037
Rat Dawg 64	AA 29583	Clay #40	AA 52674	Connexion #21	AA 53038
Rat Dawg #65	AA 29584	Clay #41	AA 52675	Connexion #22	AA 53039
Rat Dawg 66	AA 29585	Clay #42	AA 52676	Connexion #23	AA 53040
Rat Dawg #67	AA 29586	Clay #43	AA 52677	Connexion #24	AA 53041
Rat Dawg #68	AA 29587	Clay #44	AA 52678	Connexion #25	AA 53042
Rat Dawg #75	AA 29588	Clay #45	AA 52679	Connexion #26	AA 53043
Rat Dawg #76	AA 29589	Clay #46	AA 52680	Connexion #27	AA 53044
Rat Dawg #77	AA 29590	Clay #47	AA 52681	Connexion #28	AA 53045
Rat Dawg #85	AA 29591	Clay #48	AA 52682	Connexion #29	AA 53046

Claim Name	BLM No.	Claim Name	BLM No.	Claim Name	BLM No.
Rat Dawg #86	AA 29592	Clay #49	AA 52683	Connexion #30	AA 53047
Rat Dawg #87	AA 29593	Clay #50	AA 52684	Connexion #31	AA 53048
Jarvis 1	AA 51511	Clay #51	AA 52685		
Jarvis 2	AA 51512	Clay #52	AA 52686		

Source: Constantine (2018)

Table 4-3: List of AMHT Lands

Mineral Lease File Number	MHT Parcel Number	Rights	Ownership
MHT 9100759	C70451	Subsurface & Surface	Palmer Project Joint Venture Agreement
	C81210	Subsurface	100% CMR

Source: Constantine (2018)

Additional information on parcels is available at:

http://dnr.alaska.gov/projects/las/#filenumber/9100759/filetype/MHT/landflag/y/searchtype/casefile/reporttype/abstract

4.3 Property Interests, Royalties, and Other Legal Obligations

Constantine Metal Resources Ltd., incorporated March 3rd, 2006, was created for acquiring a 100% interest in the Palmer Project held by Toquima North Inc., a wholly owned subsidiary of Toquima Minerals Corporation (Toquima). Constantine acquired Toquima's interest by means of a Plan of Arrangement and assignment of its interest in Toquima North Inc., now Constantine North Inc.

Constantine, through its wholly owned US subsidiary Constantine North Inc. (formerly Toquima North Inc.), has a 99-year Mineral Lease Agreement on the 340 federal unpatented lode mining claims. The Mineral Lease, dated effective December 19th, 1997 and originally signed by Rubicon Minerals Corporation, is with Alyu Mining, Inc. and Haines Mining-Exploration Inc. (collectively the "Owners") both of Haines, Alaska.

The material terms of the Mineral Lease are as follows in Sections 4.3.1 and 4.3.2.

4.3.1 Advance Royalty Payments to the Owners

Constantine North Inc. is to make annual aggregate advance royalty cash payments to the Owners of $42,500. The initial advance royalty payments are to be paid in quarterly tranches of $10,625 each, commencing on November 10th, 1997 and continuing up to and including the 98th anniversary of the Mineral Lease. The advance royalty payments are fully paid to date. To maintain the Mineral Lease, Constantine North Inc. is also required to make annual maintenance fee payments to the Bureau of Land Management ("BLM"). Maintenance fee payments are currently $155/claim*, totaling $52,700 per year.

(*https://www.blm.gov/programs/energy-and-minerals/mining-and-minerals/locatable-minerals/mining-claims/fees).

4.3.2 Net Smelter Return Royalty

The Owners will each be entitled to half of a 2.5% net smelter return royalty on the Palmer Project. The advance royalty cash payments shall be recouped from the net smelter return royalty payable in that year or in subsequent years; however, in no year shall the amount of the aggregate of the net smelter return royalty and the advance royalty cash payment be less than US$ 42,500. The obligation to pay annual advance royalty cash payments shall be extinguished once the Owners have received a total of US$ 4,500,000 in advance royalty cash payments. Constantine North Inc. has a right of first refusal to purchase the net smelter return royalty, or any portion thereof, at any time during the term of the mineral lease.

4.3.3 Option Agreement with Dowa Mining & Metals Co., Ltd.

Constantine signed an Option and Joint Venture Agreement (the "Agreement") with Dowa Metals & Mining Co., Ltd. of Japan (Dowa) on the Project on February 1st, 2013.

Under the terms of the Agreement, Dowa had the option to earn a 49% interest in the Project by making aggregate expenditures of US$ 22 M over a four-year period. Included in the aggregate expenditure were cash payments to Constantine totaling US$ 1,250,000 over four years. The Agreement also included terms allowing Dowa to acquire 100% of the zinc off-take rights at arms-length commercial terms

On January 5th, 2017, the Company announced that Dowa had completed its US$ 22 million earn-in to the Project and had exercised its option to participate as a partner in the Project. A Joint Venture has now been formed (the Dowa JV) for the purpose of further exploring and developing the Project, with Constantine owning a 51% participating interest and Dowa owning a 49% participating interest.

Approximately US$ 2 M in unspent earn-in funds were used to form the starting cash balance of the Joint Venture.

4.3.4 Other Underlying Agreements or Obligations

There are no other underlying agreements or obligations encumbering the Project. As the claims are unpatented, no local or county-based property taxes have been assessed against them.

The federal claims are located on federal lands that are managed (both surface and mineral estates) by the United States Department of the Interior, Bureau of Land Management ("BLM"). The State claims are located on Alaska State lands that are managed by the Alaska Department of Natural Resources. Both state and federal claims are in good standing as of the date of this report.

4.4 Alaska Mental Health Trust Land

During Alaska's transition to a state, the US Congress passed the *Alaska Mental Health Enabling Act* of 1956 (http://mhtrust.org/about/history/). This act transferred the responsibility for providing mental health services from the federal government to the territory of Alaska and ultimately the state, by creating the Alaska Mental Health Trust ("(AMHT or MHT). To fund it, the state selected one million prime acres of land that would be managed to generate income to help pay for a comprehensive and integrated mental health program in Alaska.

Though the Alaska Legislature held a fiduciary responsibility to manage the land on behalf of Alaskans with mental disabilities, it did not do so. Instead, by 1982, only about 35 percent of the trust land remained in state ownership. Most of the land had been transferred to individuals or municipalities, or designated as forests, parks or wildlife areas.

In 1982, Vern Weiss filed a lawsuit on behalf of his son, who required mental health services that were not available in Alaska. Other beneficiary groups joined Weiss v State of Alaska in a class action suit. The case was ruled on in 1984 by the state Supreme Court, which ordered that the original trust be restored. Ten years later in 1992, a final settlement reconstructed the Trust with 500,000 acres of original Trust land and 500,000 acres of replacement land, plus $200 million in cash. As part of the settlement, the Trust's cash assets are managed under a contract with the Alaska Permanent Fund Corporation, and the land and non-cash assets are managed under a contract with the Trust Land Office within the Department of Natural Resources. The settlement also established an independent board of trustees, which is appointed by the governor and confirmed by the Legislature.

The AMHT lands in the region surrounding the Project were selected based on their mineral resource potential. The AMHT selection also includes area that overlaps the Project federal claims. Under the terms of the grant, mineral title of any lapsed federal claim would automatically revert to AMHT.

In 2014, Constantine was the successful applicant in a competitive lease process for the 'Haines Block' (MH Parcels C81209, C81210 and C70451) offered by the AMHT (Figure 4-3). The Trust owns the subsurface mineral estate of the Haines Block, <u>and for a small subset of the block, located adjacent to the Palmer Project, land is held fee simple for which the Trust owns both the surface and subsurface estate</u>. The Upland Mining Lease MHT No. 9100759 was finalized and signed with an effective date of September 1st, 2014, thereby consolidating a district-scale property position totaling approximately 108,000 acres (circa 2014), inclusive of the Palmer state and federal claims (Figure 4-3). Acquisition of the Haines Block lands provides protection of existing interests, unfettered access for ongoing exploration and future development, and strategic control of the entire tract of land with known volcanogenic massive sulfide potential.

The Haines Block occurs within the Area of Interest of the Palmer Project Option and Joint Venture Agreement with Dowa.

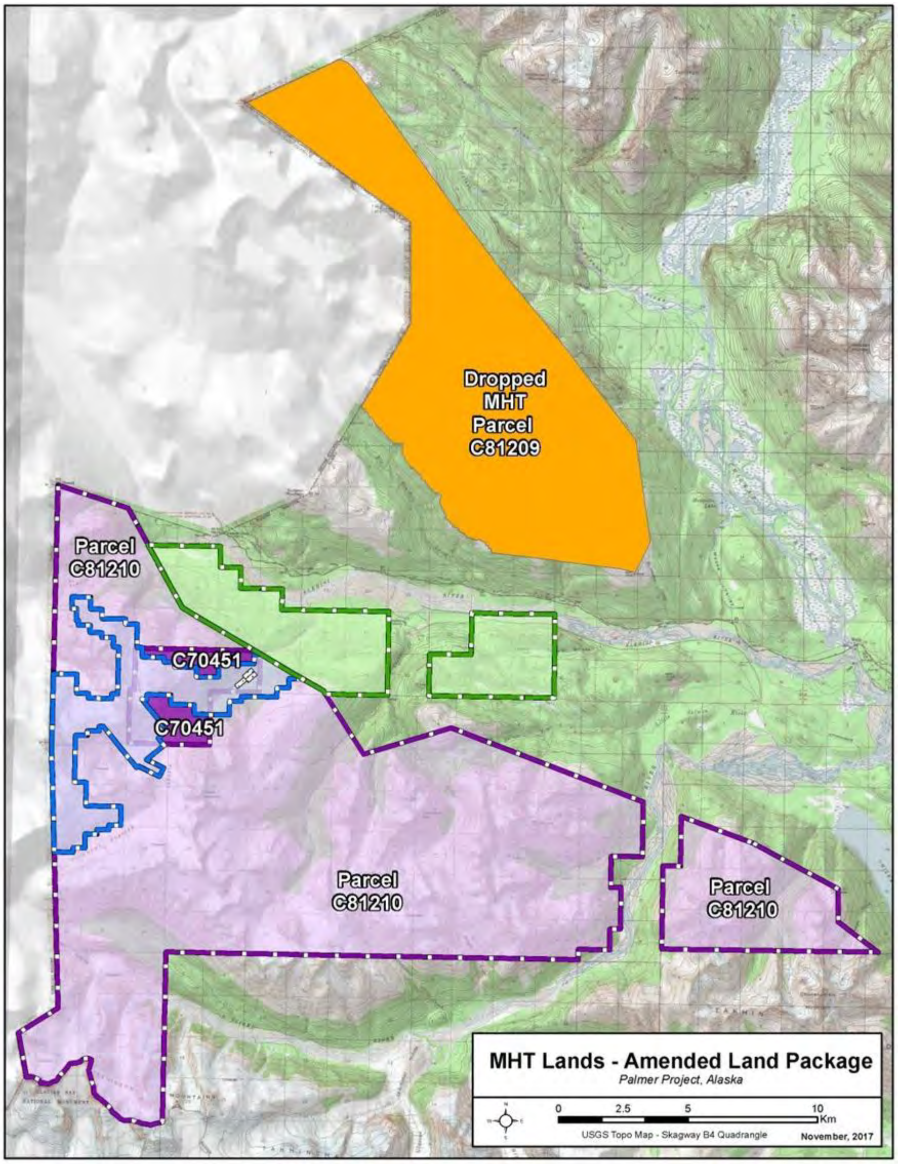
4.4.1 MHT Location Overview

The Mental Health Trust lands (the Trust) are located on the flanks of the Chilkat Mountains in the Juneau Mining District, 30 miles northwest of Haines, Alaska, USA comprising approximately 99,257 acres in three parcels, of which 65,772 acres in two parcels are currently under lease to the Company for mineral exploration and development. This land package is referred to as the "Haines Block" (Figure 4-3).

A small portion of the Haines Block, near the Palmer Project, is held fee simple with the Trust owning both the surface estate and subsurface mineral estate (dark purple). For most of the acreage, the Trust owns the mineral estate and the State of Alaska owns and manages the surface. Near the Palmer Project, the Haines Block is subject to mining claims controlled by Constantine North Inc.

The Haines Block is situated in the Porcupine mining district which has recorded intermittent production of approximately 82,489 ounces of placer gold but no lode gold.

Figure 4-3: Project Map showing AMHT Lands, Haines Block and area Under Lease (purple)



Source: Constantine (2018)

4.4.2 Description of the MHT Leased Lands ("Haines Block")

The subject parcels are in the Copper Skagway C-3, C-4, B-3, and B-4 quadrangle, Alaska, USA and originally included parcels C81210, C70451 and C81209 (now dropped) totaling 99,257 acres (now 65,772 acres). Portions of parcel C81210, specifically the East ½ Section 11, West ½ Section 12, West ½ Section 13, and East ½ Section 14, T 29 S, R. 54 E, CRM are subject to a placer mining lease with Blue Ribbon Gold, Inc. Four federal mining claims on Cahoon Creek constitute a federal in-holding in this area. Near the Palmer Project, the Trust's mineral estate is subject to the federal mining claims owned by Alyu Mining Inc.

and Haines Mining-Exploration, Inc., which are under lease to Constantine North Inc. These federal mining claims and the lands they encumber are managed by the BLM.

4.4.3 Terms of MHT Lease Agreement ("TLO Lease")

The Upland Mining Lease MHT. No.9100759 between the Alaska Mental Health Trust Authority acting by and through the State of Alaska, Department of Natural Resources, Mental Health Trust Land Office {collectively the "TLO") and Constantine North, Inc. (Lessee) was made effective September 1st, 2014 {the TLO Lease).

A general summary of the TLO Lease terms includes annual rental of US$25,000 per year for the initial 3-year TLO Lease term, US$40,000 for years 4 to 6, US$55,000 for years 7 through 9, with work commitments of US$75,000 per year, escalating US$50,000 annually. There is a mandatory acreage reduction of 25,000 acres at the end of the first and second 3-year TLO Lease terms. The TLO Lease can be extended beyond year 9 by making annual rental payments and continuing to diligently pursue exploration and development on the TLO Lease. Annual rental payments are replaced by royalty payments upon achieving commercial production. Production royalty's payable to the TLO include a sliding scale 1% to 4.5% royalty for gold based on gold price, and a 3.5% royalty on minerals other than gold. The TLO Lease in its entirety is provided as a reference and forms part of the Selection Agreement.

4.4.4 Description of the MHT Leased Lands added to the Dowa JV ("Selection Area")

In a letter dated October 16th, 2014, the Company advised Dowa that they were a successful applicant in a competitive lease offered by TLO and that they had signed a lease (the TLO Lease). On January 19th, 2015, Dowa advised the Company that Dowa had selected a portion of the TLO Lease area (Selection Area) to be included as part of the Project for which expenditures will apply to Dowa's 49% Earn-in Expenditures during the Option phase of the Agreement. The Selection Area that was requested by Dowa and accepted by the Company constitutes part of the Project as represented in the TLO Lease by parcel C70451 with surface and mineral estate (to the extent owned by the State of Alaska, Department of Natural Resources, Mental Health Trust Land Office (TLO) and comprising approximately 3,483 acres that lies within T.028S., R.053E. Sections 33, 34 and 35, T.029S., R.053E. Section 1 and T.029S., R.054E. Section 6 (Figure 4-4).

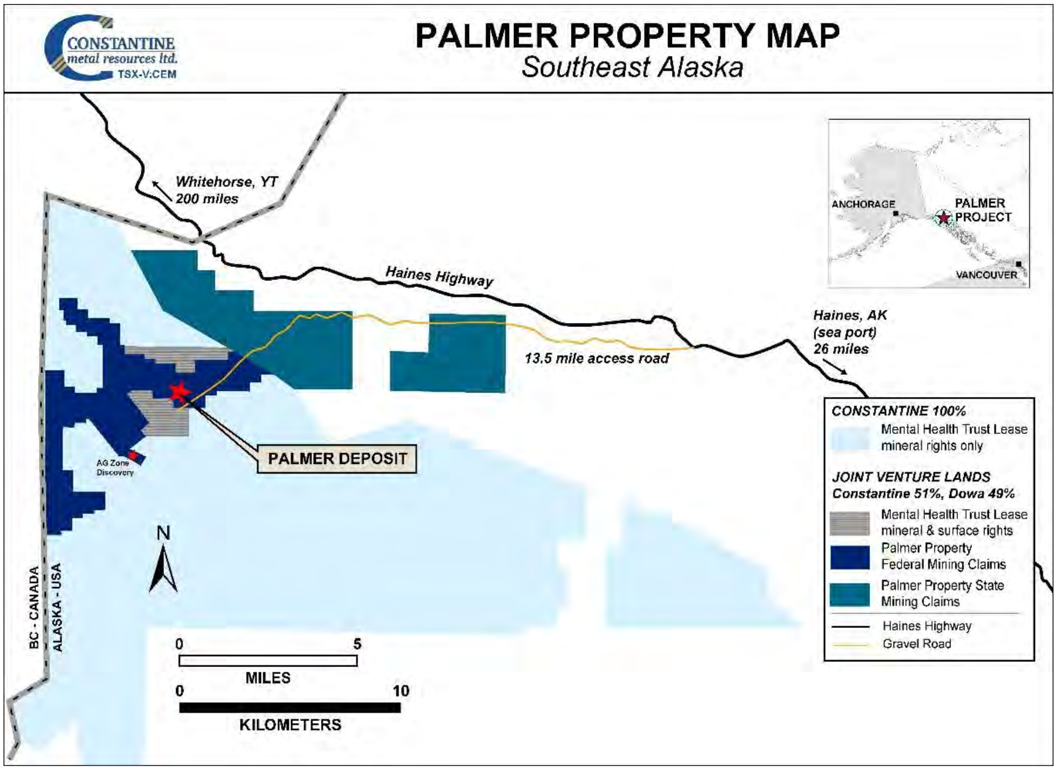
Upon the formation of the Dowa JV, the Company assigned the Selection Area to the Joint Venture, such that the interest that Dowa had earned pursuant to the Agreement would include a 49% interest in the Selection Area, subject to the approval of the TLO at the time, as provided for in Section 15 of the TLO Lease.

4.4.5 Terms of Amended MHT Lease Agreement

The Upland Mining Lease MHT No. 9100759 was amended on September 1st, 2017 when the Company notified TLO of their intent to drop MHT Parcel C81209 on the north side of the Klehini River, and reduced the TLO Lease from three parcels to two (Figure 4-4) including:

1) C70451 surface and mineral estate comprising approximately 3,483 acres and
2) C81210 mineral estate comprising approximately 62,289 acres.

Figure 4-4: Project Map showing location of Current AMHT Lands



Source: Constantine (2018)

4.5 Annual Property Maintenance

The State claims and Federal claims that comprise a portion of the Palmer property are all in good standing for September 1st, 2018 through August 31st, 2019. Annual payments, rental fees, and necessary filings (e.g. affidavits of labor and intent to hold) have been properly documented and recorded with the various government agencies. The MHT lands under lease have been amended as of September 1st, 2017 and the necessary annual lease payment and work expenditure requirements have been met.

4.5.1 Federal Claims

Annual maintenance fee payment of $52,700 ($155/claim) was made prior to Sept 1st, 2018 for the 340 Federal mining claims managed by the Bureau of Land Management (BLM). A notarized Affidavit of Payment and Notice of Intent to Hold were filed with the BLM and subsequently recorded with the State. **Federal claims are in good standing until September 1st, 2019.**

4.5.2 State Claims

An annual rental payment of $42,840 ($680/claim) was made prior to November 30th, 2018 for the 63 State mining claims managed by the Department of Natural Resources (DNR). Note that rental payment of $680/claim is the upper limit of payment (based on 11 or more years of a claim being located) and this rental fee is not expected to increase in the future (Table 4-4).

Table 4-4: Annual Rental fees for State Claims as per Department Regulation 11 AAC 86.221(b)

Number of Years for Location	Quarter-Section Size MTRSC Location (160 Acres)	Billing Date	Payment Due
Year 1 *Day 1 - September 1st* of Mining Year Location Is Staked	\$140	Same Day Claims were Located	45 Days from Posting Location
2 - 5	\$140	September 1st	November 30th
6 - 10	\$280	September 1st	November 30th
11 or More	\$680	September 1st	November 30th

Source: Constantine (2018)

In addition to rental payments, annual labour of \$400/quarter section (\$25,200 total) is due November 30th of each year. Work expenditure performed on adjacent Federal claims or MHT lands can be applied to State claims to satisfy this requirement. Excess work expenditures can be carried forward and be applied in subsequent years for as many as four years. Credits from previous years were available and expenditures from the 2017 drilling program were filed for future use. An Affidavit of Annual Labour was recorded with the State prior to November 30th, 2018. Sufficient credits have been incurred to cover work expenditure requirements for the next four years.

4.5.3 Mental Health Trust Lands

The Upland Mining Lease MHT no. 9100759 (the Lease) was made between Constantine North Inc. and the Alaska Mental Health Trust Authority on September 1st, 2014. Pursuant to the Lease, Constantine is required to surrender a minimum of 25,000 acres at the end of the three-year term. On September 1st, 2017, an amendment (amendment No. 1) was made effective to the Lease to exclude Trust Parcel C81209 (26, 493 acres) from the Leased Area. The Lease term was also extended for an additional three years, and the new expiry date for the remaining two Trust parcels is August 31st, 2020. On August 31st, 2020, the Company is required to surrender another minimum of 25,000 acres and can then renew the Lease for another three-year term (and longer) after which there are no additional requirements to reduce the land package.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Project is located adjacent to the paved all-weather Haines Highway (Alaska Highway #7), which connects the town of Haines, AK, USA situated 65 km to the southeast, with the town of Haines Junction, YK, Canada located 200 km to the north.

Access to the northern, central and eastern portions of the Project from the Haines Highway is achieved by crossing the Klehini River via the Porcupine Bridge located at '26 Mile' (or 42 km from Haines). Travel continues westward along the graded Porcupine Creek access road for 11 km to the Company's camp at the Big Nugget mine site located on private land in the Porcupine Creek Valley (Figure 5-1). Travel from camp to the centre of the Project area (an additional 11 km) is afforded by a series of logging roads maintained by the Haines Borough, which connect to the Glacier Creek Access Road which was constructed (and is currently maintained) by the Company.

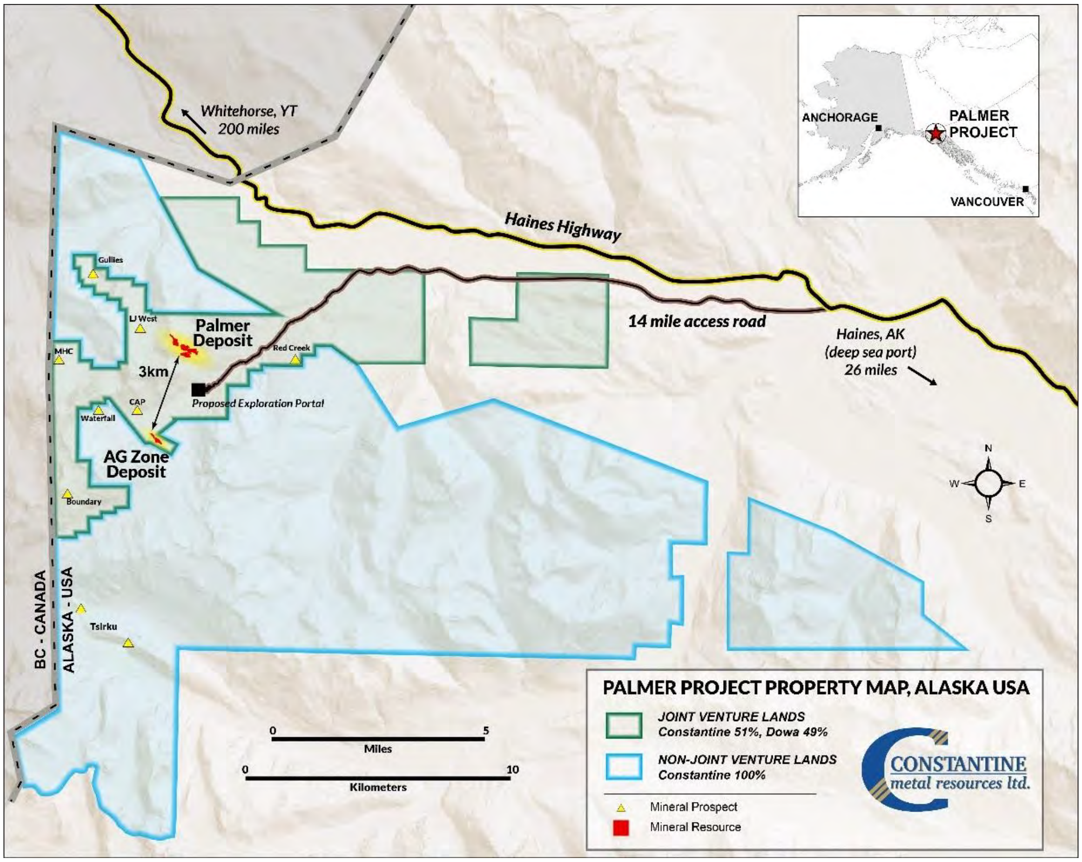
The Glacier Creek Access Road provides 2-wheel drive access to within a short distance of the mineral resource. However, practical access to majority of the Project, including nearly all exploration drill sites, is by helicopter. Drill core and camp facilities are currently based at the Big Nugget Camp.

The nearest major economic centers are Whitehorse, YK (400 km by paved road), and Juneau, AK (4½ hours by ferry). Daily scheduled flights connect Haines, AK with Juneau, AK which, in turn, has daily connections with the continental US, and via Whitehorse, YK to Vancouver, BC in Canada.

Haines (population 2,400) is a year-round deep-sea port at the north end of the Alaska Marine Highway, and the town boasts infrastructure to support exploration and mining operations. Many residents commute daily via ferry to Juneau to the Kensington Gold Mine, operated by Coeur Mining, and the Green's Creek Silver-Gold-Lead-Zinc Mine operated by Hecla Mining Company.

Temperatures are typical of the north coast of Southeast Alaska, with lows of -25º C in the winter, to highs of 25º C in the summer. At higher elevations, fieldwork is limited to late May through early October because of snow. Recent warming trends have resulted in rapid glacial retreat and better outcrop exposure over the past years.

Figure 5-1: Project Access Routes and Local Infrastructure



Source: Constantine (2018)

Figure 5-2: Westerly View up Glacier Creek Valley Towards Saksaia Glacier (Prior to Access Road)

Source: Constantine (2018)

6 History

6.1 Historical Work Completed by Previous Owners

Previous exploration work on the Project is summarized in Constantine Metal Resources Ltd. was formed out of Rubicon Minerals Corporation in 2006 with the primary purpose of exploring the Palmer Exploration Project. Constantine have completed a variety of exploration surveys and approximately 51,239 m of diamond drilling in 131 holes to the end of 2017. This work has led to the discovery of the massive sulfide deposits at the South Wall and RW Zones at the Glacier Creek prospect area and the AG Zones in the Nunatak prospect area.

A summary of previous exploration programs can be found in Table 6-5.

Total cumulative diamond drilling by the Company is 60,200 m in 156 drill holes to the end of 2018. Total cumulative diamond drilling on the Project by all operators to the end of 2018 is 67,745 m in 193 drill holes. All drilling has been summarized in Table 6-6.

Based on the encouraging exploration results to date, the Authors believe that continued delineation drilling is warranted and has the potential to increase the size of the known mineral resources laterally and to depth. There is also potential to discover additional mineralized zones within the greater Project area

6.1.1 1960s and 1970s

- Base metal sulfides and barite were first discovered in the Glacier Creek prospect area (Main and Upper Main occurrences) in 1969 by local prospector Merrill Palmer. Palmer staked the discoveries and continued to prospect the area in subsequent years.
- Barite was the original focus of exploration and prospecting led to the discovery of outcropping massive barite beds (+/- sphalerite/galena/sulfosalts) at the Nunatak Showing in eight distinct locations with individual barite lenses ranging in thickness from 0.5 to 6.0 m within a thick (>100 m) section of intensely QSP-altered basalt and rhyolite.
- A 91 kg (200 lb) bulk sample from one of three stacked barite beds at the Nunatak showing yielded an average grade of 11.84 oz/ton Ag and 0.092 oz/ton Au (Tobey 1988).
- Merrill Palmer arranged for tests to be conducted on bulk samples by B.P. Alaska Inc. and Lutak Trading & Stevedoring Company. Although the baritic material was determined to be suitable for production of drilling mud concentrates, none of the prospects were developed.

Discovery History

- From 1969 to 1971, the United States Geological Survey (USGS) completed regional mapping, which provided a geological framework for the Project (MacKevett et al., 1974).
- Subsequent grab samples collected by the USBM at the Nunatak showing graded up to 335.3 ppm Ag (10.8 oz/ton), 2.58 ppm Au (0.083 oz/ton), 2.38% Zn, 0.18% Cu, 2.0% Pb and 48% Ba (Still, 1991).

- In 1979, Anaconda Copper Company drilled the first three (3) diamond drill holes (totaling 801 m) on the Project. Although all holes failed to intersect the main mineralized barite and base metal sulfide horizons, one hole (GC-2) cored 426m of rock containing pyrite and sericite alteration, and the hole reportedly ended in siliceous sulfide breccia containing pyrite and sericite. Anaconda began a mapping program the following year (1980) in efforts to resolve structural problems during the drill program; however, Anaconda terminated the option before follow-up drilling.

6.1.2 1980s

- In the early 1980s, exploration successes at nearby Windy Craggy and Greens Creek improved the understanding and base metal potential at Palmer.
- In 1983, high-grade massive sulfide boulders up to 1.8 m (6 ft) in diameter, and grading up to 33% Zn and 2.5% Cu, were discovered at the base of a small ice sheet near Mount Henry Clay (Still et al., 1991). Twenty-six (26) samples of various boulders collected by the U.S. Bureau of Mines returned an average grade of **19.3% Zn, 1.0% Cu, 0.4% Pb, 38.2 g/t Ag, 0.22 g/t Au, and 20.6% Ba** (Still, 1984). The discovery of these boulders was followed up with four consecutive drill programs by operators Bear Creek Mining/Kennecott (1984 and 1985), Granges Exploration Inc. (1989), and Rubicon Minerals Corporation (1999). Over this period, a total of thirteen (13) holes were drilled and totaled 2,958 m of core. None of the drill programs located the source of the boulders.
- In the mid-late 1980s, Newmont Exploration Ltd. conducted exploration on the Project, and focused primarily on the Cap and Nunatak prospects. The Cap prospect was drilled by Newmont in 1988 and again by Rubicon in 1998, with the best intercept of the four (4) holes containing 23.2 m @ 134 g/t Ag within massive pyritic barite and baritic breccia. At the Nunatak prospect, a bulk sample (91 kg (200 lb) divided into thirteen separate samples) returned an arithmetic average grade of 11.84 oz/ton Ag and 0.092 oz/ton Au.

6.1.3 1990s

- In the early 1990s, ice retreat exposed an outcrop of massive sulfide in the Glacier Creek prospect area that is now known as the Little Jarvis occurrence. The best grades received by Kennecott from chip samples from the Little Jarvis occurrence contained up to 4.6 m @ 13.0% Zn, 7.0% Cu, 0.02 oz/ton Au, and 7.0 oz/ton Ag; (Wakeman, 1995); however, Rubicon Minerals Corp were unsuccessful at reproducing these grades, with their best grade containing 3.05 m @ 10.8 % Zn, 0.27% Cu, 0.17 ppm (0.005 oz/ton) Au, and 44.2 ppm (1.29 oz/ton) Ag.
- In 1999, Rubicon Minerals Corp. interpreted that the Little Jarvis occurrence was correlative with the Upper Main occurrence on the other side of the mountain to the southeast, which led to the discovery of the RW Zone. Semi-massive to massive sulfide and leached, oxidized equivalent of the RW Zone was intersected in six (6) drill holes and was open at depth. No additional drilling occurred on the Project until the acquisition by Constantine in 2006.
- Several geophysical surveys have been conducted on the Palmer Project over time, the most significant of which was a helicopter-borne magnetic-EM survey, planned by Kennecott in the mid-1980’s, and it covered most of the main mineral occurrences. A follow-up survey was conducted in 1991, and Cominco detailed three of the airborne EM surveys with TDEM (EM-37) ground surveys.

One of the TDEM surveys confirmed that an airborne EM anomaly 750 m eastward along strike of the mineral occurrences at the Glacier Creek prospect represented a significant conductor, with a geophysical signature consistent with that of a large massive sulfide deposit (Cominco, 1993). Cominco Alaska proposed three drill holes to test the different geophysical interpretations of the conductor (based on spatial orientation; flat lying versus steeply dipping), however, the holes were not drilled before Cominco's option lapsed.

- In 1993, Kennecott drilled one hole (P94-1) to test an interpretation that the conductive anomaly was flat-lying and in 1998, Rubicon Minerals drilled a second hole (RMC98-4). No significant mineralization was intersected in either hole. It was proposed that significant problems with locating the original survey grid may have been a factor in the holes missing their intended target.

6.1.4 Early 2000s

No significant field work was carried out on the Project.

6.2 Historical Work by Constantine Metal Resources Ltd.

6.2.1 2006 Field Season

Constantine Metal Resources Ltd. was formed in 2006 with the primary purpose of exploring the Palmer Project. Three (3) holes (CMR06-01 to 03) were drilled totaling 829 m, all of which targeted the eastern extension of RW Zone mineralization that was discovered by Rubicon Minerals Corporation in 1999. Two (2) drill holes intersected baritic massive sulfide mineralization, with grades including: 5.1 m @ 0.25% Cu, 11.18% Zn, 0.14 g/t Au, and 47.6 g/t Ag.

6.2.2 2007 Field Season

Seven (7) drill holes (CMR07-04 to 10) were drilled totaling 2,314 m, two of which targeted the Cap prospect and five targeted the Glacier Creek prospect area. Two holes in the Glacier Creek area contained significant massive sulfide intercepts in two separate horizons referred to the 'RW' and 'South Wall' zones including:

- *14.0 m @ 4.09% Cu, 7.35% Zn, 0.40 g/t Au and 50.9 g/t Ag, in hole CMR07-07 and*
- *24.2 m @ 1.21% Cu, 7.15% Zn, 0.78 g/t Au and 55.4 g/t Ag, in hole CMR07-09*

These two holes were most significant drilled to date and provided recognition of the potential for a major massive sulfide deposit, with all subsequent drilling focused entirely on deposit definition and expansion within the Glacier Creek prospect area.

6.2.3 2008 Field Season

Twelve (12) holes (CMR08-11 to 22) were drilled totaling 4,241 m, rapidly expanded the known extent of mineralization.

Between the 2006 and 2008 exploration programs, an 11 line-km grid was established down slope in vegetated cover along trend to the east of the South Wall Zones. The area covered the projected mineral horizons and was utilized for collection of soil samples as well as conducting geophysical surveys. Soil samples were collected at 25 m intervals along the 100 m spaced grid lines and identified several multi-element geochemical anomalies. Ground magnetic and CSAMT (Controlled Source Audio-Magneto

Telluric) geophysical surveys were completed over the grid area. Two additional CSAMT lines were completed over known, or suspected, RW Zone mineralization at higher elevations.

6.2.4 2009 Field Season

Ten (10) drill holes (CMR09-23 to 32) were completed totaling 4,561 m. The results of the 2008 and 2009 drilling rapidly expanded the known extent of mineralization and provided sufficient drill density to calculate a mineral resource estimate for the deposit (see below). Eight of the ten holes drilled in 2009 were surveyed using 3D downhole Time Domain Electro Magnetic (TDEM) geophysics, which proved to be effective at identifying copper-rich portions of South Wall Zone massive sulfide.

Regional geological field mapping and prospecting was completed over the federal lode mining claims throughout the 2007, 2008 and 2009 exploration programs and aided the understanding of the regional geological setting. Surface samples collected from the JAG showing, exposed 535 m to the southeast of the Nunatak showing on the southern slope of the Nunatak above South Saksaia Glacier with outcroppings of barite and massive galena + sphalerite, returned assays up to 10.4% Zn, 20% Pb, 537 g/t Ag, and 0.73 g/t Au (Wasteneys, 2009)

High-definition metallurgical and mineralogical work and benchmarking was completed on six core samples of mineralization from South Wall Zones I and II (SGS Canada Inc., 2009). The samples were chosen to represent the main styles of mineralization recognized in drilling to date. Although preliminary in nature, the work suggested that the mineralogy was simple, with coarse grained sulfides that would likely yield good recoveries and high-grade concentrates having low milling costs.

Following completion of the 2009 drill campaign, independent consultant Gary Giroux, P. Eng. of Giroux Consultants Ltd. was commissioned to prepare an NI43-101 compliant initial resource estimate for the Palmer project (Grieg and Giroux, 2010). The Inferred mineral resource estimate for the RW and South Wall Zones is outlined in Table 6-1 for a range of NSR (Net Smelter Return) cut-offs.

Table 6-1: 2010 Inferred Mineral Resource

Cut-off (NSR)	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb (%)	NSR (US)
$50	**4,750**	**1.84**	**4.57**	**29.1**	**0.28**	**0.15**	**$119.87**
$75	4,120	2.01	4.79	30.5	0.30	0.16	$128.49
$100	3,000	2.31	5.14	33.3	0.33	0.17	$143.75

Source: Giroux (2010)

Notes

1. Assumed metal prices for gold (Au), silver (Ag), copper (Cu), and zinc (Zn) were US$700/oz, US$12/oz, US$2.25/lb, US$0.85/lb, and estimated metal recoveries were 55%, 55%, 90%, 90% respectively.
2. The NSR formula equaled US$36.87 x Cu% + US$9.54 x Zn% + US$11.12 x Au g/t + US$0.18 x Ag g/t. NSR formula was based on assumed values for offsite costs, metal recovery, and metal prices. Offsite costs included transportation of concentrate, smelter treatment charges, and refining charges. An [2010] NSR cut-off grade of US$50 per tonne was considered appropriate for reporting the base case resource considering the likely underground mining extraction scenario envisioned for the project.
3. Forty-six (46) diamond drill holes were used in generating the 2010 geological model for the South Wall and RW Zones, with 26 of the holes (9,302 m) included in the resource estimate. Outlier assays were capped and all assays within the mineralized zones composited to 1.25 m lengths. Metal grades were estimated using inverse distance squared interpolation into a 3D block model with block dimensions of 10 x 10 x 5 m. Three dimensional geologic solids were constructed by Qualified Person, Darwin Green, VP-Exploration for Constantine and, in general, were limited to material grading > 0.5% Cu or > 2% Zn that could be demonstrated to be correlative with definable stratabound zones. As a general rule, solids were extended no more than 50 m up-dip, down-dip and along strike from any drill hole.

6.2.5 2010 Field Season

Ten (10) diamond drill holes (CMR10-33 to 42) were completed, totaling 4,017 m. Drilling resulted in the successful expansion of both the RW and South Wall mineralization zones and opened up expansion possibilities down dip and down slope on the South Wall mineralization.

Highlights included:

- *10.4 m @ 0.30% Cu, 4.18% Zn, 0.42% Pb, 81.60 g/t Ag and 0.87 g/t Au, in CMR10-34B (SWZI)*
- *7.10 m @ 2.10% Cu, 1,52% Zn, 16.80 g/t Ag and 0.18 g/t Au, in CMR10-035 (RW Zone)*
- *23.80 m @ 0.36% Cu, 2.95% Zn, 0.96% Pb, 123.10 g/t Ag and 0.82 g/t Au, in CMR10-38B (RW Oxide)*
- *20.80 m @ 1.03% Cu, 5.01% Zn, 11.30 g/t Ag and 0.14 g/t Au, in CMR10-040 (SWZI) and*
- *17.40 m @ 0.16% Cu, 2.25% Zn and 1.60 g/t Ag, in CMR10-040 (SWZIII)*

The program included both surface and downhole electro-magnetic (EM) geophysical surveys. Downhole EM surveys were completed on six of the ten holes drilled in 2010. Surface-based EM surveys totaled approximately 37-line km and covered areas immediately along trend from the 4.75 million tonne Inferred resource, as well as the Mount Henry Clay (MHC) prospect located 4.5 km to the west.

6.2.6 2011 and 2012 Field Seasons

No field work completed.

6.2.7 2013 Field Season

Ten (10) drill holes (CMR13-43 to 52) were completed totaling 3,747 m and targeted open edges of the South Wall and RW Zones, with step-out distances ranging from 30 to 100 m and elevations above the 1100 m level. The program was highly successful in expanding the mineralized zones with significant mineralization intersected in seven of ten holes (including five high-grade intersections of >20 m in width). The drilling also helped confirm a revised geological model for the South Wall environment, and better constrain the structural controls on the geometry and location of massive sulfide mineralization.

Highlights included:

- *21.71 m @ 2.36% Cu, 9.06% Zn, 0.13% Pb, 28.8 g/t Ag and 0.33 g/t Au, in CMR13-45*
- *20.58 m @ 0.92% Cu, 7.18% Zn, 0.25% Pb, 45.3 g/t Ag and 0.32 g/t Au, in CMR13-46 and*
- *24.66 m @ 2.02% Cu, 8.47% Zn, 31.7 g/t Ag and 0.51 g/t Au, in CMR13-49*

The 2013 program included surface geological mapping, borehole geophysical surveying of seven drill holes, completion on a M.Sc. thesis (Steeves, 2013), boulder sampling at Mount Henry Clay, metallurgical studies, initiation of access road construction, and numerous baseline environmental and geotechnical studies, community relations activities.

A total of 311 mineralized boulders were recorded over two days at the MHC prospect (ranging in size (length) from 7 cm to 240 cm. Sphalerite content averaged 24.6%, chalcopyrite 3.6%, barite 59%, and pyrite 19%. Of the 102 boulders mapped as Massive Sulfide, sphalerite content averages **37.1%, chalcopyrite**

4.2%, barite 35% and pyrite 27.3%. The largest concentration of mineralized boulders defined a roughly 50 m wide east-northeast to northeast trending corridor over approximately 250 m and suggested a bedrock source located to the southwest in the area with the greatest density of past drilling at MHC.

The 2013 metallurgical program, which included the first flotation test work done on the deposit, was carried out by SGS Canada Inc. (2013), in Vancouver, under the supervision and direction of metallurgists from Dowa. The test work demonstrated that the deposit responds very well to conventional metallurgy. Locked cycle flotation tests yielded smeltable copper and zinc concentrates, with high metal recoveries (e.g. 89.6% Cu, 84.9% Zn, 89.7% Ag, 75.0% Au) produced at moderate grind sizes.

Environmental work focused on data collection necessary for permitting an access road up Glacier Creek valley. This work included an aquatic biology survey, including fish presence or absence determinations, and wetlands delineation studies. A road layout consultant was also contracted to establish the road alignment and stream crossing requirements.

6.2.8 2014 Field Season

Sixteen (16) exploration drill holes (CMR14-53 to 68) and one geotechnical hole (GT14-01) totaling 9,796 m (32,136 feet) targeted the deeper portion of the South Wall mineral resource, and the Option 9 proposed exploration drift alignment. The program resulted in the intersection of a major new zone of massive sulfide in the South Wall area corresponding to the modeled SW EMZ geophysical target and on trend of the existing mineral resource. Massive sulfide and/or sulfide-rich massive barite was intersected in five drill holes within the SW EMZ target (CMR14-54, 63, 64, 65, 66) defining a strike length of 225 m and a vertical distance of 150 m.

The thickest mineralization was intersected on the east side of the SW EMZ target with:

- *22.10 m @ 2.48% Cu, 4.05% Zn, 24.00 g/t Ag and 0.39 g/t Au, in CMR14-54 and*
- *89.00 m @0.79% Cu, 5.03% Zn, 21.10 g/t Ag and 0.31 g/t Au, in CMR14-65*

The 89 m intersection was the widest drilled to date on the Project (approximately twice the length of the next widest intersection with a minimum true width of 65 to 75 m). Mineralization was interpreted to be an extension of SWZII and SWZIII and extends the total plunge length of continuous South Wall mineralization to 700 m. Deep step-out holes on the SW EMZ target (holes CMR14-56, 58 and 62) revealed the presence of a steep, east-southeast trending fault (the ‘Kudo fault’) which was interpreted to offset the down-dip extension of the massive sulfide horizon.

The 2014 field program included surface geological mapping and geochemical sampling, borehole geophysical surveying of six drill holes, numerous baseline environmental and geotechnical studies, community relations activities, and construction of the new gravel 3.6 km Glacier Creek access road and supply yard to connect the core of the Project to the existing Haines Borough logging road network.

Geotechnical studies focused on data needs to support design and permitting of an extended access road and preliminary assessment of underground exploration access. Work included the completion of a sub-horizontal geotechnical drill hole (GT14-01) at the Option 9 exploration portal site, hydrology test work and groundwater studies (SRK, 2014), surface mapping of fractures and joints, acquisition of LIDAR data for detailed topography, avalanche studies, slope stability analysis and road design work. Surface samples collected by the Company at the Nunatak showing assayed up to 1275 g/t Ag, 1.0 g/t Au, 3.7% Pb, and 1.6% Zn (CEM, 2014).

Environmental studies consisted of both long-term baseline data collection and near-term data needs to support permitting in 2015. Work included water quality sampling, aquatic survey work, wildlife observations and habitat mapping, weather station installation, acid base accounting, cultural resource studies (archeology), and hydrology work.

Following completion of the 2014 drilling campaign, an independent NI 43-101 compliant mineral resource estimate was prepared by James N. Gray, P. Geo., of Advantage Geoservices Ltd. (Gray and Cunningham-Dunlop, 2015). The Inferred Mineral Resource for the RW and South Wall Zones is tabulated below in Table 6-2 for a range of NSR (Net Smelter Return) cut-off values based on assumed underground mining and milling costs. The resource utilized a base case cut-off of $75 per tonne. A comparison to the 2010 Resource is shown in Table 6-3.

Table 6-2: 2015 Inferred Mineral Resource

Cut-off (NSR)	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	NSR (US)	CuEq (%)	ZnEq (%)
$60	9,133	1.30	5.00	30.2	0.30	$138.61	3.03	11.83
$65	8,786	1.34	5.08	30.8	0.31	$141.61	3.10	12.11
$70	8,516	1.37	5.15	31.1	0.31	$143.95	3.15	12.31
$75	**8,125**	**1.41**	**5.25**	**31.7**	**0.32**	**$147.40**	**3.23**	**12.61**
$80	7,863	1.43	5.33	32.2	0.33	$149.75	3.28	12.80
$85	7,638	1.45	5.40	32.6	0.33	$151.72	3.32	12.97
$90	7,389	1.48	5.45	33.1	0.34	$153.88	3.37	13.17
$95	7,072	1.51	5.53	33.7	0.34	$156.62	3.43	13.39

Source: Constantine (2015)

Table 6-3: Comparison to 2010 Resource above US$75 2015 NSR

Resource	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	2015 NSR (US)	CuEq (%)	ZnEq (%)
2015	8,125	1.41	5.25	31.7	0.32	$147.40	3.23	12.61
2010	4,642	1.87	4.63	29.6	0.29	$159.32	3.49	13.64
Difference	+75%	-25%	+13%	+7%	+10%	-7%	-7%	-8%

Notes

1. Assumed metal prices are US$2.75/lb for copper (Cu), and US$1.00/lb for zinc (Zn), US$1200/oz for gold (Au), US$18/oz for silver (Ag). Estimated metal recoveries are 89.6% for copper, 84.9% for zinc, 75% for gold (61.5% to the Cu concentrate and 13.5% to the Zn concentrate) and 89.7% for silver (73.7% to the Cu concentrate and 16% to the Zn concentrate) as determined from the 2013 metallurgical locked cycle flotation tests.
2. The 2015 NSR equals (US$45.69 x Cu% + US$11.70 x Zn% + US$25.04 x Au g/t + US$0.43 x Ag g/t). NSR formula is based on assumed values for offsite costs, metal recovery, and metal prices. Offsite costs include transportation of concentrate, smelter treatment charges, and refining charges.
3. An [2015] NSR cut-off grade of US$75 per tonne was considered appropriate for reporting the base case resource considering the likely underground mining extraction scenario envisioned for the project.
4. Eighty-two (82) diamond drill holes were used in generating the geological model for the South Wall and RW zones, 48 of which intersect the interpreted mineralized zones in 19,000 m of core. Outlier assays were capped, and assays were composited to 1.5 m lengths within the boundaries of interpreted mineralized zones. Three dimensional geologic solids were constructed by Darwin Green, Vice President of Exploration and reviewed by Ian Cunningham-Dunlop, P. Eng., Vice President, Advanced Projects, and, in general, were limited to material grading > 0.5% Cu or > 2% Zn that could be demonstrated to be correlative with definable stratabound zones. As a general rule, solids were extended no more than 50 m up-dip, down-dip and along strike from a drill hole; the Inferred Mineral Resource includes only mineralization within 75 m of a drill hole. A total of five solids were constructed for sulfide mineralization: South Wall Zone 1, South Wall Zone 2-3, South Wall EM Zone, RW West, and RW East. Metal grades were estimated using inverse distance cubed interpolation into a 3D block model with block dimensions of 6x6x6 m.

Source: Constantine (2018)

The 2015 resource estimate significantly increased the size of the deposit, highlighting the success of the prior drill campaigns and the growing potential of the project. The resource was interpreted as open for expansion in most areas with the thickest part of the deposit located at the down-dip limit of the South Wall Zone. Barium content within the resource averaged approximately 13 to 15%, equating to a barite mineral content of approximately 22 to 25% by weight.

6.2.9 2015 Field Season

Ten (10) drill holes (CMR14-56Ext, CMR15-69 to 76) totaling 7,735.8 m (25,380 feet) were completed at the South Wall resource area. The primary objective was the expansion of the South Wall Zone with focus on targets surrounding the 2014 South Wall EM Zone discovery area, and areas south of the Kudo fault to define extensions of the EM Zone at the Lower Offset target. South Wall EM Zone mineralization was successfully intersected in three holes, extending the known extent of the mineralized system approximately 100 m east (CMR15-72 and CMR15-73) and 65 m up dip (CMR15-75).

Significant intersections include:

- *4.2 m @ 0.5% Cu, 3.98% Zn, 60.4 g/t Ag and 0.65 g/t Au, in hole CMR15-72*
- *3.0 m @ 2.32% Cu and 14.9 g/t Ag, in hole CMR15-75*

Drill hole CMR15-69 was planned to test eastern extensions of the EM Zone and intersected EM Zone equivalent stratigraphy and mineralization at greater depth, south of the North Kudo fault and within the Lower Offset target area and returned 7.2 m @ 0.43% Cu and 0.46% Zn and intense footwall QSP alteration, in hole CMR15-69. Three holes planned to test the Lower Offset target area, south of the Kudo fault, did not intersect South Wall mineralization, alteration or obvious mineralized horizon stratigraphy suggesting that the postulated movement on the Kudo Fault was incorrect. Borehole and surface EM geophysical surveys were completed on eight holes and over 7.5 surface line km. Data collected suggested that conductors are beyond the depth of the deepest holes drilled and therefore the exact definition of these conductors remains ambiguous.

The 2015 program also included regional geological mapping at the Glacier Ck (South Wall)/Red Creek/Flower Mountain/Jasper Mountain areas, detailed geological mapping at the Little Jarvis prospect areas in support of a B.Sc. thesis, surface electromagnetic and seismic geophysical surveys, downhole electromagnetic surveys of eight drill holes, exploration access road construction, and ongoing engineering, environmental, geotechnical, permitting and community relations-related work.

6.2.10 2016 Field Season

Seven (7) diamond drill holes totalling 1,967.7 m were completed, including four holes (CMR16-78 to 81B)/1,464.7 m of exploration drilling on the South Wall QSP, Pump Valley and Cap targets and three holes (GT16-02 to 04)/502.0 m of geotechnical drilling. No significant results were returned from drilling at the South Wall or Pump Valley, while drilling at the Cap target successfully expanded the known extent of the mineral system with a 21-m section of chert +/- semi-massive pyrite returning 3.1 g/t Ag over 7.7 m, including 0.5 m of 10 g/t Ag, in hole CMR16-79.

Other field work included downhole electromagnetic surveys of three drill holes, regional geological mapping at the Nunatak prospect area, structural studies on the South Wall resource area, access road construction, and ongoing engineering, environmental, geotechnical, permitting and community relations-related work.

The detailed mapping and sampling work at the Nunatak prospect refined the understanding of the geological stratigraphy and the potential controls on mineralization. Notably, the work revealed the width of the barite-sulfide beds to be thicker than previously thought, with true-width sections measuring up to 5 m or more at both the upper and lower showings. Chip sampling across the beds has yielded up to **4.7 m @ 128 g/t Ag, 0.49 g/t Au, 0.29% Zn, 0.59% Pb and 39.1% Ba**. Individual samples from the prospect area assay as high as **778 g/t Ag, 0.89 g/t Au, 4.04% Zn, 3.53% Pb and 44.3% Ba**. The inferred surface trace of the folded upper mineral horizon from the northernmost outcrop exposure to the southernmost outcrop exposure was found to be ~ 275 m in length, and extending over an elevation range of ~ 90 m. It was interpreted that the scale of the alteration system at Nunatak, coupled with the extent of the barite-dominant mineralization, supported potential for a significant massive sulfide system at Nunatak and a recommendation for follow-up drilling was made.

6.2.11 2017 Field Season

Thirty-two (32) diamond drill holes totaling 10,631 m, including 26 holes (CMR17-82 to 107)/9,221.9 m of exploration drilling and six holes (GT17-05 to 10)/1,409.1 m of geotechnical drilling, were completed. The top priority was given to the preliminary testing of the Nunatak Prospect Area, as well as, follow-up drilling at the Cap target, and ongoing resource infill and expansion drilling of the South Wall Zone II-III zones. The results from South Wall Zones II/III were very encouraging with holes CMR17-82, 84 and 86 intersecting wide intervals of chalcopyrite- and sphalerite-rich baritic massive sulfide overlying massive pyrite and chalcopyrite which dramatically increased the width and grade of mineralization on section 4211000E.

Holes CMR17-95, -97, and -100 along section 421050E also successfully extended South Wall Zones II/III by 50 to 60 m to the west and confirmed continuity of wide high-grade mineralization over a dip length of approximately 90 m.

SW Zone highlights included:

- 45.4 m @ 2.5% Cu, 7.4% Zn, 39 g/t Ag and 0.3 g/t Au, in CMR17-82
- including 10.9 m @ 6.2% Cu and 13.8% Zn
- 18.7 m @ 2.3% Cu, 6.9% Zn, 33 g/t Ag and 0.3 g/t Au, in CMR17-84
- 13.4 m @ 1.7% Cu, 5.4% Zn, 11 g/t Ag and 0.2 g/t Au, in CMR17-88
- 14.5 m @ 1.9% Cu, 7.5% Zn, 66 g/t Ag and 0.4 g/t Au, in CMR17-97

The Nunatak prospect area was drilled for the first time with 13 drill holes that resulted in the discovery of the new AG Zones. Mineralization was intersected over an area measuring approximately 225 m x 50 m, and over a vertical distance of approximately 200 m (vertical dip length ~ 275 m) and remained open in all directions. Mineralization consists of stacked massive and semi-massive sulfide and barite, and feeder-style stringers and replacement, including a high-grade silver-gold upper zone, and a zinc-rich lower zone.

Significant upper silver-gold zone intersections include:

- 9.2 m @ 312 g/t Ag and 0.9 g/t Au, in CMR17-89
- 24.6 m @ 260 g/t Ag, 0.5 g/t Au, 1.4% Zn and 0.5% Pb, in CMR17-94
- Including 10.3 m @ 461 g/t Ag, 0.9 g/t Au, 2.0% Zn and 0.7% Pb
- Including 2.7 m @ 1214 g/t Ag and 1.3 g/t Au
- 3.8 m @ 256 g/t Ag and 1.1 g/t Au, in CMR17-96

Significant lower zinc zone intersections include:

- 17.8 m @ 11.7% Zn, 0.2% Cu, 6.3 g/t Ag and 0.2 g/t Au, and
- 20.4 m @ 9.9% Zn, 0.2% Cu, 14.4 g/t Ag and 0.1 g/t Au, and
- 41.3 m @ 5.8% Zn, 0.2% Pb, 0.1% Cu, 9 g/t Ag and 0.1 g/t Au, in CMR17-96

The 2017 field program also included downhole electromagnetic surveys of three drill holes, a 1,137 line km SkyTEM airborne electromagnetic and magnetic survey, ongoing geological mapping and rock sampling at the Nunatak prospect area, structural studies at the South Wall resource area, detailed geological mapping at the AG Zone areas in support of an Applied Master's thesis project, a subglacial sampling program at Mount Henry Clay, access road construction, and a wide range of engineering, environmental, geotechnical, permitting, and community relations-related work. The engineering work in 2017 included a focused effort on evaluation of underground exploration options and collecting data necessary for design, decision making, and potential future permitting.

A preliminary scoping level barite metallurgical test program on the Glacier Creek deposit was initiated in March 2018 with the primary objective of determining if the deposit can produce a marketable barite concentrate as a co-product for the copper-zinc-gold-silver flotation process. A secondary objective was to collect additional copper-zinc flotation data based on fresher sample material than was used in the previous metallurgical program. Detailed testwork included: sample preparation; sample characterization; grindability testing; copper and zinc rougher/cleaner/locked cycle testing with up to ten (10) preliminary cleaner flotation tests and six (6) locked cycle tests; followed by barite rougher/cleaner/locked cycle testing with up to eight (8) preliminary cleaner flotation tests and six (6) locked cycle tests. The final barite concentrates also underwent QEMSCAN™ mineralogical testing. SGS (2018) reached the following conclusions based on their test program:

- Mineralogy indicated that the economically recoverable minerals in the High Ba Composite head sample are chalcopyrite (5.07%), sphalerite (13.4%), barite (40.9%), and less than 1% of galena (0.66%)
- A Bond Ball Mill Work Index test found the sample to be categorized as very soft with a BWI of 6.3 kWh/t
- The final copper concentrate projected recovery was 88.9% at a grade of 24.5% Cu and a final zinc concentrate projected recovery was 93.1% at a grade of 61.3% Zn
- The developed flowsheet for barite recovery involved a pyrite pre-float prior to barite flotation stage, and the barite rougher concentrate was cleaned in three stages, producing a final barite concentrate projected recovery of 91.1% at a grade of 52.3% Ba (88.8% $BaSO_4$). Analysis of a final concentrate by the more reliable XRF76V method reported a higher final concentrate grade of 55.9% Ba (95.0% $BaSO_4$) and
- Mineralogy indicates that the final barite concentrate is 95.6% barite with the remainder comprised of various silicates and other minerals. Barite is 99.8% liberated with very trace attachments of 0.2%. Silicates and other minerals in the concentrate are also well liberated.

Following completion of the 2017 drilling campaign, an independent mineral resource estimate was prepared by James N. Gray, P. Geo., of Advantage Geoservices Ltd. (Gray and Cunningham-Dunlop, 2018). The resource incorporates all exploration drilling in the Palmer deposit area completed to the end of 2017. The Indicated and Inferred Mineral Resource for the RW and South Wall Zones is tabulated below in Table 6-4 for a range of NSR (Net Smelter Return) cut-off values. Based on assumed underground mining and milling costs, the resource utilizes a base case cut-off of $75 per tonne. The resource had an effective date of September 27th, 2018 based on a data cut-off of May 1st, 2018.

Table 6-4: 2018 Palmer Deposit Mineral Resource Estimate at a $75/t NSR Cut-off

Category	Tonnes	Cu	Zn	Ag	Au	Barite	ZnEq*	CuEq*
	(1,000s)	(%)	(%)	(g/t)	(g/t)	($BaSO_4$ %)	(%)	(%)
Indicated	4,677	1.49	5.23	30.8	0.30	23.9	11.67	3.84
Inferred	5,338	0.96	5.20	29.2	0.28	22.0	9.90	3.26
Contained Metal								
Category		**Cu**	**Zn**	**Ag**	**Au**	**Barite**	**ZnEq**	**CuEq**
		(M lbs)	(M lbs)	(M oz)	(K oz)	(K tonnes)	(M lbs)	(M lbs)
Indicated		154	539	4.6	45.1	1,118	1,203	396
Inferred		113	612	5.0	48.1	1,174	1,166	383

Source: Constantine (2018)

Notes

1. The cut-off date for drill data included in the resource is May 1st, 2018.
2. Net Smelter Return (NSR) equals (US$16.01 x Zn% + US$48.67 x Cu% + US$23.45 x Au g/t + US$0.32 x Ag g/t). NSR formula is based on estimated metallurgical recoveries, assumed metal prices, and assumed offsite costs that include transportation of concentrate, smelter treatment charges, and refining charges.
3. Assumed metal prices are US$1.15/lb for zinc (Zn), US$3.00/lb for copper (Cu), US$1250/oz for gold (Au), US$16/oz for silver (Ag).
4. Estimated metal recoveries are 93.1% for zinc, 89.6% for copper, 90.9% for silver (70.8% to the Cu concentrate and 20.1% to the Zn concentrate) and 69.6% for gold (49.5% to the Cu concentrate and 20.1% to the Zn concentrate) as determined from metallurgical locked cycle flotation tests completed in 2018.
5. Barite is not included in the NSR value.
6. Zinc equivalent (ZnEq%) and Copper equivalent (CuEq%) values calculated based on the NSR formula above plus an assumed net-value for barite as described below (e.g. CuEq = (total NSR value + $BaSO_4$ net-value)/US$48.67.
7. $BaSO_4$ net-value equals US$0.566 x $BaSO_4$% (e.g. a resource grade of 24% $BaSO_4$ x $0.566 = US$13.6 per tonne or 0.85% ZnEq). Formula based on barite recovery of 91.1% from metallurgical tests, assumed wholesale drilling-grade barite price in nearest North American markets of US$227/metric tonne, and assumed all-in transportation cost of US$150/tonne.
8. One hundred and eight exploration (108) diamond drill holes for 44,900 m and geological surface mapping were used to generate the geological and structural model for the South Wall and RW zones. Sixty (60) of the holes intersect the interpreted mineralized solids. Outlier assays were capped and all assays within the mineralized zones composited to 1.5-m lengths. Metal grades were estimated using inverse distance cubed interpolation into a 3D block model with block dimensions of 6 x 6 x 6 m. Density was estimated by inverse distance squared interpolation, with unique density values determined by conventional analytical methods for virtually all assay samples. Three dimensional geologic solids were constructed by Darwin Green, Vice President of Exploration and reviewed by Ian Cunningham-Dunlop, P.Eng., Vice President, Advanced Projects, and, in general, were limited to material grading > 0.5% Cu or > 2% Zn that could be demonstrated to be correlative with definable stratabound zones. As a general rule, solids were extended no more than 50 m up-dip, down-dip and along strike from a drill hole except where geology supports extension in the plunge direction of mineralization. A total of four solids were constructed for sulfide mineralization: South Wall Zone 1, South Wall Zone 2-3-EM, RW West, and RW East.
9. Indicated Resources include only a portion of the upper part of the South Wall Zone, where drill density and confidence in the geological model are highest. Indicated Mineral Resource blocks meet the criteria of being a minimum 25-m distance away from the outer edge of the mineralized geological solid, estimated by a minimum of three holes, and have an average distance to three holes of less than or equal to 50 m; remaining estimated blocks are classified as Inferred Mineral Resource.

The 2018 mineral resource highlights include:

- Indicated Resource of 4,677,000 tonnes grading 11.67% zinc equivalent (3.84% CuEq). This represented the first Indicated Resource for Palmer, and accounts for 47% of the total resource.
- Inferred Resource of 5,338,000 tonnes grading 9.90% zinc equivalent (3.26% CuEq).
- First resource to report barite mineralization for the Palmer deposit, highlighting the opportunity for barite to contribute value as an industrial mineral co-product.

6.2.12 2018 Field Season

Constantine completed exploration, definition and geotechnical drill programs during 2018 field season. The Company drilled 30 new diamond drill holes totaling 10,359 m since the last NI 43-101 technical report (Gray and Cunningham-Dunlop, 2018) for a cumulative total of 67,745 m in 193 diamond drill holes since start of drilling on the Project in 1979. Drilling focused on the Palmer deposit, the Nunatak-AG Zone, and the Boundary Prospect.

Infill drilling at the Palmer deposit was successful with holes CMR18-108/CMR18-108W, CMR18-111 and CMR18-113 demonstrating continuous zinc and copper mineralization down-dip from historic hole CMR14-59 in the western down-plunge portion of SWZ II/III. Drill hole CMR18-129, in the eastern up-plunge portion for SWZ II/III, also intersected 80 meters of stringer-style mineralization. Significant 2018 SW Zone II/III assay intersections include:

- 15.5 m @ 4.8% Zn, 1.6% Cu, 24.6 g/t Ag, 0.1 g/t Au, in hole CMR18-108, including
- 4.1 m @ 15.9% Zn, 0.3% Cu, 5.3 g/t Ag, and
- 6.1 m @ 3.6% Cu, 0.3% Zn, 56.0 g/t Ag, and 0.2 g/t Au
- 1.3 m @ 1.7% Zn, 0.2% Cu, 7.1 g/t Ag, and 0.1 g/t Au, in hole CMR18-111
- 10.9 m @ 1.8% Zn, 0.3% Cu, 6.1 g/t Ag, in hole CMR18-113, including
- 5.3 m @ 1.7% Zn, 0.5% Cu, 9.9 g/t Ag
- 12.2 m @ 3.1% Zn, 0.4% Cu, 7.0 g/t Ag, 0.1 g/t Au, and
- 6.0 m @ 3.1% Zn, 0.4% Cu, 5.2 g/t Ag, 0.1 g/t Au, and
- 3.9 m @ 1.1% Zn, 1.4% Cu, 17.0 g/t Ag, 0.1 g/t Au, in hole CMR18-129

The Kudo Wedge Target, a postulated offset of the Palmer deposit between the Kudo North and Kudo Main Faults was tested with drill hole, CMR18-123, a 150 m deep eastern step-out from CMR14-69 and intersected 15 meters of jasper/chert at the projected target horizon along with strong footwall alteration but with no significant base metal mineralization.

The 2018 drilling program on the AG Zone deposit was highly successful and effectively tripled the strike length and vertical extent of mineralization from 180 m x 50 m to a minimum of **550 m x 250 m**. Mineralization at the AG Zone deposit consists of massive and semi-massive sulfide and barite, and feeder-style stringers and replacement, including a high-grade silver-gold upper zone, and a zinc-rich lower zone. Highlights from 2018 AG Zone deposit drill holes included:

- 4.8 m @ 436 g/t Ag, 1.3 g/t Au, 3.6% Zn, 1.6% Pb and 61.6% barite, and
- 12.5 m @ 217 g/t Ag, 1.8 g/t Au, 5.2% Zn, 0.7% Pb and 29.7% barite, in CMR18-109
- 43.3 m @ 143 g/t Ag, 0.5 g/t Au, 6.5% Zn, 2.5% Pb and 41.1% barite, in CMR18-110, incl.
- 28.8 m @ 141 g/t Ag, 0.5 g/t Au, 9.0% Zn, 3.5% Pb and 21.5% barite
- 21.1 m @ 92 g/t Ag, 0.5 g/t gold, 1.0% Zn, 0.4% Pb, 55% barite, in CMR18-114
- 14.0 m @ 163 g/t Ag, 0.5 g/t Au, 5.6% Zn, 1.0% Pb, 60.7% barite, in CMR18-125, incl.

- 4.1 m @ 336 g/t Ag, 0.6 g/t Au, 14.9% Zn, 2.3% Pb, 67.0% barite
- 6.8 m @ 247 g/t Ag, 0.8 g/t Au, 5.5% Zn, 2.8% Pb, 69.6% barite, and
- 34.4 m @ 152 g/t Ag, 0.4 g/t Au, 1.6% Zn, 0.5% Pb, 63.6% barite, in CMR18-128
- 33.5 m @ 98 g/t Ag, 0.4 g/t Au, 5.0% Zn, 1.1% Pb, 0.2% Cu, 41.5% barite, in CMR18-130
- 14.4 m @ 23 g/t Ag, 0.2 g/t Au, 5.5% Zn, 0.2% Pb, 0.4% Cu, in CMR18-132, incl.
- 3.4 m @ 44 g/t Ag, 0.1 g/t Au, 10.8% Zn, 0.2% Pb, 0.4% Cu

Drilling at the new Boundary Prospect Area with four drill holes (CMR18-117/119/121/122) successfully intersected the targeted contact between graphitic argillite and QSP-altered rhyolite and pyrite stringers with local barite, sphalerite, chalcopyrite and sulfosalt mineralization. While massive sulfide mineralization was not encountered and the EM anomalies may reflect graphitic argillite, barite-sulfide stringer and thin massive barite mineralization was intersected, the source of the high-grade boulders and the massive hydrothermal carbonate still remains a target and the prospective stratigraphy, strong and widespread hydrothermal alteration of the rhyolite with trace base and precious metal mineralization distributed throughout remains encouraging.

The 2018 field program also included downhole electromagnetic surveys of three drill holes, a 1,137-line km SkyTEM airborne electromagnetic and magnetic survey, ongoing geological mapping and rock sampling at the Nunatak prospect area, structural studies

6.2.13 Ongoing Exploration Access Road Construction

The Glacier Creek access road was extended to the head of Glacier Creek valley in 2018, for a total length of 5.9 km to provide access to MHT fee-simple lands near the base of the South Wall resource (Figure 6-1). Construction was completed in three phases, with 0.6 km completed in 2015, 1.4 km in 2016, 0.5 km in 2017, and 0.92 km in 2018. The road is single-lane with a 4.5-m wide running surface and passing pull-outs spaced approximately 300 m apart. Laydown areas for equipment and supply storage were also built.

The Glacier Creek access road connects with existing gravel logging roads to the all season paved Haines highway. Total road distance from the end of Glacier Creek road to Haines is approximately 60 km.

Figure 6-1: Glacier Creek Access Road, upper Glacier Creek Valley

Source: Constantine (2018)

6.2.14 Ongoing Geotechnical and Engineering

Many geotechnical and engineering studies have been completed in support of assessing options for advanced exploration and potential future preliminary economic assessment to feasibility level analysis. Evaluation of a conceptual exploration drift for continued drill expansion and drill definition on the deeper portion of the existing resource was a focus of several of the studies. Key studies completed to date have included:

- geotechnical drilling for rock quality, hydrogeology, and environmental geochemistry
- shallow seismic surveys for determination of overburden depth
- options assessment of potential exploration portal site and access drift locations
- avalanche studies and protection berm design
- civil engineering for road alignments and water collection systems
- soils studies and infiltration tests
- hydrogeology studies and modelling ground water flow to conceptual exploration drift
- glacier ice depth studies, and
- scoping level assessment of potential infrastructure sites for mining operations

6.2.15 Summary of Historic Work

Constantine Metal Resources Ltd. was formed in 2006 with the primary purpose of exploring the Palmer Exploration Project. Constantine have completed a variety of exploration surveys and approximately 51,239 m of diamond drilling in 131 holes to the end of 2017. This work has led to the discovery of the massive sulfide deposits at the South Wall and RW Zones at the Glacier Creek prospect area and the AG Zones in the Nunatak prospect area.

A summary of previous exploration programs can be found in Table 6-5.

Total cumulative diamond drilling by the Company is 60,200 m in 156 drill holes to the end of 2018. Total cumulative diamond drilling on the Project by all operators to the end of 2018 is 67,745 m in 193 drill holes (Table 6-6).

Based on the encouraging exploration results to date, the Authors believe that continued delineation drilling is warranted and has the potential to increase the size of the known mineral resources laterally and to depth. There is also potential to discover additional mineralized zones within the greater Project area.

Table 6-5: Summary of Previous Exploration Programs on the Palmer Project

Year	Company	Work Completed	Area/ Prospect
1969	Merrill Palmer, prospector	Prospecting. Discovery of Palmer Main Zone base metal and barite occurrences	Glacier Creek Prospect
1969-1971	United States Geological Survey	Regional government mapping	Skagway B-3 and B-4 Quadrangles
1971-1977	Alyu Mining Corporation & B.P. Alaska Inc.	Barite flotation and recovery tests	Glacier Creek Prospect
1979-1980	Anaconda Copper Company	Diamond drilling (3 holes/801m). Geological mapping	Glacier Creek Prospect
1980-1983	Southeastern Minerals	Prospecting and sampling. Discovery of several new base metal and barite occurrences.	Property-wide
1983-1985	Bear Creek Mining Company (Exploration Division of Kennecott)	Diamond drilling (7 holes/1,720m at MHC). Ground and airborne geophysics (mag & EM). Ground penetrating radar used to determine ice thickness. Geological mapping.	Property-wide (focus on Mount Henry Clay Prospect)
1983-1986	Alaska Division of Geological and Geophysical Surveys & United States Bureau of Mines	Geological mapping of the Porcupine Mining area. Sampling and study of Palmer mineral occurrences.	Property-wide
1987-1989	Newmont Exploration	Diamond drilling (4 holes/419m). Detailed mapping. Rock and soil sampling.	Cap, Nunatak, and Glacier Creek Prospects
1989	Granges Exploration Ltd.	Diamond drilling (4 holes/932m)	Mount Henry Clay Prospect
1990-1993	Cominco Alaska	Time Domain Electro Magnetic (TDEM; EM-37) ground geophysics survey. Geological mapping and prospecting.	Glacier Creek Prospect, Red Creek and Gullies

Year	Company	Work Completed	Area/ Prospect
1993-1997	Kennecott	Diamond drilling (3 holes/823m)	Glacier Creek Prospect
1998-2000	Rubicon Minerals Corporation	Diamond drilling (14 holes/2,769m). M.Sc. Thesis (Green, 2001). Geological mapping and prospecting.	Property-wide
2004	Toquima Minerals Corp.	Geological mapping. Rock and soil sampling.	Glacier Creek Prospect
2006	Constantine Metal Resources Ltd.	CEM formed. Diamond drilling (3 holes/829m). Started 11 line-km grid cutting.	Glacier Creek Prospect
2007		Diamond drilling (7 holes/2,314m). Regional field mapping and prospecting on federal claims. Completed 11 line-km grid cutting.	Glacier Creek Prospects
2008		Diamond drilling (12 holes/4,395m; 2 holes abandoned at <100m depth). Regional field mapping and prospecting (federal claims).	Property-wide (Cap)
2009		Diamond drilling (10 holes/4,643m). Metallurgically-focused high-definition mineralogical work and benchmarking. Downhole EM surveys. Limited regional field mapping and prospecting (federal claims).	Glacier Creek Prospect, Property-wide
2010		Diamond drilling (10 holes/4,017m). Surface and downhole EM surveys (surface, totaling approx. 37-line km).	Glacier Creek Prospect, Property-wide
2011-2012		No field work completed	
2013	Constantine Metal Resources Ltd. & Dowa Metals and Mining Co. Ltd.	CEM-Dowa JV formed. Diamond drilling (10 holes/3,745m). Surface and downhole EM surveys. M.Sc. thesis (Steeves, 2013). Surface seismic surveys. MHC boulder sampling. Metallurgical testing. Baseline environmental and geotechnical studies.	Glacier Creek Prospect, Property-wide (MHC)
2014		Diamond drilling (16 exploration holes and 1 geotechnical hole/7,736m). Downhole EM surveys. LiDAR survey. Geotechnical studies (hydraulic/groundwater testing and slope stability analysis). Avalanche studies. Baseline environmental studies. Construction of 3.6km access road from existing road network to prospect area.	Glacier Creek Prospect, Property-wide (Cap, Nunatak, MHC reconnaissance)
2015		Diamond drilling (10 exploration holes and 1 geotechnical hole/9,796m). Surface and downhole EM surveys. Detailed mapping. Baseline environmental, geotechnical, permitting and community-relations work. Ongoing access road construction (0.6km).	Glacier Creek Prospect (South Wall, Red Ck, Flower Mt., Jasper Mt), Property-wide (Little Jarvis)
2016		Diamond drilling (4 exploration holes and 3 geotechnical holes/1,968m). Downhole EM surveys. Structural Studies. Regional mapping. Baseline environmental, geotechnical, permitting	Glacier Creek Prospect, Property-wide (SW QSP, Pump Valley, Cap, Nunatak)

Year	Company	Work Completed	Area/ Prospect
		and community-relations work. Ongoing access road construction (1.4km).	
2017		Diamond drilling (26 exploration holes and 6 geotechnical holes/10,631m). Surface and downhole EM surveys. Airborne EM and Mag Survey (1,137-line km). Applied M.Sc. Thesis Project (Doherty, 2018). Structural Studies. Detailed mapping. Baseline environmental, geotechnical, permitting and community-relations work. Ongoing access road construction (0.5km).	Glacier Creek Prospect, Nunatak Prospect, Property-wide (Cap, MHC)
2018		Diamond drilling (28 exploration holes and 2 geotechnical holes, totaling 10, 359m). Geological mapping and rock sampling. Structural analysis (J. Proffett). GPR Glacial Surveys. Geotechnical studies (incl. drilling, surface mapping, avalanche modelling, hydraulic/groundwater testing). Metallurgical Testing. Baseline environmental studies. Construction of a 920 m switchback road extension to terminus of Saksaia Glacier	Glacier Creek Prospect, Nunatak Prospect, Property-wide (Boundary)

Source: Constantine (2018)

Table 6-6: Summary of Previous Drill Programs on the Palmer Project

Year	Hole_#	Company	Area	Hole_ID	Length_m	nlength_m
1979	3	Anaconda	Main	GC-01 to GC-03	801	**801**
1884	2	Kennecott - Bear Creek Mining	MHC	K84-01 to K84-02	596	**1,397**
1985	5	Kennecott - Bear Creek Mining	MHC	K85-03 to K85-07	1,129	**2,526**
1989	4	Granges	MHC	G89-08 to G89-11	932	**3,458**
1994	3	Kennecott - Bear Creek Mining	EM-37/Main/Jarvis	P94-01 to P94-03	800	**4,258**
1998	4	Newmont Mining	Main/Cap	MZ-01, CAP-01 to Cap-03	419	**4,677**
1998	6	Rubicon Minerals Corp.	Cap, Main, 737	RMC98-01 to RMC98-04	992	**5,670**
1999	10	Rubicon Minerals Corp.	MHC/Glacier Ck	RC99-06 to RMC99-14	1,875	**7,545**
2006	4	Constantine Metal Resources	Main/Glacier Ck	CMR06-01 to CMR06-03A	830	**8,375**
2007	7	Constantine Metal Resources	Cap, Glacier Ck	CMR07-04 to CMR07-10	2,315	**10,689**
2008	13	Constantine Metal Resources	Glacier Creek	CMR08-11 to CMR08-22	4,241	**14,931**
2009	11	Constantine Metal Resources	Glacier Creek	CMR09-23 to CMR09-32	4,562	**19,492**
2010	12	Constantine Metal Resources	Glacier Creek	CMR10-33 to CMR10-42	4,018	**23,510**
2013	10	Constantine Metal Resources	Glacier Creek	CMR13-43 to CMR13-52	3,747	**27,257**
2014	20	Constantine Metal Resources	Glacier Ck, RW, GT	CMR14-53 to CMR14-68, GT14-01	9,796	**37,052**
2015	10	Constantine Metal Resources	South Wall, GT	CMR14-56Ext, CMR15-69 to 76	7,736	**44,788**
2016	7	Constantine Metal Resources	SW, Cap, Pump Valley, GT	CMR16-78 to 81B, GT16-02 to 03	1,968	**46,756**
2017	32	Constantine Metal Resources	SW, Cap, Nunatak, GT	CMR17-82 to 107, GT17-05 to 10	10,631	**57,387**
2018	30	Constantine Metal Resources	SW, Nunatak, Boundary	CMR-108 to 134B, GT18-11 to 12	10,359	**67,745**
Total	**193**			**Total**	**67,745**	

Source: Constantine (2018)

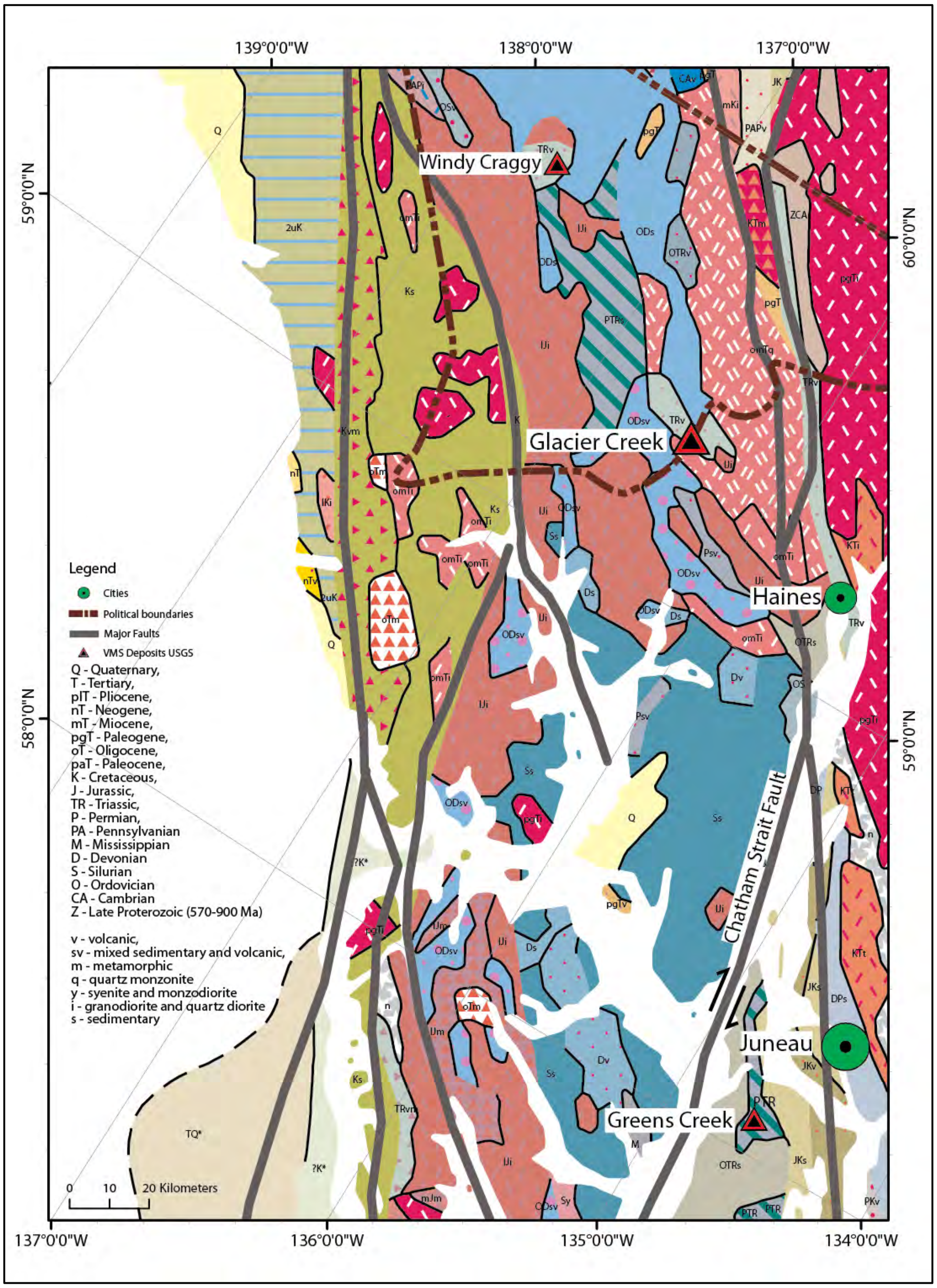
7 Geological Setting and Mineralization

7.1 Regional Geology

The Palmer Project is underlain by a mafic-dominated, bimodal sequence of submarine volcanic rocks that host volcanogenic massive sulfide (VMS) mineralization. These rocks are part of a ~600-km-long, discontinuously exposed belt of Late Triassic, rift related volcanic and sedimentary rocks belonging to the Alexander Terrane. Throughout southeast Alaska and northwest British Columbia, the Alexander Terrane hosts numerous VMS occurrences, prospects and deposits, including the giant Windy Craggy deposit in British Columbia, and the precious metals-rich Greens Creek deposit in southeast Alaska (Taylor, 1997) (Figure 7-1).

The Alexander Terrane (Figure 7-2) extends along the coast of northwest British Columbia northward through the Alaskan panhandle (southeast Alaska), through the Saint Elias Mountains of British Columbia and the Yukon, and westward into the Wrangell Mountains of Alaska (Wheeler and McFeely, 1991). The Alexander Terrane evolved along a convergent plate margin from the Precambrian-Cambrian to Early Devonian, with continuous deposition of arc-type igneous and sedimentary rocks (Gehrels and Berg, 1994). The latest Precambrian and early Paleozoic strata were subsequently deformed and metamorphosed during Middle Cambrian-Early Ordovician and Middle Silurian-earliest Devonian orogenies (Gehrels and Berg, 1994). During a period of relative tectonic stability from Middle Devonian to Early Permian, shallow marine carbonates, clastic rocks and mafic-intermediate volcanic rocks were deposited (Gehrels and Saleeby, 1987). Late Triassic rift-related volcanic and sedimentary rocks were deposited unconformably over the Permian and older rocks (Gehrels et al., 1986). Overprinting deformation and metamorphism occurred mainly throughout the mid-Jurassic to Cretaceous accretion of the Alexander Terrane to inboard Cordilleran terranes (Berg et al., 1972; Coney et al., 1980), with further dismemberment occurring along regional-scale right-lateral strike slip faulting during Tertiary to Recent time.

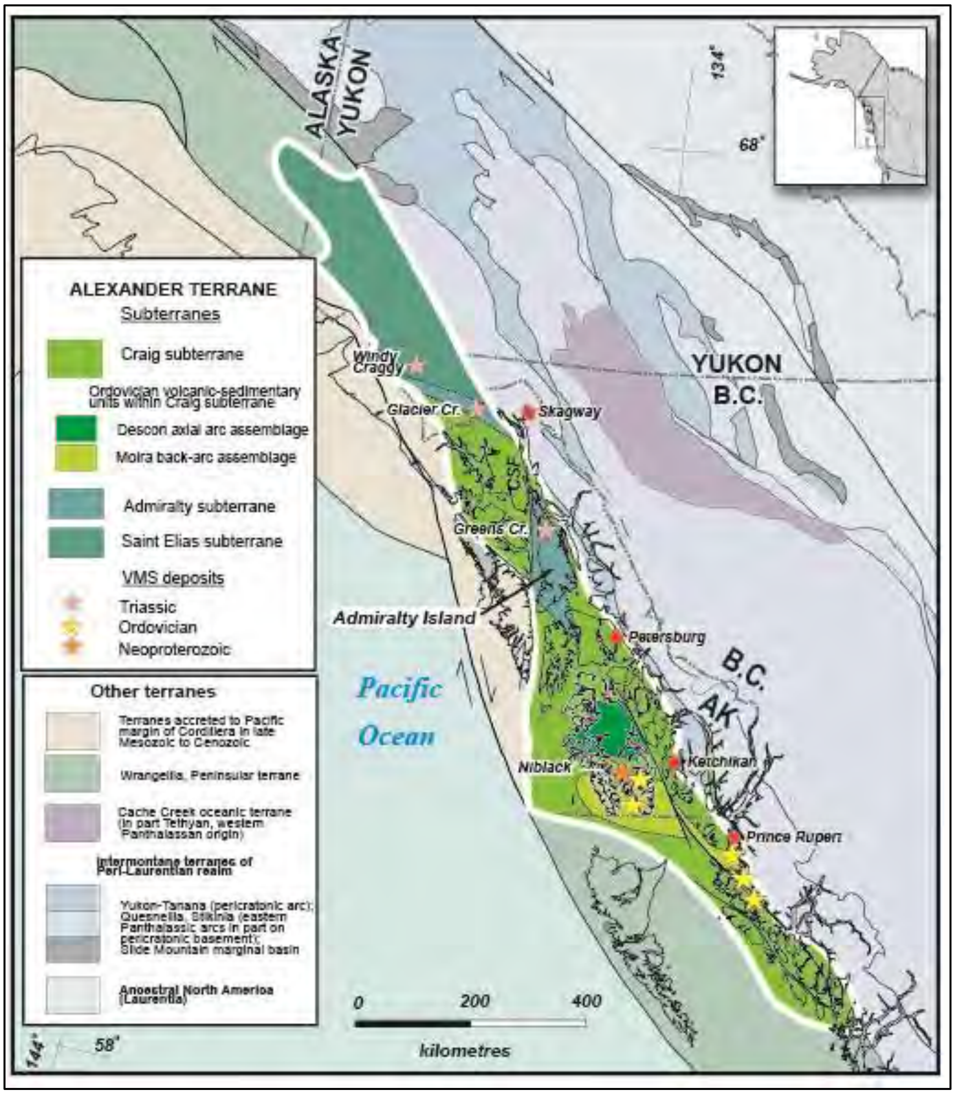
Figure 7-1: Regional Scale Geology Map Centered on Northern Southeast Alaska (Steeves, 2013)



Source: Steeves (2013)

Notes: The Greens Creek, Windy Craggy and Glacier Creek deposits are all situated within Triassic volcanic and sedimentary units. Reconstructing offset along the Chatham-Strait fault places Greens Creek <50 km from Glacier Creek.

Figure 7-2: Terrane Map of the Northwestern Cordillera. Modified from Nelson et al. (2013a)



Source: Nelson et al. (2013)

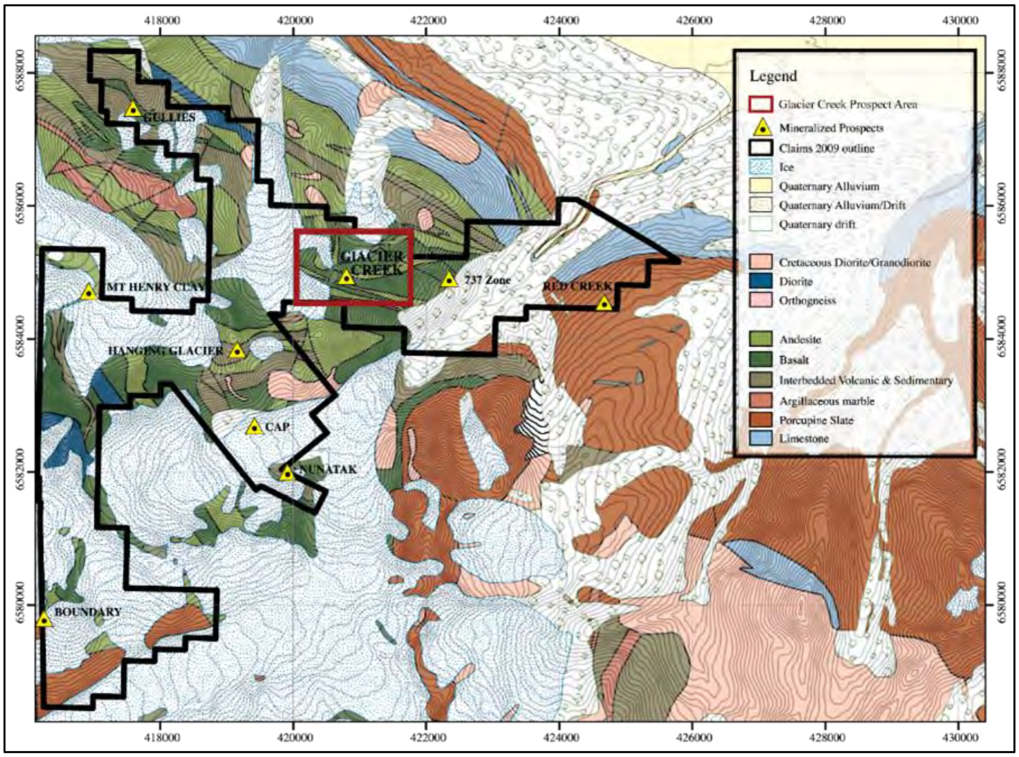
Notes: The Saint Elias, Admiralty and Craig Sub terranes are outlined along with notable VMS deposits.

7.2 Property Geology

The Project is underlain by Paleozoic and lower Mesozoic metasedimentary and metavolcanic rocks that have been intruded locally by Cretaceous and Tertiary granitic plutons (Figure 7-3). Thin-bedded limestone and massive marble that contain fossils of Devonian to Carboniferous-age appear to be the oldest rocks in the area, and they are apparently conformably overlain by pelitic rocks of the Porcupine Slate, which are of probable Late Triassic age (Redman et al. 1985; MacKevett et al. 1974). The hosts to VMS mineralization at the Project, a dominantly mafic volcanic package, are the youngest stratified rocks in the area, and they are locally interbedded with rocks of the Porcupine Slate (Redman et al., 1985). The previously assumed

Late Triassic age of the mafic volcanic sequence was confirmed by microfossil data and U-Pb dating of volcanic rocks from the Glacier Creek prospect area (Green, 2001). Elsewhere in the Alexander terrane, an unconformity separates Late Triassic from Paleozoic rocks (Gehrels et al., 1986) but this has yet to be recognized on the Project.

Figure 7-3: Geological Map of the Palmer Project. Modified from Redman et al. (1985)



Source: Redman et al. (1985)

The aforementioned Late Triassic rocks predominate on the Project and consist of massive to pillowed basalt, fragmental basalt, and possible andesite, with intercalations of calcareous siltstone and tuff, and rare rhyolite flows and dykes. Folding and faulting likely repeats stratigraphy and may in part be responsible for the broad distribution of exhalative mineralization and associated quartz-sericite-pyrite alteration across the Project. Alteration is commonly several hundred meters in extent and is of such strong intensity that discrimination of the protolith is difficult without the use of immobile element geochemistry.

7.3 Metamorphism

Late Triassic rocks on the Project have undergone lower to mid-greenschist facies regional metamorphism (Green et al., 2003).

7.4 Structural Geology

The rock units on the Project record at least three different episodes of deformation.

a. The earliest and most evident deformation event (D1) is a north-south contractional event characterized by a slatey to schistose cleavage (S1), which is likely axial planar to south-verging folds and thrust faults (Lewis, 1998). Fabric intensity is highly variable, reflecting the strong contrast in rheology between massive basalts and thin-bedded silty rocks comprising most of the section. In most places, preservation of primary rock textures is very good, although the rocks are locally deformed beyond recognition of the protolith. In general, the S1 fabric is most strongly developed within sedimentary strata and intensely altered volcanic rocks. Map-scale D1 folds have km-scale wavelengths, close to tight forms, and are commonly overturned. Outcrop-scale D1 folds are rare, and typically restricted to intercalations of sedimentary strata.

b. The second phase of deformation (D2) is less readily observed and has no associated fabric. It is evidenced by map-scale folds that affect bedding and S1 foliation. The folds are generally tight and have sub-angular hinges with axes that plunge variably to the northwest. Although important at the property scale, the effects of the D2 deformation event are not apparent in the Glacier Creek prospect area.

c. The D3 deformation is manifest by weakly developed north-easterly trending crenulation fabrics that are present locally within some of the more schistose rocks (Lewis, 1998). They are interpreted to post-date the D2 event because the orientation of the crenulation cleavage is independent of position on D2 folds. Regional strain associated with D3 is minor and does not appear to have produced any megascopic structures.

In a regional structural context, the Project host rocks are correlative with the Hyd formation in central and southeastern Alaska (Loney, 1964; Gehrels et al. 1986), as well as with the informally named Tats group, exposed in the Saint Elias Mountains of British Columbia within fault bounded blocks to the northwest of the Project (MacIntyre, 1986; Mihalynuk et al., 1993). The Hyd formation hosts the Greens Creek deposit, and the Tats group hosts the Windy Craggy deposit. After restoration of 150 km of Tertiary dextral offset along the Chatham Strait – Denali fault system (Hudson et al., 1982), the Project would be located less than 50 km from the Greens Creek deposit, and perhaps not surprisingly, the two share similarities in their style of alteration and mineralization.

7.5 Local Geology of the Glacier Creek Prospect

The Glacier Creek prospect, host to the South Wall and RW Zones (collectively known as the Palmer deposit), is exposed on flanks of Mount Morlan (Figure 7-4, Figure 7-8, and Figure 7-9). The structure of Mount Morlan is that of a large, overturned, south verging anticline with an axial plane that dips moderately to the northeast. The stratigraphic section (Figure 7-5) is dominated by massive to pillowed basalt flows, with subordinate impure carbonate rocks, tuff, and felsic volcanic rocks. The rocks have undergone prolonged hydrothermal activity and host stacked zones of massive sulfide.

7.5.1 Hanging Wall Units

Younger, amygdaloidal, massive to pillowed and locally spherulitic basalts are considered the unaltered hanging wall sequence. These basalts can be differentiated chemically and also appear on the steeper South Wall limb.

7.5.2 Mineralized Horizon

VMS mineralization at the RW zone is hosted by rhyolite while the massive sulfide unit at South Wall consists of stratiform massive barite and sulfide, and rare black shaley limestone.

7.5.3 Footwall Units

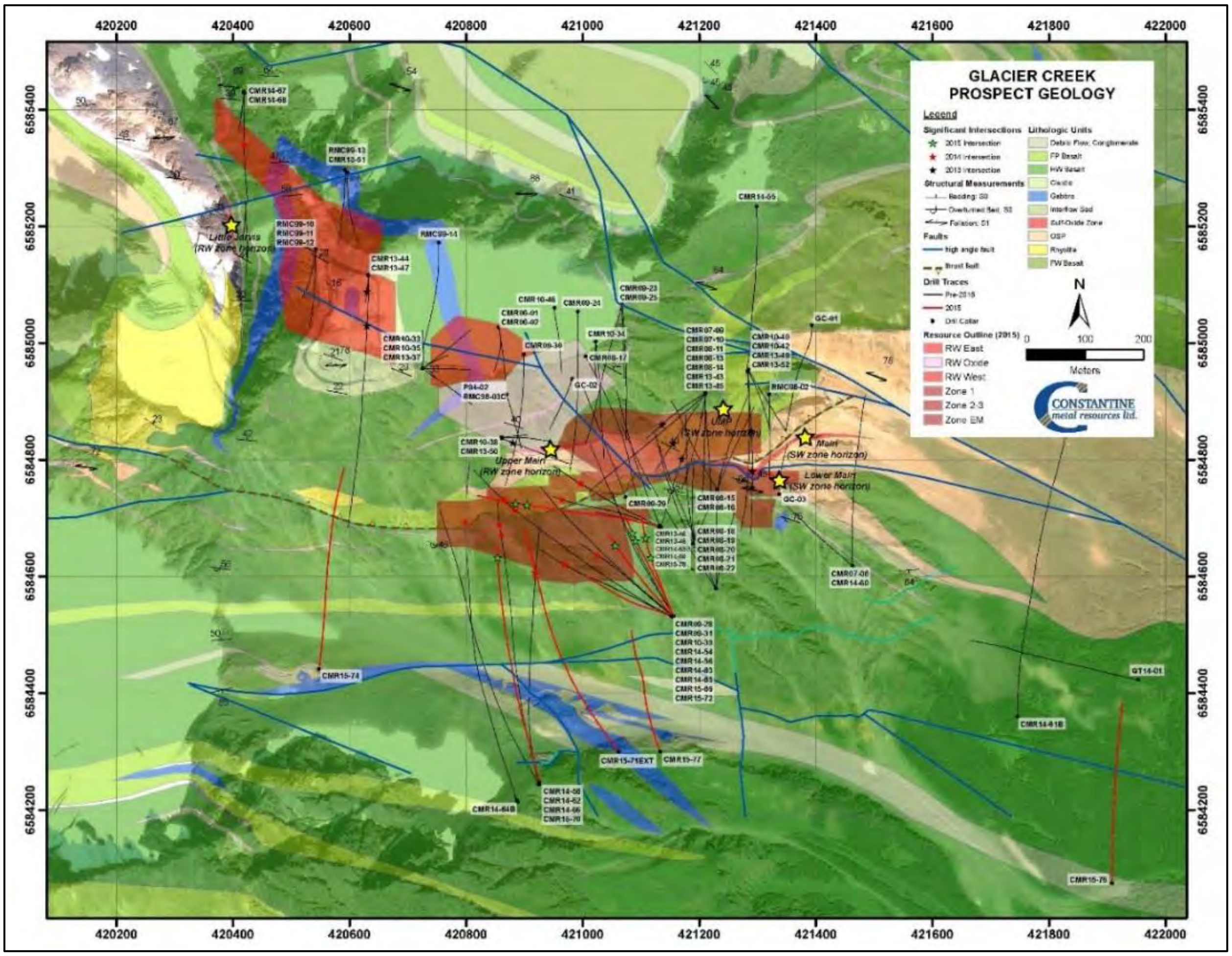
Feldspar-phyric basalt underlies the host rhyolite and the RW Zone. Intense alteration footwall obscures primary protolith stratigraphically below the South Wall Zones, although it appears to be similar to the footwall rocks to the RW Zone; the primary difference being that the stratigraphic footwall to the South Wall Zones, has a lower volume of feldspar-phyric rocks, a much greater proportion of fragmentals (volcaniclastic), and a higher percentage of aphyric basalt. The extent and intensity of quartz-sericite-pyrite alteration may be controlled by the permeability of the volcanic precursor, with volcaniclastic units likely being the focus of more widespread alteration.

7.5.4 Structure

Prospect-Scale Anticline – The large, overturned, northeast trending, south-verging anticline (the “Anticline”) is the dominant feature of the Glacier Creek prospect area (Figure 7-6). The stratigraphy of the upright fold limb around the RW Zone is generally intact and has been relatively undisturbed by folding or faulting. The upper limb is upright, moderately northeast dipping, and host to RW Zone mineralization. The lower limb is overturned to sub-vertical, and is host to South Wall zone mineralization

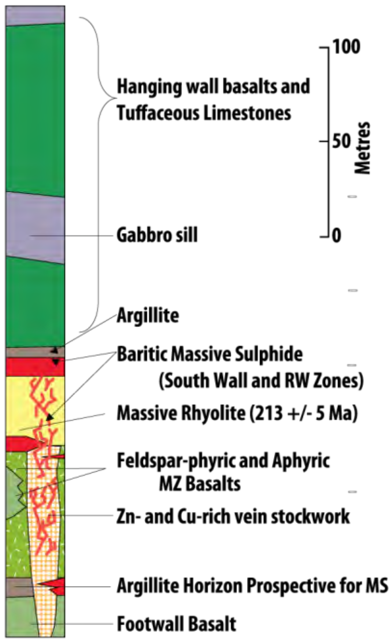
MZ Fault – A poorly constrained, shallow, prospect-scale thrust fault (termed the “MZ Fault” or “Main Zone Fault”) with an orientation similar to the axial plane of the Anticline offsets the upper limb from the lower limb of the fold (Figure 7-6). Offset on the thrust is ‘top to the south’ with an estimated 300 m or less of displacement. A single, discrete, clearly defined structure representative of the thrust fault and that can be correlated from drill hole to drill hole has not been observed in the mineral resource area drilling; instead, the thrust fault appears to be manifest as a structural zone of variable thickness, characterized by a highly strained variety of feldspar-phyric basalt; referred to as ‘FP Lentil’ for its characteristic flattened and lenticular shaped feldspar phenocrysts.

Figure 7-4: Geological Map of Glacier Creek Prospect Area (SW/RW Zones)



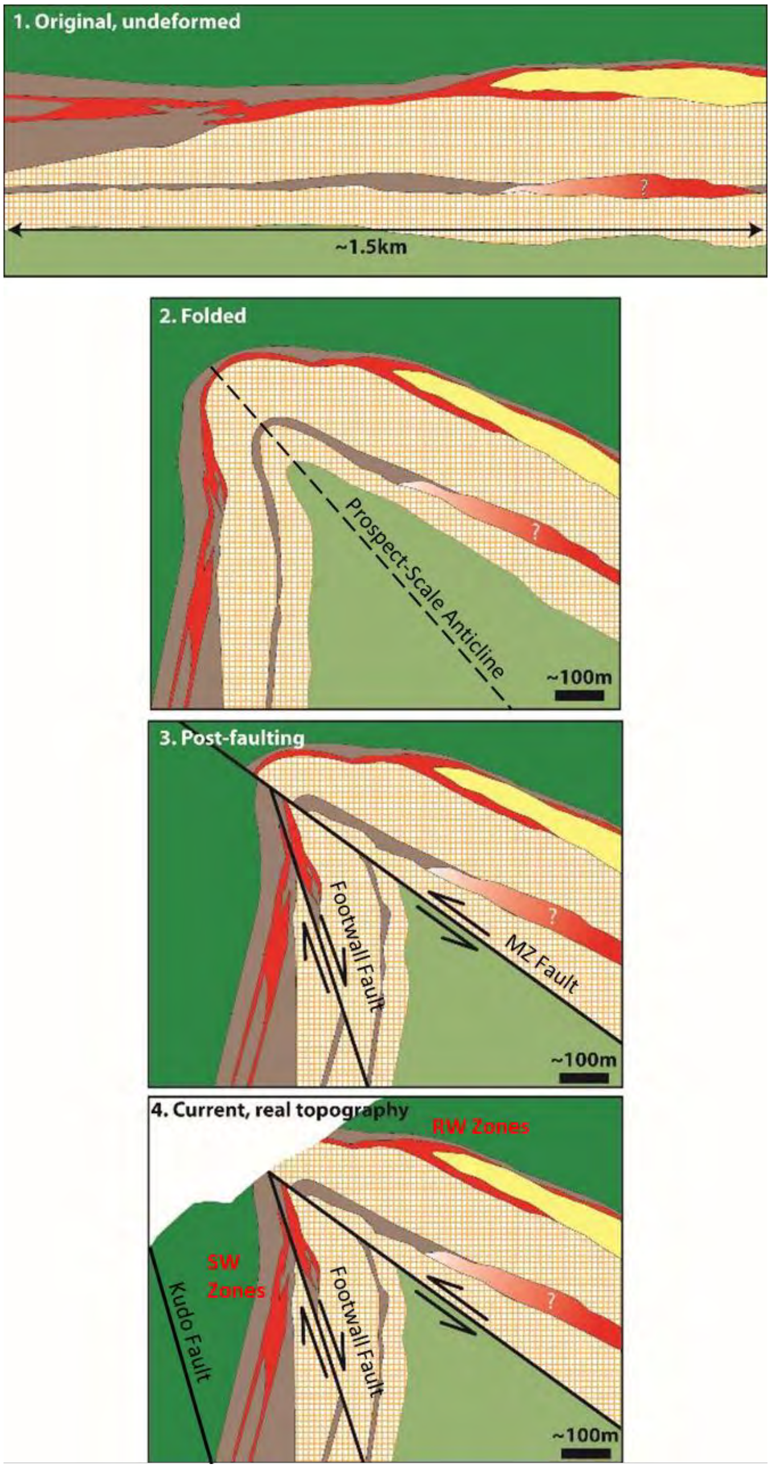
Source: Constantine (2018)

Figure 7-5: Schematic Stratigraphic Column for the Glacier Creek Prospect Area (SW/RW Zones)



Source: Constantine (2018)

Figure 7-6: Schematic Geological Reconstruction and Structural Evolution of Glacier Creek Prospect

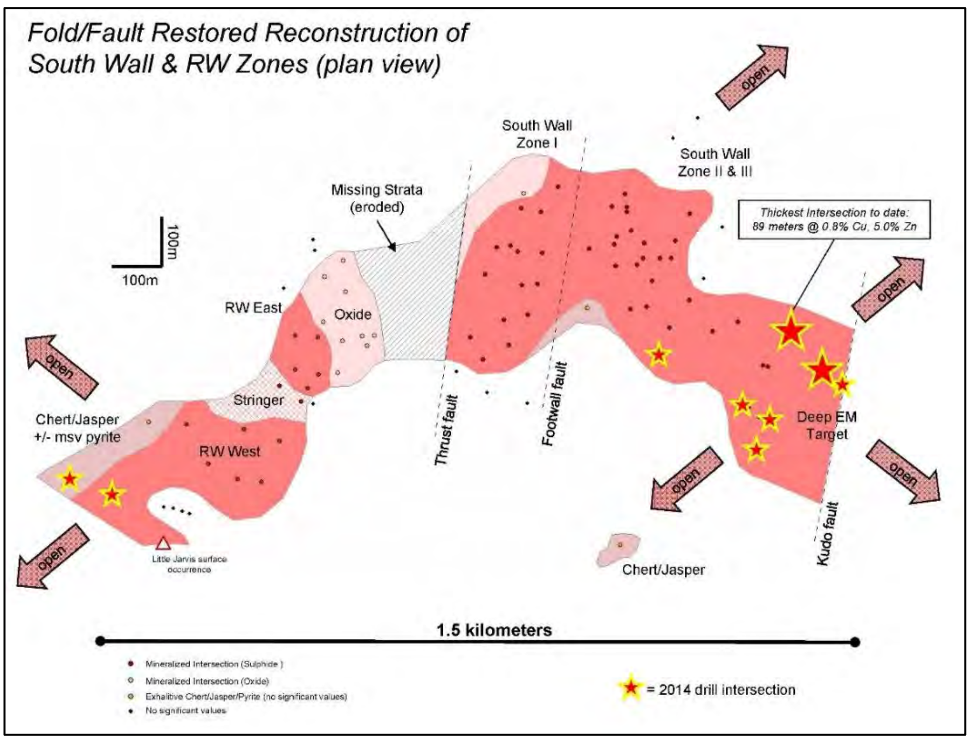


Source: Constantine (2018)

Footwall Fault – The area between massive sulfide lenses within the South Wall Zones (subdivided as Zones I, II and III) is also complicated by structure. Recent work by Steeves (2013) strongly supports all three zones being located within the steep overturned lower limb of the prospect-scale anticline, and that the three zones are crudely time-stratigraphic equivalent. A moderate to steep north dipping, normal fault (termed the "Footwall Fault") is interpreted to offset Zone I from Zone II, which are believed to have originally been contiguous (Figure 7-6). An extension of this interpretation is that South Wall Zones I, II, III and EM Zone, and the RW Zones all represent, more or less, a single time/stratigraphic equivalent body of mineralization (Figure 7-7).

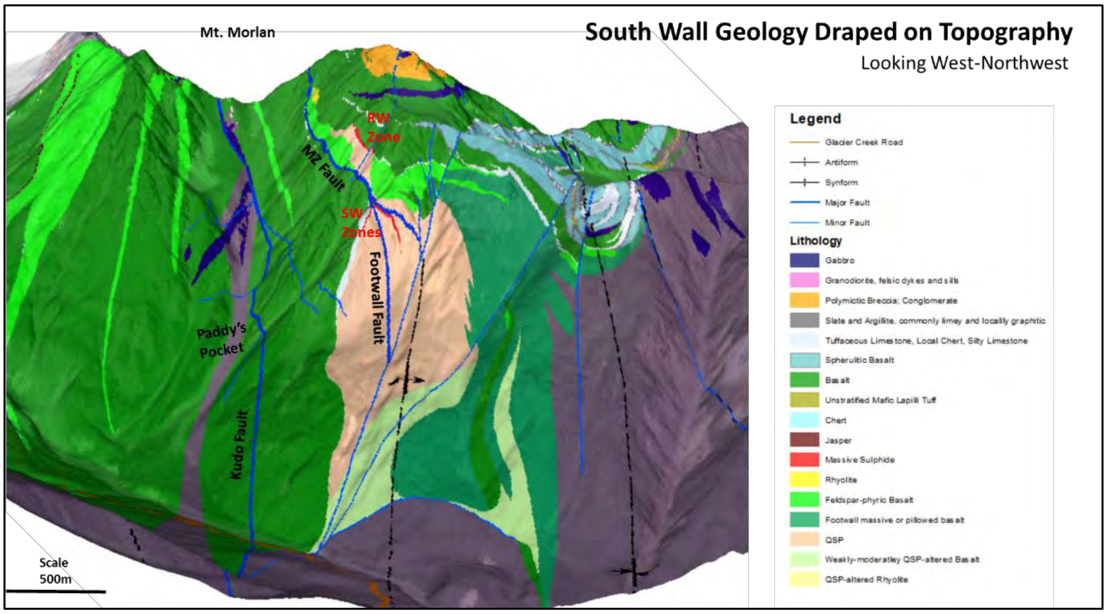
Kudo Fault – The lower portion of the South Wall resource area has been offset by a major, steeply dipping, east-west striking fault system termed the "Kudo Fault". Displacement is interpreted to include both reverse, ~180 m north-side up, and left-lateral, strike-slip, with movement interpreted to be on the order of ~350 m (Proffett, 2016, see Figure 7-6).

Figure 7-7: Fold-Fault Restored Reconstruction of South Wall and RW Zones



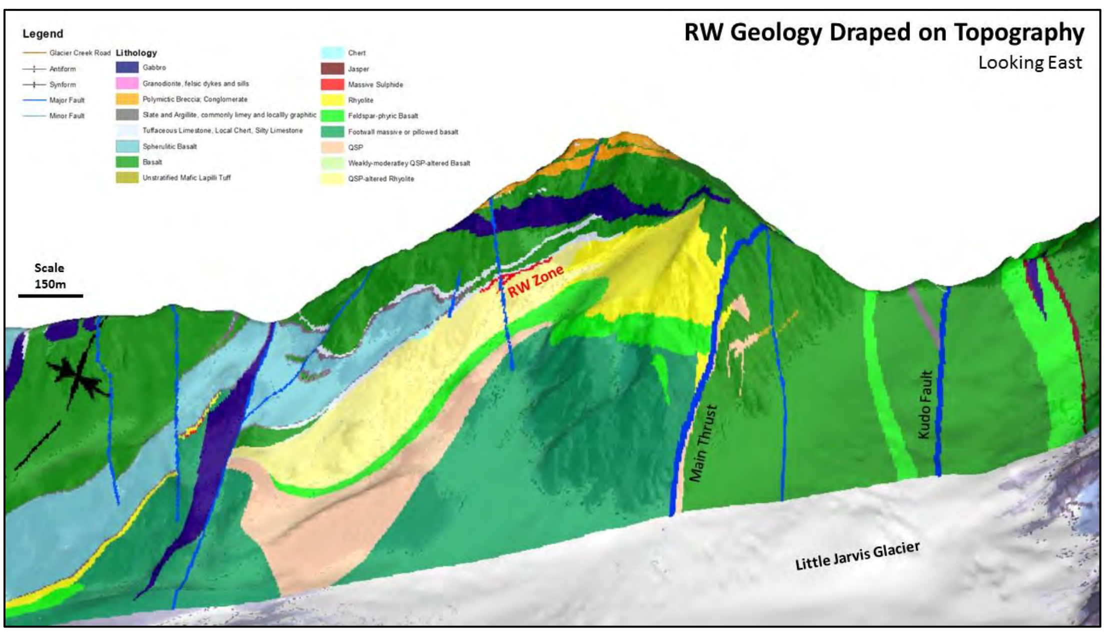
Source: Constantine (2018)

Figure 7-8: South Wall Area with Geology Draped on Topography (Looking West-Northwest)



Source: Constantine (2018)

Figure 7-9: RW Area with Geology Draped on Topography (Looking East)



Source: Constantine (2018)

7.6 Local Geology of the Nunatak Prospect

The Nunatak prospect area, host to the AG Zones, is located 3 km southwest from the South Wall zones, on a steep Nunatak between two branches of the Saksaia Glacier. The area is underlain by a folded sequence of bimodal volcanic flows, fragmental volcanic units, volcaniclastics, tuffs, and limey argillites and siltstone (Figure 7-10, Figure 7-11, Figure 7-13, and Figure 7-12).

VMS mineralization, mainly massive barite beds with variable base and precious metal mineralization hosted in sphalerite, galena and sulfosalts, outcrops in several places on the north-northwestern flanks of the mountainside. The eastern aspect of the slopes at the “Jag” showing includes an outcrop of barite and massive galena with sphalerite. Talus and glaciers cover a large portion of the prospect area and limits the ability to confidently demonstrate continuity of the mineral horizons between outcrops, creating the potential for differing geological interpretations. Nevertheless, it is clear that exhalative mineralization along this horizon is widespread both on surface and at depth, and also attains significant width.

7.6.1 Hanging Wall Units

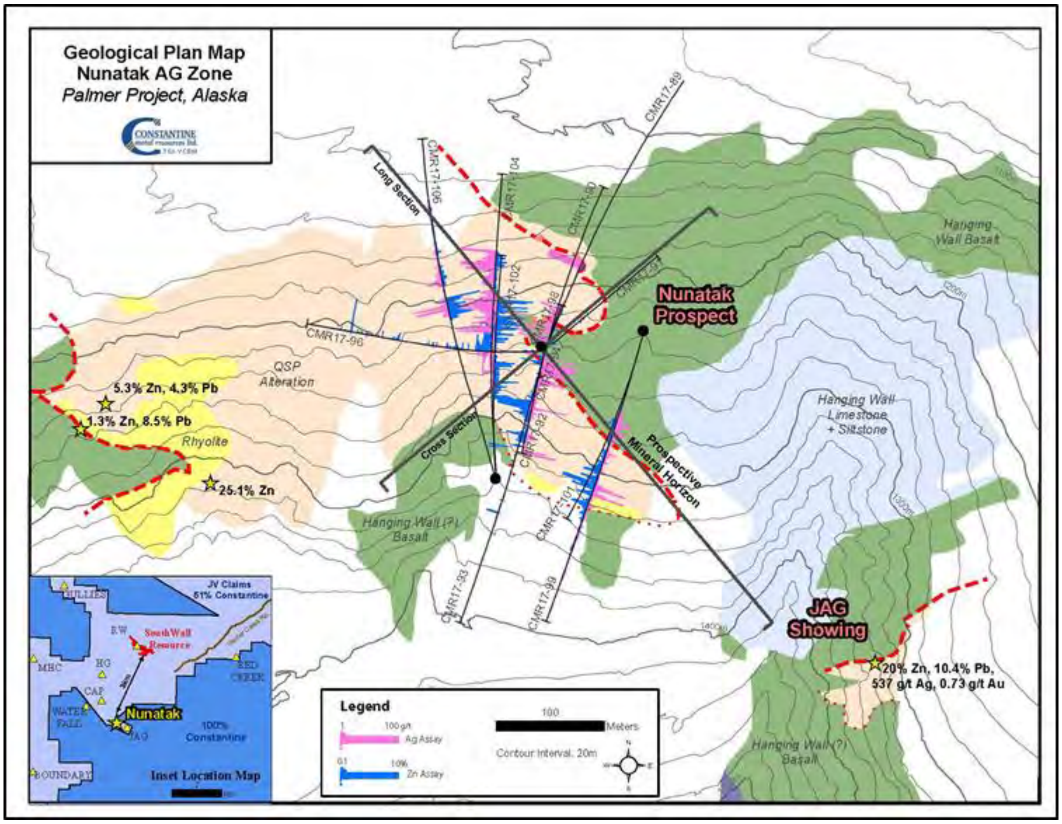
The immediate hanging wall to AG Zones mineralized horizon is a thick sequence of mafic volcaniclastics and coherent massive, pillowed and amygdaloidal basalts with local jasper beds/lenses. A distinct spherulitic basalt unit is also observed above the mineralized zone in places but is also observed within and below the mineralized zone. These mafic volcanic rocks are, in turn, overlain by mixed sedimentary rocks (limestone and siltstone) and mafic tuffs that are intruded by abundant dykes of various compositions. Faulted and folded limey argillites overlie the mixed sedimentary package.

7.6.2 Mineralized Horizon

The AG Zones are spatially associated with rhyolite and is hosted within a folded and faulted, steeply dipping exhalative chert-barite horizon and that is stratigraphically underlain by a massive pyrite-rich sulfide zone and an extensive discordant sulfide stringer zone. The AG Zones consists of two major components:

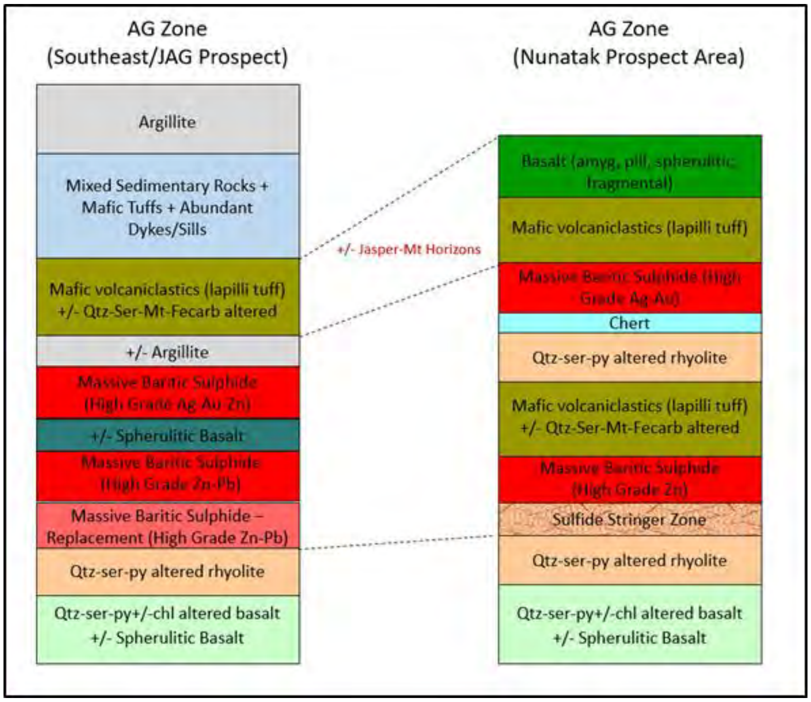
- An upper, Ag+Au-rich folded barite bed defined by a tight synform and antiform pair with fold axial traces trending WNW-ESE (supported by both surface mapping and drilling). The large-scale anticline is defined by an upright, moderately SW dipping south limb and an overturned steeply SW dipping north limb. This upper stratigraphy is interpreted to be displaced from the lower zone by a shallow, normal, listric fault (dubbed ‘McFault’) and
- A lower, sheet-like, Zn-rich baritic core that is slightly warped (by probable broad D2 folds) and dips steeply back and forth towards the NE and SW. The mineralization is stratigraphically underlain by a massive pyrite-rich sulfide zone and an extensive discordant Zn-Pb-rich stringer zone.

Figure 7-10: Geological Map of the Nunatak Prospect (AG Zones) including 2017 Drill Traces



Source: Constantine (2018)

Figure 7-11: Schematic Stratigraphic Columns for the AG Zones (Nunatak & JAG Prospect areas)



Source: Constantine (2018)

The immediate hanging wall to the mineralized horizon is dominated by mafic volcaniclastics that are locally altered to a distinct assemblage of quartz-sericite-magnetite+/-chlorite+/-iron-carbonate+/-jasper (referred to as “QSM”). This package of intense QSM alteration also occurs between what is interpreted as two separate stratigraphic units of mineralization.

7.6.3 Footwall Units

The footwall sequence is composed of fragmental to coherent strongly quartz-sericite-pyrite (QSP) altered rhyolites, minor exhalative chert deposits, and a thick sequence of variably QSP-chlorite altered heterogeneous basaltic flows that have textures ranging from massive to pillowed, amygdaloidal, spherulitic, and brecciated. The volcanics were geochemically identified as tholeiitic to calc- alkaline bimodal volcanic flows (Doherty, 2018). Massive to pillowed and amygdaloidal basalt is dominant in outcrop in the southern part of the map area and is interpreted as the footwall to the mineralized horizon. The footwall alteration has a large surface footprint of ~500 x 200m.

7.6.4 Structure

Two-fold generations are observed. F1 folds are the most easily visible in outcrop and have amplitudes ranging from several metres to tens of metres, with small parasitic folds with amplitudes of tens of centimetres. F1 folds are parasitic folds on the north-facing limb of a north-closing synform with the hinge interpreted to occur below the Saksaia glacier to the north and stratigraphy is repeated around the CAP prospect on the other limb of the synform. F1 folds are refolded by open F2 folds with poorly understood, N-NW trending fold axial traces. Fold interference and faulting complicates mapping and correlation of surface geology to drill holes.

Upper Zone - The dominant structural style of the upper zone consists of tight folds with fold axial traces trending WNW-ESE and folds plunging to the ESE. Distribution of rock units is dominated by a large-scale anticline defined by an upright, moderately SW dipping south limb and an overturned, steeply SW dipping north limb. Synclines are mapped to the south and north of the anticline. Small scale parasitic folds are also inferred based on mapped distribution of rock units and bedding measurements. A second phase of deformation (noted to be manifested in a macroscale and NNW trending in R. Greig's 2014 mapping), may be the cause of further deformation/warping of the stratigraphy in the map area.

Lower Zone - The lower zone is steeply dipping, well zoned, and comprises a large majority of the drill defined extent of the AG Zone deposit. The zone has a northwesterly trend (~310-320 deg Az) and a dip that changes orientation along strike, presumably the result of a second deformation event. The northwest half of the zone is sub-vertical to locally overturned with a southwest dip, whereas the southeast half is upright with a moderate to steep dip to the northeast. To date, the thickest and best developed mineralization is defined in the southeast from holes drilled in 2018.

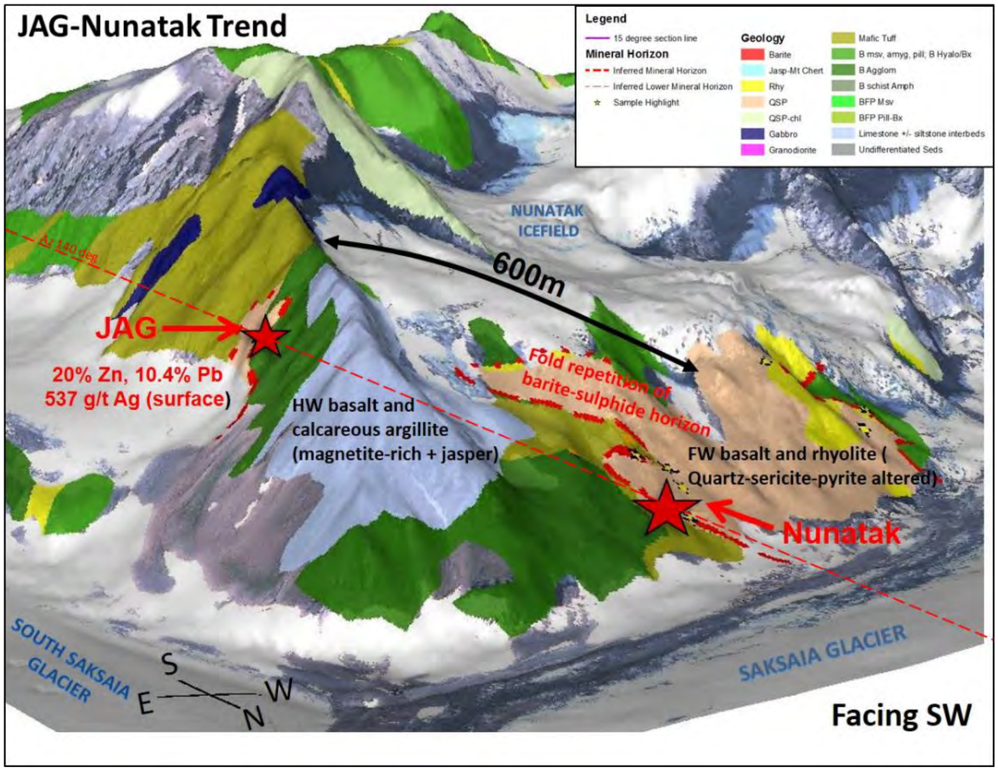
7.6.5 Alteration

Footwall alteration at the AG Zone deposit is similar to the Palmer deposit, dominated by sericite-pyrite-quartz alteration increasing in intensity towards mineralization. Hanging wall alteration is typified by magnetite, jasper, Fe-carbonate, minor chlorite, and locally sericite alteration up to tens of meters above mineralization.

Figure 7-12: Surface Outcropping of AG Zone Deposit Massive Barite – Nunatak Prospect Area

Source: Constantine (2018)

Figure 7-13: JAG-Nunatak Prospect Area with Geology draped on Topography (looking southwest)



Source: Constantine (2018)

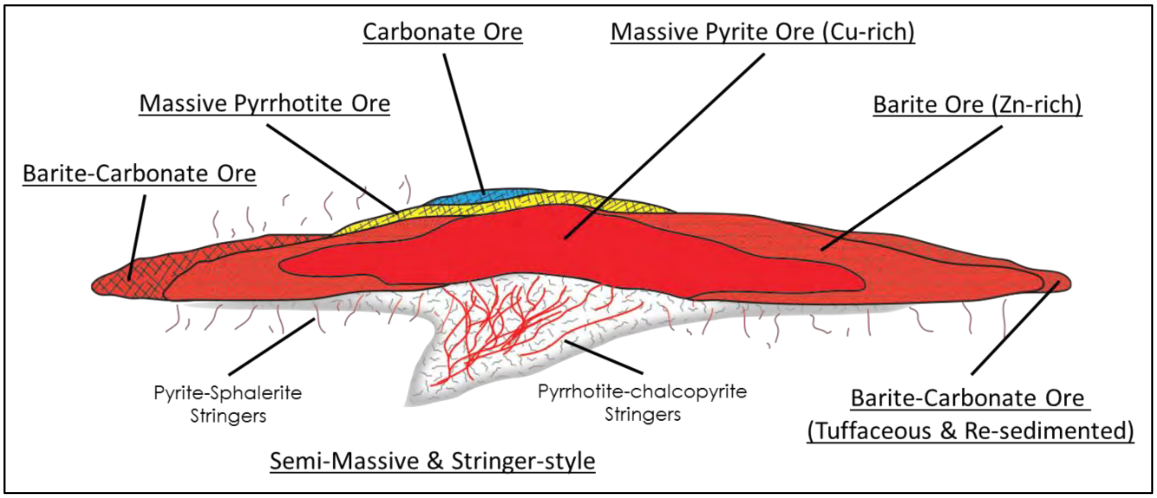
7.7 VMS Mineralization

The Project hosts two known volcanogenic massive sulfide (VMS) deposits, the Palmer deposit, which consists of the South Wall and RW Zones, and the newly discovered AG Zone deposit located 3 km to the southwest. Numerous other mineralized prospects are also present throughout the property. The various prospects and deposits share similar alteration and mineralogical characteristics, suggesting a large-scale, property-wide Late Triassic mineralizing event with multiple hydrothermal vent centers.

South Wall and RW Zone mineralization is the best studied, and a summation of principal mineralization styles, metal zonation and alteration can be found below. A more detailed description can be found in the Master of Science thesis completed on the Glacier Creek prospect by Nathan Steeves (2013). Descriptions of the regional mineral prospects on the Project can be found in the NI 43-101 technical report by Greig and Giroux (2010).

Six mineralization styles have been identified and are grouped according to dominant mineral assemblages and texture (Figure 7-14, Table 7-1, and Table 7-2). More details can be found in Gray and Cunningham-Dunlop (2015).

Figure 7-14: Principal Mineralization Styles with Typical VMS Lens



Source: Constantine (2018)

7.7.1 Barite Mineralization (Zn-rich)

Barite-rich ores contain abundant pale honey-brown, low-Fe sphalerite with variable amounts of pyrite, locally minor galena, tetrahedrite, arsenopyrite and variable concentrations of late chalcopyrite. Barite ores contain >50% barite and 30-50% sulfides and resemble some types of “black ore” in the Kuroko deposits (Eldridge et al., 1983). Barite is the dominant gangue mineral. Compositional bands dominated by either barite, sphalerite, pyrite or chalcopyrite occur locally. This compositional banding is likely primary, by comparison with Kuroko analogs, but may have been enhanced by deformation. The Barite-rich mineralized rock grades locally into massive pyrite mineralized rock as sulfide content increases and quartz becomes the dominant gangue. Barite mineralized rock also grades into a barite-carbonate mineralized rock at the upper limits and flanks of South Zones I, II and III, with carbonate increasing in abundance at the margins of the lens.

7.7.2 Massive Pyrite Mineralization (Cu-rich)

Massive pyrite mineralization typically occupies the core of the lens and is dominant in SW Zone I and parts of SW Zones II and III. They are classified as having >50% sulfide content, typically as pyrite and chalcopyrite with lesser sphalerite and associated minor quartz and/or barite gangue. These ores resemble the ‘yellow ore’ of the Kuroko deposits (Eldridge et al., 1983). The massive pyrite ores commonly exhibit compositional banding with variable amounts of sphalerite and chalcopyrite. They also show fine-grained, dispersed pyrite followed by later, coarser, anhedral pyrite-with remobilized intergranular chalcopyrite or sphalerite.

7.7.3 Semi-massive and Stringer-style Mineralization

Semi-massive and stringer-style pyrite ± sphalerite ± chalcopyrite zones stratigraphically underlie and form the feeder-zones to massive sulfide mineralization. They consist of 30-50 vol.% and 15-30 vol.% sulfide, respectively. Pyrite grains occur as very fine disseminated grains and as coarser grains within stringers. Sphalerite and chalcopyrite are also disseminated with pyrite and within stringers. Locally, stratigraphically below SW Zone I/II/III massive sulfide mineralization and within the alteration zone, stringer-style mineralization is dominated by pyrrhotite-chalcopyrite, rather than pyrite-sphalerite. This facies is characterized by <50 vol.% sulfides as stringer-style and stockwork veins and >50 vol.% gangue of quartz and muscovite. The chalcopyrite content ranges from 3 to 15 vol.%, with pyrrhotite content up to 40 vol.%. Trace sphalerite is present in most samples as dark red, anhedral grains.

7.7.4 Massive Pyrrhotite Ore

Massive pyrrhotite ores occur both above and below massive pyrite ores and barite ores (Figure 7-14) within SW Zones I and II, and generally represents a volumetrically small portion of mineralized zones. They contain >50 vol.% sulfide with up to 15 vol.% chalcopyrite, up to 20 vol.% sphalerite and <1 vol.% pyrite. Chalcopyrite and sphalerite occur within massive pyrrhotite and in fractures in pyrrhotite. The dominant gangue minerals are quartz and carbonate with minor very fine-grained muscovite. Trace hematite and rare molybdenite are present within muscovite-rich patches cross-cutting quartz and sulfide grains.

7.7.5 Carbonate Ore

Carbonate-rich ores are found at the stratigraphic top of SW Zone I. Carbonate ores typically contain 60 vol.% coarse-grained carbonate, with minor quartz, muscovite and dark green chlorite, up to 35 vol.% dark red-burgundy sphalerite and up to 5 vol.% chalcopyrite. Trace amounts of partially replaced (Ba-K) feldspar are also observed. More massive carbonate contains relatively coarse, subhedral to euhedral, interlocking crystals of calcite (up to 3 mm). Late chalcopyrite stringers cross-cut carbonate as thin veinlets, and sphalerite is disseminated throughout, locally as relatively coarse anhedral to euhedral grains (up to 1 mm) forming aggregates. These rocks appear to be highly recrystallized.

7.7.6 Barite-Carbonate Mineralization (Tuffaceous and Re-sedimented)

At the margins of SW Zone I and SW Zone II and to a lesser extent in SW Zone III, there is a mix of carbonate-rich and sulfide-rich mineralization. These include a finely-layered barite-carbonate-sulfide facies, some sulfide-clast and barite crystal-rich facies, and a variably mineralized tuffaceous and cherty facies. Barite mineralized rock grades outward into a barite-carbonate mineralized rock as the disseminated carbonate content increases and gradually becomes more tuffaceous, with interlayered barite and carbonate laminae.

Above this unit is a weakly mineralized, barite-free tuffaceous horizon. This capping tuffaceous horizon overlies the entire lens and is characterized by weak mineralization, (Ba-K)-feldspar, barian muscovite, local albite, cherty patches/layers and is strongly calcaerous. It is intercalated with cherts and altered volcaniclastics and may continue laterally along the mineral horizon. Cherts and tuffaceous units above SW Zones I and II are locally cross-cut by thin pyrite ± sphalerite stringers containing quartz, carbonate, albite and muscovite, which is suggestive of continued hydrothermal activity after to the deposition of each lens. Tuffaceous units locally contain chalcopyrite, possibly as replacement of amygdules and/or feldspars and sphalerite.

Locally within SW Zone II, the barite-carbonate mineralized rock has a distinctive clastic texture. These rocks contain euhedral, and locally broken, barite crystals and clasts of massive barite, sulfide and/or quartz within a very fine-grain carbonate matrix. Massive barite clasts are irregular, angular and can reach up to 1 cm across. Other minerals include minor to trace albite, muscovite and chlorite. Sulfides are dominantly pyrite and pale honey-brown sphalerite, with minor to trace chalcopyrite, galena, tetrahedrite, and arsenopyrite. Chalcopyrite typically replaces pyrite, which is framboidal and less recrystallized than in the main massive sulfide lenses. The clastic nature of this facies and the abundant broken crystals of barite suggest that this material was re-sedimented.

Table 7-1: Description of Principal Mineralization Styles

	Description
Barite-Carbonate Mineralization (Tuffaceous & Re-sedimented)	<50% py ± sph ± cpy within variable carbonate rich rock; located on the margins of massive sulfide lenses
Carbonate Ore	>50% carbonate; capping massive sulfides and possibly as infilling and replacement of calcareous tuffs
Barite Ore	<50% sulfide with dominantly barite gangue; banded to massive texture; dominated by py-sph, with varying cpy
Massive Pyrite Ore	>50% sulfide; pyrite and chalcopyrite with quartz/barite gangue
Massive Pyrrhotite Ore	>50% sulfide, dominated by pyrrhotite, quartz and carbonate mega-crysts; up to 15% cpy, 20% sph
Semi-massive & Stringer Style Pyrrhotite-Chalcopyrite	<50% po-cpy within quartz-muscovite altered rock within footwall
Semi-massive & Stringer Pyrite-Sphalerite (± cpy)	<50% py-sph (± cpy) within quartz-muscovite altered rock, above and below mineralized lenses; >2% sph ± cpy

Source: Constantine (2018)

Table 7-2: Mineralogy of Principal Mineralization Styles

Cherty, Tuffaceous and Barite-Carbonate Ore	Carbonate Ore	Barite Ore	Massive Pyrite Ore	Massive Pyrrhotite Ore	Po-Cpy Stringers	Semi-massive & stringer Py-Sph (± Cpy)
Pyrite Chalcopyrite Sphalerite ± Galena ± Tetrahedrite ± Arsenopyrite	Fe-Rich Sphalerite Chalcopyrite ± Pyrite ± Pyrrhotite	Fe-poor Sphalerite Pyrite Chalcopyrite ± Tetrahedrite ± Galena ± Arsenopyrite ± Covellite	Pyrite Chalcopyrite Sphalerite ±Tetrahedrite ±Arsenopyrite ± Covellite	Pyrrhotite Chalcopyrite Sphalerite Pyrite	Pyrrhotite Chalcopyrite Pyrite ± Fe-rich Sphalerite	Pyrite Fe-poor Sphalerite ± Chalcopyrite ± Tetrahedrite
Associated Gangue Minerals						
Barite Carbonate ± Muscovite ± Chlorite ± Ba-Feldspar ± Albite	Carbonate ± Quartz ± Chlorite ± Muscovite ± Ba-Feldspar	Barite ± Quartz ± Muscovite ± Chlorite	Quartz ± Muscovite ± Chlorite ± Barite ± Carbonate ± Albite ± Ba-Feldspar	Carbonate ± Quartz ± Muscovite ± Chlorite	Quartz Muscovite ± Chlorite	Quartz Muscovite Carbonate ± Albite ± Chlorite

Note - Minerals are listed in approximate, relative order of abundance (± indicates presence in only a few samples)
Source: Constantine (2018)

7.8 VMS Metal Zonation

Massive sulfide lenses SWZII and SWZIII exhibit typical, vertical metal zonation, with Cu-rich zones underlying Zn-rich zones and zones elevated in Pb. SWZI is the exception with generally Cu-rich mineralization at the tops of the drill holes and Zn and Pb enrichment at the bottom, suggesting that the lens is overturned (Steeves, 2013).

The RW Zone (both East and West zones) show unequivocal metal zonation (i.e., Cu below Zn below Pb, with a deeper stringer zone). The RW Zone is spatially associated with a large rhyolite body and may represent a more proximal setting during deposition. Massive sulfides have been intersected both above and below the rhyolite unit. This contrasts with the clastic-associated, locally re-sedimented SWZI, SWZII and SZWIII mineralized intersections, which may have formed in topographic lows or on the flanks of the volcanic center.

No large, significant, pipe-like feeder zone has been discovered at the deposit to date. This may be due to a lack of focused, high temperature hydrothermal up-flow through the permeable volcaniclastic rocks, that host SWZII and SWZIII (and part of SWZI) or transposition of originally discordant stringer networks (Steeves, 2013). Local, small stringer zones have been intersected below the RW Zone and SWZI, within more competent feldspar-phyric basalts and further exploration may reveal more feeder-style mineralization and the roots to the system.

7.9 VMS Alteration

Four alteration facies are associated with the mineralized zones; Quartz-Pyrite, Muscovite, Carbonate-Chlorite, and Epidote (Table 7-3). The laterally extensive alteration zone is typical of VMS deposits with permeable volcaniclastic footwalls (Franklin et al., 2005; Large et al., 2001c).

7.9.1 Quartz-Pyrite Facies

Quartz-pyrite facies occurs immediately below massive sulfide mineralization, and forms partially transposed feeder zones to the mineralized lenses. Quartz is extensively recrystallized and forms polygonal, granoblastic texture of varying grain size. Directly underlying the center of SWZI, this quartz-pyrite dominated assemblage contains minor chlorite associated with stringers. The quartz-pyrite facies likely grades into the pyrrhotite-chalcopyrite stringer zone underlying SWZI.

7.9.2 Muscovite Facies

The dominant footwall facies is a muscovite>quartz+pyrite assemblage which forms a large alteration zone referred to as QSP (quartz-sericite-pyrite) schist. In general, alteration intensity increases towards ore. Muscovites throughout the deposit are barium-rich and will be discussed below. Rocks show a simple mineralogy and, in strongly altered zones, have lost nearly all primary textures. Within weakly altered feldspar-phyric basalts, muscovite selectively replaces the igneous feldspar. In the moderately altered volcaniclastic rocks, muscovite replaces the matrix or clasts. Locally quartz alteration may also be selective, replacing clasts, amygdules or matrix material.

7.9.3 Carbonate-Chlorite Facies

Moderately altered rocks containing minor carbonate ± chlorite (up to 10 vol.%) form a stratabound alteration facies 20-40 m below SWZII and just below SWZIII massive sulfide lenses. This facies is also observed locally below SWZI. Carbonate and chlorite are also found enveloping massive sulfide lenses where the rocks are thought to be tuffaceous. This carbonate alteration may be from an earlier alteration phase or even diagenesis of the volcanic precursor.

7.9.4 Epidote Facies

Stratigraphically below the mineralized zones (>100 m), muscovite+quartz+pyrite alteration grades into an epidote+muscovite+quartz+pyrite/pyrrhotite alteration facies. The least altered volcanic rocks typically have a greenschist facies metamorphic mineral assemblage of chlorite, carbonate, feldspar and locally minor quartz and epidote. These least altered rocks are typically relatively calcareous, magnetic and are cut by numerous thin calcite veinlets.

Alteration zonation, except for weakly transposed quartz-pyrite facies ‘conduits’, appears to be parallel stratigraphy and may outline previous lithologies (Steeves, 2013).

Table 7-3: Dominant Alteration Facies

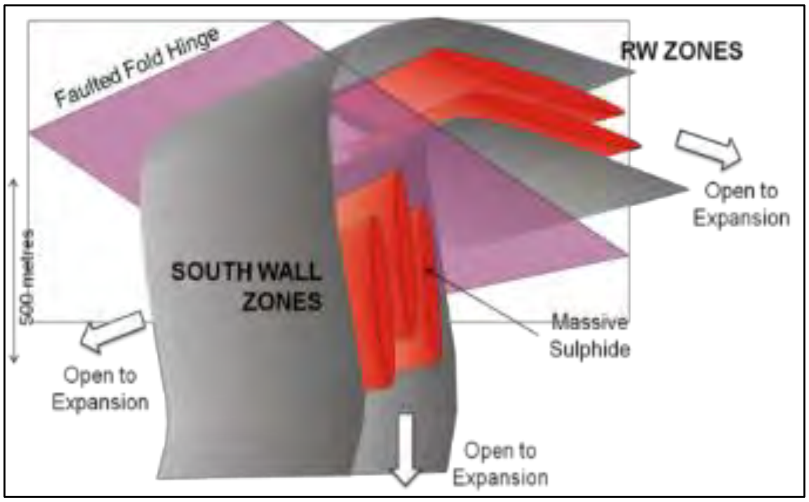
Alteration Facies	Mineral Assemblage	Description
Quartz-Pyrite	Quartz > Pyrite + Muscovite	Underlies massive sulfide and forms feeder zones
Muscovite	Muscovite > Quartz + Pyrite	Dominant, pervasive footwall alteration
Carbonate-Chlorite	Quartz + Muscovite + Pyrite > Carbonate ± Chlorite	Up to 10% carbonate ± chlorite. Stratabound footwall and mineralized horizon alteration.
Epidote	Epidote > Muscovite > Quartz + Pyrite/ Pyrrhotite	Distal alteration/metamorphism

Source: Constantine (2018)

7.10 Principal Mineralized Zones

The Palmer deposit (or main area) consists of six distinctive zones (lenses) of strataform massive sulfide-sulfate. South Wall Zones I, II-III and EM lenses, located on the south-facing, steeply dipping limb of megascopic, deposit-scale anticline, disrupted by significant faulting, are referred to as the ‘South Wall’ (Figure 7-15). The RW Zones, which includes RW East, RW West, and RW Oxide, are located on the north-facing, gently dipping upper limb. The RW Oxide Zone is the near surface equivalent of the RW East Zone where sulfide minerals of massive barite-sulfide mineralization have been oxidized and leached, depleting the zone of copper and zinc and enriching the silver and gold grades.

Figure 7-15: Schematic Diagram of Principal RW and SW Mineralized Zones

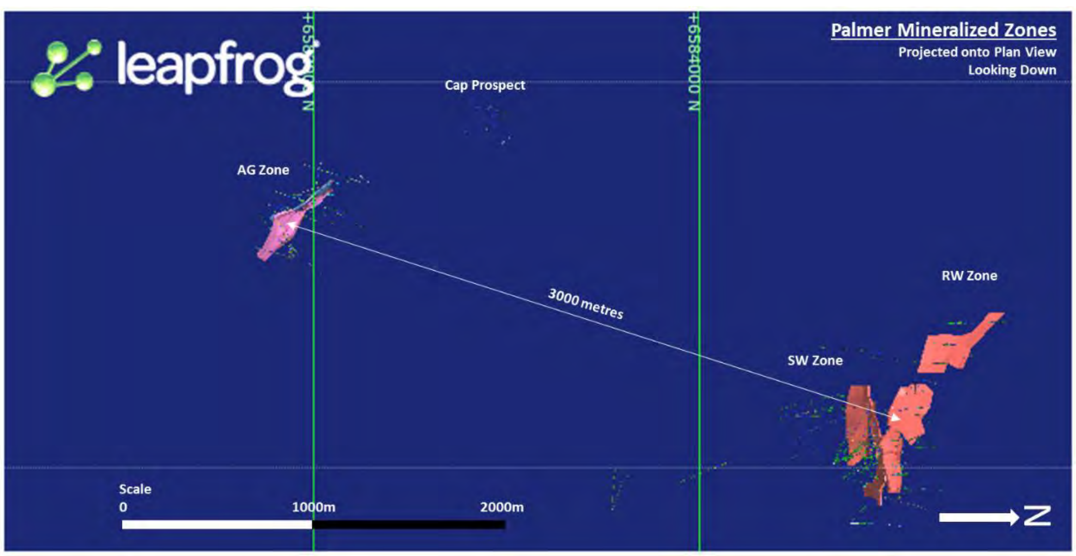


Source: Constantine

The recently discovered AG Zone deposit, which includes the AG Upper and Lower Zones, is located 3,000 m to the southwest, on a steep Nunatak between the Saksaia and South Saksaia Glaciers.

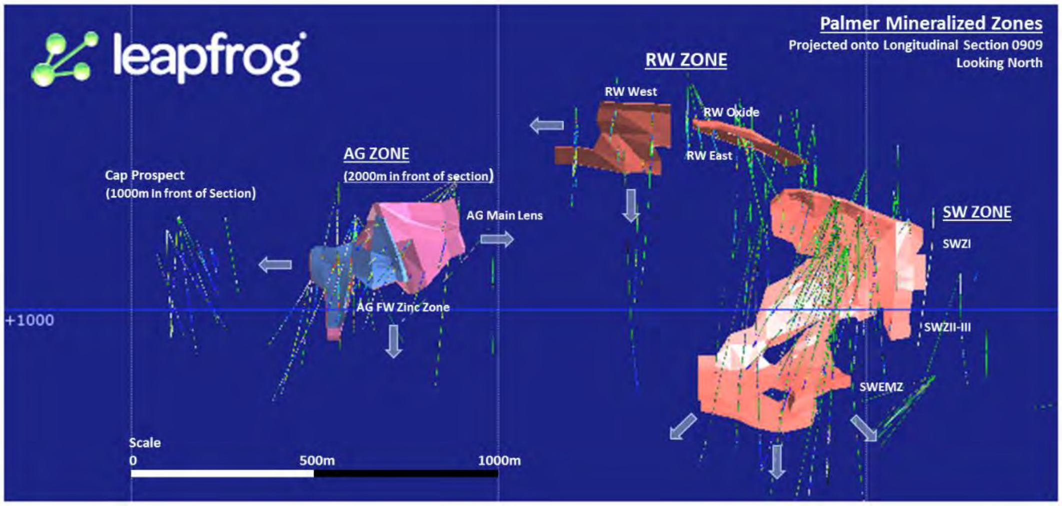
The spatial distribution of the RW/SW and AG Zones can be seen in Figure 7-16, Figure 7-17, and Figure 7-18).

Figure 7-16: 3D Leapfrog View of RW/SW Zones and AG Zones (Plan View – Looking Down)



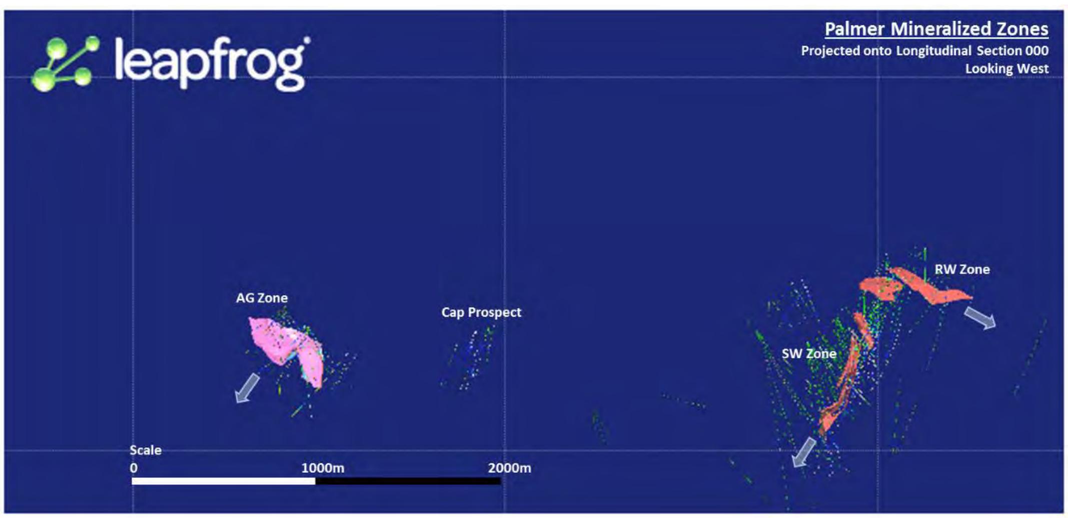
Source: Constantine (2018)

Figure 7-17: 3D Leapfrog View of RW/SW Zones and AG Zones (Vertical Section – Looking North)



Source: Constantine (2018)

Figure 7-18: 3D Leapfrog View of RW/SW Zones and AG Zones (Vertical Section – Looking West)



Source: Constantine (2018)

7.10.1 South Wall Zones

Drilling to date at South Wall has defined four zones of VMS-style mineralization with a total plunge length of ~ 700 m and the total strike length ~ 550 m. All four zones are open to expansion along strike, and both up and down dip.

South Wall Zone I (SWZI) occurs at the up-dip, overturned, edge of the South Wall and consists of a single tabular lens of massive sulfide. SWZI is interpreted to be offset from stratigraphically correlative mineralization in South Wall Zone II ("WZII) and South Wall Zone III (SWZIII) by normal displacement along the high angle 'footwall fault'. Exhalative mineralization occurs at more than one stratigraphic level within a section that measures 40 to 80 m in thickness. SWZIII is located at the stratigraphic top near the contact with overlying argillite, whereas SWZII is stratigraphically lower and is generally the thicker and better developed of the two. In places, the two zones merge or coalesce into a single sulfide body. This feature occurs up-dip toward SWZI and may reflect proximity to the axis or core of the mineralizing system. South Wall EM Zone (SWEMZ) is located down-dip of SWZII and SWZIII and based on additional drilling in 2017 is now demonstrated to be contiguous with them. Like SWZII and SWZIII, the SWEMZ includes an upper and lower exhalative horizon. In one drill hole (CMR14-65), replacement and stringer-like sulfide mineralization links the upper and lower massive sulfide lenses to produce a continuous zone of mineralization with a true width of 65 to 75 m. Continuity of mineralization is good between drill holes, which are generally spaced 50 to 100 m apart.

Details of the SW Zones are given below and shown in Figure 7-16, Source: Constantine (2018)

Figure 7-17 and Figure 7-19, and Figure 7-20. A typical cross-section is shown in Figure 7-19.

7.10.1.1 SWZ1

South Wall Zone I outcrops for over 120 m along the southern slope of Mt. Morlan where it is largely oxidized and leached of sulfide (Greig and Giroux, 2010). The massive sulfide lens is located at the core of the deposit-scale anticline and appears to be bound above and below by faults (thrust fault above, footwall fault below). SWZI has an approximate maximum true thickness of 30 m, dip length of 220 m, and strike length 350 m (based on resource wireframes). It is composed mainly of massive pyrite (Py>Cpy>Qtz), semi-massive pyrite (Py>Qtz>Cpy-Sph) and massive to layered barite mineralization (Brt>Sph~Py>Cpy). Pyrrhotite-chalcopyrite and pyrite±sphalerite stringers overlie and underlie massive pyrite and barite ores. Cherty, tuffaceous, or carbonate-rich ores presently underlie the massive ores, supporting the interpretation that SWZI is overturned and on the south-facing limb of the anticline.

7.10.1.2 SWZII / SWZIII

South Wall Zone II-III outcrops discontinuously for over 100 m as a 2-3 m thick, leached, stratiform massive barite-sulfide and chert horizon (Greig and Giroux, 2010). SWZII-III have an approximate combined maximum true thickness of 24 m, dip length of 350, and strike length of 425 m (based on resource wireframes). Most of SWZII and SWZIII consists of massive, mineralized barite ore, with thin mineralized chert and carbonate horizons stratigraphically above and below. SWZIII also contains significant re-sedimented barite-sulfide mineralization, like that seen in several Kuroko-style deposits in Japan (Eldridge et al., 1983).

7.10.1.3 SWEMZ

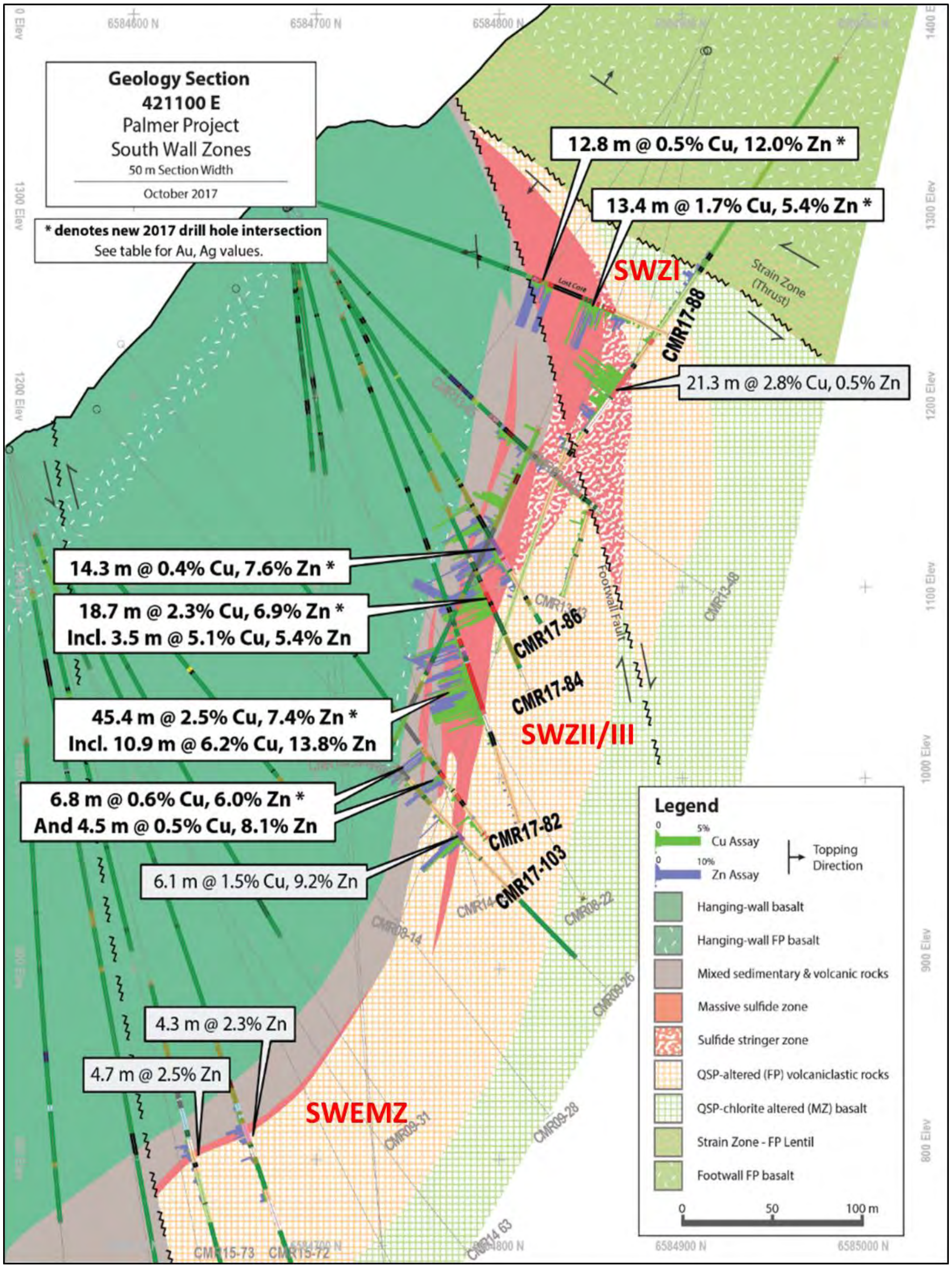
South Wall EM Zone has a more moderate dip than the overlying steeply dipping SWZII and SWZIII, which may suggest the presence of a synclinal hinge to the south. SWEMZ has an approximate maximum true thickness of 74 m, dip length of 250 m, and strike length 370 m (based on resource wireframes). SW EMZ is composed of most of the same mineralization as those present in SWZI, SWZII and SWZIII, and includes both lateral and vertical mineral zonation within the massive sulfide lenses. A well-developed copper-rich footwall stringer zone has yet to be defined at SW EMZ. The down-dip edge of SW EMZ is truncated by a high angle north dipping reverse fault referred to as the Kudo fault. Apparent vertical offset of approximately 200 m is estimated for the fault, and a component of left-lateral strike-slip displacement is also interpreted. Strong QSP alteration and lower grade mineralization (e.g. 7.3 m @ 0.43% copper and 0.46% zinc in hole

CMR15-69) has been identified on the south side of the Kudo fault, suggesting potential for continuation of the South Wall zone.

Mineralogy

Primary mineralogy of the South Wall Zones consists of barite, sphalerite, pyrite, chalcopyrite, quartz, and galena, with lesser calcite, magnetite, pyrrhotite, arsenopyrite, chalcocite, tetrahedrite and tenantite. Typical zoning consists of copper-rich massive pyrite-chalcopyrite mineralization grading laterally and vertically outwards into zinc dominant barite-sphalerite-pyrite +/- chalcopyrite mineralization. Further outward, mineralization locally grades into massive carbonate-sphalerite or variably precious-metal enriched low sulfide chert-barite mineralization. Other types of mineralization include copper-rich pyrite and/or pyrrhotite stockwork, and massive pyrrhotite-chalcopyrite. Continuity of mineralization is good between drill holes, which are generally spaced 50 to 100 metres apart. All three zones are open to expansion along strike, and both up and down dip.

Figure 7-19: Typical Cross-Section through SWZI, SWZII-III and SWZEM (Looking West)



Source: Constantine (2018)

Figure 7-20: Surface Exposure of SWZI with Structural Consultant J. Proffett

Source: Constantine (2018)

7.10.2 RW Zones

The RW Zone, on the upright limb of the prospect-scale anticline, was the initial massive sulfide lens discovered at Glacier Creek. The RW Zone outcrops discontinuously along both the western and southern faces of Mt. Morlan. A coherent rhyolite flow is associated with RW Zone mineralization. Exhalative massive barite-sulfide occurs at both the upper and lower contact of the rhyolite, with RW West predominantly overlying the rhyolite and RW East predominantly underlying the rhyolite or occurring east of where the rhyolite pinches out. The western and eastern sections of the RW Zone have been partially defined and traced to within approximately 100 m of one another but have yet to be demonstrated to be contiguous by drill holes. A large portion of the eastern section, the RW Oxide Zone, has been oxidized and leached of much of its sulfide content. The mineralized zone grades laterally and vertically into tuffaceous and argillaceous rocks, much like the other lenses (Green, 2001).

Details of the various RW Zones are given below and shown in Figure 7-16, Figure 7-17, Source: Constantine (2018)

Figure 7-17, and Figure 7-22. A typical cross-section is shown in Figure 7-21.

7.10.2.1 RW Zone West

RW Zone West has an approximate *maximum true thickness of 6 m, a strike length of 375 m, and a dip length of 325 m (based on resource wireframes)*. It remains open both up and down dip, and along strike.

7.10.2.2 RW Zone East

The RW East Zone has an approximate *maximum true thickness of 11 m, a strike length of 150 m, and a dip length of 165 m (based on resource wireframes).* It remains open along strike and down-dip. Notably, the area between the RW East and the RW West Zones is untested except for one hole (RMC99-14) that intersected 25.2 m of stockwork mineralization @ 0.52% copper and 0.40% zinc.

7.10.2.3 RW Oxide Zone

The RW Oxide Zone transitions to oxide facies mineralization to the south and east consisting of vuggy, porous silica-barite rock in which primary sulfide minerals have been oxidized and largely leached away in the near surface environment. It has an *approximate true thickness of 24 m, a strike length of 190 m, and a dip length of 260 m.* Oxidized parts of the RW Zone typically contain negligible copper and zinc, whereas lead, gold, and silver grades remain similar or higher than those of non-oxidized parts. Locally, remnant blocks or lenses of weakly oxidized to unoxidized RW zone sulfide mineralization are also present.

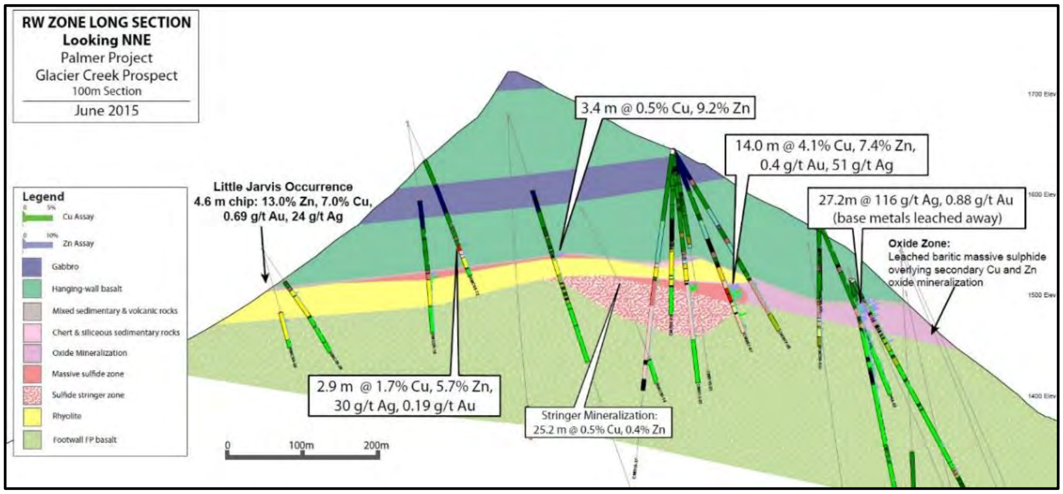
7.10.2.4 RW Zone Surface Occurrences

Well exposed on the west side of Mt. Morlan at the Little Jarvis occurrence, where it can be traced discontinuously along the slope for about 50 m, varying in thickness from 4-5 m to a few tens of cm. On the southeast side of Mt. Morlan, the RW Zone is exposed at the Upper Main and UMP (Upper Merrill Palmer) occurrences, as well as in local exposures in between.

7.10.2.5 Mineralogy

The primary (hypogene) mineralogy of the RW Zone consists of relatively coarse-grained barite, sphalerite, pyrite, chalcopyrite, quartz, and galena, with lesser calcite, magnetite, pyrrhotite, arsenopyrite, tetrahedrite and tenantite. Overall character of the mineralization is like the South Wall Zones. Exhalative chert is common distal to massive barite-sulfide.

Figure 7-21: Typical Longitudinal Section through the RW Zone (Looking NNE) (mod. Green 2001)



Source: Green (2001)

Figure 7-22: Close-up View of RW West Zone Exposed at Surface

Source: Constantine (2018)

7.10.3 AG Zones

7.10.3.1 Introduction

The AG Zone deposit is located three km southwest from the Palmer deposit on the north and northwestern flanks of a steep Nunatak between the Saksaia and South Saksaia Glaciers (aka the Nunatak Prospect Area). Drilling to date has defined a combined total strike length of ~ 550 m within two zones and a vertical dip length of 250 m.

The Nunatak Prospect Area was drilled for the first time in 2017 and targeted the downdip extension of the exposed barite beds while drilling across an interpreted anticline-syncline fold pair to test for possible repetition of the mineralized zone. The drilling was highly successful with the upper sections of initial drill holes intersecting gold- and silver-rich beds of massive barite within an area referred to as the Upper Zone. The Upper Zone consists of a folded massive barite+/-sulfide bed(s) that is structurally offset along a shallow angle (~35^{o}) normal fault, north-side down, from a steeply dipping, relatively planar zone of mineralization at depth.

Deeper sections of initial drill holes, and subsequent drilling thereafter has defined the main body of mineralization at AG Zone deposit. It consists of strataform massive barite-sulfide, the *AG Main Lens*, and feeder-style replacement and stringer mineralization in the stratigraphic footwall referred to as the AG Footwall Zinc Zone.

Details of the various zones are given below and shown in Figure 7-16, Figure 7-17, and Figure 7-18. A typical cross-section is shown in Figure 7-23. An aerial view of the AG Zone deposit appears in Figure 7-24.

7.10.3.2 AG Main Lens

The AG Main Lens has a drill defined strike length of approximately 500 m, vertical extent of approximately 400 m, maximum true thickness of approximately 35 m, and remains open to expansion in most directions (all dimensions as defined by 3D resource wireframes). The zone has a northwesterly trend (~310-320 deg Az) and a dip that changes orientation along strike, presumably the result of a second deformation event. The northwest half of the zone is sub-vertical to locally overturned with a southwest dip, whereas the southeast half is upright with a moderate to steep dip to the northeast. To date, the thickest and best developed mineralization is defined in the southeast from holes drilled in 2018. Mineralization is stratiform and varies from silver-rich massive barite to zinc-lead rich massive sulfide, with both vertical and lateral zonation between these two end members.

7.10.3.3 AG Footwall Zinc Zone

The AG Footwall Zinc Zone has a drill defined strike length of approximately 365 m, vertical extent of approximately 315 m and maximum true thickness of approximately 14 m. It is located 5 to 40 m stratigraphically below the AG Main Lens and has the same general trend and orientation (320 deg Az and 80 deg dip north). The Footwall Zinc Zone is typically zinc +/- lead rich, with significantly lower grade precious metals than observed in the AG Main Lens. Mineralization is characterized by veins, irregular seams and patches, and what appears to be replacement of a volcaniclastic protolith.

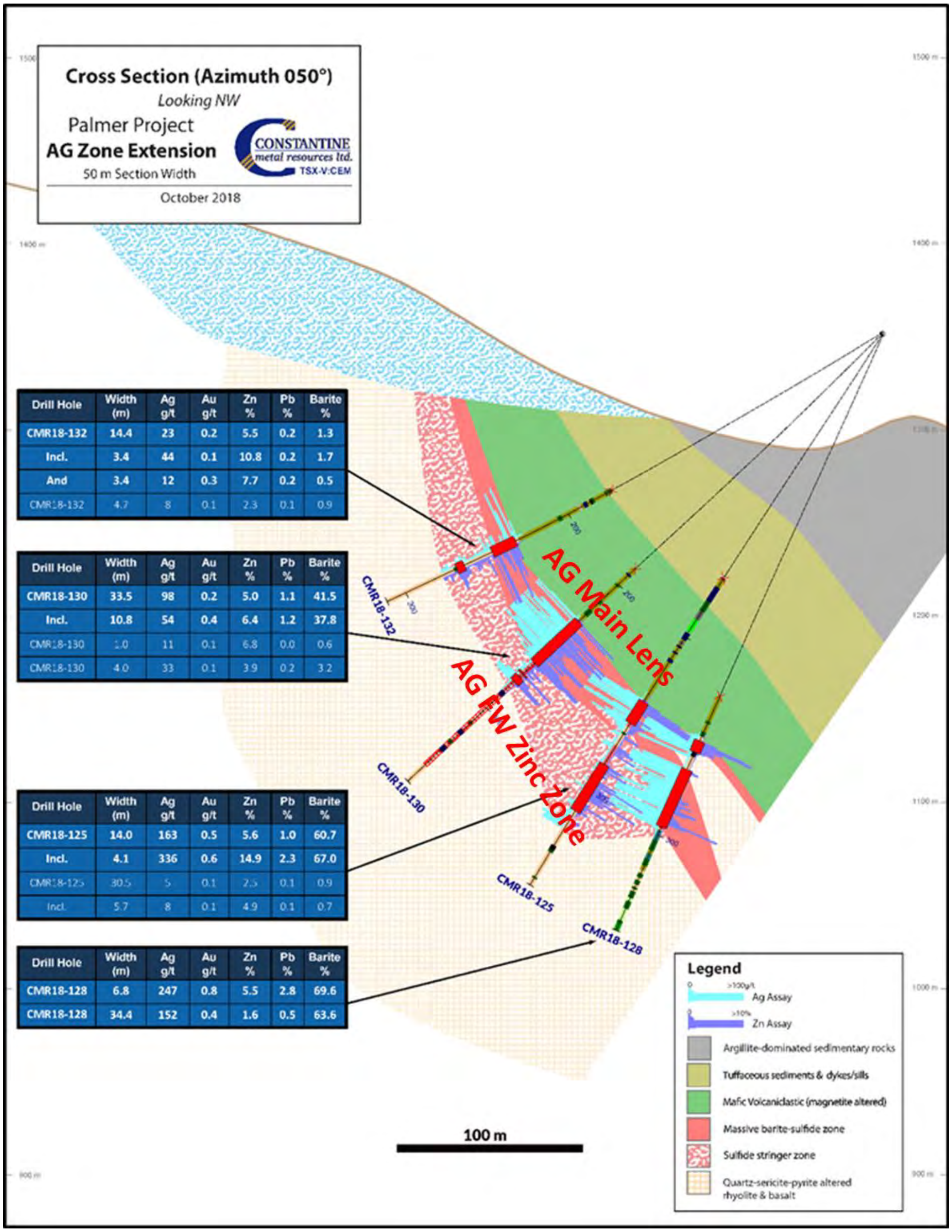
7.10.3.4 Mineralogy

Primary mineralogy of the AG Zone deposit includes zinc occurring in low-Fe sphalerite, lead within galena and within the sulfate anglesite, silver predominantly in tetrahedrite-tennantite and in the rare lead-silver-antimony sulfosalt, diaphorite, lesser copper in chalcopyrite and rare gold in discrete grains of electrum. Barite is abundant in the AG Main Lens and locally as stringers and patches in the footwall. Pyrite is common but occurs in significantly lower concentration than barite within the AG Main Lens. Doherty (2018) noted that nearly all sulfide phases in the AG Zone deposit were fully recrystallized during regional greenschist metamorphism, which resulted in coarsening of grain sizes and the simplification of mineral boundaries.

7.10.3.5 Alteration

Footwall alteration at AG Zone deposit is similar to the Palmer deposit, dominated by sericite-pyrite-quartz alteration increasing in intensity towards mineralization. Hanging wall alteration is typified by magnetite, jasper, Fe-carbonate, minor chlorite, and locally sericite alteration up to tens of meters above mineralization.

Figure 7-23: Typical Cross-Section of AG Main Lens and AG FW Zinc Zone (Looking Northwest)



Drill Hole	Width (m)	Ag g/t	Au g/t	Zn %	Pb %	Barite %
CMR18-132	14.4	23	0.2	5.5	0.2	1.3
Incl.	3.4	44	0.1	10.8	0.2	1.7
And	3.4	12	0.3	7.7	0.2	0.5
CMR18-132	4.7	8	0.1	2.3	0.1	0.9

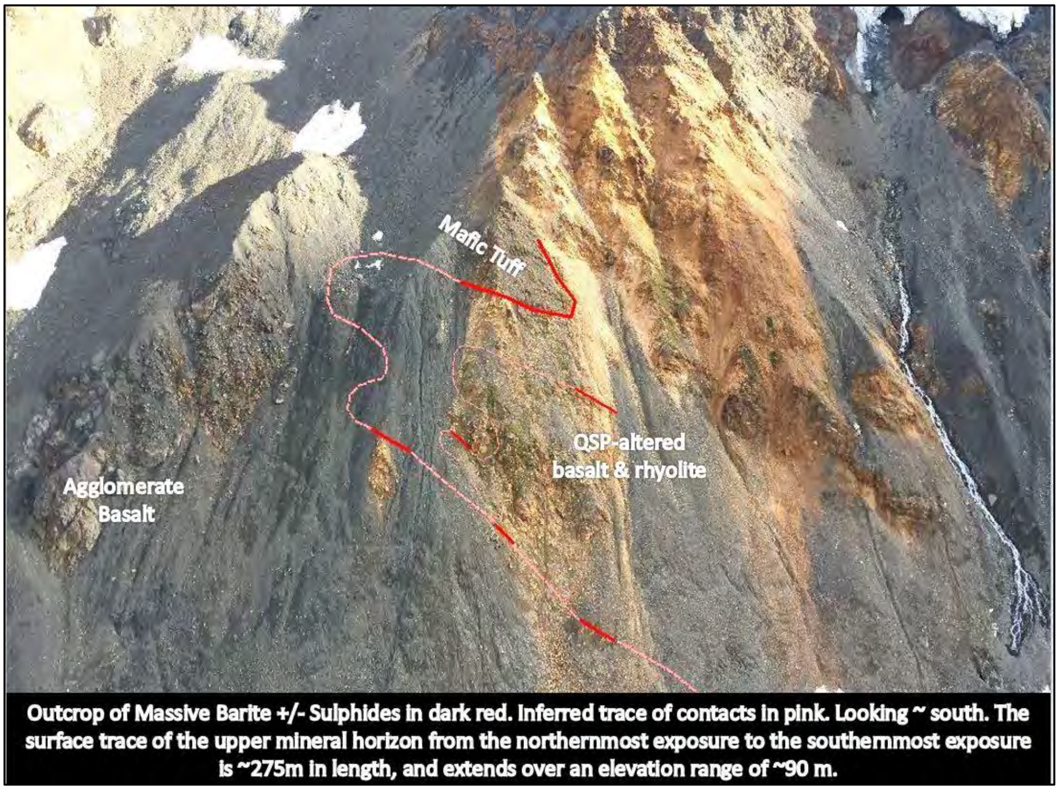
Drill Hole	Width (m)	Ag g/t	Au g/t	Zn %	Pb %	Barite %
CMR18-130	33.5	98	0.2	5.0	1.1	41.5
Incl.	10.8	54	0.4	6.4	1.2	37.8
CMR18-130	1.0	11	0.1	6.8	0.0	0.6
CMR18-130	4.0	33	0.1	3.9	0.2	3.2

Drill Hole	Width (m)	Ag g/t	Au g/t	Zn %	Pb %	Barite %
CMR18-125	14.0	163	0.5	5.6	1.0	60.7
Incl.	4.1	336	0.6	14.9	2.3	67.0
CMR18-125	30.5	5	0.1	2.5	0.1	0.9
Incl.	5.7	8	0.1	4.9	0.1	0.7

Drill Hole	Width (m)	Ag g/t	Au g/t	Zn %	Pb %	Barite %
CMR18-128	6.8	247	0.8	5.5	2.8	69.6
CMR18-128	34.4	152	0.4	1.6	0.5	63.6

Source: Constantine (2018)

Figure 7-24: Aerial View of Nunatak-AG Zone (Looking South)

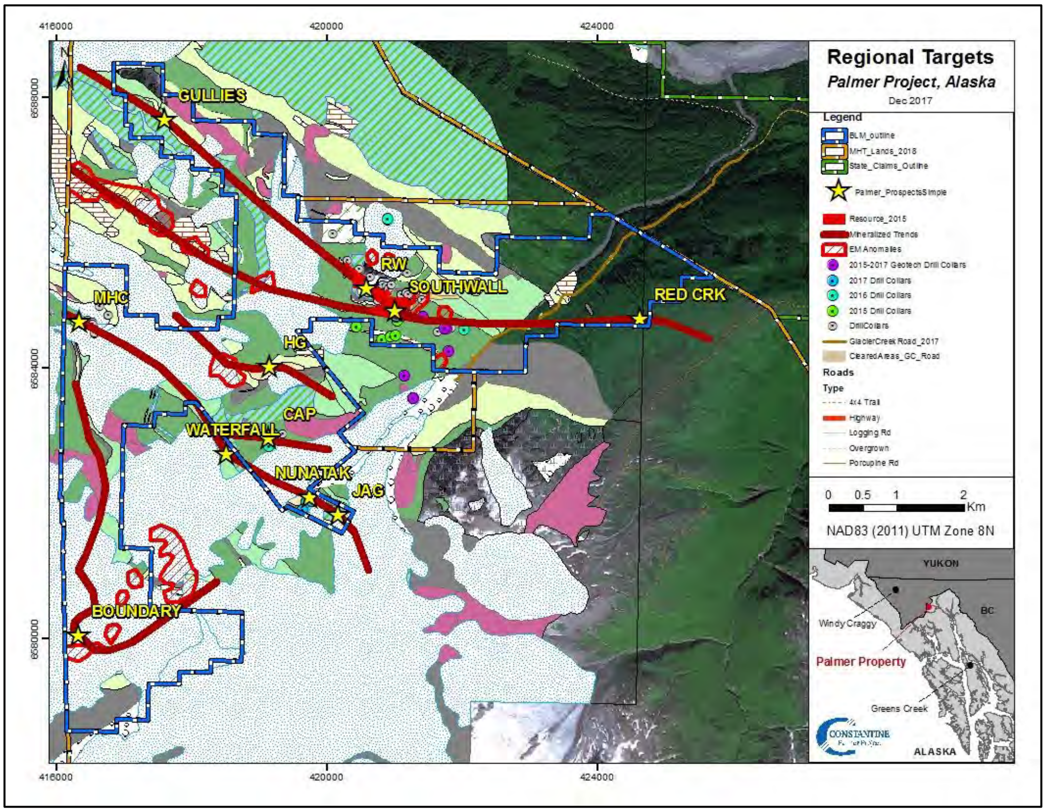


Source: Constantine (2018)

7.11 Regional Mineralized Prospects

The Project hosts a significant number of regional mineralized prospects, six of which (Cap, HG, MHC, Boundary, Red Creek, and Gullies are shown in Figure 7-25 and described briefly below. Only two of the prospects, Cap and MHC, have been tested with drilling.

Figure 7-25: Regional Mineralized Prospects on Palmer Project

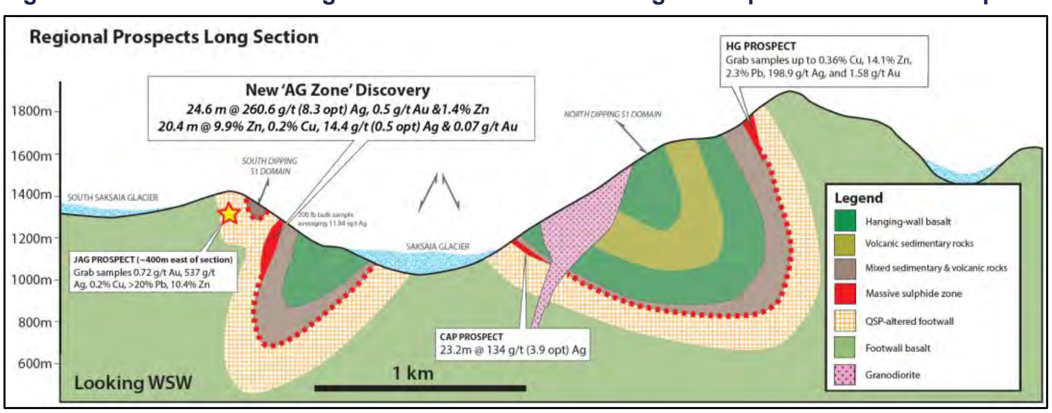


Source: Constantine (2018)

7.11.1 Cap Prospect

The Cap prospect is located 0.5-1.0 km north-northwest of the Nunatak-AG Zone and is interpreted as the same mineralized horizon (Rosenkrans, 1991) (Figure 7-26).

Figure 7-26: North-South Regional Cross-Section showing HG-Cap-Nunatak-JAG Prospects



Source: Constantine (2018)

The Cap prospect is a silver-rich, barite-dominated system that contains locally elevated concentrations of zinc, lead, and gold. Base and precious metal mineralization is hosted within veined and brecciated, intensely QSP altered basalt, which is in turn “capped” by a bed of massive pyritic barite.

The target includes an upper horizon composed of several 5-8 m thick beds of Ag-rich, variably pyritic massive barite within brecciated and veined QSP. Sample highlights include 23.2 m @ 134 g/t Ag in hole CAP01, 90.6 m @ 31 g/t Ag in hole RMC98-01, a surface grab sample of 1,828 g/t Ag, 11.2 g/t Au and 0.83%, and a 5.5 m continuous channel sample averaging 265 g/t Ag and 0.27% Zn. The upper horizon mineralization is open down-dip of the horizon to the northwest and north-northwest. A deeper, less-understood, stratabound horizon of semi-massive to massive sulfide bands/veins hosted within intensely siliceous and altered amygdaloidal basalt returned the highest base metal values along with elevated precious metal values (1.9 m @ 3.75% Zn, 1.91% Pb, 92.1 g/t Ag, 0.47 g/t Au in hole CMR07-04). The lower horizon mineralization is open to the west.

Regionally, the CAP stratigraphy is postulated to be on the southwestern limb of a regional-scale synform, which may connect the mineralized horizon at CAP with the mineralized, discontinuous sulfide-bearing barite lenses exposed in QSP-altered rocks at HG.

The Cap prospect was drilled by Newmont in 1988, by Rubicon in 1998, and the Company in 2016-2017, with the best intercept containing 23.2 m @ 134 g/t Ag, in hole CAP-01, within massive pyritic barite and baritic breccia. Both surface and drill data suggest that the hydrothermal system is diminishing in strength along strike to the northeast, but down-dip and to the southwest (below the ice), it maintains its intensity and has been only partially tested with drilling.

7.11.2 Hanging Glacier (“HG”) Occurrence

The HG occurrence, located 2 km southwest of the Glacier Creek Prospect, consists of discontinuous sulfide-bearing barite lenses up to a few metres thick over 610 m strike length. The mineral horizon dips steeply to the north within the overturned northern limb of a large-scale syncline. Mineralization occurs stratigraphically above an extensive zone of strong QSP +/- chlorite alteration and is overlain by calcareous siltstone and black, slatey limestone and interbedded pillow basalt flows and associated fragmental units. Grab samples contain up to 0.36% Cu, 14.1% Zn, 2.3% Pb, 198.9 ppm Ag, and 1.58 ppm Au (Still et al., 1991). No drilling has been completed at HG.

The west-northwest extension of the HG alteration zone projects beneath the glacier in the direction of the Mt. Henry Clay Prospect, located 2 km away. Results from the 2017 Airborne EM survey show a large conductive response under the ice in this same area (HG West) which warrants follow-up.

The HG occurrence is interpreted to be stratigraphically equivalent to the CAP prospect on the southern limb of the same regional syncline, located 1100m to the south and 700m lower in elevation. A large area of the target mineral horizon is preserved in the syncline between HG and CAP (estimated at +2000m of dip length and +2000m of strike). The majority of this key stratigraphy is accessible to exploration with moderate length holes. If an interpretation of normal offset on the Kudo fault system is assumed, then the HG mineral horizon may also be stratigraphically equivalent to the South Wall – RW zones and is perhaps the fault displaced down plunge continuation of the South Wall system.

7.11.3 Mount Henry Clay (“MHC”) Prospect

High-grade massive sulfide boulders were discovered by prospector Merrill Palmer at the base of a stranded glacier at the Mt. Henry Clay Prospect in 1983. The average grade of several types of boulders, as sampled by the United States Bureau of Mines (USBM) (Still, 1984), are as follows:

- 26 Boulders of Barite-sulfide (zinc-rich):
    - 1% Cu, 0.4% Pb, 19.3% Zn, 38.2 g/t Ag, 0.22 g/t Au, and 20.6% Ba
    - 33% Zn, 2.5% Cu, and 5% Ba from a 6.0 ft (1.83 m) chip of the largest boulder.
- 4 Boulders of massive pyrite and chalcopyrite:
    - 5.18% Cu, 0.03% Pb, 1.00% Zn, 44.1 g/t Ag, tr. Au, and 0.12% Ba
- 6 Boulders of mineralized volcanic host rocks (lacking barite):
    - 2.83% Cu, 0.02 % Pb, 3.90% Zn, tr Au, 9.8 g/t Ag, and 0.41% Ba.
- The mean grade of all the boulders sampled by the USBM:
    - 18.5% Zn, 0.87% Cu, 1.3 oz/ton Ag, 0.02 oz/ton Au, and 5.9% Ba

The MHC mineralization appears to be comprised of primary sphalerite, chalcopyrite, barite, and pyrite with minor late stage galena, tetrahedrite, native silver, and quartz-carbonate gangue. Two principle styles of mineralization occur on the prospect: (1) stratiform Zn-Cu-Ba {sphalerite, chalcopyrite, barite), and (2) stringer (feeder zone) chalcopyrite. MHC is associated with thin intercalated beds of volcanic flows, carbonates, and elastic rocks and conglomeratic textures are frequently observed in the sulfide boulders.

The MHC massive sulfide target has not been located in outcrop although the high-grade Zn-Cu-rich and precious metals-enriched massive sulfide boulders found scattered along the margins and near the terminus of the MHC glacier suggest a source beneath the glacier. Thirteen (13) holes for a total of 2,957 m; seven holes by Kennecott Exploration, four holes by Granges Exploration Ltd, and two holes by Rubicon Minerals Corporation. The drilling identified two mineralized horizons beneath the MHC Glacier but did not intersect mineralization with grades equivalent to those in the boulders. Several holes did intersect lower-grade mineralization within broad pyrite-sericite alteration zones, including 49.1 m @ 0.19% Cu in hole K85-3, 10.7 m @ 0.44% Cu in hole K84-2, and 36.6 m @ 0.29% Cu in hole G89-9 (Still et al., 1991 and Rubicon, 1998). Annually retreating ice led to a discovery by Rubicon Minerals Corporation of an intensely foliated chlorite-sericite ~alteration zone containing pyrite-chalcopyrite stockwork veins that was dubbed the ‘P2’ zone. It is speculated that the P2 zone may represent footwall feeder mineralization and alteration to the horizon from which the high-grade boulders were sourced (Bull, 1998). As ice continues to retreat, high-grade boulders appear to be vectoring back towards their bedrock source. Results from the Company’s 2017 MHC subglacial sampling program showed a distinct anomalous Zn trend and a potential sub-ice sulfide source in the vicinity of borehole BH13. Further work was recommended to target this area.

7.11.4 Boundary Occurrence

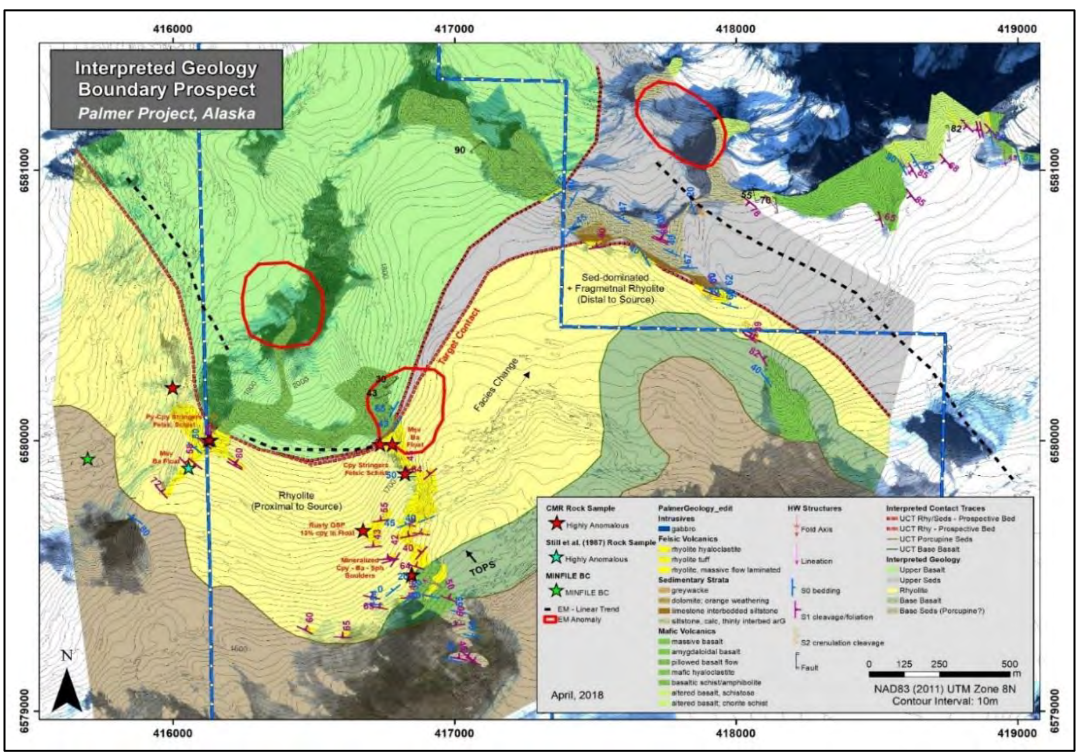
The Boundary target is defined by mineralized boulders, favourable stratigraphy including the thickest bed of rhyolite on the Palmer property, the presence of chalcopyrite-stringers and anomalous barium in outcrop, altered volcanics, and EM anomalies at depth (Figure 7-27).

The Boundary prospect is exposed as a ridge of outcrop in a large ice field near the international border. It consists of chalcopyrite mineralization and anomalous barium within quartz-sericite-pyrite schist and rhyolite that is intermittently exposed over a distance of 2-3 km. A marker bed of iron-stained meta-sediments (phyllite, pelitic schist, argillaceous sediments) are overlain by unaltered hanging wall basalt and underlain by the altered rhyolite. The stratigraphy may be correlative to occurrences outside the Property (e.g. the Herbert showings), located by Stryker resources Ltd. on the Canadian side of the border (McDougal et al., 1983).

Grab samples returned up to 6.6% Cu, 3610 ppm Zn, 12 ppm Ag, and 1.98 ppm Au (Wakeman, 1995). More recently, prospecting by Constantine has documented barite-sulfide boulders grading up to 2.28% Cu, 19.7% Zn, 49.7 ppm Ag and 0.61 ppm Au. The mineralized boulders are located directly downslope from the fall-line of the upper contact of the rhyolite.

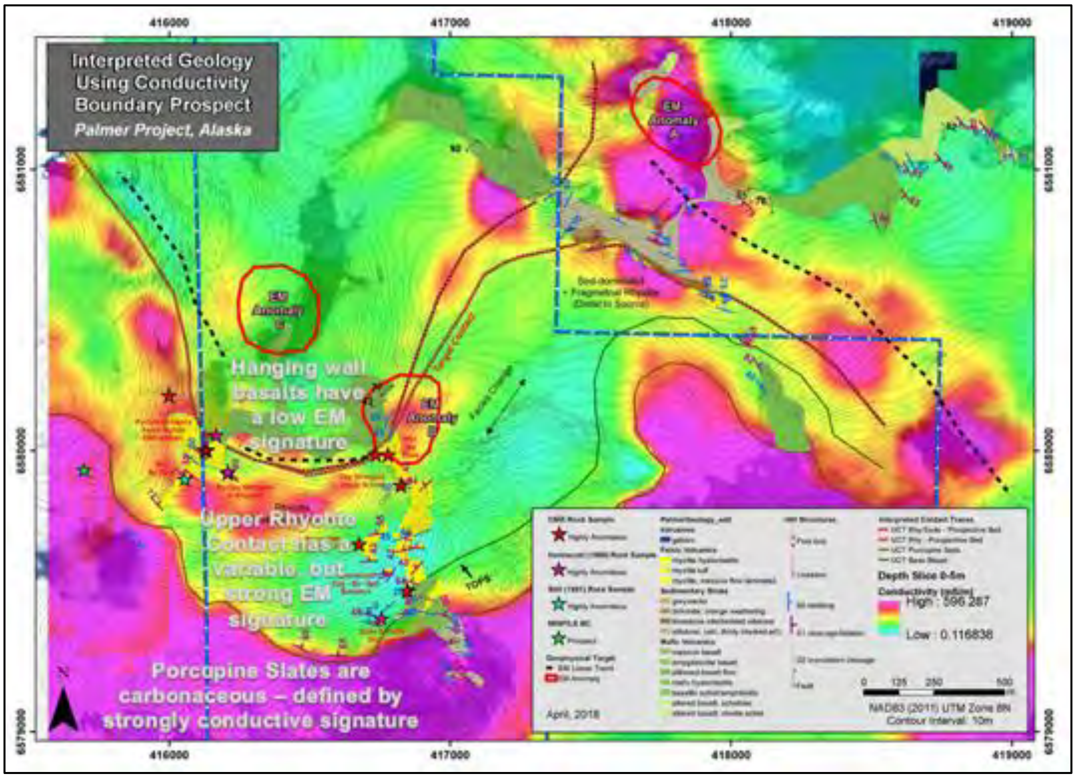
Two EM anomalies are located at depth, somewhat along the projection of the upper rhyolite contact, which may correlate to massive sulfide mineralized bodies (Figure 7-28). Rhyolite is documented to occur spatially proximal to base- and precious-metal mineralization elsewhere on the Palmer property (RW Zone, AG Zones, MHC). Regionally, the Boundary stratigraphy is near the base of the Triassic section.

Figure 7-27: Boundary Target with Interpreted Geology



Source: Constantine (2018)

Figure 7-28: Boundary Target with Electromagnetic Results (Inversion) (Depth 0-5 m)



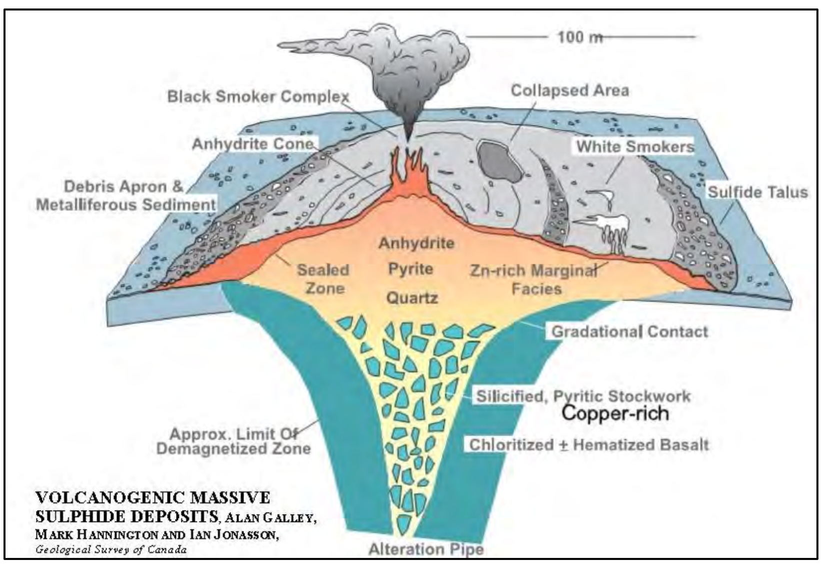
Source: Constantine (2018)

8 Deposit Types

The Palmer Project is host to volcanogenic massive sulfide (VMS) style mineralization. As a group, VMS deposits are stratiform accumulations of sulfide minerals that formed on or near the seafloor, by precipitation near a discharge site, or vents, of hydrothermal fluids (Franklin et al., 1981, see Figure 8-1). They form polymetallic mineralized bodies, and commonly contain economic concentrations of Zn, Cu, Pb, Ag, and Au. Many VMS deposits occur in clusters, with several individual mineralized bodies occurring within a radius of a few km, and they are often stacked above one another at different stratigraphic levels. Late Triassic, rift-related volcanic and sedimentary rocks within the Alexander Terrane are host to numerous VMS occurrences, prospects, and deposits throughout southeast Alaska and northwest British Columbia. Major deposits in the belt include the Windy Craggy Cu-Co-Au deposit, the fourth largest VMS deposit by size in the world, and the largest of the copper-rich (Besshi-type) VMS deposits, and the Greens Creek Ag-Zn-Pb-Au Mine, one of the world's richest large tonnage VMS deposits (Galley et al., 2007).

The Project most closely resembles the Greens Creek deposit. Significant differences exist however, most notably the much higher copper/zinc and zinc/lead ratios present at Palmer, which more closely resemble deposits in Noranda, Quebec or at Kidd Creek, Ontario. Zinc is the dominant base metal at both the Greens Creek deposit (Swainbank et al., 2000) and the Palmer. Silver grades are locally similarly enriched but are much lower within the mineral resource area at Palmer than at Greens Creek. Gold grades are commonly elevated at the Palmer (e.g. 0.5 to 1.5 g/t) but are lower than the average at Greens Creek (0.12 oz/ton (4.11 g/t)). Barite is common in both and is the dominant gangue mineral for parts of the ore body at the Greens Creek deposit. Deformation at the Greens Creek deposit is much more ductile in style than at the Palmer, resulting in sometimes tight and complex folding of the mineralized zones and host stratigraphy at Greens Creek.

Figure 8-1: Cross-sectional View of Typical VMS Deposit



Source: Franklin et al. (1981)

9 Exploration

Constantine, including its current JV partner, Dowa Metals & Mining Co. Ltd., and predecessor company Rubicon, have carried out approximately US$ 46.1 million in exploration work over a 20-year period from 1998 to the end of 2018, including US$ 9.1 million in Project expenditures incurred in 2018. The work has included prospecting, regional and detailed geological mapping, structural studies and thesis work, line cutting, soil and rock sampling, airborne, ground and downhole geophysics, satellite imagery, diamond drilling, geotechnical, metallurgical, engineering and environmental baseline studies, and access road construction.

During the June to September 2018 summer field season, continued exploration included detailed geological mapping at the Nunatak and Boundary Prospects, and Terminus, East Pump Valley, Khyber Pass Areas, diamond drilling at the Palmer and AG Zone deposits and the Boundary Prospect, glaciology studies included ground penetrating radar over the Saksaia and South Saksaia Glaciers, ongoing engineering, environmental, geotechnical, permitting and community relations-related work.

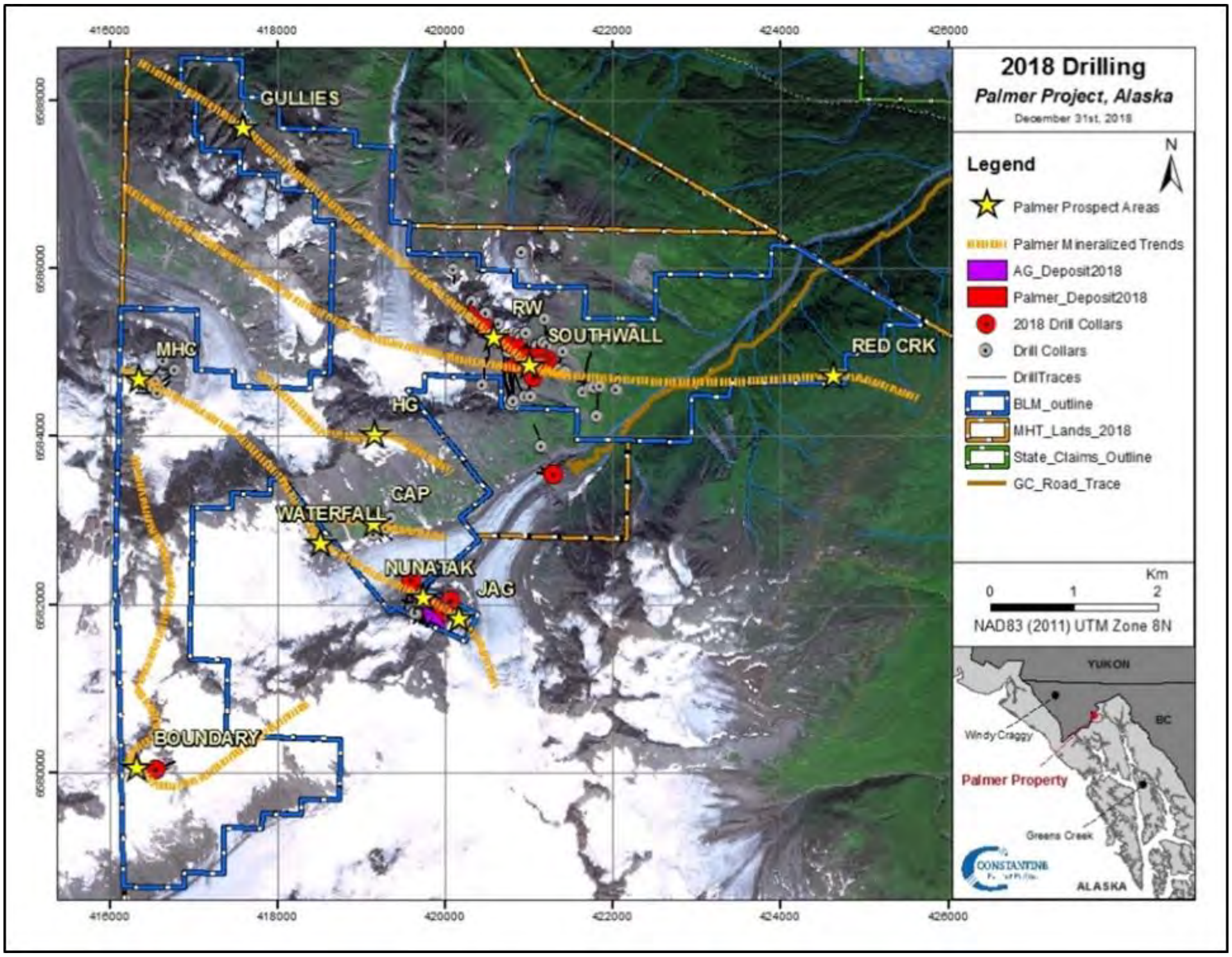
The Glacier Creek access road was extended 0.92 km to the head of Glacier Creek valley and the terminus of the Saksaia Glacier, for a total length of 6.83 km (see Figure 9-1 for an aerial view and Figure 9-2 for a plan map). This final road section provides exploration access to MHT fee-simple lands near the base of the Palmer deposit mineral resource area with a total road distance to Haines, Alaska of approximately 60 km.

Further details of the 2018 Exploration and Development Programs can be found in Gray and Cunningham-Dunlop (2019) and in Section 6.2.12.

Figure 9-1: Glacier Creek Access Road, Lower Glacier Creek Valley

Source: Constantine (2018)

Figure 9-2: Plan Map of Palmer Exploration Project showing Key Prospect Areas (2018 DDH in red)



Source: Constantine (2018)

10 Drilling

Diamond drilling programs have been carried out on the Project over a period of 40 years since 1979. Total cumulative diamond drilling by the Company since 2006 is 60,200 m in 156 drill holes. Total cumulative diamond drilling on the Project by all operators is 67,745 m in 193 drill holes (Table 6-6).

No new diamond drilling has been completed by the Company since the issuance of its most recent NI43-101 technical report on January 31st, 2019 (Gray and Cunningham-Dunlop, 2019).

In its most recent phase of drilling in June to September 2018, the Company carried out exploration, definition and geotechnical programs on the Palmer deposit (comprised of the RW and SW Zones), the AG Zone deposit, and the Boundary Prospect. The primary goals of the 2018 drill program were:

- Definition and expansion drilling on the Palmer deposit mineral resource and conversion of Inferred to Indicated categories within the South Zone
- Infill/step-out drilling on the AG Zone deposit to support an initial mineral resource estimate
- Testing of new property-wide targets such as the Boundary Prospect
- Geotechnical drilling to support ongoing engineering and permitting for the Option 7 proposed exploration portal site near the terminus of the Saksaia Glacier

The Company drilled a total of 30 diamond drill holes totaling 10,094 m, including 28 exploration holes (CMR18-108 to 134B; 108W) totaling 9,694 m and two geotechnical holes (GT18-11 to 12) totaling 400 m. The Company met its seasonal objectives with encouraging drill hole intercepts at both the Palmer deposit and the AG Zone deposit (See Significant Intersections in Table 10-1 and Table 10-2. The new results expanded the SW Zone at the Palmer deposit to the west and confirmed continuity of grade and width down-plunge to the southwest towards the deeper SW EM zone. The Palmer deposit is open along strike and at depth and the potential to expand the known zones is considered good. The 2018 drill results also expanded the strike length of AG Zone deposit mineralization to over 550 meters and included multiple, thick, high-grade intersections of massive barite-sulfide mineralization with excellent continuity between holes. The AG Zone deposit remains open along strike and at depth.

Based on the results of the 2017 and 2018 drilling campaigns, the Company completed the following tasks:

- An updated mineral resource estimate for the Palmer deposit (Figure 10-1) by the Company on September 27th, 2018 (based on a data cut-off of May 1st, 2018) with a NI43-101 Technical Report issued by the Company on November 9th, 2018 (Gray and Cunningham-Dunlop, 2018); and
- A new mineral resource estimate for the AG Zone deposit (Figure 10-2) by the Company on December 18th, 2018 (based on a data cut-off of November 15th, 2018) with a NI43-101 Technical Report issued by the Company on January 31st, 2019 (Gray and Cunningham-Dunlop, 2019)

Further details of the 2018 Drilling Program can be found in Gray and Cunningham-Dunlop (2019) and in the History Section 6.2.12. Drill hole collar information can be found in Appendices I and II, significant assay intercepts are in Appendices III and IV, and representative drill cross-sections can be found in Appendix V.

Table 10-1: 2018 Significant DDH Assay Intercepts – Palmer Deposit – SW Zones

Drill Hole	From (meters)	To (meters)	Intercept (meters)	Cu %	Zn %	Pb %	Ag (g/t)	Au (g/t)	Zone
CMR18-108	328.4	343.9	15.5	1.61	4.76	<0.01	24.6	0.1	SW Zone II
Includes	**328.4**	**332.5**	**4.1**	**0.3**	**15.93**	**<0.01**	**5.3**	**0.02**	**SW Zone II**
Includes	**337.8**	**343.9**	**6.1**	**3.64**	**0.3**	**<0.01**	**56**	**0.22**	**SW Zone II**
CMR18-111	319.1	320.4	1.3	0.17	1.72	0.04	7.1	0.05	SW Zone II
CMR18-113	302.2	313.1	10.9	0.29	1.82	0.02	6.1	0.03	SW Zone II
Includes	**302.2**	**307.5**	**5.3**	**0.48**	**1.73**	**0.01**	**9.9**	**0.04**	**SW Zone II**
CMR18-123				*No significant intersection*					
CMR18-126				*No significant intersection*					
CMR18-127				*No significant intersection*					
CMR18-129	145.8	158.0	12.2	*0.36*	*3.1*	*0.01*	*7*	*0.07*	SW Zone II
CMR18-129	162.5	168.5	6.0	*0.39*	*3.06*	*0.02*	*5.15*	*0.08*	SW Zone II
CMR18-129	174.5	178.4	3.9	*1.44*	*1.05*	*0.01*	*16.96*	*0.13*	SW Zone II

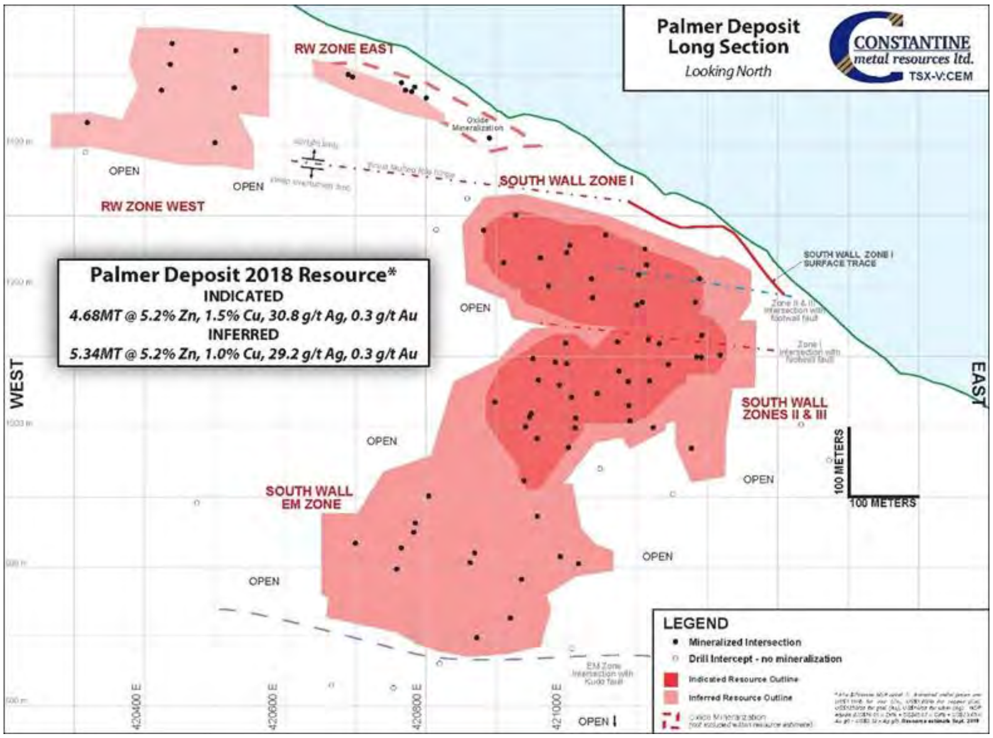
Drill intercepts reported as core lengths are estimated to be 50-100% true width. Bold text denotes intervals at >2 meters at >2% copper and/or 10% zinc OR >20 meters at >1% copper and/or 5% zinc and/or 100 g/t Ag. Averages are length x density weighted using density data obtained for each sample within a given interval (where density data is available). Length x density averages more accurately represent the metal content of a given interval and is common practice in reporting on massive sulfide deposits because of the wide range of densities they exhibit. The Company has adopted length x density weighting as standard procedure for this project. For QAQC and other sample related procedures please visit the Company's technical report entitled, "NI 43-101 Technical Report and Updated Resource Estimate for the Palmer Exploration Project, Porcupine Mining District, Southeast Alaska, USA" dated November 9th, 2018, on SEDAR at www.sedar.com. Darwin Green, VP Exploration for Constantine Metal Resources Ltd. and a qualified person as defined by Canadian National Instrument 43-101 has reviewed and verified the information within this table.

(2) Part of a continuous 43 meter wide intersection from 143.7m to 186.7m that includes 16.8 meters of lost core (not included in reported assay intersections)

(3) The 50.9 meter intersection represents the total width of the mineralized zone, consisting of 3 separate but closely spaced intersections totaling 28.9 meters, separated by intervals up to 9.5 meters of below cut-off grade

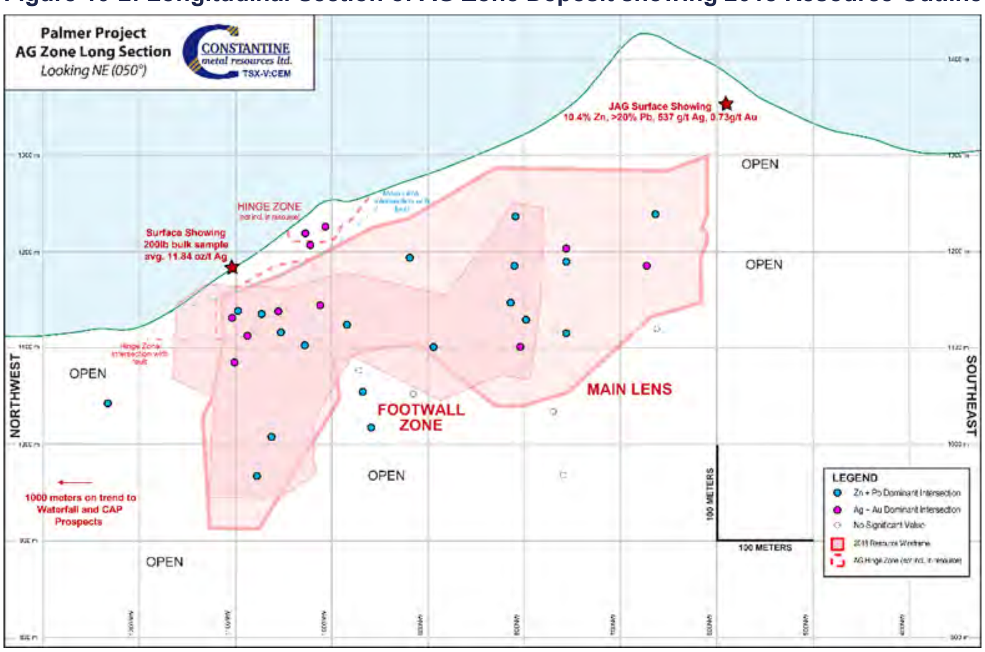
Source: Constantine (2018)

Figure 10-1: Longitudinal Section of Palmer Deposit showing 2018 Resource Outline



Source: Constantine (2018)

Figure 10-2: Longitudinal Section of AG Zone Deposit showing 2018 Resource Outline



Source: Constantine (2018)

Table 10-2: Significant DDH Assay Intercepts – AG Zone Deposit

Drill Hole	From (meters)	To (meters)	Intercept (meters)	Zn %	Pb %	Cu %	Ag (g/t)	Au (g/t)	$BaSO_4$ (% Barite)	Zone
CMR18-109	203.6	208.4	4.8	3.59	1.61	0.10	436	1.25	61.6	AG Zone
CMR18-109	219.2	231.7	12.5	5.20	0.72	0.27	217	1.81	29.7	AG Zone
Includes	219.2	223.4	4.2	3.87	1.09	0.08	388	3.08	50.5	AG Zone
CMR18-110	**238.8**	**282.1**	**43.3**	**6.54**	**2.51**	**0.16**	**143**	**0.47**	**41.1**	**AG Zone**
Includes	**253.3**	**282.1**	**28.8**	**8.98**	**3.55**	**0.24**	**141**	**0.49**	**21.5**	**AG Zone**
CMR18-112				*No significant intersection*						
CMR18-114	204.8	226.1	21.3	1.03	0.42	0.07	92	0.47	55.0	AG Zone
CMR18-115				*No significant intersection*						
CMR18-116	73.5	75.6	2.1	1.74	0.86	0.15	49	0.12	8.4	AG Zone
CMR18-116	121.1	124.0	2.9	1.77	0.78	0.02	14	0.09	2.8	AG Zone
CMR18-116	130.7	132.6	1.9	1.59	1.45	0.04	17	0.19	1.6	AG Zone
CMR18-116	347.2	349.8	2.6	2.31	0.03	0.03	1	0.02		AG Zone
CMR18-118				*No significant intersection*						
CMR18-120				*No significant intersection*						
CMR18-124				*No significant intersection*						
CMR18-125	242.7	256.7	14.0	5.58	1.04	0.10	163	0.48	60.7	AG Zone
Includes	**243.6**	**247.7**	**4.1**	**14.87**	**2.33**	**0.21**	**336**	**0.63**	**67.0**	**AG Zone**
CMR18-125	283.6	314.1	30.5	2.49	0.08	0.06	5	0.05	0.9	AG Zone
Includes	299.7	305.4	5.7	4.85	0.10	0.09	8	0.08	0.7	AG Zone
CMR18-128	243.4	250.2	6.8	5.46	2.79	0.06	247	0.79	69.6	AG Zone
CMR18-128	260.6	295.0	34.4	1.56	0.53	0.04	152	0.39	63.6	AG Zone
CMR18-130	230.3	263.8	33.5	4.97	1.11	0.22	98	0.39	41.5	AG Zone
Includes	253.0	263.8	10.8	6.37	1.23	0.41	54	0.18	37.8	AG Zone
CMR18-130	272.1	273.1	1.0	6.79	0.02	0.23	11	0.09	0.6	AG Zone
CMR18-130	275.1	279.1	4.0	3.90	0.18	0.39	33	0.14	3.2	AG Zone
CMR18-131				*No significant intersection*						
CMR18-132	232.9	247.3	14.4	5.48	0.17	0.41	23	0.16	1.3	AG Zone
Includes	**234.8**	**238.2**	**3.4**	**10.79**	**0.18**	**0.36**	**44**	**0.13**	**1.7**	**AG Zone**
Includes	**243.9**	**247.3**	**3.4**	**7.72**	**0.19**	**1.02**	**12**	**0.33**	**0.5**	**AG Zone**
CMR18-132	264.3	269.0	4.7	2.34	0.09	0.07	8	0.09	0.9	AG Zone
CMR18-133	165.7	178.6	12.9	0.85	0.27	0.02	37	0.15	1.8	AG Zone
CMR18-133	186.5	190.8	4.3	2.01	0.08	0.02	13	0.06	4.3	AG Zone
CMR18-134B	192.2	195.9	3.7	0.92	0.31	0.07	78	0.26	35.7	AG Zone

Source: Constantine (2018)

11 Sample Preparation, Analyses and Security

11.1 Sample Collection and Security

Since 2006, samples collected have been prepared by properly trained and supervised Constantine employees at a secure facility on site. Sample collection and security has been undertaken in accord with currently acceptable methods and standards in use in the mining exploration industry. The sampling methodology and approach applied by Constantine are deemed by the Authors to be appropriate for the styles of mineralization exhibited on the Project.

11.1.1 Soil Geochemical Sample Collection

Any soil geochemical samples were collected from the B horizon, or C horizon in underdeveloped soil if on talus slopes, at an average depth of 10 to 15 centimetres. A shovel or mattock was used to dig a hole at each station, and the soil was placed in a standard kraft paper soil sample bag that was labeled with a sample number.

11.1.2 Rock Geochemical Sample Collection

Any surface rock geochemical sampling included grab samples of alteration and mineralization in outcrop and float, and randomly spaced grab samples of outcrop for alteration and lithogeochemical discrimination studies. Rock chip sampling was carried out along outcrops of prospective rocks, for geochemical characterization.

11.1.3 Drill Core Sample Collection

All 2018 drill core samples were selected by core logging geologists based on mineralization, alteration and lithology observations. All samples were analyzed by 4-acid digestion multi-element ICP, most were analyzed by gold fire assay, and select samples were analyzed with a complete lithogeochemical characterization package, including whole rock by XRF. This is used to obtain major oxide XRF data plus additional elements (i.e. rare earths, volatiles, and some trace elements such as Hg and Tl) and is particularly useful for identifying different basaltic flows. Samples through significant mineralization were also analyzed for barium by XRF

Samples were prepared by properly trained and supervised Constantine employees at a secure facility on site. Samples of drill core were cut by a diamond blade rock saw, with half of the cut core placed in individually labeled and sealed polyurethane bags and half placed in the original core box for permanent storage. Sample lengths typically vary from a minimum 0.3 m interval to a maximum 2.0 m interval, with an average 1.0 to 1.5 m sample length. Samples were placed in sealed woven plastic bags and driven by Constantine personnel to Manitoulin Transport in Whitehorse, Yukon, Canada. Drill core samples were trucked by Manitoulin to the ALS Geochemistry prep facility in Kamloops, BC, Canada, and analyzed at ALS Minerals Canada Ltd., in North Vancouver, BC, Canada. Rock samples were trucked by Manitoulin to ALS Minerals Canada Ltd., in North Vancouver, BC, Canada for prep and analysis.

Sample collection and security were undertaken in accordance with currently acceptable methods and standards in use in the mining exploration industry.

The sampling methodology and approach applied by Constantine are deemed by the Authors to be appropriate for the styles of mineralization exhibited on the Project.

11.2 Sample Preparation and Analyses

All 2018 drill core samples were prepped by ALS Geochemistry in Kamloops, British Columbia, Canada and analyzed by ALS Minerals Canada Ltd. (ISO 9001) in North Vancouver, British Columbia, Canada.

For samples not being analyzed by metallic screening, the raw samples were crushed in an oscillating steel jaw crusher (>70% of the sample passing through a 6 mm screen), followed by a riffle split of 250 grams using a Boyd crusher/rotary splitter combination, then pulverized in a chrome steel ring mill (>85% of the sample passing through a 75 µm screen) (ALS prep codes: CRU-21q, PUL-31, SPL-22Y, WEI-21).

For samples analyzed by metallic screening, the raw samples were crushed in an oscillating steel jaw crusher (>70% of the sample passing through a 2 mm screen), followed by a riffle split of 1000 grams using a Boyd rotary splitter, then pulverized in a chrome steel ring mill (>85% of the sample passing through a 75 µm screen) (ALS prep code: PREP-31B).

Gold analysis was performed on a 30 g sub-sample using ALS Method Au-AA23; fire assay fusion with atomic absorption spectroscopy (AAS) finish.

Metallic screening (ALS method ME-SCR24) was performed for analysis of samples with coarse-grained Au and Ag mineralization. The method utilizes 1000 g of prepared pulp material screened through a 100 micron stainless steel mesh to separate the oversize fractions. Any material larger than 100 microns is analyzed by fire assay fusion with gravimetric finish and reported as the positive fraction result. Material that is smaller than 100 microns is then homogenized, and two sub-samples are analyzed using fire assay with gravimetric finish. The average of the two sub-samples' gravimetric results is reported as the negative fraction result. Results of all three analyses are used to calculate the metal content across the positive and negative fractions.

Four acid digestion ICP (ALS method ME-ICP61) was performed for analysis of 33 elements: Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, and Zn. The method utilizes inductively coupled plasma-atomic emission spectrometry (ICP-AES) conducted on 0.25 g of prepared sample digested in perchloric, nitric, hydrofluoric and hydrochloric acids. For samples in which Cu, Zn, Pb, or Ag values exceeded the ME-ICP61 upper detection limit, ALS Method OG62 was utilized – a four-acid ICP-AES technique calibrated for potentially economic grade mineralization. For samples in which Zn or Ag exceeded the OG62 upper detection limits, Zn titration (ALS method Zn-VOL50) or Ag by fire assay and gravimetric finish (Ag-GRA21) were used, respectively.

A complete characterization package (ALS method CCP-PKG03) that consists of several methods was performed for the analysis of 65 oxides and elements. This analytical package also includes measurement of loss on ignition (LOI). Individual methods consist of ALS methods ME-XRF26, ME-MS81, ME-4ACD81, ME-MS42, and ME-IR08. ALS method ME-XRF26 is a 13-element oxide package where the sample is prepared utilizing lithium borate fusion into a fused disc where it is then analyzed by XRF spectrometry. This method yields Al2O, BaO, CaO, Cr2O3, Fe2O3, K2O, MgO, MnO, Na2O, P2O5, SO3, SiO2, and TiO2. The ALS Method ME-MS81 is a 31-element package that includes Ba, Ce, Cr, Cs, Dy, Er, Eu, Ga, Gd, Hf, Ho, La, Lu, Nb, Nd, Pr, Rb Sm, Sn, Sr, Ta, Tb, Th, Tm, U, V, W, Y, Yb, Zr is a lithium borate fusion technique followed by acid dissolution and ICP-MS analysis. The elements As, Bi, Hg, In, Re, Sb, Se, Te and Tl were analyzed using the aqua regia digestion and ICP-MS method (ALS method ME-MS42) while elements C, and S were analyzed by combustion furnace (ALS method ME-IR08). The ME-4ACD81 is an identical method to the main four acid digestion ICP method (ME-ICP61) except it yields results for only 10 elements: Ag, Cd, Co, Cu, Li, Mo, Ni, Pb, Sc, Zn. This method is already built into the whole-rock characterization package in ALS Minerals' price schedule and is more cost-efficient to process this method despite the duplicate analyses.

The barium analysis utilized lithium borate fusion into fused discs for XRF analyses (ALS Method ME-XRF26).

All pulps and selected coarse rejects that may merit additional analyses or are near or within the zone of mineralization were retrieved from the lab and stored in Constantine's storage locker in Vancouver.

All historic and recent whole and split diamond drill core at the Porcupine Creek exploration camp, Big Nugget mine site, Alaska (Figure 11-1).

Figure 11-1: Aerial view of Porcupine Creek Exploration Camp

Source: Constantine (2018)

12 Data Verification

Previous Authors Gray and Cunningham-Dunlop (2019), Gray and Cunningham-Dunlop (2018), Gray and Cunningham-Dunlop (2015), Greig and Giroux (2010) and Greig (2006) all performed verification of the diamond drilling data that supports the Palmer Exploration Project mineral resource estimate.

12.1 Drill Hole Database

The Project data is stored in a custom Microsoft Access® (Access) database (termed the "database"). This database is secure, operated by a single database administrator, and contains data checking routines designed to prevent common data entry errors.

An export of the database was provided to the Authors for auditing purposes and mineral resource estimation. This audit consisted of checking the digital data against source documents to ensure proper data entry, as well as, data integrity checks (checking for overlapping intervals, data beyond total depth of hole, unit conversion, etc.). Minor errors identified during this review were corrected within the master database and passed back to the Company. All original assay certificates have been found and catalogued for drill holes included within the mineral resource estimate.

12.2 Drill Hole Collar and Downhole Surveying

All drill hole collar and down-hole surveys were loaded into the database by the database manager.

In 2016, the Company undertook a drill hole collar survey program to convert the existing Project survey datum from UTM NAD27 to UTM NAD83 (Gray and Cunningham-Dunlop, 2018). Out of 134 known drill hole collars, 101 were re-surveyed. The priority was to acquire drill hole collar coordinates for holes that contributed to the mineral resource estimate and these holes were successfully surveyed. Fifteen holes (13 from MHC plus CMR06-01 and CMR06-02) had coordinates that did not change and were identical to the pre-2016 coordinates. Eighteen holes (18) had co-ordinates adjusted on a case-by-case basis. To assess the quality and accuracy of the 2016 collar re-survey program, a selection of the 2014 monuments (14-01, 14-07 and 14-09) were also surveyed each day. Based on the known locations of the 2014 monuments, the 2016 survey returned an average x,y positional accuracy to within 1-2 cm and an average elevation accuracy to within 2cm. All the results from the 2016 collar re-survey program have been incorporated in the database.

The Authors are confident that the Company has made best efforts to confirm all existing drill hole collar locations and that the resulting data was acquired using adequate quality control procedures that generally meet industry best practices for a drilling-stage exploration project, and the data are adequate for purposes of mineral resource estimation.

12.3 Drill Hole Logs

All drill logs including collar, lithology, alteration, and geotechnical data were loaded into the database by the database manager from the Microsoft Access tables generated by Constantine geological staff.

Gray and Cunningham-Dunlop (2019) reported that 5% of the Project drill holes were selected and content of the original drill logs were compared against the records in the database. No significant issues were noted and the lithology codes in the drill logs matched the records in the database. These drill logs were used to inform the geological model used in mineral resource estimation.

Gray and Cunningham-Dunlop (2019) also reported that drill core from selected drill holes was reviewed from each year's drill campaign and compared against logged lithologies in the database and that the data concurs with the descriptions.

12.4 Drill Hole Assays

Drill Assays were loaded into the database in their original units from files received directly from the assay laboratories.

Gray and Cunningham-Dunlop (2019) reported that 5% of the Project drill holes were randomly selected and values in the original assay certificates were compared against the records in the database and no significant data entry errors were found.

12.5 Specific Gravity

12.5.1 Field Specific Gravity Measurements

Bulk specific gravity (SG) is measured by trained and qualified Company personnel performing the industry standard "weight-in-water/weight-in-air". Representative sections of halved core, generally consisting of one to five 10-30 cm long pieces, are measured and averaged for all assay sample intervals within mineralized intersections with the potential to be included in resource wireframes. SG measurements are also carried out on wall rock adjacent to mineralized intervals. Samples containing significant void space, such as those from the RW Oxide Zone, are first coated in paraffin wax to ensure more accurate and representative SG measurements.

A project total of 3,911 specific gravity measurements have been carried out on core samples with an overall average SG of 3.21 (average is inclusive of unmineralized wall rock). In the opinion of Gray and Cunningham-Dunlop (2019), the number of SG measurements is adequate to support mineral resource estimation.

12.5.2 Laboratory Specific Gravity Measurements

Field SG measurements were verified by ALS Minerals Canada Ltd. who used a pycnometer to measure specific gravity for randomly selected sample pulps from the mineralized zones (ALS Method OA-GRA08).

A project total of 232 laboratory specific gravity measurements have been carried out. Gray and Cunningham-Dunlop (2019) reviewed the laboratory SG results which generally showed good correlation and validated the field measurements. It should be noted that whole core samples were used to obtain field data and sample pulps were used to obtain laboratory data. Void space error was minimized in the field by using the wax immersion method, and therefore field data are considered more accurate and representative than the laboratory method.

12.6 Assaying and Quality Control-Quality Assurance (QA-QC)

Quality control data for the Project has included both internal and external quality control measures. ALS Minerals Canada Ltd. implemented internal laboratory measures consisting of quality control samples (blanks and certified reference materials and duplicate pulp) within each batch of samples submitted for assaying.

All results have been recorded in an Excel document containing all QA/QC data and charts by the Company's database manager and vetted by Senior Company technical staff. In the opinion of Gray and Cunningham-Dunlop (2019), the current analytical quality control program developed by Constantine for the Project is mature and is overseen by appropriately qualified geologists. The exploration data was acquired using adequate quality control procedures that generally meet industry best practices for a drilling-stage exploration project, and the data are adequate for the purposes of mineral resource estimation.

12.7 Metallurgy

Metallurgical test data was verified through a review of testwork reports and an analysis of the new results from the 2018 metallurgical testwork program completed by SGS, specifically:

- SGS Canada Inc, "The Recovery of Copper, Zinc, Silver and Gold from the Palmer Samples", Project No. 14063-001, (Issued: 28 October 2013)
- SGS Canada Inc, "Barite Metallurgical Testwork on the Palmer VMS Project", Project No. 14063-002, (Issued: 30 July 2018)
- SGS Canada Inc, "Comminution and Mineralogy on the Palmer VMS Deposit", Project No. 14063-03 Revision 1, (Issued: 14 December 2018)

Metallurgical test data was verified through a review of testwork reports and an analysis of the new results from the 2018 metallurgical testwork program completed by SGS. The sample selection for the 2013 test program was representative of the Palmer deposit. The 2018 samples were limited to two drill holes from one area of the mineralized zone but produced similar results to previous test work. The samples tested had head grades much higher than those anticipated from the mine plan and recoveries may be lower than the results indicate. 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.

12.8 Mining and Underground Geotechnical

Mining and geotechnical data was verified during the site visit, through review of previous studies and historical data and is adequate for the PEA as required by NI 43-101 guidelines.

12.9 Surface Geotechnical

With the exception of limited geophysical survey information near the mill site, no subsurface information currently exists to allow for geotechnical characterization of mill site foundations or the TMF/WRSF. Therefore, geotechnical engineering design parameters were assumed based on experience from other projects and the current understanding of the site geologic setting. Surface conditions and the geologic setting were confirmed by the QP during his site visit and engineering parameters selected are adequate for PEA design. Geotechnical investigation and testwork will be required to support future studies.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Metallurgical testing was performed on Palmer samples by SGS Canada Inc. (SGS), Burnaby, BC, in 2013 and 2018. The most recent test program was completed by SGS in 2018 and was used as the basis for the process design and recovery method outlined in Section 17. A full breakdown of the results for each test program can be found in the following reports:

- SGS Canada Inc, "The Recovery of Copper, Zinc, Silver and Gold from the Palmer Samples", Project No. 14063-001, (Issued: 28 October 2013)
- SGS Canada Inc, "Barite Metallurgical Testwork on the Palmer VMS Project", Project No. 14063-002, (Issued: 30 July 2018)
- SGS Canada Inc, "Comminution and Mineralogy on the Palmer VMS Deposit", Project No. 14063-03 Revision 1, (Issued: 14 December 2018)

Based on the results from SGS a Cu, Zn, Py and Ba sequential flotation can produce saleable concentrates at a primary grind size of 80% passing (P_{80}) 72 µm, and rougher concentrate regrind sizes of 35 µm for Cu and 50 µm for Zn. Locked cycle flotation test results carried out on the High Ba Composite achieved recoveries of 88.9% Cu, 93.1% Zn and 91.1% Ba at concentrate grades of 24.5% Cu, 61.3% Zn and 52.3% Ba.

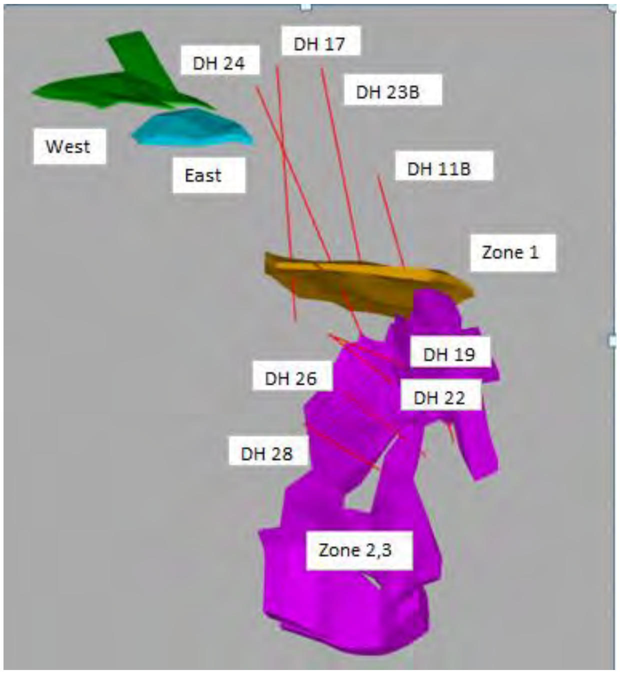
13.2 Summary of SGS Test Program (2013)

The following section summarizes the main results from the metallurgical test programs conducted by SGS Canada Inc. in 2013. The results were previously reviewed in Section 13 of the Palmer 43-101 Technical Report and Updated Resource Estimate for the Palmer Exploration Project May 11th, 2015.

13.2.1 SGG Test Program (2013)

Approximately 500 kg of the Palmer deposit material from exploration rejects was used to create a master composite for flotation testwork. Drill holes CMR08-11B, 17, 19, 22 and CMR09-23B, 24, 26, 28 were blended to make up the master composite used in the test program. The location of the drill holes are shown in Figure 13-1. The composite assayed 1.56% Cu, 6.47% Zn, 0.19 g/t Au, 28.5 g/t Ag, 0.20% Pb, 13.1% Fe and 19.2% S. The test program included mineralogical analysis and sequential flotation of the Cu and Zn which included rougher flotation, cleaner flotation and locked cycle tests (LCT). The mineralogy indicated the material is relatively coarse grained with most of the Cu as Chalcopyrite and the Zn as Sphalerite. Secondary Cu minerals were noted in the mineralogy and depressant reagents were required to reduce Zn activation. Of the material tested at P_{80} target grind size of 72 µm results indicated that sequential Cu, Zn flotation could achieve recoveries in the range of 87% to 93% Cu and 90% Zn in concentrate grades of 27% to 30% Cu and 55% Zn. Further optimization of the flowsheet was recommended moving forward in addition to variability testwork and mapping of secondary minerals throughout the deposit that could influence recovery. The averages of the two LCTs completed are shown in Table 13-1.

Figure 13-1: Master Composite Drill Hole Locations – Palmer Deposit



Source: Constantine (2013)

Table 13-1: LCT 1 & 2 Results (Average)

Concentrate	Grade (%, g/t)							Recovery (%)					
	Cu	Zn	Fe	Au	Ag	S	Pb	Cu	Zn	Fe	Au	Ag	S
Cu Cleaner	25.5	8.54	26.2	3.11	393	34.2	4.22	89.6	7.44	10.7	61.5	73.7	9.39
Zn Cleaner	0.91	59.1	5.7	0.41	51.4	34.8	-	5.27	84.9	3.87	13.5	16.0	15.8

Source: SGS (2013)

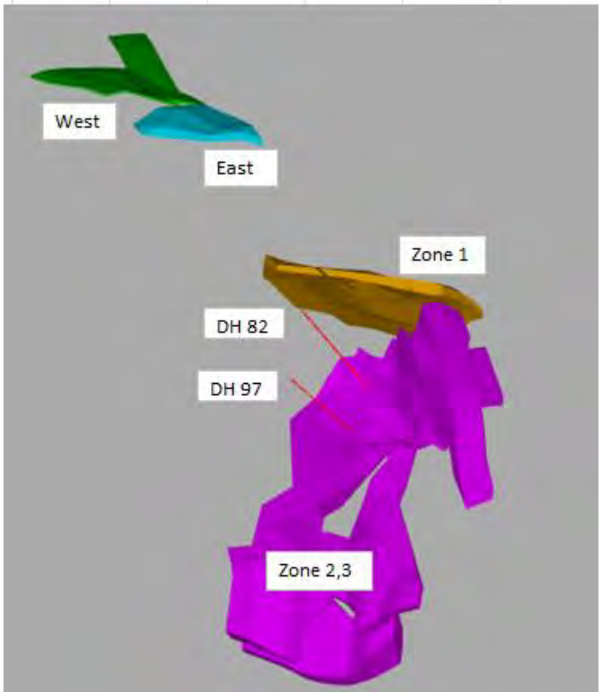
13.3 Summary of SGS (2018), 14063-002

In March 2018, a metallurgical test program was commenced at SGS in Burnaby, BC (Project No. 14063-002). Drill core rejects from two drill holes were used to blend a High Ba composite that was submitted for metallurgical testing in support of this preliminary economic assessment. The test program focused on mineralogy and sequential flotation of Cu, Zn, Py and Ba.

13.3.1 Sample Selection

A blend of exploration reject material from two drill holes CMR17-82 and CMR17-97 were used to create the High Ba composite. The location the two drill holes are shown in Figure 13-2.

Figure 13-2: High Ba Drill Hole Locations - Palmer Deposit



Source: Constantine (2018)

Measured head assays for the High Ba composite are shown in Table 13-2.

Table 13-2: Head Assays for SGS (2018) High Ba Composite

Composite	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Ba (%)	Au (g/t)	Ag (g/t)
High Grade Ba Composite	1.61	0.37	10.3	13.6	26.2	23.2	0.29	49.0

Source: SGS (2018)

13.3.2 Mineralogy

A sample of the High Ba composite was ground to a P_{80} of approximately 70 µm and analyzed using QEMSCAN and PMA routine to determine mineral content and mineral fragmentation. A summary of mineral content is presented in Table 13-3. The sample was screened into +38 µm and -38 µm size fractions and analyzed separately.

Table 13-3: Mineral Content for the High Ba Composite

Fraction		Combined	+38µm		-38µm	
Mass Size Distribution (%)		100.0	53.5		46.5	
Calculated ESD Particle Size		17.2	43.1		11.0	
		Sample	Sample	Fraction	Sample	Fraction
Mineral Mass (%)	Pyrite	24.8	16.0	29.9	8.80	18.9
	Pyrrhotite	0.01	0.01	0.01	0.01	0.01
	Chalcopyrite	5.07	2.29	4.29	2.78	5.98
	Sphalerite	13.4	8.34	15.6	5.03	10.8
	Galena	0.66	0.16	0.30	0.49	1.06
	Arsenopyrite	0.07	0.04	0.08	0.03	0.06
	Other Sulfides	0.12	0.05	0.10	0.06	0.14
	Quartz	8.79	4.82	9.00	3.97	8.54
	Feldspar	1.04	0.58	1.09	0.45	0.97
	Celsian	0.62	0.34	0.64	0.28	0.60
	Mica	1.36	0.66	1.23	0.71	1.52
	Chlorite	1.44	0.47	0.88	0.97	2.09
	Clays	0.06	0.02	0.04	0.04	0.09
	Other Silicates	0.24	0.13	0.25	0.11	0.23
	Oxides	0.61	0.18	0.34	0.43	0.93
	Carbonates	0.48	0.26	0.48	0.22	0.48
	Smithsonite	0.19	0.09	0.16	0.10	0.22
	Barite	40.9	19.0	35.6	21.8	46.9
	Other	0.18	0.02	0.03	0.17	0.36
	Total	100.0	53.5	100.0	46.5	100.0

Source: SGS (2018)

Copper is associate with chalcopyrite (Cp) (96.5%) and to a minor extent tennantite-tetrahedrite (Tn) (2.8%). The Cp and Tn were equally distributed between the two size fractions with respect to Cu deportment. Zinc was predominately associated with sphalerite (Sp) (98.6%) and to a lesser extent smithsonite (1.1%). Approximately 89.6% of the Sp was liberated indicating high recovery potential. Sphalerite was evenly distributed between +38 µm and -38 µm size fractions. Previous mineralogy indicated a coarse grind could produce high recoveries. Based on the 2013 test program a target grind size of P_{80} 72 µm was chosen with minimal regrind. The Ba mineralogy indicates the majority of the Ba is associated with Barite and is 95.4% liberated. The Ba final concentrate was analyzed and the majority of the sample (95.6%) was barite.

13.3.3 Comminution

A bond ball mill work index (BWi) test was completed on the High Ba composite at a sieve size of 106 µm. The results are summarized in Table 13-4. The results indicate the sample can be classified as very soft.

Table 13-4: Bond Ball Mill Work Index Results for Base Met (2018) Global Composites

Composite	Sieve Size (µm)	Feed Size, F_{80} (µm)	Product Size, P_{80} (µm)	Grams per Revolution (g)	Bond Ball Mill Work Index (kWh/t)
High Ba Composite	150	2,363	85	3.88	**6.3**

Source: SGS (2018)

13.3.4 Flotation

Rougher and cleaner flotation tests were conducted on the High Ba composite to refine the test conditions from the 2013 program for Cu and Zn. From the Zn tailings a pyrite rougher floatation and Ba flotation tests were completed to investigate the parameters and flowsheet that would produce a saleable Ba concentrate. Locked cycle testing was completed to produce results closer to what would be expected in an operating plant for Cu, Zn and Ba concentrates.

13.3.4.1 Cu-Zn Rougher Flotation

Rougher flotation tests were completed as a confirmation of the previous test program (SGS 14063-001). The main conditions included a primary grind size targeting a P_{80} of 72 µm followed by sequential rougher flotation of the Cu and then Zn. Other important parameters tested included PAX as a collector, MIBC as a frother, the effect of sodium cyanide (NaCN) and $ZnSO_4$ dosage on Zn depression, $CuSO_4$ as a depressant in the Zn circuit and lime as a pH modifier. The results from the rougher test work indicated a lower depressant dosage improved Cu recovery.

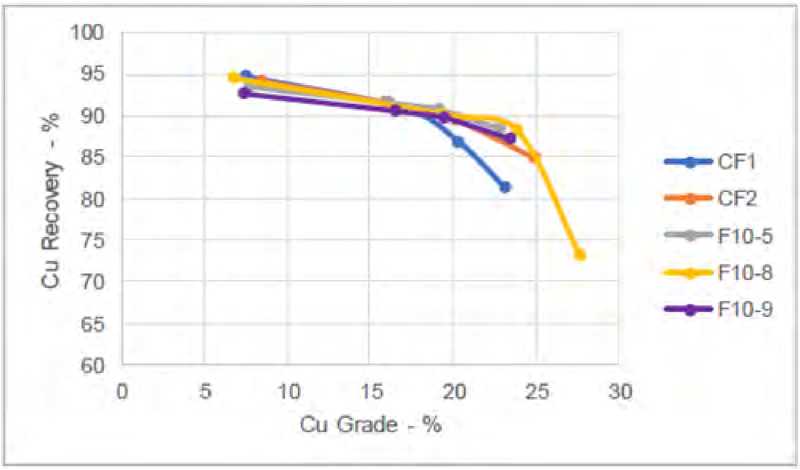
After completing Cu rougher flotation, the pH of the flotation pulp was adjusted up to 11.5 with lime, and copper sulfate was added to activate sphalerite. Initial Zn rougher tests indicated relatively good Zn rougher performance with conventional reagent dosages. Overall recoveries in the mid 90’s were achieved to the Zn rougher concentrate.

13.3.4.2 Cleaner Flotation

Two batch cleaner flotation tests were conducted to determine if higher concentrate grades could be achieved while maintaining Cu and Zn recovery. Sequential Cu and Zn flotation at primary P_{80} grind size target of 72 µm was completed with 3 stages of cleaning. The rougher concentrates regrind sizes of 44 µm and 30 µm for the Cu circuit and 62 µm and 40 µm for the Zn circuit were tested. In the first cleaner test (CF1) no NaCN was added to the grinding stages for Cu and a coarser regrind size was targeted for both Cu and Zn. The final Cu recovery was 81.3% at a grade of 23.1% Cu. There were minimal losses of Zn to the Cu concentrate. The Zn recovery was low at 57.2% at a grade of grade of 64.7% Zn. The second cleaner test (CF2) carried out with NaCN added to the primary grinding and copper regrind stages as well as at a higher PAX dosage in the copper rougher flotation stage. The main changes to CF1 for the Zn circuit included a higher $CuSO_4$ dosage in the rougher flotation circuit and lime was added to the regrind mill. The final Cu concentrate recovered 84.4% Cu at a grade of 24.9% Cu. The Zn concentrate recovered 40.3% Zn at a grade of 62.8% Zn. The stage recovery for Zn in CF2 was slightly higher at 74.3% compared to 70.3%.

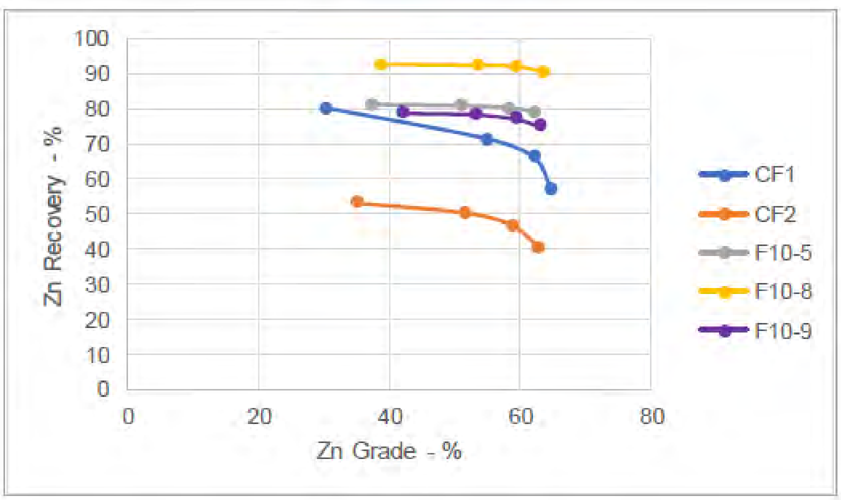
Three additional cleaner tests were completed (F10-5, F10-8 and F10-9) to produce a bulk sample for the Ba flotation test work. The tests were also used for additional optimization to improve the Cu and Zn concentrate recoveries. The best results were produced using conditions from test F10-8 with a final Cu concentrate recovering 73.2% Cu at a grade of 27.7% Cu and Zn concentrate recovering 90.5% Zn at a grade of 63.4% Zn. The test parameters from F10-8 were used for the Cu and Zn LCT tests. The grade recovery curves for Cu and Zn cleaner tests are illustrated in Figure 13-3 and Figure 13-4.

Figure 13-3: Copper Concentrate Grade vs. Recovery Curves



Source: SGS (2018)

Figure 13-4: Zinc Concentrate Grade vs. Recovery Curves

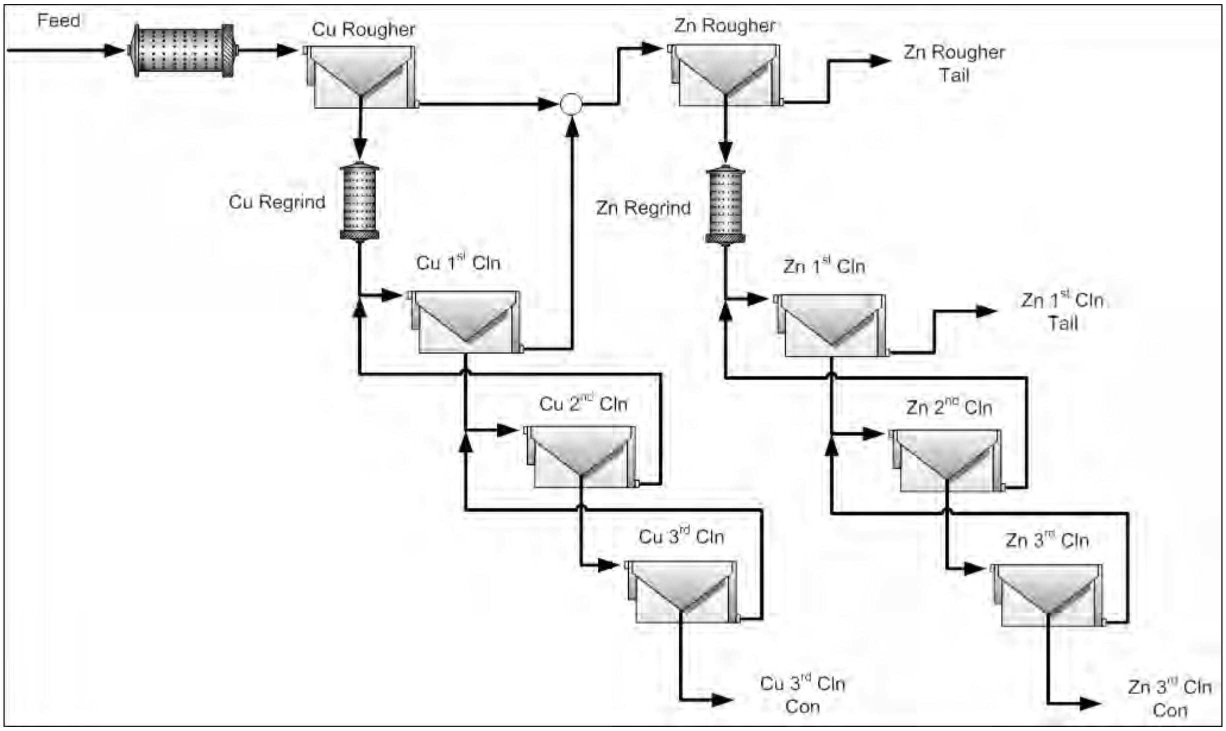


Source: SGS (2018)

13.3.4.3 Locked Cycle Flotation

Using the optimized conditions developed from test F10-8, a single locked cycle test was completed on the High Ba Composite to obtain metallurgical performance data and predict grade and recovery values for economic analysis. An illustration of the flowsheet and a list of the test conditions are shown in Figure 13-5 and in Table 13-5.

Figure 13-5: SGS (2018) Locked Cycle Test Flowsheet



Source: SGS (2018)

Table 13-5: LCT Parameters

LCT 1 - F10-8	Reagents added, g/t						Time, minutes			Pulp
	Lime	NaCN	ZnSO4	CuSO4	PAX	MIBC				
	g/t	g/t	g/t	g/t	g/t	g/t	Grind	Cond.	Froth	pH
Grind										
Grind Size, 68 µm	400		150				19.5			9.0
Cu Circuit										
Cu Rougher										
Cu Rougher 1	60				10	15		2	2	10.0
Cu Rougher 2	35				10	5		2	4	10.0
Cu Rougher 3	45				10	5		2	3	10.0
Cu Ro Con Regrind	100	20	30				20			9.8
Grind Size, 49 µm										
Cu Cleaner										
Cu 1st Cleaner 1	40				6	6		2	4	10.5
Cu 1st Cleaner 2	45				4	4		2	2	10.5
Cu 1st Cleaner 3	20				4	4		2	2	10.5
Cu 2nd Cleaner	20				2	3		2	4	10.5
Cu 3rd Cleaner	12					2		1	5	10.5
Zn Circuit										
Zn Rougher										
Zn Conditioning	1280			400				3		11.5
Zn Rougher 1					35	30		2	2	11.6
Zn Rougher 2	100				25	10		2	5	11.5
Zn Ro Con Regrind	100			30			10			11.0
Grind Size, 47 µm										
Zn Cleaner										
Zn 1st Cleaner	430				15	15		2	7	11.6
Zn 2nd Cleaner	235				5	5		2	4	11.6
Zn 3rd Cleaner	85					10		1	4	11.5
Total:	**3007**	**20**	**180**	**430**	**126**	**114**				

Source: SGS (2018)

The final results produced Cu concentrate recovering 88.9% of the Cu at a concentrate grade of 24.5% Cu; while the Zn concentrate recovered 93.1% of the Zn at a concentrate grade of 61.3% Zn. These values were used to estimate grades and recoveries for the PEA economic analysis. The final results based on five cycles are shown in Table 13-6.

Table 13-6: Cu and Zn LCT Results

Product	Weight		Assays, % g/t							% Distribution						
	Dry	%	Cu	Zn	Pb	Fe	Au	Ag	S	Cu	Zn	Pb	Fe	Au	Ag	S
Cu 3rd Cleaner Con	**120.8**	**6.0**	**24.5**	**8.21**	**4.39**	**26.6**	**3.17**	**521.4**	**35.2**	**88.9**	**4.8**	**81.1**	**12.1**	**49.5**	**70.8**	**8.3**
Zn 3rd Cleaner Con	**315.0**	**15.7**	**0.67**	**61.3**	**0.29**	**3.49**	**0.49**	**56.7**	**33.7**	**6.3**	**93.1**	**14.2**	**4.1**	**20.1**	**20.1**	**20.8**
Zn 1st Cleaner Tail	164.4	8.2	0.40	0.92	0.11	24.0	0.46	18.1	32.1	2.0	0.7	2.8	14.9	9.8	3.3	10.4
Zn Rougher Tail	1406	70.1	0.07	0.20	0.01	13.0	0.11	3.67	22	2.8	1.4	1.9	68.8	20.6	5.8	60.5
Head (calculated)	2006	100	1.66	10.3	0.33	13.2	0.39	44.4	25.4	100	100	100	100	100	100	100

*Average Cycles D, E & F
Source: SGS (2018)

13.3.4.4 Py-Ba Rougher Flotation

Five rougher flotation tests were completed using the Zn bulk tailings sample from Tests F10-5, 8 & 10. The initial stage includes flotation of the pyrite to produce a pyrite concentrate that could potentially be acid generating material (PAG) for deposition underground as paste followed by Ba flotation to produce a saleable cleaner concentrate. The chemical analysis of the Zn tailings sample feeding the Py rougher flotation circuit is shown in Table 13-7.

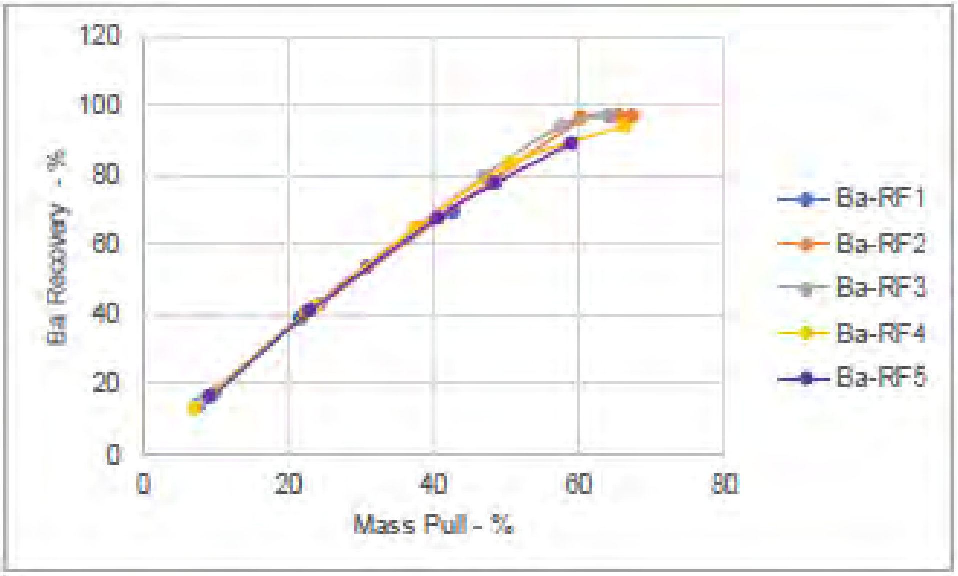
Table 13-7: Py-Ba Flotation Sample Feed Grade

Sample	Ba, %	Au, g/t	Ag, g/t	Cu, %	Pb, %	Zn, %	Fe, %	S, %
Zn Tailings	28.0	0.07	3.5	0.07	<0.01	0.26	11.8	20.7

Source: SGS (2018)

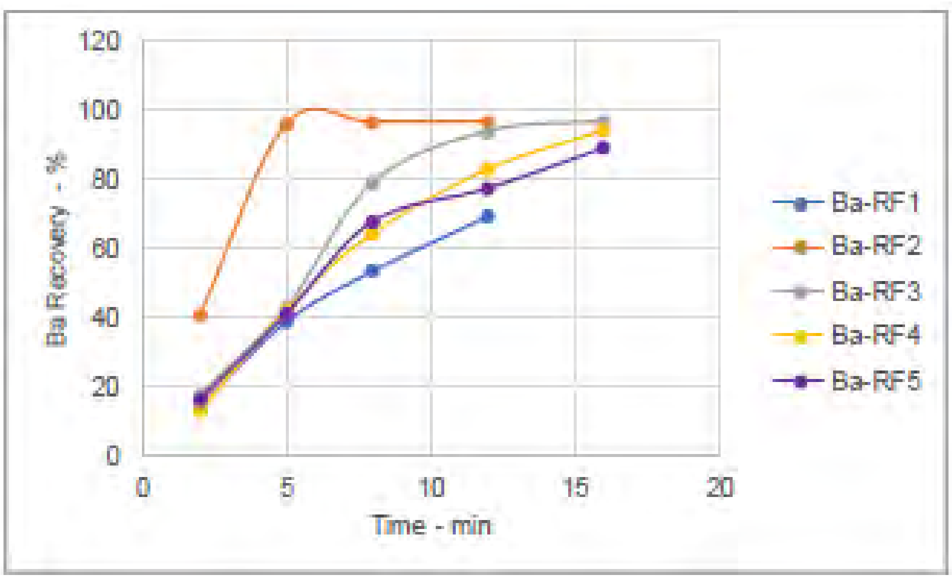
Of the five rougher tests, Ba-RF1 through Ba-RF5, only Ba-RF2 included a pyrite flotation prior to the Ba float. PAX and MIBC were used to produce the Py concentrate. The results indicate that 64.5% S and 3.0% Ba reported to the Py concentrate. The Ba rougher flotation tests were carried out using Soda Ash, Aero 845, Fuel Oil, MIBC and Guar Gum at varying dosages and float times (12 to 16 minutes) maintaining a pH of 10. The tests produced similar mass pulls and grades for Ba to the concentrate. The Ba-RF2 and Ba-RF3 were the only tests where the recovery curves leveling off after the final stage of flotation. The mass versus recovery and grade versus recovery curves are shown in Figure 13-6 and Figure 13-7. Parameters from rougher tests Ba-RF2 and Ba-RF3 parameters were used for the rougher flotation in the next stage of test work.

Figure 13-6: Py-Ba Rougher Flotation – Mass Pull versus Ba Recovery



Source: SGS (2018)

Figure 13-7: Py-Ba Rougher Flotation – Ba Grade versus Recovery

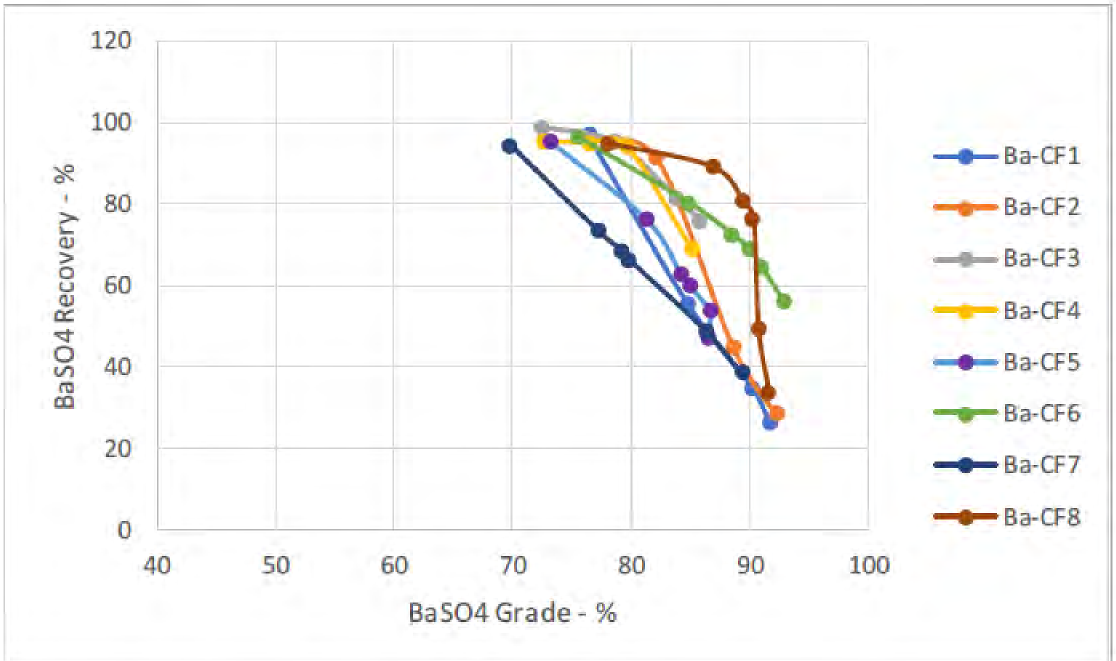


Source: SGS (2018)

13.3.4.5 Ba Cleaner Flotation

A total of eight cleaner tests were completed on the Zn tailings sample (Ba-CF1 through Ba-CF8). The test work included three to five stages of cleaning at varying slurry densities utilizing Aero 845, Fuel Oil, MIBC and Sodium Silicate to facilitate the upgrade of the rougher concentrate. The tests were run at a pH in the range of 9.2 to 9.7 with the exception of test Ba-CF7 where lime was added to maintain a pH of 10.5. The tests were conducted with pyrite pre-float with the exception of tests Ba-CF1 and Ba-CF3. The best grade and recovery results were produced from test Ba-CF8. These results were used as the criteria for the LCT. The grade versus recovery curves are shown in Figure 13-8.

Figure 13-8: Ba Grade versus Recovery



Source: SGS (2018)

Test Ba-CF8 included a pyrite rougher flotation stage with the rougher tailings feeding the Ba rougher circuit. The rougher flotation was followed by five stages of cleaning. The addition of sodium silicate as a depressant in the first cleaner and performing the tests at a low slurry density of 20% solids provided the best conditions. The test parameters are shown in Table 13-8 and the results in Table 13-9.

Table 13-8: Ba-CF8 Flotation Parameters

Flotation Stage	Float Time (min)	Reagent Dosage (g/t)						pH
		PAX	MIBC	Soda Ash	Aero 845	Fuel Oil	Sodium Silicate	
Pyrite Rougher	9	200	15	-	-	-	-	9.1
Barite Rougher	5	-	5	340	200	15	-	10
Barite Cleaner - 5 stages at 20% solids	7+5+5+5+3	-	20	-	85	12.5	50	9.6

Source: SGS (2018)

Table 13-9: Ba-CF8 Cleaner Test Results

Test	Product	Mass Pull	Grade			Recovery		
			%	%	%	%		
		%	Ba	S	$BaSO_4$	Ba	S	$BaSO_4$
Ba-CF8	Ba 5th Cleaner Con	18.2	53.9	13.9	91.6	33.6	12.0	33.6
	Ba 4th Cleaner Con	26.9	53.4	13.9	90.7	49.1	17.7	49.1
	Ba 3rd Cleaner Con	41.8	53.1	13.8	90.3	76.0	27.3	76.0
	Ba 2nd Cleaner Con	44.8	52.6	13.7	89.4	80.5	29	80.5
	Ba 1st Cleaner Con	50.9	51.1	13.3	86.9	89.0	32.1	89.0
	Ba Rougher Con	60.4	45.9	11.8	78.1	94.9	33.7	94.9
	Py Rougher Con	28.7	4.6	48.8	7.82	4.5	66.2	4.5
	Ba Rougher Tail	10.9	1.7	0.19	2.89	0.6	0.1	0.6
	Head (calc.)	100	29.2	21.1	49.7	100	100	100

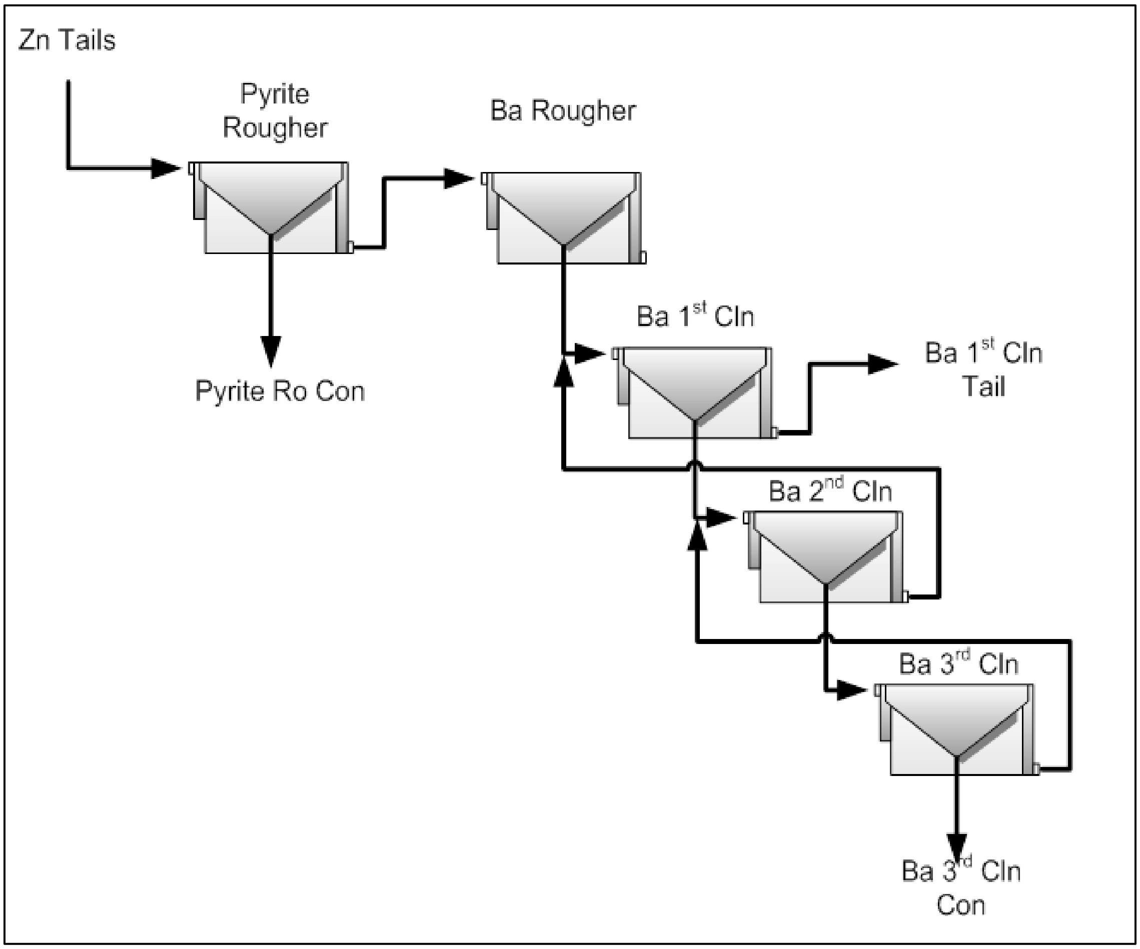
Source: SGS (2018)

The results of the test work indicate at three stages of cleaning a Ba recovery of 75% at a grade of 53.1% Ba can be achieved. The fourth and fifth stages of cleaning did not provide additional benefit.

13.3.4.6 Ba Locked Cycle Flotation

Using the conditions developed from test Ba-CF8 with three stages of cleaning, a single locked cycle test was completed on the High Ba Composite Zn tailings. An illustration of the flowsheet and test conditions is shown in Figure 13-9 and in Table 13-10.

Figure 13-9: SGS (2018) Locked Cycle Test Flowsheet



Source: SGS (2018)

Table 13-10: LCT Parameters

LCT 1 - Ba-CF8	Reagents added, g/t						Time, min		Pulp
	Soda Ash	Aero 845	Fuel Oil	MIBC	Na_2SiO_3	PAX	Cond.	Froth	pH
	g/t	g/t	g/t	g/t	g/t	g/t	Cond.	Froth	pH
Pyrite Pre-float									
Py Rougher 1				5		100	1	2	8.3
Py Rougher 2				5		50	1	3	8.5
Py Rougher 3				5		50	1	4	8.5
Ba Rougher									
Conditioning	410						2		10.0
Conditioning							1		10.0
Ba Rougher 1		100	10	5			1	2	10.0
Ba Rougher 2		100	5				1	3	9.8
Ba Cleaner									
Conditioning					50		2		9.8
Ba 1st Cleaner 1		10	5	1			1	2+2	9.8
Ba 1st Cleaner 2		25	0	1			1	2+1	9.3
Ba 2nd Cleaner 1		10	5	1	10		1	2+1	9.5
Ba 2nd Cleaner 2		25	0	1			1	2	9.0
Ba 3rd Cleaner 1		0	0	2	10		1	2	9.2
Ba 3rd Cleaner 2		15	2.5	2			1	2	8.9
Total	**410**	**285**	**27.5**	**28**	**70**	**200**			

Source: SGS (2018)

The final results produced Ba concentrate recovering 91.1% at a concentrate grade of 52.3% based the average of cycles E and F. These values were used to estimate grades and recoveries in the economic model. The final results based on 6 cycles are shown in Table 13-11.

Table 13-11: Cu and Zn LCT Results

Product	Weight Dry	%	Grade, %			Recovery, %		
			Ba	S-total	$BaSO_4$	Ba	S-total	$BaSO_4$
Py Rougher Con	614.6	30.7	6.70	47.1	11.4	7.0	66.9	7.0
Ba 3rd Cleaner Con	**1021.3**	**50.9**	**52.3**	**13.9**	**88.8**	**91.1**	**32.7**	**91.1**
Ba 1st Cleaner Tail	281.6	14.0	2.71	0.41	4.61	1.3	0.3	1.3
Ba Rougher Tail	87.7	4.4	3.75	0.43	6.38	0.6	0.1	0.6
Head (calculated)	2005	100	29.2	21.6	49.6	100	100	100

Average Cycles E & F
Source: SGS (2018)

13.3.5 Concentrate Quality

The Cu and Zn concentrates from locked cycle testing were not analyzed for minor elements in the 2018. The SGS 2013 program did an ICP scan of the LCT2 concentrates and the results indicate relatively high As (2,210 ppm) in the Cu concentrate and Cd (2,720 ppm) in the Zn concentrate. A multi-element analysis was completed on the SGS (2018) Py rougher concentrate and Ba-CF2. Elements were analyzed using lithium borate fusion and ICP or ICP mass spectroscopy. The results for the final Ba concentrate are summarized in Table 13-12.

Table 13-12: Concentrate Quality for SGS (2018) Ba LCT1-Ba-CF8 3rd Cleaner Concentrate

Element	Unit	Ba Concentrate
Barium (Ba)	%	56.1*
$BaSO_4$	%	95.3*
Sulfur (S)	%	13.9*
Aluminum (Al)	g/t	2,900
Arsenic (As)	g/t	<200
Beryllium (Be)	g/t	<0.8
Bismuth (Bi)	g/t	80
Calcium (Ca)	g/t	2,900
Cadmium (Cd)	g/t	< 10
Cobalt (Co)	g/t	< 20
Chromium (Cr)	g/t	< 40
Copper (Cu)	g/t	< 40
Iron (Fe)	g/t	<500
Potassium (K)	g/t	< 400
Lithium (Li)	g/t	110
Magnesium (Mg)	g/t	220
Manganese (Mn)	g/t	< 20
Molybdenum (Mo)	g/t	< 300
Niobium (Nb)	g/t	< 50
Lead (Pb)	g/t	< 800
Antimony (Sb)	g/t	< 100
Selenium (Se)	g/t	< 200
Tin (Sn)	g/t	< 100
Strontium (Sr)	g/t	4,060
Titanium (Ti)	g/t	37
Thallium (Tl)	g/t	< 100
Vanadium (V)	g/t	< 80
Yttrium (Y)	g/t	14
Zinc (Zn)	g/t	400

*Lithium Borate Fusion
Source: SGS (2018)

13.4 Summary of SGS (2018), 14063-03

Samples from the AG Zone deposit were sent to SGS by Constantine for comminution and mineralogical analysis. The SMC and bond ball mill work index test work was completed on the 'South West' sample. Mineralogy was also the focus with no flotation test work.

13.4.1 Comminution

Drill core, CMR18-110, from the AG Zone was collected by Constantine and sent to SGS for comminution test work. The work included SMC, abrasion index and bond ball mill work index test work. The results indicate the mineralized rock is soft with an A x b value of 89.6 and mildly abrasive with an abrasion index (Ai) of 0.119 g. The bond ball mill index work index indicates the material is very soft with a BWi of 6.9 kWh/t at a P_{80} of 79 µm. Similar results, BWi 6.3 kWh/t, were produced as part of the SGS (2013) test work. The results are shown Table 13-13.

Table 13-13: Comminution Test Results

Sample ID	A	b	A x b	t_a	SCSE (kWh/t)	DWI (kWh/m^3)	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	Ai (g)	BWi (kWh/t)
South West Sample	82.2	1.09	89.6	0.66	7.42	3.92	9.7	6.5	3.3	0.119	6.9

Source: SGS (2018)

13.4.2 AG Zone Mineralogy

Coarse rejects representing the AG Zone were collected by Constantine and sent to SGS for mineralogical analysis using QEMSCAN. The analysis indicated that the copper mineral was predominately tennantite-tetrahedrite (0.6%), zinc mineral was sphalerite (11%), barite content was 38% and the main sulfide mineral was pyrite (25%). In addition to these minerals lead was present as galena (3%) and the main non-sulfide minerals are mica (14%) followed by quartz (6%).

The mineralogy indicates tennantite-tetrahedrite is 80.9% liberated (76.9% fully exposed) and sphalerite is 92.5% liberated (88.6% exposed). Liberation was higher in the -38 µm size fraction for both tennantite-tetrahedrite and sphalerite. Barite was found to be 97% liberated (94.5% exposed). The results indicate minimal regrind will be required to recover Cu, Zn and Ba. Similar liberation results were seen for the High Ba composite for sphalerite and barite.

A comparison of mineral content in the AG Zone, High Ba composite and Master composite (SGS 2013) is shown in Table 13-14. Copper in the AG Zone was mainly associated with tennantite-tetrahedrite (0.62%) followed by chalcopyrite (0.01%) and the High Ba composite copper was mainly associated with chalcopyrite (5.07%). In the AG Zone there was 3.30% galena (61.4% liberated) and minimal galena in the Palmer deposit. Sphalerite and Barite have similar mineral mass with a few percent lower in the AG Zone compared to the High Ba composite. Pyrite content was approximately 25% for both 2018 samples.

Table 13-14: Mineral Content Comparison

Mineral		AG Zone	High Ba Comp	Master Comp
Mineral Mass (%)	**Pyrite**	**25.0**	**24.8**	**19.1**
	Pyrrhotite	0.00	0.01	2.3
	Tennantite-Tetrahedrite	**0.62**	**-**	**-**
	Chalcopyrite	**0.01**	**5.07**	**5.0**
	Sphalerite	**11.0**	**13.4**	**8.8**
	Galena	**3.30**	**0.66**	**1.0**
	Arsenopyrite	0.08	0.07	0.0
	Other Sulfides	0.01	0.12	0.1
	Quartz	5.95	8.79	18.0
	Feldspar	0.28	1.04	2.7
	Celsian	0.48	0.62	2.4
	Mica	14.2	1.36	3.3
	Chlorite	0.04	1.44	7.1
	Clays	0.09	0.06	0.1
	Other Silicates	0.15	0.24	2.2
	Oxides	0.35	0.61	0.5
	Carbonates	0.07	0.48	3.7
	Smithsonite	0.00	0.19	0.7
	Barite	**38.3**	**40.9**	**22.2**
	Other	0.06	0.18	0.6
	Total	100.0	100.0	100.0

Source: SGS (2018)

In the mineralogy report of the AG Zone prepared by SGS they have used benchmarking against the High Ba composite and Master composite to estimate the Zn and Ba recoveries. They indicate that recoveries could potentially be 90% based on the mineral association and liberation. The estimated SGS recoveries for AG Zone and the test results from the other two composites are shown below in Table 13-15.

Table 13-15: Recovery Estimation for the AG Zone

Sample	Mineral Exposure			Flotation Recovery (%)		
	Chalcopyrite	Sphalerite	Barite	Cu	Zn	Ba
Master Comp	95.5	98.2	-	90.0	90.0	-
High Ba Comp	93.6	97.5	99.1	88.9	88.9	88.8
AG Zone	96.0*	98.2	99.2	90.0*	~90	~90

Mineralogy and Benchmarking on Palmer AG Zone, *Tennantite-tetrahedrite
Source: SGS (2018)

13.5 Relevant Results

Based on the results from the SGS (2018) test program, the process flowsheet will include one stages of crushing underground. Through the test work, the material will then be processed through two stages of grinding to achieve a target P_{80} of 72 µm. The grinding circuit will be designed using SMC results and Bond ball mill work index of 6.9 kWh/t.

Cyclone overflow from the secondary grinding circuit will then be subjected to Cu, Zn, Py and Ba sequential flotation. To improve concentrate grade, Cu and Zn rougher concentrates will be reground to P_{80} grind sizes of 35 µm and 50 µm respectively, before being cleaned in three stages of cleaner flotation. Tailings from the Zn rougher and 1st cleaner circuits will be combined and feed the Py rougher flotation circuit. The rougher concentrate will be filtered and mixed as paste for deposition underground. The Py rougher tailings feeds the Ba rougher flotation circuit. The Ba concentrate will be upgraded in three stages of cleaning and the Ba tailings will be processed as filtered tailings. The test conditions from SGS (2018) LCT1 (F10-8) for Cu and Zn and LCT1 (BA-CF8) for Py and Ba were used to size the flotation circuit and predict reagent consumable rates. Section 17 provides more detail on each process unit operation.

A preliminary estimate of Cu, Zn and Ba recoveries and concentrate grades are summarized in Table 13-16 and provide the basis for the economic analysis presented in Section 22. These projections are a combination of SGS (2018) locked cycle test results and SGS (2018) “Mineralogy and Benchmarking on Palmer AG Zone report”. The test results were conducted on samples with head grades that are higher than the average life-of-mine (LOM) head grades and additional test work at varying head grades is recommended to confirm the concentrate grades and recoveries. Test work has not been completed on AG Zone and the results are only estimates based on SGS 14063-03 (2028) mineral analysis and SGS (2018) benchmarking comparisons to previous test work and similar projects.

Table 13-16: Preliminary Recovery Projections

Description	Concentrate Grade			Recovery		
	Cu (%)	Zn (%)	Ba (%)	Cu (%)	Zn (%)	Ba (%)
Palmer Deposit (LCT SGS 2018)	24.5	61.3	52.3	88.9	93.1	91.1
AG Zone Deposit	24.5	61.3	52.3	78.9*	88.1**	91.1

*10% Reduction in Cu Recovery based on the slower and more complex metallurgy of Cu associated with Tn and lack of flotation results to confirm recovery.

**5% Reduction in Zn Recovery based on potential losses to the Cu concentrate and lack of flotation results to confirm recovery.

Source: JDS (2019)

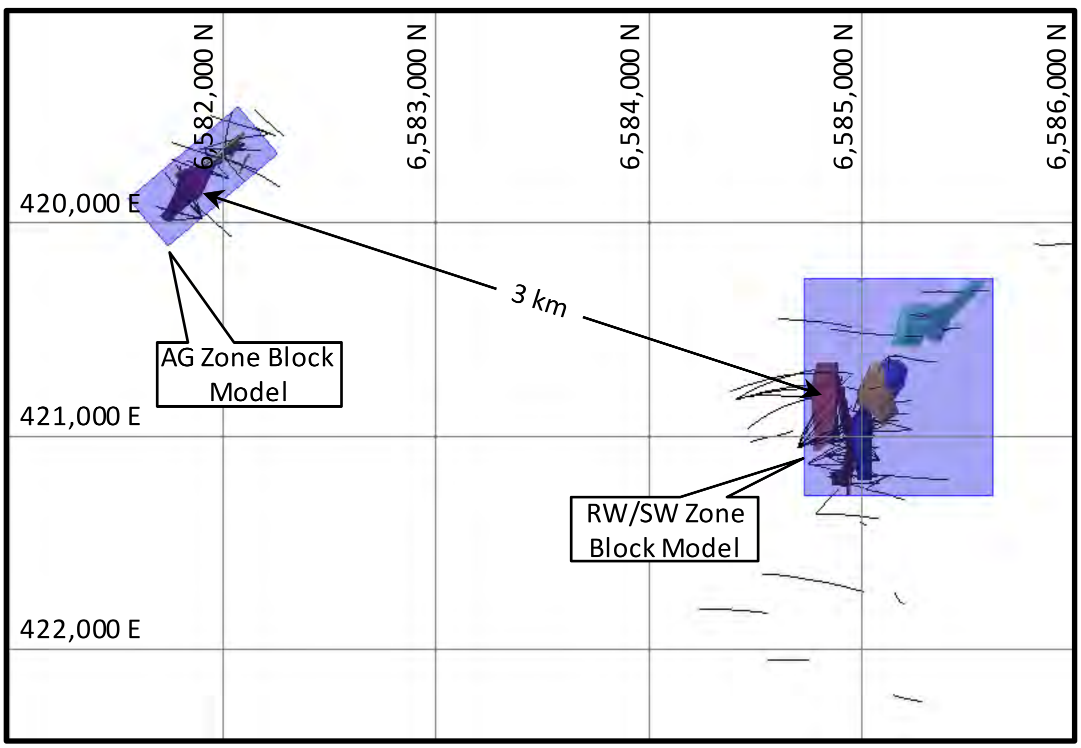
14 Mineral Resources Estimate

14.1 Introduction

This mineral resource estimate is an update of the resource documented in the NI 43-101 Technical Report by Gray and Cunningham-Dunlop dated November 9th, 2018. That report described the Mineral Resource in the RW and SW areas – collectively referred to here as the main area (or the Palmer deposit). This update includes that documentation in Section 14.1, but also describes the estimation of the additional AG Zone (or the AG Zone deposit) Mineral Resource, released December 18th, 2018, in Section 14.2. The Palmer Project Mineral Resource Statement is found in Section 14.3.

Figure 14-1 highlights the location of the RW/SW & AG Zones Mineral Resource areas.

Figure 14-1: Palmer Project Mineral Resource: Block Model Locations



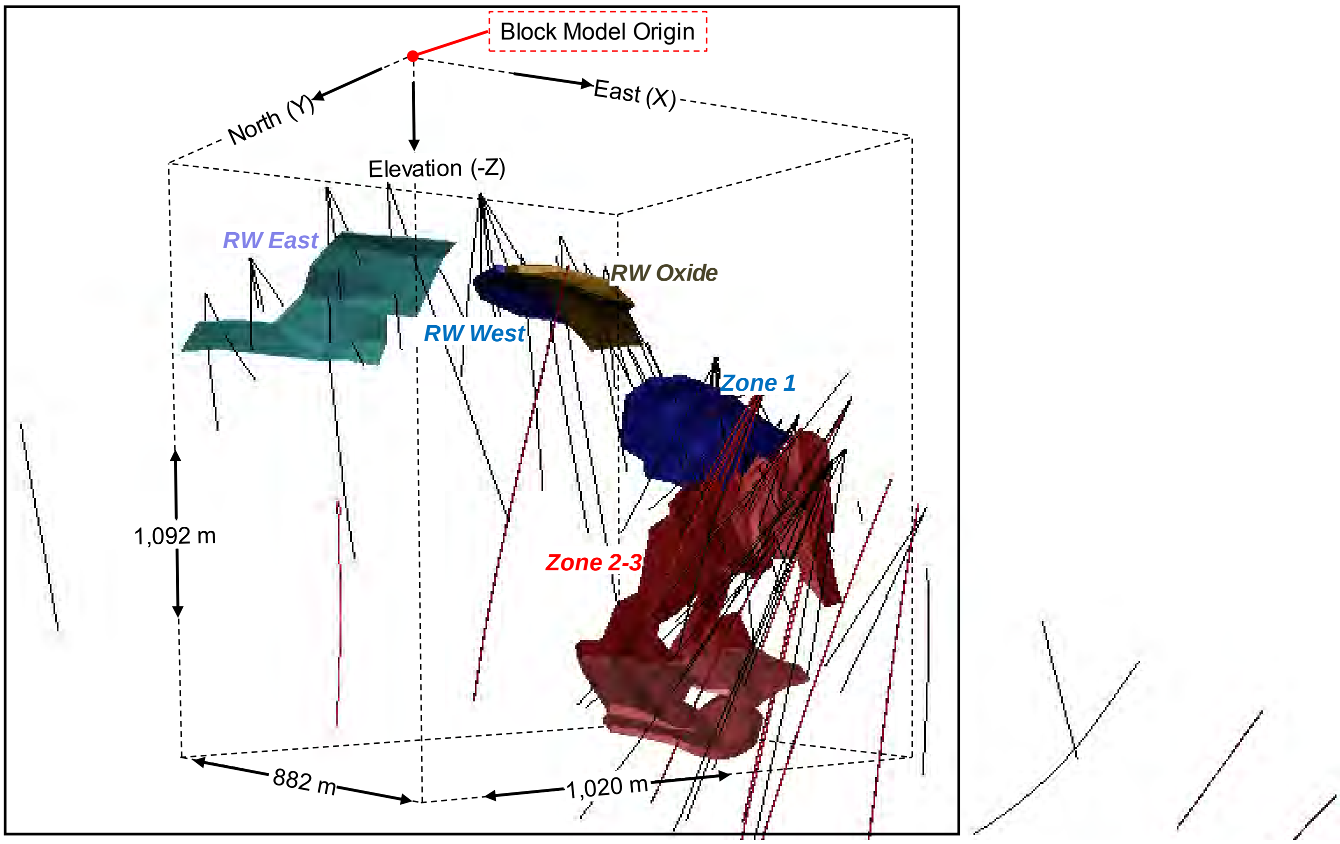
Source: Constantine (2018)

14.2 RW and SW Zones Resource Estimation

14.2.1 Currently Available Drill Data and Model Setup

The Mineral Resource Estimate for the RW and SW Zones is based on assay data available as of May 1st, 2018. Results from 44,900 m of diamond drilling in 108 holes have been used in the interpretation of geology in support of this resource estimate; 60 holes (23,700 m) intersected the mineralized zones and have been used for grade estimation. This includes 26 holes that have been drilled since the previous main area resource, reported in 2015. Figure 14-2 illustrates drill hole locations, the extents of the resource block model and the interpreted zones of mineralization; holes drilled since the 2015 resource are shown in red. Table 14-1 lists the main area block model setup.

Figure 14-2: Palmer Drilling, Mineralized Zones and Block Model Extents (Looking Northeast)



Source: Constantine (2018)

Table 14-1 Main Area Block Model Setup

Block:	X	Y	Z
origin(1)	420,260	6,584,730	1,740
size (m)	6	6	6
n blk	170	147	182

No Rotation

4,548,180 blocks

(1) SW model top, block edge

Source: Constantine (2018)

14.2.2 Geologic Model

Consistent with previous estimates, the geologic model used for control of grade interpolation was derived from wireframes constructed by Constantine personnel. Darwin Green, P. Geo, Vice President of Exploration, oversaw development of the mineralization solids. As a general rule, mineralization included in sulfide zone wireframes was limited to material grading >0.5% copper or >2% zinc, and that could be demonstrated to be correlative with definable stratabound zones. In some cases, lower grade material was included where geology was clearly correlative (i.e. massive barite and/or barren pyrite) and its inclusion supported a more meaningful and geologically consistent volume. Distances applied in the interpretation of geologic units were based on geologic understanding of the type of mineralization being enveloped. For example, stringer style would not be as laterally persistent as strong massive sulfide mineralization. Mineralization was outlined in five zones. Previous geologic models have separated the lower portion of Zone 2-3 as the EM Zone; subsequent drilling supports interpretation as a single mineralized zone.

A zone of significant stringer mineralization has been intersected by a single hole in the lower portion of Zone 2-3. This zone has been separately wireframed and is included in the Inferred Mineral Resource (Zone 2-3 Mid). An RW Oxide zone has been wireframed as in the 2015 model. Estimation of this zone is documented here; however due to uncertain metallurgical properties, it is not included in the Mineral Resource Estimate.

Separate ‘fringe’ wireframes were developed as part of 2018 geologic model that essentially envelope the main mineralization solids. Interpolation of grade and density values inside the fringe wireframes was carried out separately and have been included in this resource estimate where mineralization is less than 3 m in horizontal thickness or where horizontal gaps in the mineralized zones are less than 3 m; in this way, the tabled resource reflects a minimum 3 m mineralization thickness as well as internal dilution where unmineralized zones are less than of 3 m in horizontal thickness (not left as pillars on mining). The main purpose of the modelled ‘fringe’ material will be to quantify mining dilution in the upcoming PEA study.

14.2.3 Assay Compositing

Assays were composited to a length of 1.5 m within the bounds of the mineralized wireframes. A 1.5 m composite length was chosen based on the fact that 1.5 m was the most common assay interval as well as its correspondence to the selected resource bock size (4:1).

As was the case for the 2015 resource estimate, less than half-length composites (<0.75 m), resulting from compositing within the mineralized solids, were handled in such a way as to appropriately preserve their influence. The composite length across these zone intersections was recalculated such that composite lengths were equal, and as close to 1.5 m as possible. This technique resulted in composite lengths ranging between 0.76 and 1.72 m.

Correlation between grade and density indicated that the calculation of density weighted grade composites was appropriate. Ninety-five (95) percent of assays inside the reported resource volumes have density measurements; for those without a measured density value, the average density per mineralized solid, was used in the compositing process. This approach is also consistent with the 2015 estimate.

14.2.4 Grade Capping

Grade capping is used to control the impact of extreme, outlier high-grade samples on the overall resource estimate. For this estimate, assay grades were examined in histograms and probability plots to determine levels at which values are deemed outliers to the general population. These cap values were applied by metal, by mineralized zone prior to compositing (see Table 14-2). Uncapped and capped composite statistics are presented in Table 14-3.

Table 14-2 Grade Capping Levels

MinSolid		Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Ba (%)
1	RW East	*uncap*	*uncap*	*uncap*	*uncap*	14.0
2	RW Oxide	1.3	*uncap*	*uncap*	*uncap*	*uncap*
3	RW West	*uncap*	*uncap*	*uncap*	*uncap*	*uncap*
4	Zone 1	*uncap*	*uncap*	150	*uncap*	*uncap*
5 / 51	Zone 2-3	*uncap*	*uncap*	*uncap*	2	*uncap*
52	Zone 2-3 Mid	*uncap*	6	*uncap*	*uncap*	*uncap*

Source: Constantine (2018)

The impact of grade capping can be measured by comparing estimated uncapped and capped block grades above a zero cut-off. At Palmer, metal removed by capping is very low reflecting the fact that relatively few assays were capped. Metal removed through the capping process amounts to: 0.5% for copper, 0% for zinc, 0.4% for silver, 2.0% for gold and 0.7% for barium.

Table 14-3 Composite Statistics (CV (coefficient of variation) = standard deviation ÷ mean)

MinSolid		Uncapped Cu (%)				Capped Cu (%)			
		count	mean	max	CV	#Cap	mean	max	CV
1	RW East	20	2.26	8.76	1.2	0	2.26	8.76	1.2
2	RW Oxide	95	0.24	1.60	1.1	3	0.23	0.90	1.0
3	RW West	18	0.57	3.07	1.6	0	0.57	3.07	1.6
4	Zone 1	242	1.65	10.70	1.0	0	1.65	10.70	1.0
5 / 51	Zone 2-3	415	0.96	11.01	1.5	0	0.96	11.01	1.5
52	Zone 2-3 Mid	18	0.27	0.98	1.2	0	0.27	0.98	1.2
	Total:	**808**	**1.09**			**3**	**1.09**		

MinSolid		Uncapped Zn (%)				Capped Zn (%)			
		count	mean	max	CV	#Cap	mean	max	CV
1	RW East	20	6.49	16.51	0.7	0	6.49	16.51	0.7
2	RW Oxide	95	1.68	9.95	1.5	0	1.68	9.95	1.5
3	RW West	18	5.58	19.59	0.8	0	5.58	19.59	0.8
4	Zone 1	242	4.15	24.01	1.1	0	4.15	24.01	1.1
5 / 51	Zone 2-3	415	5.02	34.64	1.0	0	5.02	34.64	1.0
52	Zone 2-3 Mid	18	3.14	12.50	1.0	8	2.35	6.00	0.8
	Total:	**808**	**4.37**			**8**	**4.36**		

MinSolid		Uncapped Ag (g/t)				Capped Ag (g/t)			
		count	mean	max	CV	#Cap	mean	max	CV
1	RW East	20	35.4	109.1	0.8	0	35.4	109.1	0.8
2	RW Oxide	95	74.5	323.4	0.8	0	74.5	323.4	0.8
3	RW West	18	39.2	139.8	1.0	0	39.2	139.8	1.0
4	Zone 1	242	23.5	140.2	1.1	8	23.3	123.8	1.1
5 / 51	Zone 2-3	415	29.0	378.8	1.2	0	29.0	378.8	1.2
52	Zone 2-3 Mid	18	7.3	20.3	0.9	0	7.3	20.3	0.9
	Total:	**808**	**32.6**			**8**	**32.5**		

MinSolid		Uncapped Au (g/t)				Capped Au (g/t)			
		count	mean	max	CV	#Cap	mean	max	CV
1	RW East	20	0.25	0.81	0.9	0	0.25	0.81	0.9
2	RW Oxide	95	0.49	3.17	1.1	0	0.49	3.17	1.1
3	RW West	18	0.22	0.64	0.8	0	0.22	0.64	0.8
4	Zone 1	242	0.31	2.45	1.2	0	0.31	2.45	1.2
5 / 51	Zone 2-3	415	0.24	5.46	1.5	5	0.22	1.78	1.1
52	Zone 2-3 Mid	18	0.06	0.19	0.9	0	0.06	0.19	0.9
	Total:	**808**	**0.28**			**5**	**0.28**		

MinSolid		Uncapped Ba (%)				Capped Ba (%)			
		count	mean	max	CV	#Cap	mean	max	CV
1	RW East	20	7.02	36.37	1.3	6	4.95	14.00	1.0
2	RW Oxide	95	32.41	55.74	0.6	0	32.41	55.74	0.6
3	RW West	18	11.33	35.09	0.9	0	11.33	35.09	0.9
4	Zone 1	242	12.73	47.49	1.1	0	12.73	47.49	1.1
5 / 51	Zone 2-3	415	12.44	47.91	1.0	0	12.44	47.91	1.0
52	Zone 2-3 Mid	18	5.11	11.04	0.6	0	5.11	11.04	0.6
	Total:	**808**	**14.55**			**6**	**14.50**		

Source: Constantine (2018)

14.2.5 Grade Estimation

The generally narrow and branching nature of massive sulfide mineralization at Palmer make the calculation and meaningful interpretation of variograms difficult. Global variograms are useful for directional analysis; however, there are insufficient sample numbers to fit compelling variogram models. Grades were therefore interpolated by inverse distance cubed weighting (ID3). Examination of downhole variograms and the comparison of ID3 and inverse distance squared (ID2) estimates, led to the choice of interpolation by ID3. At the current drill density, the ID2 estimate appeared unrealistically smooth between drill intersections. Sample search distances and orientations were derived to best fit the zones of mineralization. Search details are presented in Table 14-4.

Table 14-4 Main Area Estimation Search Parameters

MinSolid		Axis	Direction (dip/azimuth)	Range (m)
1	RW East	X	00/145	150
		Y	-23/055	150
		Z	67/055	20
2	RW Oxide	X	00/153	150
		Y	-22/063	150
		Z	68/063	20
3	RW West	X	00/126	150
		Y	-41/036	150
		Z	49/036	20
4	Zone 1	X	00/095	150
		Y	-66/005	150
		Z	24/005	20
5	Zone 2-3 Upper *	X	19/282	150
		Y	-28/232	200
		Z	-20/005	20
51 / 52	Zone 2-3 Lower *	X	13/291	125
		Y	-53/210	175
		Z	-50/005	20

* Upper and Lower refer to elevation; mineralization dip changes at depth.

Source: Constantine (2018)

Grades for all elements were estimated using a minimum of two and a maximum of 16 samples and a maximum of six samples per hole; except for the interpolation of zinc grade where estimation by a maximum of 24 samples produced results that were deemed more valid. All grade estimates were hard-bounded by mineralized zone.

14.2.6 Density Interpolation

In total, 1,105 density measurements have been made on core samples within the main area mineralized zones (1,091 excluding RW Oxide Zone). The abundance of samples coupled with their variability illustrated that interpolation of density values was more appropriate than assignment of a mean value by zone. Density values were estimated by inverse distance squared weighting, within each wireframe domain using a minimum of two and a maximum of 24 samples and the same geologic controls as used for grade estimation. Table 14-5 lists simple statistics of available density measurements, as well as the average interpolated block values, by mineralized zone.

Table 14-5 Density Samples and Interpolated Values, Palmer Main Area

MinSolid		Density Measurements (t/m^3)				Block Estimate
		Count	Avg	Min	Max	Avg Density
1	RW East	33	3.18	2.59	4.16	3.20
2	RW Oxide	14	3.28	2.54	4.04	3.26
3	RW West	30	3.21	2.55	4.41	3.24
4	Zone 1	348	3.59	1.89	4.70	3.56
5 / 51	Zone 2-3	654	3.50	2.39	4.78	3.41
52	Zone 2-3 Mid	26	3.02	2.81	3.43	3.03
Total:		**1,105**	**3.50**	**1.89**	**4.78**	**3.39**

Source: Constantine (2018)

14.2.7 Model Validation

Estimated grades for all elements were validated visually by comparing composite to block values in plan-view and on cross-sections. There is good visual correlation between composite and estimated block grades for all modelled elements.

Two additional check models were estimated for all metals. To appropriately match the composite length, a nearest neighbour model (NN) was estimated using a block size of 6 x 6 x 1.5 m and then re-blocked (4:1) to the resource model grid. Another estimate was made by compositing single intervals across mineralization intersections and interpolating grade with a nearest neighbour approach. This zone composite model (ZC) produces results akin to a polygonal estimate. All models were hard-bounded by mineralized zone.

The ID3 estimates were compared spatially to the check models, using swath plots and deemed appropriately smooth. Globally, model average grades above zero cut-off (shown on swath plots) compare very closely, indicating no bias (Table 14-6).

Table 14-6: Palmer Deposit Estimate Compared to Check Models

MinSolid		Block	Cu (%)			Zn (%)			Ag (g/t)			Au (g/t)			Ba (%)		
		Count	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC
1	RW East	3,324	1.78	1.35	1.35	6.65	6.37	7.16	34.3	26.4	28.7	0.22	0.19	0.19	5.86	5.24	5.71
2	RW Oxide	10,621	0.18	0.17	0.15	1.68	1.66	1.27	58.4	53.6	53.6	0.42	0.38	0.39	31.20	27.23	27.77
3	RW West	12,676	0.44	0.50	0.41	6.36	7.12	6.77	40.9	31.9	41.5	0.22	0.19	0.22	11.45	10.37	11.96
4	Zone 1	28,025	1.39	1.33	1.48	4.20	4.55	4.42	25.8	21.4	28.0	0.31	0.27	0.36	14.24	12.87	15.43
5/51	Zone 2-3	71,959	0.75	0.77	0.82	4.43	4.51	4.35	24.2	27.2	24.8	0.19	0.22	0.19	11.02	11.13	10.68
Total:		**126,605**	**0.84**	**0.83**	**0.88**	**4.40**	**4.59**	**4.42**	**29.4**	**28.6**	**29.7**	**0.24**	**0.24**	**0.25**	**13.33**	**12.64**	**13.16**

Source: Constantine (2018)

14.2.8 Resource Classification and Tabulation

Additional drilling since the 2015 estimate has given sufficient confidence in the geologic continuity to upgrade a portion of the steeply dipping South Wall mineralization (Zones 1 and 2-3) to Indicated Mineral Resource. As a test of confidence in geologic interpretation, the 2015 mineralized solids were pierced by new drilling to allow comparison to intervals actually encountered in Zones 1 and 2-3. Results showed that new drilling enlarged the extents of mineralized zones in some cases, and while not always intersected at exactly the predicted depth, intersected widths were always present and generally thicker that predicted by the 2015 solids. This exercise qualitatively led to confidence in the continuity of the mineralized zones as interpreted from drilling.

Blocks were classified as Indicated Mineral Resource where:

- They are ≥ 25 m inside the extents of mineralized solids,
- They are estimated by composites from at least three holes, and
- The average distance to three holes is ≤ 50 m.

Remaining estimated material within the interpreted wireframes constitutes the Inferred Mineral Resource.

Measures were taken to ensure the Mineral Resource meets the condition of “reasonable prospects of eventual economic extraction” as required under NI 43-101. Potential extraction of the resource will be by underground techniques and while the 6 x 6 x 6 m block size is not a true underground selective mining unit, it is deemed reasonable to tally the resource by NSR cut-off, by-block.

As a selectivity test, of by-block versus some larger mining volume, the resource was broken down into groups of contiguous blocks and tabled by NSR cut-off. For the grouping, blocks were accumulated by model column/level through the mineralized solids (model rows). Grouped in this way, the 36,700 blocks were assigned to 11,700 units of grouped blocks. At a $75 NSR cut-off, the value of blocks totalled in this way was within 1% of the total by-block, giving credibility to the by-block tabulation presented here.

Block NSR value (US$/t) was calculated using a formula supplied by Constantine based on metallurgical test work to date, as well as other relevant project experience. The calculation includes metal price and recoveries as listed in Table 14-7 as well as offsite costs that include concentrate transportation, smelter treatment charges and refining charges. Block NSR was calculated as follows:

$$NSR_{block} = \$48.67 \times \%Cu + \$16.01 \times \%Zn + \$0.32 \times g/t\ Ag + \$23.45 \times g/t\ Au$$

Table 14-7 NSR Parameters

Metal	Price (US$)	Recovery (%)
Cu	3.00/lb	89.6
Zn	1.15/lb	93.1
Ag	$16/oz	90.9
Au	1,250/oz	69.6

Source: Constantine (2018)

The main area resource is stated at a cut-off of US$ 75/tonne (Table 14-8). This is considered appropriate given the likely costs of underground mining and processing of the Palmer deposit. Table 14-9 represents the 2018 estimate at a range of NSR cut-offs. Barite is included in the tables below as opposed to barium, as that is the mineral that will be marketed. The conversion from Ba to $BaSO_4$ is 1.7 times (based on atomic weights).

Table 14-8 2018 RW and SW Mineral Resource Estimate

Category	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$%)	NSR ($/t)
Indicated	4,677	1.49	5.23	30.8	0.30	23.9	173.26
Inferred	5,338	0.96	5.20	29.2	0.28	22.0	146.09

Contained Metal

Category	Cu (M lbs)	Zn (M lbs)	Ag (M oz)	Au (K oz)	Barite (K tonnes)
Indicated	154	539	4.6	45.1	1,118
Inferred	113	612	5.0	48.1	1,174

Source: Constantine (2018)

Table 14-9 2018 Mineral Resource by NSR Cut-off

INDICATED

Cut-off ($/t NSR)	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$%)	NSR ($/t)
65	4,843	1.46	5.12	30.3	0.30	23.7	169.72
70	4,765	1.47	5.17	30.6	0.30	23.8	171.41
75	4,677	1.49	5.23	30.8	0.30	23.9	173.26
80	4,575	1.51	5.29	31.2	0.31	24.1	175.41
85	4,476	1.53	5.35	31.5	0.31	24.2	177.45

INFERRED

Cut-off ($/t NSR)	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$%)	NSR ($/t)
65	5,795	0.91	5.02	28.0	0.27	21.8	140.09
70	5,565	0.94	5.11	28.6	0.27	21.9	143.09
75	5,338	0.96	5.20	29.2	0.28	22.0	146.09
80	4,846	1.03	5.38	30.5	0.30	22.0	153.06
85	4,487	1.09	5.54	31.3	0.30	22.3	158.70

Source: Constantine (2018)

The previous RW and SW resource, reported in 2015, is presented below in Table 14-10 for comparative purposes. The 2018 Resource Estimate includes material classified as Indicated for the first time. Also, metal prices and recoveries used in the calculation of NSR have been adjusted to reflect current conditions.

Table 14-10 Previous (2015) Mineral Resource

INFERRED

Cut-off ($/t NSR)	Tonnes (1,000s)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	NSR ($/t)
65	8,786	1.34	5.08	30.8	0.31	141.61
70	8,516	1.37	5.15	31.1	0.31	143.95
75	8,125	1.41	5.25	31.7	0.32	147.40
80	7,863	1.43	5.33	32.2	0.33	149.75
85	7,638	1.45	5.40	32.6	0.33	151.72

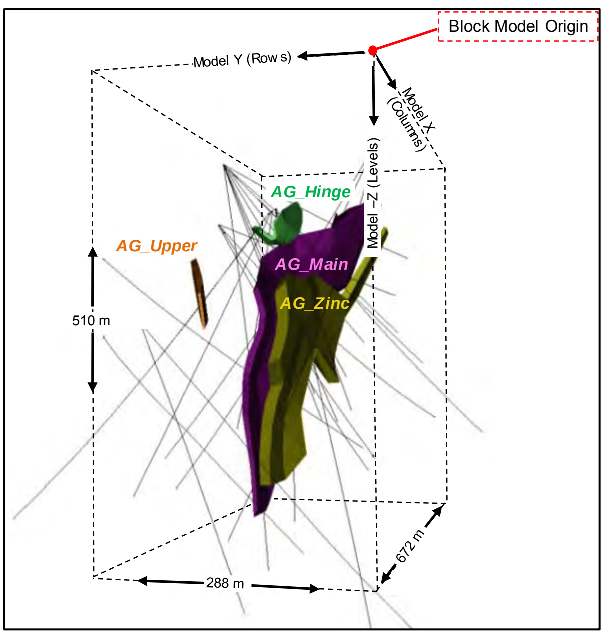
Source: Constantine (2018)

14.3 AG Zone Resource Estimation

14.3.1 Currently Available Drill Data and Model Setup

This AG Zone resource estimate for the Palmer Exploration Project is based on assay data available as of November 15th, 2018. Results from 10,800 m of diamond drilling in 29 holes have been used in the interpretation of geology in support of this resource estimate; 20 holes (7,400 m) intersected the mineralized zones and have been used for grade estimation. Figure 14-3 illustrates drill hole locations, the extents of the resource block model and the interpreted zones of mineralization. Table 14-11 lists the AG Zone block model setup.

Figure 14-3: AG Zone Drilling, Mineralized Zones and Block Model Extents (Looking Southeast)



Source: Constantine (2018)

Table 14-11 AG Zone Block Model Setup

Block:	X	Y	Z
origin[1]	419,457.019	6,582,070.716	1,370
size (m)	6	6	6
nblk	112	48	85

50° Clockwise Rotation about Origin

456,960 blocks

[1] SW model top, block edge

Source: Constantine (2018)

14.3.2 Geologic Model

Consistent with the Palmer main area resource estimate, the geologic model used for control of grade interpolation was derived from wireframes constructed by Constantine personnel; Darwin Green, P. Geo, Vice President of Exploration, oversaw development of the mineralization solids. As a general rule, mineralization of >2% zinc or >60 g/t silver was used to define solids. In several cases, grade cut-off criteria are overruled on the basis of geology where mineralization and rock types are deemed to be geologic horizon equivalents. For example, below cut-off semi-massive to massive sulfide-barite or equivalent exhalative mineralization (e.g. chert). An attempt was made to include all intersections of massive barite/sulfide within the mineralized wireframes.

The AG Zone consists of four mineralized zones:

- A single large lens of massive-barite sulfide – *AG_Main* lens;
- A separate zinc-rich footwall zone of stringer/feeder and replacement style mineralization - the *AG_Zinc* zone;
- The upper NW corner of the deposit is cut by a shallow angle fault, with two small mineralized solids defined above the fault – the *Hinge_Zone;* and
- The *AG_Upper* zone

As was done in the Palmer main area resource estimate, a separate 'fringe' wireframe was developed that essentially envelopes the AG Zone mineralization solids. Interpolation of grade and density values inside the fringe wireframes was carried out separately and will be used to quantify mining dilution in the upcoming PEA study.

14.3.3 Assay Compositing

Assays were composited to a length of 1.5 m within the bounds of the mineralized wireframes. A 1.5 m composite length was chosen based on the fact that 1.5 m was the most common assay interval as well as its correspondence to the selected resource bock size (4:1).

As was the case for the Palmer main area resource estimate, less than half-length composites (<0.75 m), resulting from compositing within the mineralized solids, were handled in such a way as to appropriately preserve their influence. The composite length across these zone intersections was recalculated such that composite lengths were equal, and as close to 1.5 m as possible. This technique resulted in composite lengths ranging between 1.33 and 1.73 m.

Correlation between grade and density indicated that the calculation of density weighted grade composites was appropriate. Ninety-seven (97) percent of assays inside the AG Zone resource volumes have density measurements; for those without a measured density value, the average density per mineralized solid, was used in the compositing process. This approach is consistent with the main area resource estimate.

14.3.4 Grade Capping

Grade capping is used to control the impact of extreme, outlier high-grade samples on the overall resource estimate. For this estimate, assay grades were examined in histograms and probability plots to determine levels at which values are deemed outliers to the general population. These cap values were applied by metal, by mineralized zone prior to compositing (see Table 14-12). Uncapped and capped composite statistics are presented in Table 14-13.

Table 14-12 AG Zone Grade Capping Levels

	MinSolid	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Au (g/t)	Ba (%)
11	AG_Upper	*uncap*	*uncap*	0.4	*uncap*	*uncap*	*uncap*
12	AG_Zinc	*uncap*	0.6	*uncap*	100	*uncap*	3.5
13	AG_Main	*uncap*	1.1	*uncap*	800	*uncap*	*uncap*
14	AG_Hinge	0.4	0.04	*uncap*	*uncap*	*uncap*	*uncap*

Source: Constantine (2018)

The impact of grade capping can be measured by comparing estimated uncapped and capped block grades above a zero cut-off. At Palmer's AG Zone, metal removed by capping is very low reflecting the fact that relatively few assays were capped. Metal removed through the capping process amounts to: 0% for zinc, 3.6% for copper, 0.2% for lead, 4.0% for silver, 0% for gold (uncapped) and 0.1% for barium.

Table 14-13 AG Zone Composite Statistics

MinSolid		Uncapped Zn (%)				Capped Zn (%)			
		count	mean	max	CV	#Cap	mean	max	CV
11	AG_Upper	7	0.18	0.47	0.9	0	0.18	0.47	0.9
12	AG_Zinc	84	5.74	16.70	0.7	0	5.74	16.70	0.7
13	AG_Main	169	3.55	19.50	1.1	0	3.55	19.50	1.1
14	AG_Hinge	43	0.06	0.54	1.4	1	0.05	0.20	1.0
	Total:	**303**	**3.58**			**1**	**3.58**		

MinSolid		Uncapped Cu (%)				Capped Cu (%)			
		count	mean	max	CV	#Cap	mean	max	CV
11	AG_Upper	7	0.08	0.11	0.5	0	0.08	0.11	0.5
12	AG_Zinc	84	0.13	0.51	1.0	4	0.12	0.44	0.9
13	AG_Main	169	0.13	1.46	1.6	4	0.12	0.93	1.4
14	AG_Hinge	43	0.01	0.06	1.5	1	0.01	0.03	1.1
	Total:	**303**	**0.11**			**9**	**0.10**		

MinSolid		Uncapped Pb (%)				Capped Pb (%)			
		count	mean	max	CV	#Cap	mean	max	CV
11	AG_Upper	7	0.20	0.39	0.5	1	0.18	0.28	0.3
12	AG_Zinc	84	0.18	1.64	1.5	0	0.18	1.64	1.5
13	AG_Main	169	1.03	8.38	1.4	0	1.03	8.38	1.4
14	AG_Hinge	43	0.06	0.32	1.3	0	0.06	0.32	1.3
	Total:	**303**	**0.64**			**1**	**0.64**		

MinSolid		Uncapped Ag (g/t)				Capped Ag (g/t)			
		count	mean	max	CV	#Cap	mean	max	CV
11	AG_Upper	7	298.9	592.2	0.5	0	298.9	592.2	0.5
12	AG_Zinc	84	14.1	201.9	1.8	1	12.4	80.5	1.2
13	AG_Main	169	128.3	1,319.9	1.2	2	123.3	717.5	1.0
14	AG_Hinge	43	35.8	134.5	0.9	0	35.8	134.5	0.9
	Total:	**303**	**87.4**			**3**	**84.2**		

MinSolid		Uncapped Au (g/t)				Capped Au (g/t)			
		count	mean	max	CV	#Cap	mean	max	CV
11	AG_Upper	7	0.85	1.73	0.7	0	0.85	1.73	0.7
12	AG_Zinc	84	0.10	0.55	0.9	0	0.10	0.55	0.9
13	AG_Main	169	0.48	3.53	1.2	0	0.48	3.53	1.2
14	AG_Hinge	43	0.54	3.01	1.2	0	0.54	3.01	1.2
	Total:	**303**	**0.39**			**0**	**0.39**		

MinSolid		Uncapped Ba (%)				Capped Ba (%)			
		count	mean	max	CV	#Cap	mean	max	CV
11	AG_Upper	7	47.77	56.48	0.1	0	47.77	56.48	0.1
12	AG_Zinc	84	0.72	5.00	1.3	2	0.69	3.42	1.2
13	AG_Main	169	24.04	57.84	0.7	0	24.04	57.84	0.7
14	AG_Hinge	43	24.55	52.61	0.7	0	24.55	52.61	0.7
	Total:	**303**	**18.20**			**2**	**18.19**		

Source: Constantine (2018)

14.3.5 Grade Estimation

Compared to the Palmer main area resource estimate, the AG Zone is presently interpreted as consisting of zones of more continuous mineralization. At the present drill spacing the branching nature of mineralization, seen in the main area, is not apparent. Despite that fact, at present there is insufficient drilling to calculate meaningful variograms and estimation has been carried out by geometric methods. Examination of downhole variograms and based on the experience in the main area, grades were interpolated by inverse distance cubed weighting (ID3). Sample search distances and orientations were derived to best fit the zones of mineralization. Search details are presented in Table 14-14.

Table 14-14 Estimation Search Parameters – AG Zone

MinSolid		Axis	Direction (dip/azimuth)	Range (m)
11	AG Upper	X	0/120	150
		Y	83/030	150
		Z	7/210	50
12	AG Zinc Zone West	X	0/142	150
		Y	-82/052	150
		Z	8/052	50
112	AG Zinc Zone East	X	0/127	150
		Y	-64/037	150
		Z	26/037	50
13	AG Main Lens West	X	0/147	150
		Y	87/057	150
		Z	3/237	50
113	AG Main Lens East	X	0/125	150
		Y	-63/035	150
		Z	27/035	50
14	AG Zone Hinge SW	X	0/156	150
		Y	78/066	150
		Z	12/246	50
114	AG Zone Hinge NE	X	0/138	150
		Y	31/048	150
		Z	59/228	50

Source: Constantine (2018)

Grades for all elements were estimated using a minimum of two and a maximum of 16 samples and a maximum of six samples per hole. All grade estimates were hard-bounded by mineralized zone.

14.3.6 Density Interpolation

In total, 528 density measurements have been made on core samples within the AG Zone mineralized wireframes. The abundance of samples coupled with their variability illustrated that interpolation of density values was more appropriate than assignment of a mean value by zone. Density values were estimated by inverse distance squared weighting, within each wireframe domain using a minimum of two and a maximum of 24 samples and the same geologic controls as for grade estimation – hard bounded by mineralized domain. Table 14-15 lists simple statistics of available density measurements, as well as the average interpolated block values, by mineralized zone.

Table 14-15 AG Zone Density Samples and Interpolated Values

MinSolid		Density Measurements (t/m^3)				Block Estimate
		Count	Avg	Min	Max	Avg Density
11	AG_Upper	18	3.93	2.61	4.45	3.95
12	AG_Zinc	150	3.19	2.72	4.93	3.15
13	AG_Main	304	3.77	2.24	4.60	3.66
14	AG_Hinge	56	3.54	2.47	4.44	3.54
Total:		**528**	**3.59**	**2.24**	**4.93**	**3.53**

Source: Constantine (2018)

14.3.7 Model Validation

Estimated grades for all elements were validated visually by comparing composite to block values in plan-view and on cross-sections. There is good visual correlation between composite and estimated block grades for all modelled elements.

As was done for the Palmer main area resource estimate, two additional check models were estimated for all metals. To appropriately match the composite length, a nearest neighbour model (NN) was estimated using a block size of 6 x 6 x 1.5 m and then re-blocked (4:1) to the resource model grid. Another estimate was made by compositing single intervals across mineralization intersections and interpolating grade with a nearest neighbour approach. This zone composite model (ZC) produces results akin to a polygonal estimate. All models were hard-bounded by mineralized zone.

The ID3 estimates were compared spatially to the check models, using swath plots. Rather than reproduce the plots here, Table 14-16 list results comparing the ID3 estimates to the test models by mineralized domain. Globally, model average grades above zero cut-off (shown in the table below) compare very closely, indicating no bias. Swath plots did indicate that ID estimates are appropriately smooth in comparison to the check models.

Table 14-16 AG Zone Estimates Compared to Check Models

MinSolid		Block	Zn (%)			Cu (%)			Pb (%)			Ag (g/t)			Au (g/t)			Ba (%)		
		Count	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC	ID^3	NN	ZC
11	AG_Upper	0																		
12	AG_Zinc	4,093	5.62	5.71	5.58	0.12	0.12	0.12	0.16	0.23	0.20	11.1	12.1	13.1	0.09	0.11	0.10	0.63	0.62	0.67
13	AG_Main	10,416	3.08	2.83	3.21	0.10	0.11	0.11	0.86	0.86	0.92	121.4	123.5	134.9	0.52	0.52	0.52	22.44	22.25	23.75
14	AG_Hinge	240	0.05	0.06	0.06	0.00	0.00	0.00	0.05	0.05	0.06	36.3	42.0	36.6	0.57	0.79	0.54	24.47	26.70	23.10
Total:		14,749	3.74	3.58	3.82	0.10	0.11	0.11	0.65	0.67	0.71	89.4	91.2	99.5	0.40	0.41	0.40	16.42	16.32	17.33

Source: Constantine (2018)

14.3.8 Resource Classification and Tabulation

The current density of exploration drilling is sufficient only for the declaration of an Inferred Mineral Resource in the AG Zone; estimated blocks inside the mineralized wireframes are assigned as Inferred where estimated by:

- At least two holes the closest of which is within 75 m (98%), or
- Composites from at least three holes (1%), or
- A single hole within 30 m (1%).

Of blocks classified by the first criterion above, 98% of those were estimated by three or more holes. *AG_Upper* blocks were not included in the declared resource as there is presently only one drill intercept in the zone.

Measures were taken to ensure the Mineral Resource meets the condition of “reasonable prospects of eventual economic extraction” as required under NI 43-101. Potential extraction of the resource will be by underground techniques. The 6 x 6 x 6 m block size is not a true underground selective mining unit and as a selectivity test, of by-block versus some larger mining volume, the resource was broken down into groups of contiguous blocks and tabled by ZnEq cut-off. For the grouping, blocks were accumulated (at zero cut-off) by model column/level through the mineralized solids (model rows). Grouped in this way, the 14,700 blocks were assigned to 4,250 units of grouped blocks. At a 5% ZnEq cut-off, the tonnage and metal content of blocks totalled in this way was within 4% of the total by-block, giving credibility to the by-block tabulation presented here.

Block ZnEq values (%) were based on 100% recovery and payable for zinc, copper, lead, silver and gold, and metal prices of:

- Zinc – US$ 1.15/lb
- Copper – US$ 3.00/lb
- Lead – US$ 1.00/lb
- Silver – US$ 16/oz
- Gold – US$ 1250/oz

The zinc equivalence formula is:

$$ZnEq = (25.3 \times Zn\% + 66 \times Cu\% + 22 \times Pb\% + 0.51 \times Ag\ g/t + 40.19 \times Au\ g/t)/25.3$$

The AG Zone resource and contained metal is stated at a cut-off of 5% ZnEq (Table 14-17). A range of cut-offs are presented for comparison. Barite is included in the tables below as opposed to barium, as that is the mineral that will be marketed. The conversion from Ba to $BaSO_4$ is 1.7 times (based on atomic weights).

Table 14-17 AG Zone Inferred Mineral Resource by ZnEq Cut-off

Cut-off Grade (% ZnEq)	Tonnes (1,000s)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Au (g/t)	Barite (BaSO4%)	ZnEq (%)
4.5	4,648	4.48	0.12	0.90	114.2	0.50	34.1	8.68
5.0	**4,256**	**4.64**	**0.12**	**0.96**	**119.5**	**0.53**	**34.8**	**9.04**
5.5	3,975	4.78	0.13	1.00	122.2	0.54	34.7	9.31

Contained Metal

Cut-off Grade (% ZnEq)		Zn (M lb)	Cu (M lb)	Pb (M lb)	Ag (M oz)	Au (K oz)	Barite (K tonnes)	ZnEq (M lbs)
4.5		459	12	92	17.1	74.7	1,583	889
5.0		**435**	**11**	**90**	**16.4**	**72.5**	**1,480**	**848**
5.5		419	11	88	15.6	69.0	1,379	816

Source: Constantine (2018)

14.4 Palmer Project Mineral Resource Statement

The Palmer Project Mineral Resource includes the RW, South Wall (Section 14.1) and AG Zones (Section 14.2) as presented in Table 14-18. Copper equivalent grade is also included in the table due to the importance of copper in the SW Zone. As described above, for zinc equivalence, CuEq is calculated as:

CuEq = (25.3 x Zn% + 66 x Cu% + 22 x Pb% + 0.51 x Ag g/t + 40.19 x Au g/t)/66

Table 14-18 2019 Palmer Project Mineral Resource Statement

Zone	Cut-off	Resource Category	Tonnes (1,000s)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$%)	ZnEq (%)	CuEq (%)
RW and	$75/t NSR	Indicated	4,677	5.23	1.49	-	30.8	0.30	23.9	10.21	3.92
South Wall	$75/t NSR	Inferred	5,338	5.20	0.96	-	29.2	0.28	22.0	8.74	3.35
AG Zone	5.0% ZnEq	Inferred	4,256	4.64	0.12	0.96	119.5	0.53	34.8	9.04	3.46
Total:		Indicated	4,677	5.23	1.49	-	30.8	0.30	23.9	10.21	3.92
		Inferred	9,594	4.95	0.59	0.43	69.3	0.39	27.7	8.87	3.40

CONTAINED METAL

	Resource Category	Zn (M lb)	Cu (M lb)	Pb (M lb)	Ag (M oz)	Au (K oz)	Barite (K tonnes)	ZnEq (M lbs)	CuEq (M lbs)
Total:	Indicated	539	154	-	4.6	45.1	1,116	1,053	404
	Inferred	1,047	124	90	21.4	120.6	2,654	1,876	719

Source: Constantine (2018)

15 Mining Reserve Estimate

No Mineral Reserve has been established at the Palmer Project in this report.

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 within the PEA mine plan will be converted into mineral reserves.

16 Mining Methods

16.1 Summary

Mining of the Palmer Project will be conducted using underground mining methods. The potential mine will be accessed using the 680 Exploration Portal which will connect with the Palmer deposit at 845 masl. This ramp will serve as a personnel and material entry point to the mine, the haulage route from the AG Zone deposit to the Palmer deposit, and the primary fresh air source. The transportation of mineralized material from the mine to the mill will take place in the 510 Conveyor Drift.

The Palmer Project hosts two deposits; the Palmer deposit and the AG Zone deposit. The Palmer deposit is segregated into three zones; Lower Zone 2-3, Zone 2-3, and Zone 1. All production levels are referenced to the drift elevation in regard to elevation in meters above sea level (masl). The Lower Zone 2-3 extends from the 848 Level to 628 Level. Zone 2-3 extends from 848 Level to 1128 Level. Zone 1 extends from 1130 to 1330 Level. The AG Zone deposit extends from the 1000 Level to the 1280 Level. Mining of both the Palmer deposit and AG Zone deposit will occur in 20 m vertical increments, and will be connected by a primary spiral ramp, sized at 5.0 m W x 6.0 m H and located in the footwall of the deposit.

The selected mining method for the Palmer deposit will be transverse and longitudinal LH mining with paste backfill in the mined-out voids with a primary-secondary stoping sequencing. Thinner portions of the deposit will be mined using longitudinal longhole.

The mine production rate is approximately 3,400 t/d once full production is established in year 3. The mineralized material will be crushed underground and transported via the 510 Conveyor Drift to the mill. Mineralized material from the Palmer deposit will be fed into the conveyors via the production pass system; the AG Zone deposit material will be hauled to the production pass in the Palmer deposit.

Once a mining panel has been exhausted, the void will be backfilled with a cemented paste backfill composed of pyrite tails, de-sulfide tails and run of mine waste rock that is potentially acid generating (PAG). Paste backfill originating from pyrite tails will be prioritized for primary stopes. Paste for secondary stopes will be composed of the remaining pyrite tails and supplement with de-sulfide tails. Run of mine PAG will be deposed in any available open void. The remaining de-sulfide tails will be filtered and placed at the tailings storage facility (TSF).

Non-potentially acid generating (NPAG) rock generated from the mine will be used in the construction of site infrastructure and the TSF.

16.2 Geotechnical Analysis and Recommendations

16.2.1 Geotechnical Data and Characterization

Geotechnical specific drilling and testing programs have not yet been carried out for the project; however, preliminary geotechnical logging of resource core has been carried out by Constantine since acquiring the project. The geotechnical logging was conducted according to the Q rock mass classification system (Barton & Grimstad, 1994) including the joint set number (Jn), joint roughness (Jr), and joint alteration (Ja) parameters as well as rock quality designation (RQD) and total core recovery (TCR).

The Constantine geotechnical database includes intervals from a total of 11 drillholes within the SW Zone 1, 21 drillholes from SW Zone 2/3, 4 drillholes from RW East and 6 drillholes from the RW West Zone. Complete geotechnical data was available for intervals from four drillholes piercing the southeast AG Zones with partial data available for several others. JDS has estimated geotechnical logging parameters using core photographs for an additional four drillhole pierces through the AG Zones northwest section in order to develop PEA geotechnical design recommendations.

A scoping-level geotechnical assessment was carried out for the project by Henry Bogert, Ph.D., P.E. (Bogert, 2018). The Constantine geotechnical database through end of the 2017 was assessed and data was grouped into hanging wall (5 to 6 m), footwall (5 to 6 m) and resource zone for the study. A single Q value was calculated from the database to represent the hanging wall, footwall and resource zone for each drillhole by averaging the drillhole runs within the respective zone. Each interval was weighted equally, regardless of length. Empirical stope design analyses were not carried out as part of the 2018 geotechnical scoping assessment.

It was noted by JDS during review of the database that the Jn parameter used in the Q calculation was typically logged by core run rather than by rock type or zone. Often not all joint sets present in a rock mass on a stope wall scale are observed in a single run length which can lead to fewer joint sets being logged than are actually present. As a result, the Jn values used in the scoping assessment calculations (typically 0.5 to 3) may be lower than actually exist resulting in unconservatively high Q values. This would likely bias the upper bound Q values reported in the scoping study to be higher than actual in-situ values. As a result, unusually high values of Q noted in the database were excluded from the statistical calculations used for the PEA stope design. The Jn values will need to be reconciled as the project advances to further stages of study but are considered reasonable for a PEA-level assessment.

JDS has used the data developed by Bogert (2018) as the basis of the PEA geotechnical design. The 25th and 75th percentile Q' values were used to estimate the range of rock mass quality that may be anticipated on a stope wall scale for each zone. The average Q' value as well as the 25th and 75th percentiles are summarized in Table 16-1 by zone and deposit. The Q' value, which excludes the third quotient in the Q equation (stress reduction factor, Jw/SRF), was used rather than Q, which includes the third quotient, as stresses are accounted for separately in the empirical analyses by use of the 'A' (stress) parameter. For the mostly shallow deposits at Palmer, the use of the Q value as input to the empirical analyses may result in overly conservative estimates of stable stope dimensions.

Table 16-1: Summary of Rock Mass Quality Parameters by Zone

Deposit	Zone	Q' [1]			
		Intervals	25%	Mean [2]	75% [3]
SW Zone 1	Hanging Wall (North)	11	3.1 (Poor)	15.1 (Good)	27.6 (Good)
	Mineralized Zone	11	9.1 (Fair)	29.3 (Good)	59.2 (Very Good)
	Footwall (South Wall)	11	4.7 (Fair)	21.1 (Good)	47.8 (Very Good)
SW Zone 2/3 Upper	Hanging Wall (South)	15	4.6 (Fair)	18.1 (Good)	12.8 (Good)
	Mineralized Zone	34	6.4 (Fair)	25.1 (Good)	41.8 (Very Good)
	Footwall (North)	15	3.0 (Poor)	8.8 (Fair)	20.1 (Good)
	Hanging Wall (South)	6	4.3 (Fair)	22.3 (Good)	17.1 (Good)

Deposit	Zone	Q' [1]			
		Intervals	25%	Mean [2]	75% [3]
SW Zone 2/3 Lower	Mineralized Zone	14	6.9 (Fair)	16.9 (Good)	25.7 (Good)
	Footwall (North)	6	5.2 (Fair)	10.5 (Good)	32.5 (Good)
RW West	Hanging Wall	6	13.5 (Good)	25.8 (Good)	41.6 (Very Good)
	Mineralized Zone	5	5.8 (Fair)	14.7 (Good)	23.9 (Good)
	Footwall	6	10.7 (Good)	17.2 (Good)	23.0 (Good)
RW East	Hanging Wall	4	7.5 (Fair)	35.0 (Good)	60.7 (Very Good)
	Mineralized Zone	4	7.5 (Fair)	22.7 (Good)	48.9 (Very Good)
	Footwall	4	2.3 (Poor)	15.2 (Good)	27.3 (Good)
AG	Hanging Wall	48	1.2 (Poor)	8.8 (Fair)	7.1 (Fair)
	Mineralized Zone	66	3.1 (Poor)	10.5 (Good)	12.2 (Good)
	Footwall	48	1.0 (Poor)	6.1 (Good)	7.1 (Fair)

1. Q' is calculated by setting the Joint Water Factor (Jw) and Stress Reduction Factor (SRF) both equal to 1 in the Q equation.
2. Text description next to the average Q' values are according to (Barton & Grimstad, 1994) system.
3. Upper bound values may be unrepresentatively high due to Jn parameter recorded by run.

Source: JDS (2019)

With the exception of the AG Zone deposit, the overall rock mass quality between the various deposits appears to be consistent but highly variable at all of the deposits, based on the geotechnical data available. It does not appear that the hanging wall, footwall or mineralized zones consistently exhibit better or worse rock quality when compared to the other zones. The rock mass is generally of 'Fair' to 'Good' quality according to the Q (Barton & Grimstad, 1994) rock mass classification system. The 'Poor' rock quality zones encountered are anticipated to represent local shears or heavily fractured zones. Major fault structures have not been modeled for the project to date.

Based on the available geotechnical data, the southeast portion of the AG Zone deposit appears to be of significantly lower rock quality than the northwest portion and the other deposits at Palmer. The rock mass at the AG southeast zone is typically of 'Poor' to 'Fair' rock quality according to the Q (Barton & Grimstad, 1994) rock mass classification system.

16.2.2 Stope Dimensions

Maximum stope dimensions were estimated using the Potvin (2001) method for the anticipated range of ground conditions and stope dimensions. The Trueman & Mawdesley (2003) 'Stable' line was then used as a check against the Potvin (2001) results. Empirical factors used in the calculation of stability numbers (N') were based on the following assumptions:

- Induced stress parameter (A) varied depending on the depth of the stope below ground surface. A value of 1.0 (least conservative) was used for the shallow, SW Zone 1, RW and AG stopes, 0.6 to 0.8 was used for the upper SW Zone 2/3 and 0.4 to 0.6 for the lower SW Zone 2/3
- Joint orientation factor (B) was set to 0.2 (most conservative) for the stope hanging walls and footwalls based on the dominant discontinuity orientation (bedding) being sub-parallel to the mineralization and stope walls and,

- Gravity factor, C for stope walls, except for the footwall, was calculated based on the dip of the particular wall using the equation: C =8 – 6 cos (face dip angle). The footwall is assigned the maximum C value of 8.0 because of its favourable orientation.

The maximum level spacing was set at 20 m (floor to floor) and the maximum stable lengths and widths were then estimated from the stability graph for the stope hanging walls and backs. A maximum hydraulic radius was then provided for each stope wall and deposit to be used for mine design as summarized in Table 16-2. Figure 16-1 contains output plots from a typical empirical stope stability analysis for transverse LH mining of the SW Zone 2/3 deposit, Lower portion.

Table 16-2: Maximum Hydraulic Radii for Each Deposit and Stope Wall

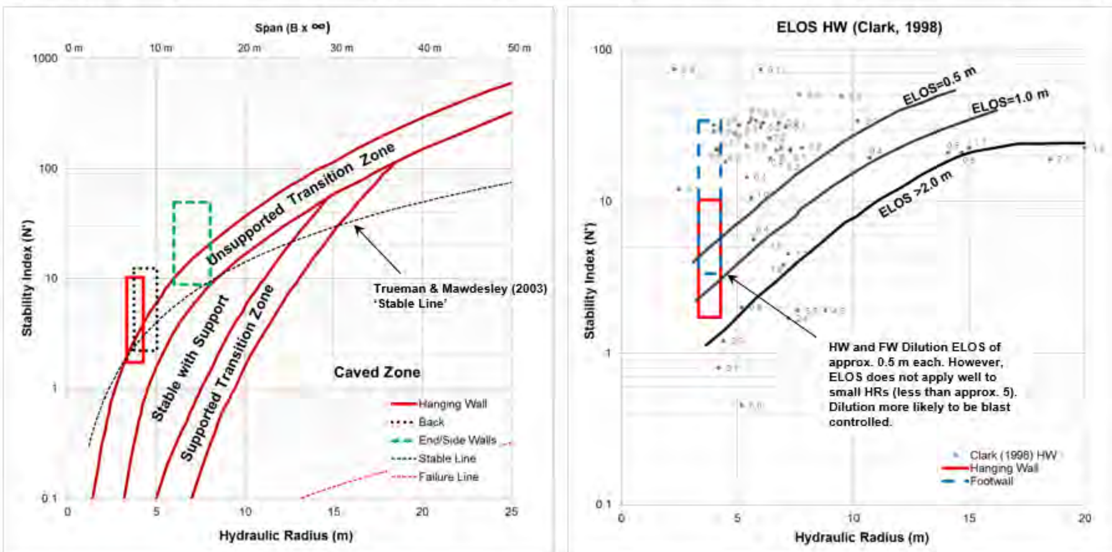
Zone	Stope Wall			Comment
	Hanging Wall (along strike)	Back (flat)	End (perp. to strike)	
SW Zone 1	5	5	10	
SW Zone 2-3 Upper	5.5	6	11	
SW Zone 2-3 Lower	5	5.5	8.5	Controlled by shallow hanging wall dip and potential for higher stress
RW East	6.5	7	12	
RW West	8	6	10	Back controls due to apparent quality of ore

Source: JDS (2019)

A minimum 30 m thick crown pillar for SW Zone 1 was estimated based on currently planned stope spans and strike lengths, at the top of the deposit. The crown pillar estimate was based on the scaled span method (Carter et al, 2008).

Figure 16-1: Example Empirical Stope Stability Analyses and Dilution Estimate (Lower SW Zone 2/3)

Wall	Dip	Q'	Empirical Parameters			N'	Max. and Min. Ranges		
			A (Stress)	B (Joint Orientation)	C (Dip Angle)		Width	Height	Length
HW (South)	60	4.3	0.4	0.2	5.0	1.7	45.0	25.0	13.0
FW (North)	50	5.2	0.4	0.2	8.0	3.4			
Back	0	6.9	0.4	0.4	2.0	2.2			
End/Side	90	6.9	0.4	0.4	8.0	8.8			
HW (South)	60	17.1	0.6	0.2	5.0	10.2	30.0	20.0	10.0
FW (North)	50	35.5	0.6	0.2	8.0	34.0			
Back	0	25.7	0.6	0.4	2.0	12.3			
End/Side	90	25.7	0.6	0.4	8.0	49.3			



Source: JDS (2019)

16.2.3 Ground Support

Based on the range of anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings, ground support requirements were initially assessed according to the Barton & Grimstad (1994) criteria. The Q-system also accounts for 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, in order to obtain the equivalent span for use in the Q Support Diagram; in effect, it imposes a higher factor of safety on critical structures with long life (such as an underground nuclear power station with an ESR rating of 0.5 to 0.8) than on temporary tunnels (such as temporary mine workings with an ESR rating of 2 to 5). An ESR of 1.6 was used for permanent development and man-entry stope development with temporary, non-entry stope development assessed assuming an ESR of 3.

Based on the Barton & Grimstad (1994) criteria, most of the temporary and permanent, non-intersection development would require only spot bolting as a minimum to remain serviceable; however, pattern bolting and welded wire mesh are recommended to control loose material for all development where miners will enter. The ground support recommended for each type of permanent and temporary/ore development include:

- Permanent Development (5 m W x 5 H):

  - 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
  - Assume 15 % of the all permanent development intersections will require 5 cm of shotcrete in addition to bolting
  - Assume 20 % of the all permanent development in the AG Zone deposit will require 5 cm of shotcrete in addition to bolting
- Temporary / Stope Development (5 m W x 5 H):
  - 2.4 m long, 39 mm split sets 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
  - No shotcrete required in temporary/ore development.

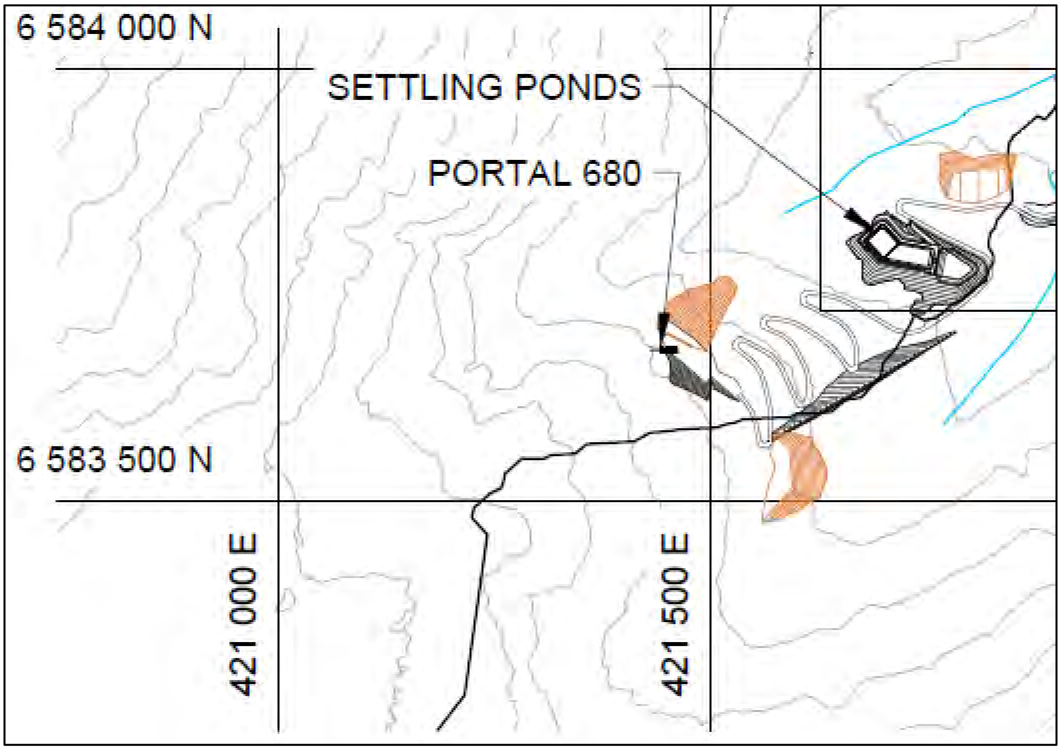
16.3 Mine Access and Development

16.3.1 Portals

The Mine will have three points of access. The 680 Exploration Portal will grant access for haulage and secondary equipment. A cross-cut near the portal entry will be blasted for installation of the main fans and heater for fresh air feed. The 510 Conveyor Drift will be used strictly for the transport of mineralized material out of the mine and secondary egress for the Palmer deposit. The AG 1000 Ramp will be used for secondary egress.

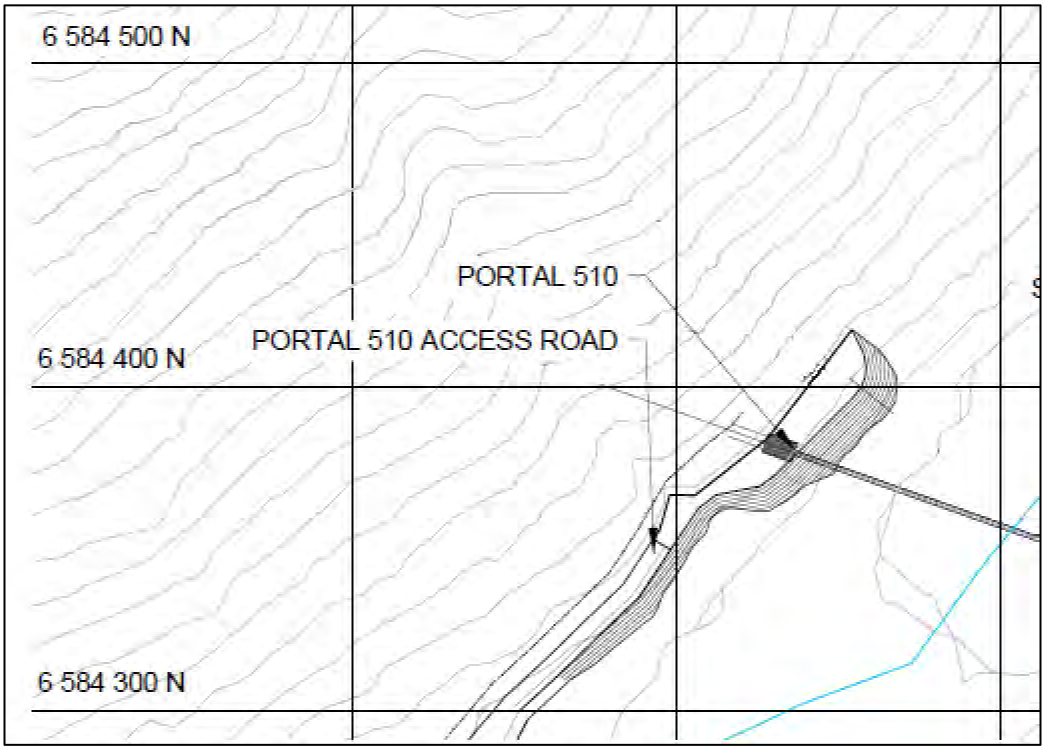
The portal laydown for the 680 Exploration Portal and 510 Conveyor Drift have been designed with a rock slope of 4:1 vertical:horizontal (V:H) with a 5 m catch bench in 10 m vertical intervals. Due to the relatively steep terrain, rock fill slopes have been set to 1:1.25 V:H. The 1000 Level Ramp is located on flat terrain, no major excavation will be required. A general arrangement of the portals is shown in Figure 16-2 and Figure 16-3.

Figure 16-2: Palmer 680 Exploration Portal Location



Source: JDS (2019)

Figure 16-3: Palmer 510 Conveyor Drift Location

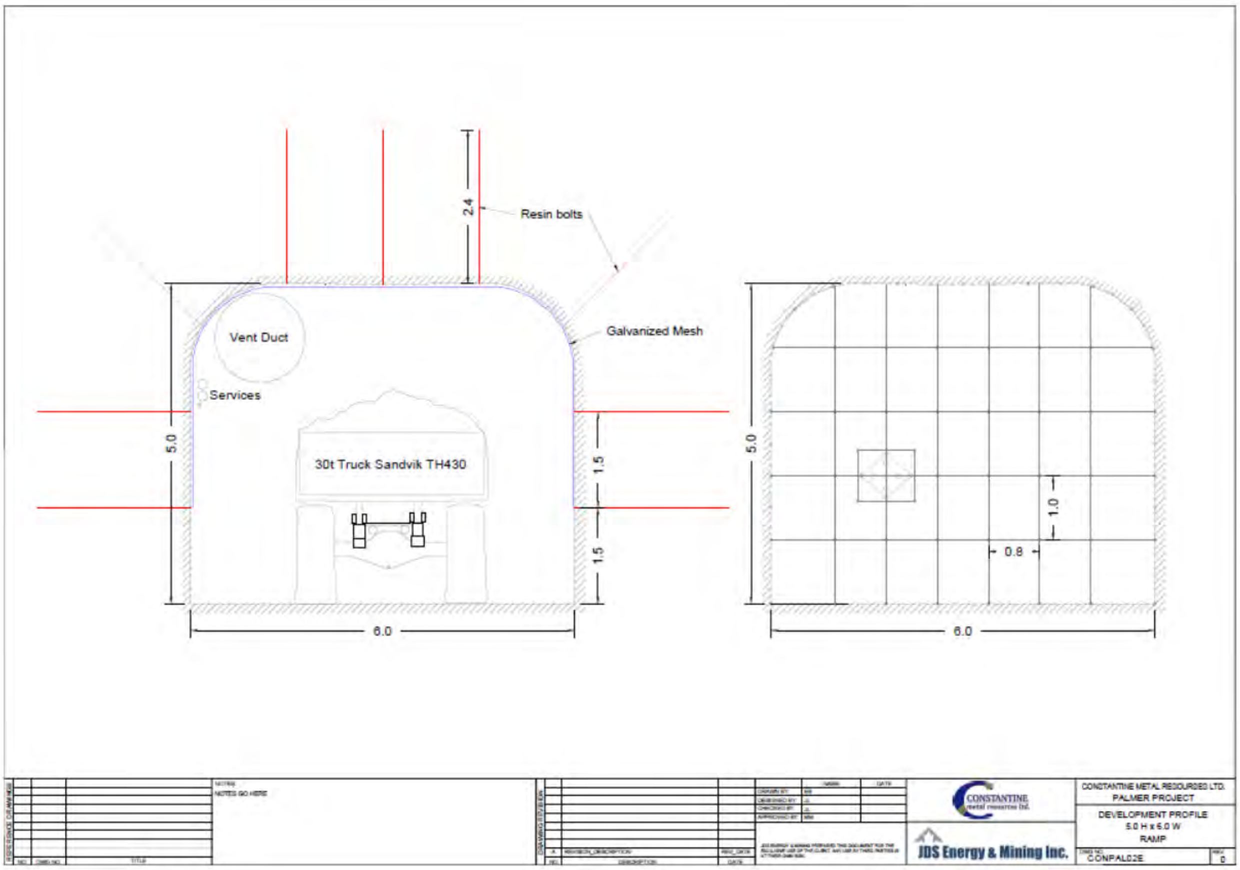


Source: JDS (2019)

16.3.2 Lateral Development

The 680 Exploration Portal will be driven to the 848 Level in Palmer deposit, a linear distance of 1,600 m. It will be driven with an arched profile at 5.0 m W x 5.0 m H, sized to accommodate the necessary ventilation ducting, services and mobile equipment. The ramp will be used for haulage between the AG Zone deposit and the Palmer deposit. It will also act as a fresh air feed into the mine, with the primary fan and heater located at the portal. Pull outs have been designed every 100 m and the ramp has been designed at a maximum gradient of 15%. A general cross section of the mine development headings is shown in Figure 16-4.

Figure 16-4: Development Heading General Cross Sections

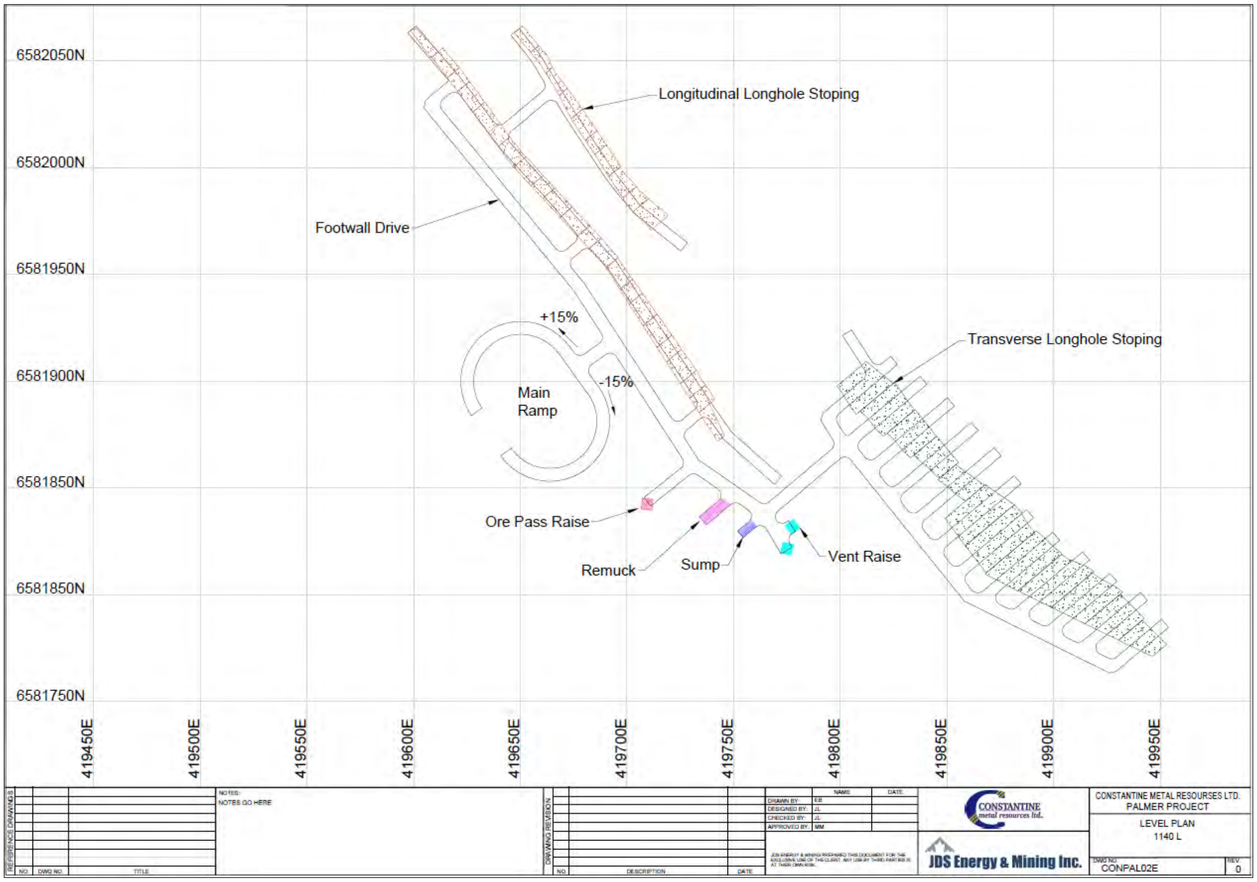


Source: JDS (2019)

Each working level of the mine will be connected using a 6.0 m W x 5.0 m H spiral ramp located in the footwall of the deposit. The spiral ramp will provide fresh air and will connect all levels of the mine, from 628 masl to 1330 masl. Access to the production pass is at every other level, the spiral ramp will be used for haulage of waste and mineralized material to the production pass. The ramp has been designed at gradients of 12-15% to accommodate the relatively close level spacing of 20 m with a minimum turning radius of 20 m on centre.

Each mining level will have a 6.0 m W x 5.0 m H footwall drive to allow haul trucks and production LHD's to access the stoping cross-cuts. The profile of these will also be arched and shall be located at a minimum offset of 10 m from the deposit in the footwall. Stope sill drifts will be driven at 5.0 m W x 5.0 m H with a flat back to accommodate remote LHD entry. The sill drift will be spaced in 16 m intervals along the footwall drift and driven through the deposit to the hanging wall. The footwall drives will house the majority of services including remuck bays, ancillary bays, ventilation raise access, and production pass access. The stope sill drifts will be driven for LH drilling, mineral extraction and backfill placement. The footwall drift will also contain the electrical sub-stations and a refuge bays, the locations will vary based on current mining activities. A plan view of a typical level is shown in Figure 16-5.

Figure 16-5: Typical Level Layout (AGZ Level Shown)



Source: JDS (2019)

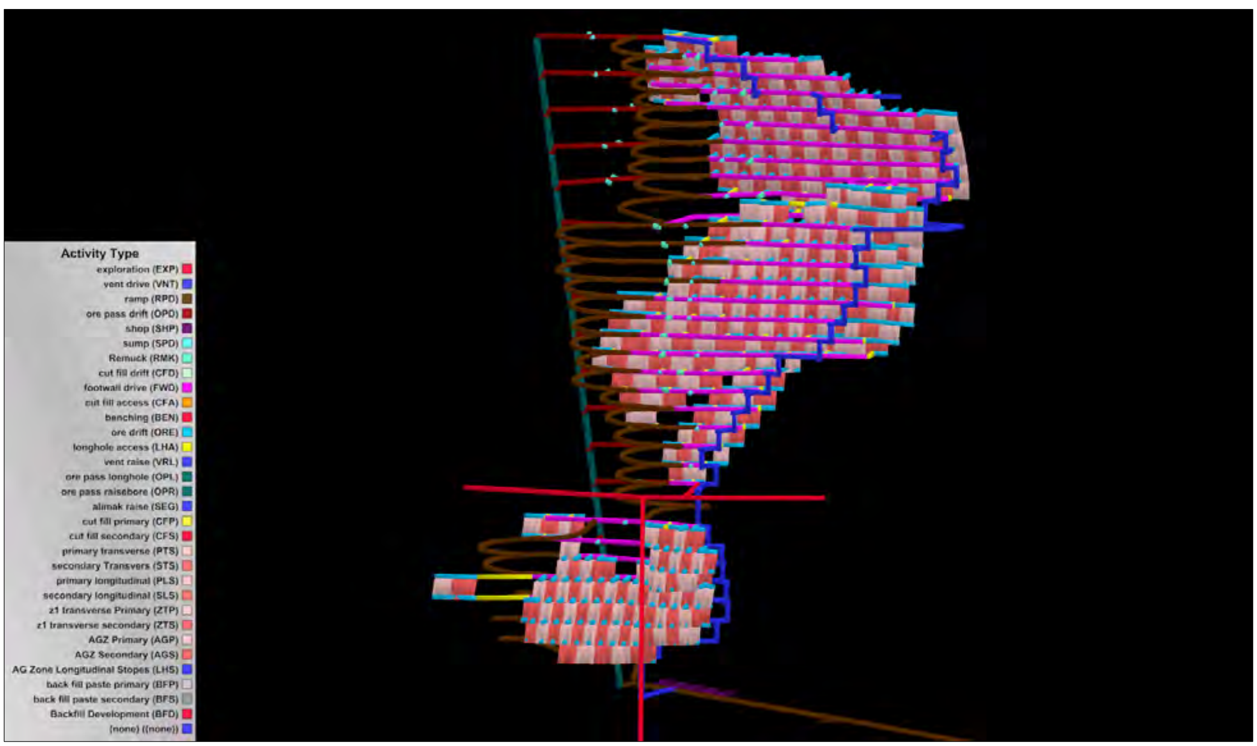
The lateral development requirement for the mine plan is shown in Table 16-3. Figure 16-6 and Figure 16-7 illustrates the mine design for the Palmer deposit and AG Zone deposit respectively. Of these total development, 7,747 m will be driven during the pre-production development period, 10,961 m as sustaining capitalized development, and 41,772 m as operating cost development.

Table 16-3: Lateral Development Summary

Items	Units	Width	Height	Pre-Production	Sustaining	Operating	Total
Ramp	m	6.0	5.0	3,795	10,961		14,756
Footwall Drive	m	6.0	5.0	1,389		9,367	10,756
Sump	m	4.0	4.0	56		937	993
Production Pass Drive	m	5.0	5.0	235		1,613	1,848
Ventilation Drive	m	6.0	5.0	426		1,894	2,320
Maintenance Shop	m	6.6	7.2	125		-	125
Long-hole Access	m	5.0	5.0	390		5,048	5,438
Remuck	m	5.0	5.0	-		600	600
Transverse Access	m	5.0	5.0	1,332		22,313	23,645
Total Waste Lateral	**m**			**7,748**	**10,961**	**41,772**	**60,481**

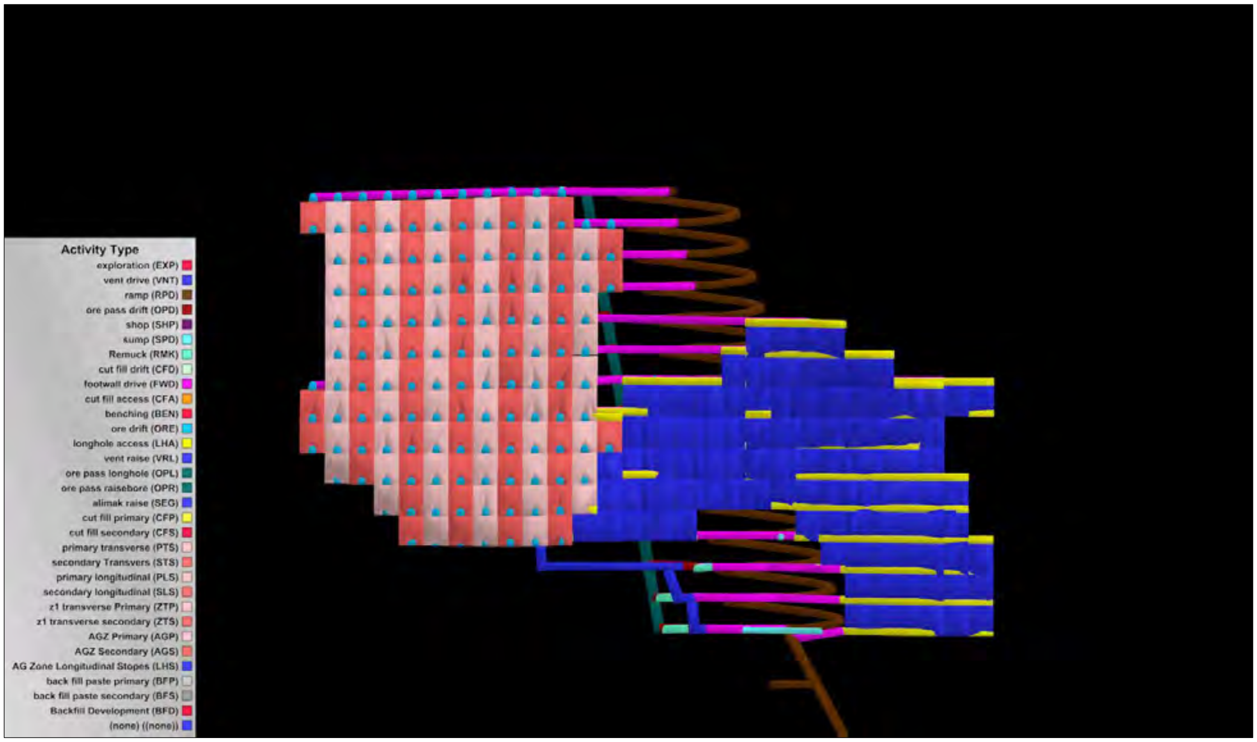
Source: JDS (2019)

Figure 16-6: Palmer Deposit Development and Stopes – Looking North



Source: JDS (2019)

Figure 16-7: AG Zone Deposit Development and Stopes – Looking South West



Source: JDS (2019)

16.3.3 Vertical Development

Vertical development will include a production pass and ventilation raise system for the Palmer deposit and the AG Zone deposit.

The production passes and ventilation raises will be drop raises. A grizzly will be installed at the top of the production pass to remove oversize material. In the Palmer deposit, every other level connects to the production pass which feeds the jaw crusher. Crushed material is then transported to the mill via the 510 Conveyor Drift.

In the AG Zone deposit, the production pass will be fed directly from drives located off the footwall. At the bottom of the production pass, material will be loaded onto 30 t trucks and hauled to the production pass in the Palmer deposit.

Fresh air will intake via the 680 Exploration Portal. In the Palmer deposit, the vent raise system will exhaust to the adit located at 1230 Level. The secondary egress to the mine will be the 510 Conveyor Drift.

For the AG Zone deposit, the vent raise system will exhaust to the 1000 Level Ramp. All ventilation raises will be equipped with ladders.

A summary of all vertical development is shown in Table 16-4.

Table 16-4: Vertical Development Summary

Type	Size	Total (m)
Ventilation Raises	5 m x 5 m	992
Production Pass Raises	5 m x 5 m	973
Total		**1,965**

Source: JDS (2019)

16.4 Mining Method Selection

Given the outcropping nature and deep vertical extension of the Palmer deposits, several zones were reviewed for both open pit and underground potential. Open pit methods were discarded due to the high probability of avalanche occurrences, the large disturbance area and high strip ratios.

16.4.1 Underground Mining Methods

Longhole stoping was selected for both zones of the Palmer deposit as the principal mining method. In the planned LH stopes, a top and bottom drift delineates the stope and blast holes are drilled between the two sub-levels via a LH drilling machine. The blast holes are loaded with explosives and the stope is blasted. The blasted material is extracted from the bottom drift by conventional and remote mucking with LHDs.

Several variations of LH stoping will be applied at Palmer, specific to the geotechnical and geometric properties of the resource zones.

16.4.1.1 Transverse Longhole

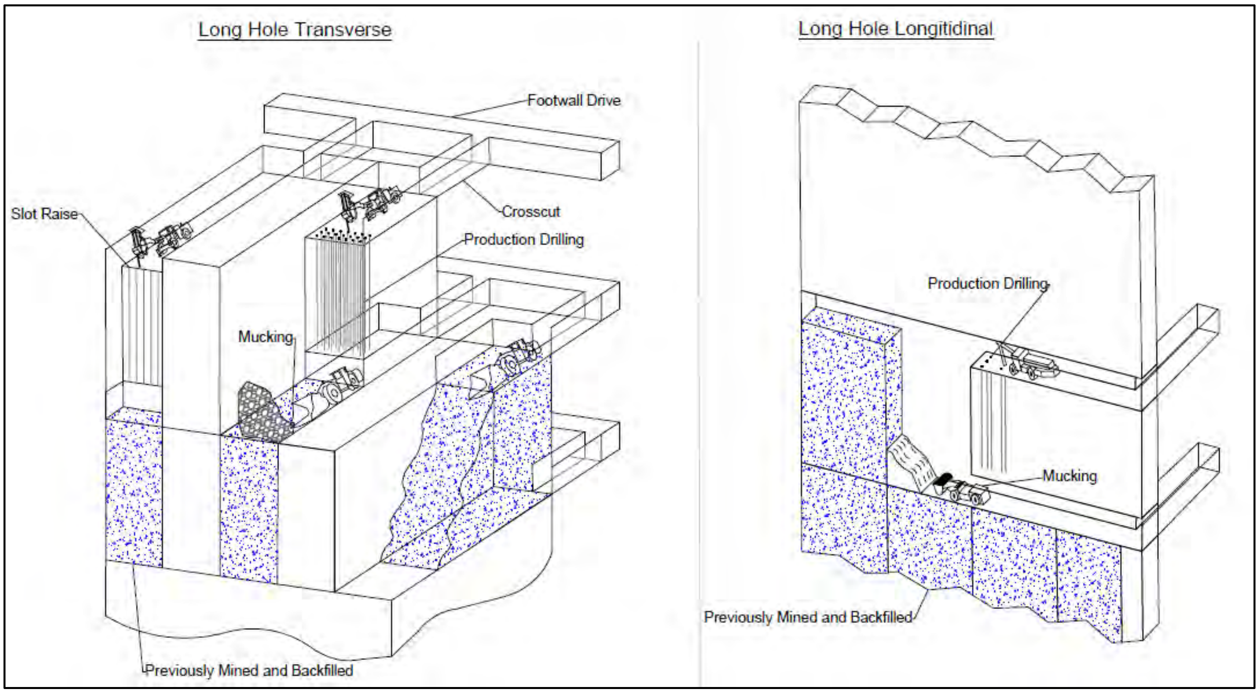
Where ground conditions are good and the resource average thickness is greater than 8 m but less than 20 m) as is the case with Zone 1 and 2-3 from the Palmer deposit and the SW zone of AG, transverse LH stoping will be used. Transverse LH stopes are mined in a primary / secondary fashion whereby primary stopes are mined and filled with a cemented paste backfill. Transverse LH stoping requires a footwall access to be driven parallel to the resource to provide cross-cut entries evenly spaced along strike. As such, this method requires more non-mineralized development than longitudinal LH stoping but offers higher productivity and selectivity given the ability to mine several high grade stopes at once.

16.4.1.2 Longitudinal Longhole

Where ground conditions are less favourable and the resource width is less than 8 m, longitudinal LH retreat (LHR) will be used. LHR mining is a variation on sub-level LH stoping where the long axis of the stope is along strike of the mineralized zone. LHR stopes are mined, retreating from the outside-in towards the level access. As LHR stopes are mined, they are simultaneously backfilled. Disadvantages to LHR mining include higher backfill dilution, lower mineral recovery, and limited selectivity and productivity. Advantages include less non-mineralized development.

The LH mining methods are illustrated in Figure 16-8.

Figure 16-8: Longhole Open Stoping Mine Method



Source: JDS (2019)

16.5 Mine Design Parameters

16.5.1 Dilution and Mining Recovery

Unplanned dilution was estimated for stope hanging walls and footwalls using the equivalent linear overbreak / slough (ELOS) method. The rock quality at Palmer is variable between mineralized zones and over break parameters were developed for each zone. Each stope received unique dilution parameters depending on the geotechnical conditions and orientation. Three types of dilution were evaluated for each stope:

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

Hanging wall and footwall dilution grades were assigned to all applicable stopes. Average dilution grades were derived for each stope from the resource block model. An ELOS value was developed for each zone to determine the amount of over break of the HW and FW. Backfill dilution was modelled based on stope length and an assumption of 0.5 m dilution from all backfill wall planes. Floor backfill was estimated at 0.3 m.

Dilution parameters are listed in Table 16-5 and Table 16-6.

Table 16-5: Estimated HW and FW Dilution by Zone

Mining Zone	Hanging Wall	Footwall
	ELOS (m)	ELOS (m)
Lower Z23 – Palmer Deposit	0.75	0.75
Upper Z23 and Z01 – Palmer Deposit	0.50	0.50
AG Zone Deposit	0.50	0.50

Source: JDS (2019)

Table 16-6: Dilution by Mining Method

Mine Method	Average	Z23	Z1	AGZ
Transverse LH	8%	9%	7%	10%
Longitudinal LH	21%	19%	10%	27%
Weighted Average Total	**12%**	**13%**	**7%**	**15%**

Source: JDS (2019)

Dilution grades by zone from the resource block model are shown in Table 16-7, inclusive of all mineral and waste contained within the mined dilution tonnes from the HW/FW.

Table 16-7: Mine Dilution Grades

Dilution Grade	Z23	Z1	AGZ
Cu (%)	0.10	0.15	0.01
Zn (%)	0.44	1.32	0.07
Ba (%)	0.71	0.38	0.98
Au (g/t)	0.02	0.03	0.04
Ag (g/t)	2.32	2.14	4.88

Source: JDS (2019)

A mining recovery of 95% was assumed for all stopes.

16.5.2 NSR Cut-off Value Criteria

NSR cut-off value (COV) calculation criteria are summarized by zone in Table 16-8.

Table 16-8: Cut-off Value Parameters

Parameter	Units	Palmer Deposit	AG Zone
Metal Price			
Cu	$USD/lb	$3.00	$3.00
Zn	$USD/lb	$1.15	$1.15
Au	$USD/oz	$1,250.00	$1,250.00
Ag	$USD/oz	$16.00	$16.00
Exchange Rate	$USD/$CAD	1.30	1.30
Cu Concentrate			
Cu Recovery	% Cu	90%	80%
Au Recovery	% Au	62%	62%
Ag Recovery	% Ag	74%	74%
Cu Concentrate Grade	%	26%	26%
Moisture Content	%	8%	12%
Cu Payable	% Payable	97%	97%
Au Payable	% Payable	91%	91%
Ag Payable	% Payable	90%	90%
Cu Treatment Charge	$USD/dmt	$80.00	$80.00
Transport Costs	$USD/dmt	$68.18	$68.18
Cu Refining Charge	$USD/Cu lbs	$0.08	$0.08
Au Refining Charge	$USD/Au oz	$6.50	$6.50
Ag Refining Charge	$USD/Ag oz	$0.75	$0.75
Total Offsite Costs	$USD/dmt	$191.59	$191.59
Royalties	%NSR	0%	0%
Zn Concentrate			
Zn Recovery	% Zn	85%	80%
Au Recovery	% Au	14%	14%
Ag Recovery	% Ag	16%	16%
Zn Concentrate Grade	%	59%	59%
Moisture Content	%	12%	12%
Zn Payable	% Payable	85%	85%
Au Payable	% Payable	70%	70%
Ag Payable	% Payable	70%	70%
Zn Treatment Charge	$USD/dmt	$225.00	$225.00
Transport Costs	$USD/dmt	$68.18	$68.18
Au Refining Charge	$USD/Au oz	$6.50	$6.50
Ag Refining Charge	$USD/Ag oz	$0.75	$0.75
Total Offsite Costs	$USD/dmt	$293.18	$293.18
Royalties	%NSR	0%	0%
Underground Cost Estimate			
Opex - Mining (inclusive of Sustaining Capital)	$USD/tonne	$38.80	$38.80
Opex - Processing	$USD/tonne	$27.10	$27.10
Opex - G&A	$USD/tonne	$14.20	$14.20
Opex - Total	$USD/tonne	$80.10	$80.10
Mine Dilution	%	15%	15%
Mine Recovery	%	95%	95%
Underground Cut-Off			
Cut-Off - NSR	$USD/tonne	$80.10	$80.10

Source: JDS (2019)

16.5.3 Mine Plan

To determine the mine plan tonnes at the Palmer deposit, 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 COV based on expected operating cost, mining recovery, mining dilution, and commodity price assumptions
- Identify the blocks in the model that are above COV, 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 COV, 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.

The mine plan for the Palmer deposit is a product of multiple runs of Vulcan Underground Stope Optimizer© software.

The total resource contained in the mine is summarized in Table 16-9. These results are based upon preliminary mineable stope designs and incorporate the factors for recovery and dilution noted in Section 16.5.1. This does not constitute a mining reserve, as the mining factors and geometries have been applied to inferred resources which are not considered to be sufficiently proven geologically for reliance in an economic model.

Table 16-9: Mine Plan Resource Classification

Palmer Deposit	Tonnes (kt)	Cu (%)	Zn (%)	Ba (%)	Au (g/t)	Ag (g/t)
Indicated	4,798	1.33	5.02	12.42	0.27	27.25
Inferred	3,553	0.90	3.47	10.56	0.25	20.39
AG Zone Deposit	**Tonnes (kt)**	**Cu (%)**	**Zn (%)**	**Ba (%)**	**Au (g/t)**	**Ag (g/t)**
Indicated	-	-	-	-	-	-
Inferred	4,130	0.11	3.99	16.71	0.46	100.56
Total Mine Plan	12,481	0.80	4.24	13.31	0.33	49.56

Notes:

1. Mineral Resources are estimated at a cut-off of 80.10US$/t NSR for both the Palmer deposit and AG Zone deposit
2. Metal prices used for determining NSR cut-off were: Copper 3.00US$/lb; Zinc 1.15US$/lb; Gold 1,250.00US$/oz; Silver 16.00US$/oz
3. Totals may not add up correctly due to rounding

Source: JDS (2019)

16.6 Material Handling

16.6.1 Mineralized Material

Mucking will be carried out using 10 t LHDs with remote tramming capabilities. Levels with a production pass connection, the mineralized material will be trammed and dumped into the production pass system. Levels that are not connected to the production pass system, will have mineralized material trammed to a re-muck, loaded onto 30 t haul trucks, and hauled and dumped to the nearest production pass. The material will be collected at the loading pocket on the 628 Level, crushed and transported out of the mine via conveyor to the mill for processing.

16.6.2 Waste Material

Development waste rock is classified as one of two types:

- Non-potentially acid generating (NPAG) and
- Potentially acid generation (PAG).

Waste rock will be mucked using 10 t LHDs and will be loaded at cross cuts, into 30 t haul trucks.

NPAG waste rock will transported to surface via conveyor. This material will be re-handled by the surface fleet, primarily to construct the TSF, avalanche berms and other earthworks.

PAG waste rock will be dump in empty stopes by an LHD were it will mix with the paste backfill. No PAG rock will remain on surface at the end of mine life.

16.7 Backfill

16.7.1 Paste Backfill

Paste backfill has been chosen as the primary backfill method. Paste will be distributed throughout the mine using overhead steel piping with thicknesses and strengths matched to pressure requirements. The paste plant has been sized for a maximum batch production rate of 1,000 m^3/d.

Backfill will consist of cemented paste and run of mine PAG. Primary stopes will be filled with self-standing cemented paste backfill composed primarily of pyrite tailings. Secondary stopes will be filled with a cemented paste composed of pyrite and de-sulfide tailings. PAG material from development will be disposed in any available void. All pyrite tailings and PAG waste will be consumed placed as backfill. De-sulfide tails will be used to supplement any shortages of pyrite tails and PAG material.

For primary stopes, a cement binder content of 4% for pyrite tailings and 3% for de-sulfide tailings was assumed. For secondary stopes, a cement binder of 1% was assumed for both pyrite and de-sulfide tailings. No cement will be added to PAG run of mine material placed as backfill. To contain the paste within the stope void, engineered bulkheads will be constructed this will be required for each stope sill drift.

Paste backfill will be produced on surface at a batch plant located adjacent to the mill. The slurry will be pumped via 6-inch pipe. Booster pumps will be located underground to distribute the paste to the desired void.

16.8 Mine Services

16.8.1 Ventilation

Airflow requirements were estimated based on Mine Safety and Health Administration (MSHA) diesel ventilation regulations. Where the engine model could not be sourced in the regulations, Canadian diesel ventilation regulations was used. The ventilation requirement was then multiplied by the overall equipment utilization and the estimated diesel engine utilization.

Ventilation for the Mine will be managed through a series of raises and one fresh air intake located at the 680 Exploration Portal. The Conveyor Drift will act as a secondary egress in the Palmer deposit and the 1000 Level Ramp will act as a secondary egress for the AG Zone deposit. The fan locations and duty points are shown in Table 16-10.

Table 16-10: Fan Location and Duty

Fan Location	Quantity	Motor Size HP	Air Flow CFM
Main Intake – 680 Exploration Portal	2	250	355
Main Return – 1230L Adit	1	250	225
AG Return – 1000L Ramp	1	250	225

Source: JDS (2019)

The underground workings will receive fresh air from the Exploration incline cross-cut, with 2 x 250 HP fans mounted in parallel. This system will provide approximately 355,000 cubic feet per minute (cfm) of fresh air. The air will travel through the decline and will exhaust out at the 1230 Level adit, 1000 Level Ramp, and the 510 Conveyor Portal. Exhaust fans will be installed at the 1230 Level adit and 1000 Level Ramp, each drawing 175,000 cfm with a 250 HP fans. The ventilation schematic during production is shown in Figure 16-9 and Figure 16-10.

The internal ventilation drop raises will be from level to level, with a square 5.0 m x 5.0 m profile. These will be developed as the mine progresses and will act as the temporary ventilation network until the mine can reach its steady state connection. Regulators will be installed at the entry to the raise to control air flow to the desired level. Refer to Table 16-11 for the mine wide airflow requirements.

Table 16-11: Ventilation Requirements

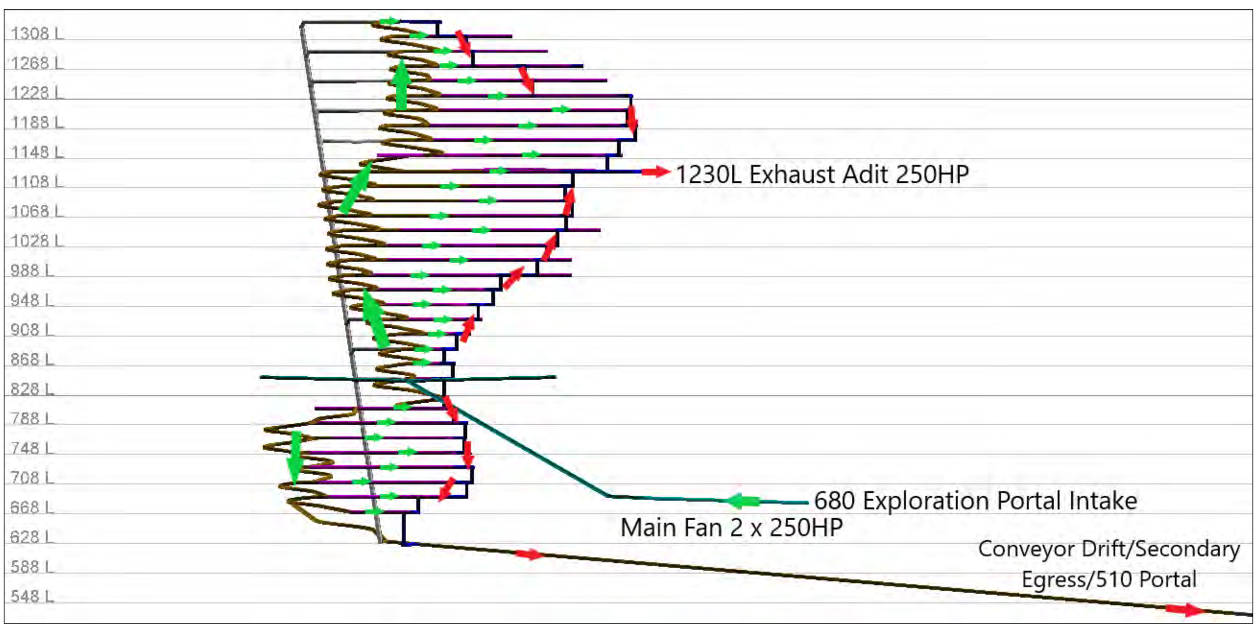
Item	Max Quantity	Engine Power (kW)	Engine Utilization (%)	Total Power (kW)	MSHA Ventilation per unit (CFM)	Total Ventilation (CFM)
LHD (4.5t/2.0m^3)	1	125	80%	100	5,000	4,000
LHD (10t/4.0m^3)	6	220	80%	1,056	9,000	43,200
Truck (30t/14.5 m^3)	9	289	80%	2,083	13,000	93,600
Jumbo - 2 Boom	4	110	15%	66	5,500	3,300
Bolter	2	110	15%	33	4,500	1,350
Longhole Drill	4	96	15%	58	5,000	3,000
Large Explosives Truck	2	129	15%	39	7,500	2,250

Item	Max Quantity	Engine Power (kW)	Engine Utilization (%)	Total Power (kW)	MSHA Ventilation per unit (CFM)	Total Ventilation (CFM)
Scissor Lift	2	129	15%	39	7,500	2,250
Shotcrete + Transmixer	1	129	15%	19	5,000	750
Fuel/Lube Truck	1	129	15%	19	7,500	1,125
Grader	1	292	15%	44	36,600	5,490
Utility Vehicle	2	16	15%	5	2,700	810
Supervisor Truck	2	118	15%	35	9,500	2,850
Electrician Truck	1	118	15%	18	9,500	1,425
Personnel Carrier	2	129	15%	39	7,500	2,250
Mine Wide Ventilation Required (CFM)						167,650
Misc. Additional Headings (CFM)						40,344
Infrastructure Allowance (CFM)						120,000
Ventilation Leakage @ 20% (CFM)						28,997
Total Mine Ventilation (CFM)						**356,991**

Source: JDS (2019)

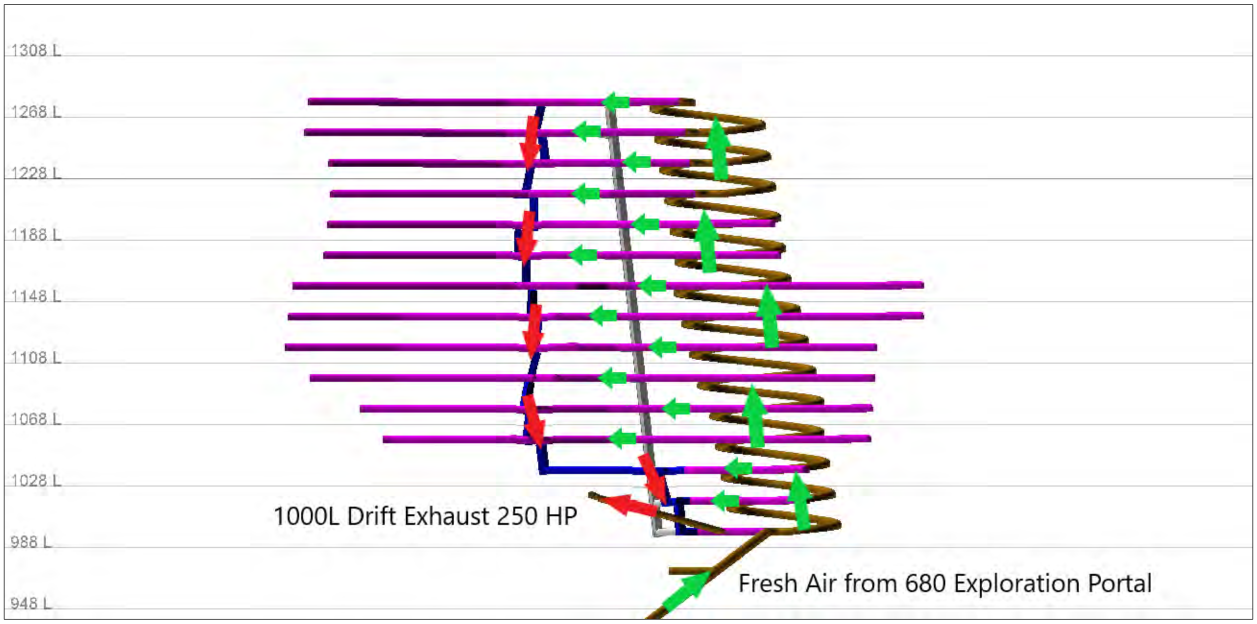
The above ventilation requirements are the maximum required between the Palmer deposit and the AG Zone deposit. First principles ventilation calculations were used to estimate power requirements for the ventilation network. Fans were planned as multiples in parallel rather than a single large fan to aid in future fan servicing and replacement.

Figure 16-9: Palmer Deposit Ventilation Schematic



Source: JDS (2019)

Figure 16-10: AG Zone Deposit Ventilation Schematic



Source: JDS (2019)

16.8.2 Mine Air Heating

Mine air would be heated to a minimum +2.0°C by a direct-fired propane heater located at the fresh air intake off the exploration ramp. 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 during operations. Heating calculations were based on average monthly temperatures collected at the site weather station. It was estimated that an average 51,700 m^3 of propane would be required annually.

16.8.3 Electrical Power

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

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

High-voltage cables would enter the mine via the Exploration and Conveyor 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 480 V at electrical sub-stations. Each working level will include a primary sub-station and power panel off the spiral ramp where power will be further stepped down and distributed to the working faces.

16.8.4 Compressed Air and Service Water Supply

Compressed air will be supplied throughout the mine from a surface compressor. To maintain consistent pressures, underground surge tanks will be positioned in ancillary bays at strategic positions throughout the mine.

Service water for drilling and dust control would be sourced from a sump and distributed in 50 mm diameter steel piping.

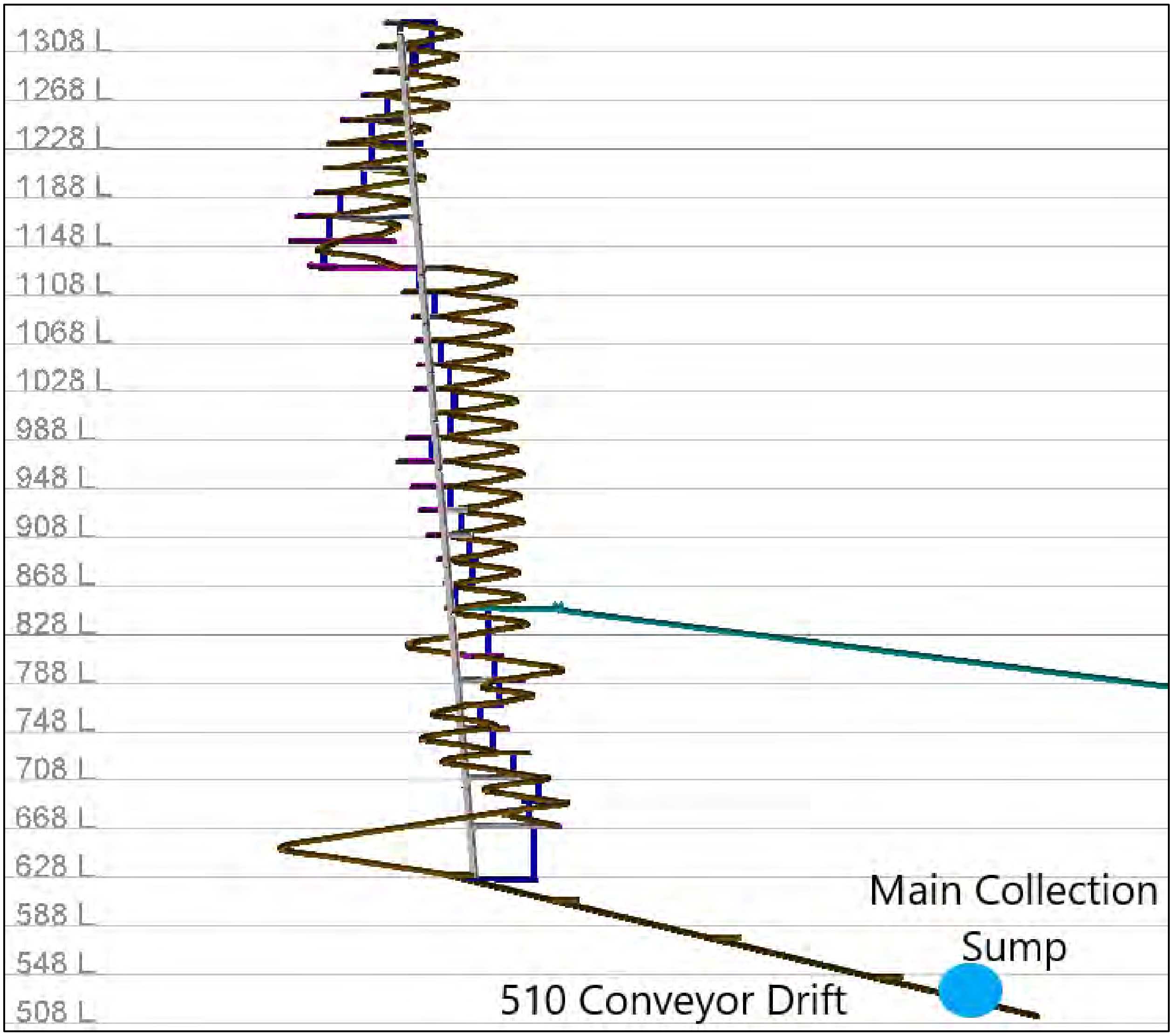
All water used for underground operations will be drawn from collected mine inflow after solids settlement and filtration and re-distributed to the working faces through 152 mm diameter overhead water lines.

16.8.5 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 from the exploration adit and a similar amount from the mining operations at Palmer and AG deposits (estimated at 30 L/s). Small sumps have been strategically designed on each level for both the Palmer deposit and the AG Zone deposit; these will be gravity fed via boreholes. A main water collection sump would be installed at the bottom of the mine near the 510 Conveyor portal. Water collected in the sump would either return underground for use or be piped to the water treatment plant.

Sumps not connected via bore holes will incorporate a small portable 22 kW submersible pump to facilitate water flow down to the main collection sump. Portable pumps will be used at the working face to handle water from drilling and other mining activities. Other small pumps will be installed in underground infrastructure such as shops and crusher chambers. The location of the main collection sump is shown in Figure 16-11.

Figure 16-11: Palmer Deposit Collection Sump Location



Source: JDS (2019)

16.8.6 Explosives Storage and Handling

Primary explosives storage magazines would be located on surface and transported 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.

A combination of bulk emulsion and ammonium nitrate fuel oil (ANFO) would be used as for mine development and production. Explosives handling, loading, and detonation would be carried out by trained and authorized personnel.

16.8.7 Fuel Storage and Distribution

Major mobile equipment, such as haul trucks, LHDs and auxiliary mobile equipment would be re-fueled on surface at a fuel station from an Enviro-Tank located close to the 680 Exploration Portal. Drilling equipment would be re-fueled underground with a fuel / lube truck.

16.9 Mine Personnel

The mine will require a full-time work force of mining, maintenance, services, technical and administrative personnel. Mine operations will be run 360 days per annum, 22 hours per day (2 x 11-hour shifts), and allowing one hour for smoke clearing between shifts. Mine operations will consist of personnel working two different rosters:

- Two weeks on / two weeks off (2x2): Mine Operations, Maintenance, Construction Labor, Site Services (11-hour shifts)
- Two weeks on / two weeks off (2x2): Mine Technicians, Surveyors, and Grade Control (12 hours shift, day shift only) and
- Four days on / three days off (4x3): Engineering, Administrative, and Management (12-hour shifts, day shift only).

During full production the mine will require 94 people on site, including those on 4x3 rotations, and a total payroll of 184 workers.

Staffing will be ramped up to full production requirements during the first year of operations. Certain production related positions are not expected to be necessary during pre-production mine development and construction.

Similarly, some positions, such as training, have been reduced and/or eliminated during the winding down period of the final years of operations.

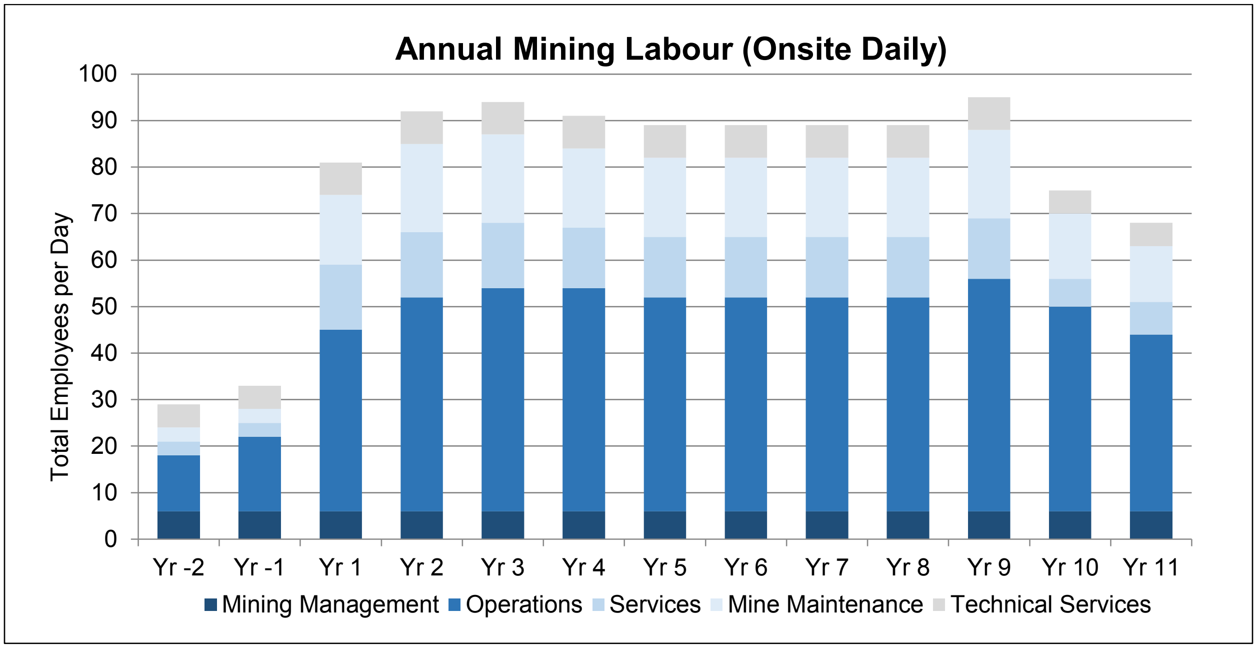
A summary of the daily on-site personnel required in the ramp-up, peak production and final years is presented in Table 16-12. Refer to Figure 16-12 for annual labour requirements.

Table 16-12: On-site Personnel, Mine Operations

On-Site Personnel	Year -2	Peak Production	Year 11
Mining Management	6	6	6
Operations	12	48	38
Services	3	14	7
Mine Maintenance	3	19	12
Technical Services	5	7	5
Grand Total	**29**	**94**	**68**

Source: JDS (2019)

Figure 16-12: Annual Mining Labour



Source: JDS (2019)

16.10 Mine Equipment

16.10.1 Mobile Equipment

Diesel and electric hydraulic equipment will be employed throughout the mine. The primary haulage fleet will consist of 30 t haul trucks and 10 t LHDs for the mineralized material, waste handling, secondary tasks, and backfill. Development drilling will be conducted using two-boom jumbos and longhole drilling will be conducted using Sandvik DL311 or equivalent drills.

Equipment requirements were developed from first principles, based on the maximum annual duty hours for an individual piece of equipment, modified for mechanical availability and projected utilization. A list of the underground production and support equipment and respective factors used in the mine plan are shown in Table 16-13.

Table 16-13: Mine Mobile Equipment Summary

Mobile Equipment	Max # of Units	Mech. Availability (%)	Average LOM Utilization (%)
Truck (30t/14.5m^3)	9	90%	88%
LHD (4.5t/2.0m^3)	1	90%	14%
LHD (10t/4.0m^3)	6	90%	77%
Jumbo - 2 Boom	4	65%	50%
Bolter	2	70%	44%
Longhole Drill	4	70%	68%
Large Explosives Truck	2	80%	50%

Mobile Equipment	Max # of Units	Mech. Availability (%)	Average LOM Utilization (%)
Scissor Lift	2	85%	53%
Shotcrete + Transmixer	1	70%	1%
Jackleg/Stoper	8	90%	1%
Grout Pump	1	70%	20%
Personnel Carrier	2	85%	21%
Fuel/Lube Truck	1	85%	28%
Electrician Truck	1	85%	76%
Grader	1	70%	41%
Utility Vehicle	2	85%	50%
Mechanics Truck	1	85%	34%
Supervisor Truck	2	85%	50%

Source: JDS (2019)

16.11 Mine Plan

16.11.1 Mine Development Schedule

The mine development schedule is based on developing the mine to maintain peak production rates. As LH stoping is a bottom up mining method and the highest-grade stopes are located at the top of the deposit, prioritizing the development for the highest-grade stopes is limited. The schedule has also been designed to provide secondary egress and positive ventilation flow throughout the mine prior to production.

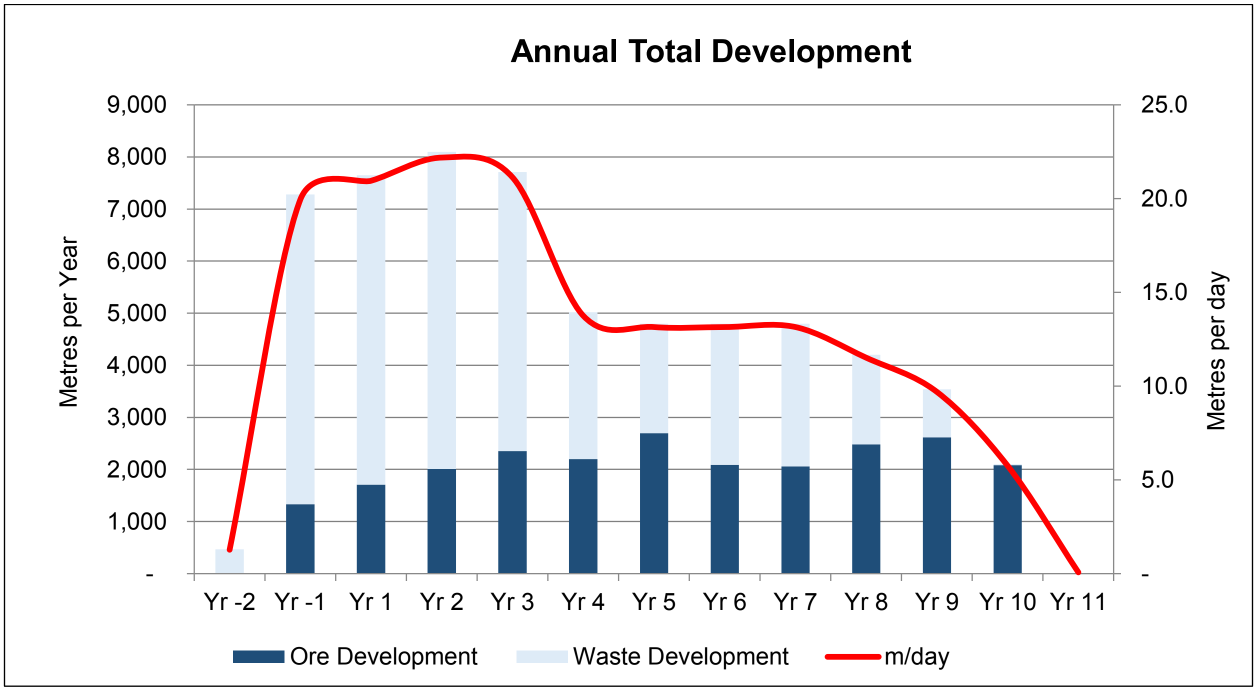
Unit rates for each development and production activity are shown in Table 16-14. The development schedule is summarized in Figure 16-13 and Table 16-15.

Table 16-14: Mining Rates for Scheduling

Heading	Units	Rate
Lateral Development		
Ramp / Access / Footwall / Cross-cut	m/d	5.6
Auxiliary / Sump / Vent Drives / Shop	m/d	5.6
Vertical Development		
Vent Raise	m/d	2.0
Production Pass	m/d	2.0
Alimak Raise	m/d	2.0
Mine Production		
Palmer Deposit Transverse LH Stoping	t/d	500
Palmer Deposit Longitudinal LH Stoping	t/d	400
AG Zone Deposit LH Stoping	t/d	400

Source: JDS (2019)

Figure 16-13: Mine Development Schedule



Source: JDS (2019)

Table 16-15: Underground Development Plan

Underground Development	Units	Total	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11
Ramp Drive	m	14,756	464	3,331	3,359	2,676	1,491	549	424	779	828	694	161	-	-
Footwall Drive	m	10,755	-	1,389	1,113	1,668	1,894	830	1,056	634	1,111	555	508	-	-
Sump	m	992	-	56	84	522	130	25	30	40	10	35	45	15	-
Production Pass Drive	m	1,848	-	235	273	306	112	150	119	134	307	135	77	-	-
Ventilation Drive	m	2,321	-	426	440	297	149	348	90	76	245	182	68	-	-
Maintenance Shop	m	125	-	125	-	-	-	-	-	-	-	-	-	-	-
Long-hole Access	m	5,438	-	390	615	586	1,355	880	378	1,047	125	61	-	-	-
Remuck	m	600	-	-	60	32	228	40	7	-	113	60	60	-	-
Transverse Access	m	23,645	-	1,332	1,705	2,012	2,354	2,198	2,698	2,091	2,060	2,481	2,618	2,084	12
Grand Total Lateral Development	m	60,480	464	7,283	7,648	8,099	7,714	5,020	4,800	4,800	4,800	4,203	3,537	2,099	12
Lateral Advance Rate	m/day	13	1.3	20.0	21.0	22.2	21.1	13.8	13.2	13.2	13.2	11.5	9.7	5.8	0.1
Jumbo productivity	m/mth	382	38	599	629	666	634	413	395	395	395	345	291	173	2
	m/mth/jumbo	149	38	200	157	166	159	138	197	197	197	173	145	173	2
Vent Raise	m	992	-	151	232	82	142	82	40	49	91	61	61	-	-
Production Pass	m	973	-	39	340	135	90	64	41	40	103	60	62	-	-
Grand Total Vertical Development	m	1,965	-	190	572	217	232	146	81	89	194	121	123	-	-

Source: JDS (2019)

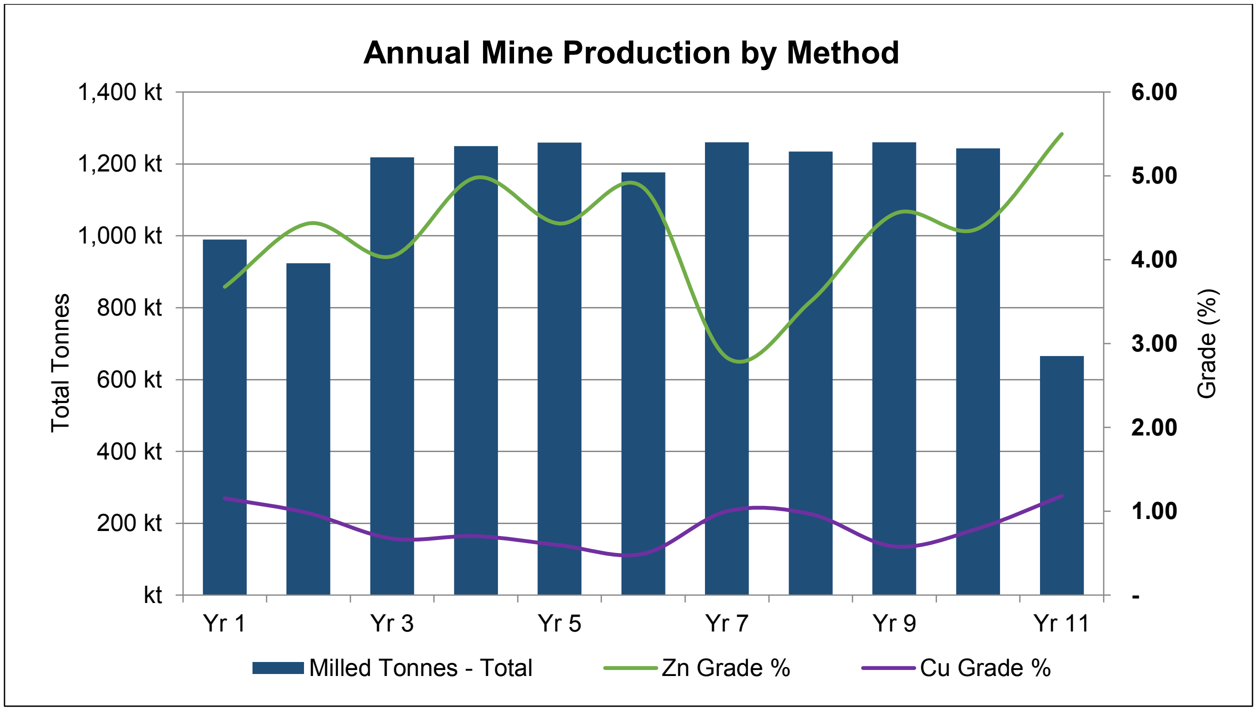
16.11.2 Mine Production Schedule

Mine production is expected to commence in year one, at 990 kt mined, approximately 81% of the steady-state production rate. At steady-state production, the mine is expected produce approximately 1,175 kt per annum for 8 years (Years 3 to 11 with production ending in Q2 of Year 11.

Determining the production rate was a build-up of the various mining activities, including: cycle times for the different mining activities, tonnes per vertical metre, and the layout of the mine. The lower production rates in Year 1 and 2 are a result of no AG Zone deposit feed. Development to the AG Zone deposit doesn't commence until Year 1.

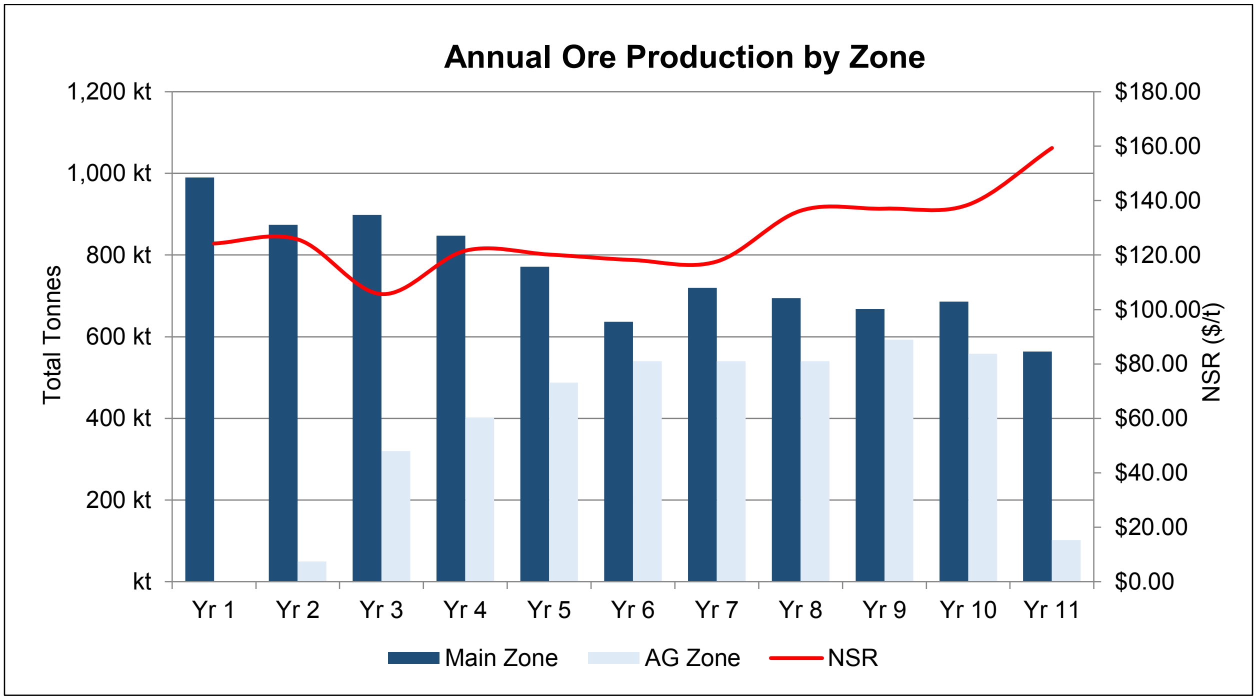
A summary of the mine production schedule is presented in Figure 16-14 to Figure 16-16 and Table 16-16.

Figure 16-14: Mine Production Schedule, with Copper and Zinc Grades



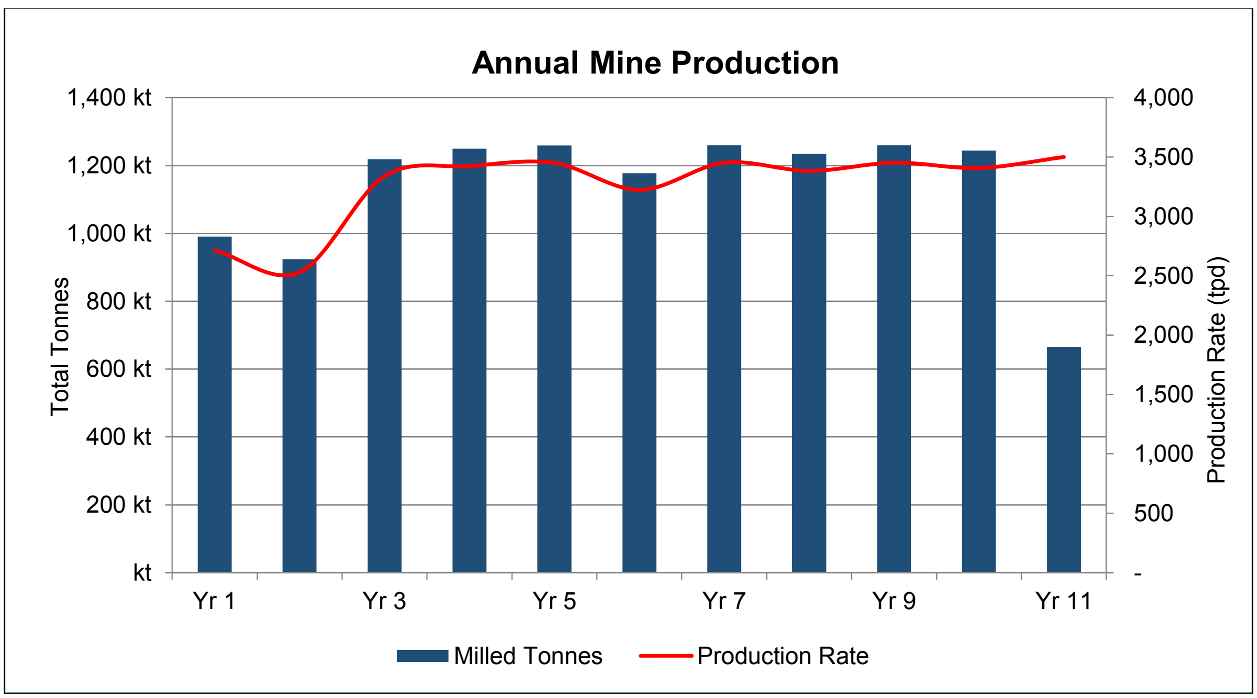
Source: JDS (2019)

Figure 16-15: Mine Production by Zone and NSR



Source: JDS (2019)

Figure 16-16: Annual Production and Daily Production Rate



Source: JDS (2019)

Table 16-16: Mine Production Plan

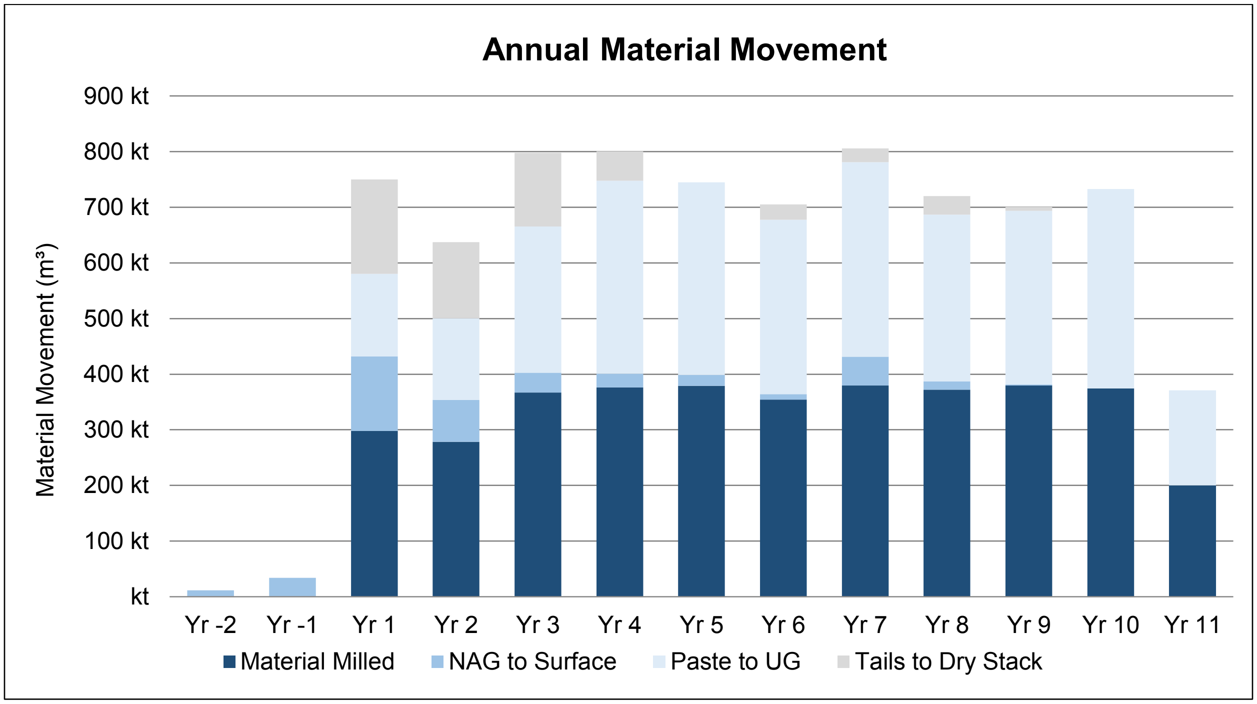
Mine Production	Units	Total	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11
Mineral	kt	12,481	-	-	990	924	1,219	1,249	1,259	1,177	1,260	1,235	1,260	1,244	665
Waste	kt	2,695	33	452	474	418	338	169	155	152	208	164	131	3	-
Cu	%	0.81	-	-	1.16	0.98	0.68	0.71	0.60	0.49	1.00	0.97	0.58	0.80	1.18
Zn	%	4.24	-	-	3.68	4.43	4.04	4.98	4.43	4.86	2.83	3.50	4.55	4.37	5.50
Ba	%	13.31	-	-	9.08	10.68	9.55	9.27	15.69	13.73	13.41	16.45	18.51	13.76	15.81
Au	g/t	0.33	-	-	0.28	0.27	0.19	0.19	0.25	0.25	0.27	0.44	0.57	0.50	0.40
Ag	g/t	49.56	-	-	18.46	18.58	28.15	32.88	59.25	53.72	57.30	74.18	80.40	66.54	33.15

Source: JDS (2019)

16.11.3 Material Movement Schedule

Material movement is based on the mining sequencing of both development waste and production tonnages. The schedule includes material to the mill, development NPAG waste to surface, and backfill to underground voids. The annual material movement schedule is summarized in Figure 16-17.

Figure 16-17: Annual Material Movement



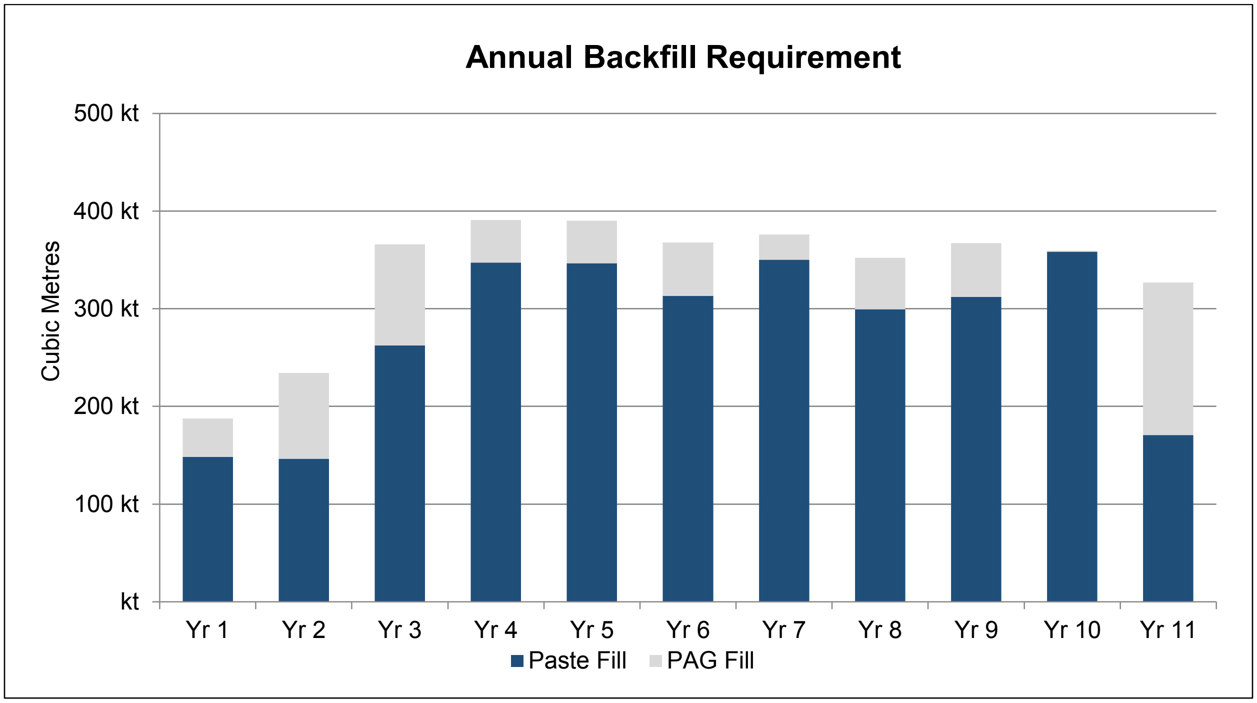
Source: JDS (2019)

16.11.4 Backfill Schedule

Backfill will consist of cemented paste and run of mine PAG material. Primary stope will be filled a self-standing cemented backfill paste composed pyrite tailings. Secondary stopes will be filled with a cemented backfill composed of a combination of a pyrite and de-sulfide tailings. Run of mine PAG will be disposed in any available open void. All pyrite tailings and PAG run of mine will be consumed within the backfill. De-sulfide tailings will be used as backfill to supplement any shortages of pyrite tailing and PAG run of mine.

Figure 16-18 outlines mine backfill placement.

Figure 16-18: Annual Backfill Schedule



Source: JDS (2019)

17 Recovery Methods

The Palmer deposit will process Cu, Zn and Ba concentrates. The recent metallurgical test program on Palmer deposit 14063-002 completed at SGS in Burnaby, BC, summarized in Section 13, demonstrates that standard Cu, Zn, Py and Ba sequential flotation can produce saleable concentrates. 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 (underground)
- Semi-Autogenous Grinding (SAG) mill operating in open circuit
- Ball mill grinding in reverse closed-circuit with cyclones
- Sequential Cu, Zn, and Ba flotation circuits, each incorporating conventional flotation, regrind for Cu and Zn and three cleaning stages
- Py rougher flotation, rougher concentrate dewatering and filtration for paste mixing
- Concentrate dewatering and filtration
- Concentrate storage and load-out facilities, and
- Tailings dewatering and filtration for deposition in the filtered tailings facility or paste for underground.

The material from underground from the Palmer deposit at average LOM head grades of 0.81% Cu, 4.24% Zn, 13.31% Ba, 0.33 g/t Au and 49.56 g/t Ag will provide a total throughput of 3,500 t/d to the process plant for the first few years of the mine life. The same flowsheet will be used for the AG Zone deposit expected to be mined at the end of Palmer deposit. The process plant will operate 24 hours per day, 365 days per year at an availability of 92%.

Primary crushing 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 72 µm, before Cu, Zn and Ba are recovered into concentrates using sequential flotation. Py concentrate, designated as paste, and Ba rougher and Ba 1st cleaner tailings, designated as final tailings, will be pumped to the paste plant for dewatering and filtration.

The process plant will consist of grinding as well as Cu, Zn, Py and Ba flotation circuits. The Cu, Zn and Ba will include rougher and cleaner flotation. The Py circuit will produce a rougher concentrate that will be mixed with cement to produce paste material for deposition underground. The three saleable concentrates will be dewatered in concentrate thickeners and pressure filters to produce a target moisture content of 8% and the Cu and Zn will be loaded into bulk concentrate trucks. The Ba concentrate will be dried to a target 1% moisture, bagged and trucked to Haines for barging to Prince Rupert, BC.

17.1 Introduction

The processing facilities will consist of the following unit operations:

- Jaw Crusher – A vibrating grizzly feeder and jaw crusher in open circuit, producing an estimated final product P_{80} of 110 mm
- Primary Grinding – A SAG mill in open circuit, producing a T_{80} transfer size of approximately 800 µm
- Secondary Grinding – A ball mill in reverse closed circuit with a cluster of hydrocyclones, producing a final target product size P_{80} of 72 µm
- Cu Flotation – Rougher and cleaner flotation to produce a saleable Cu concentrate
- Cu Rougher Concentrate Regrind – A ball regrind mill in closed circuit, reducing Cu rougher concentrate to a P_{80} of 35 µm
- Cu Concentrate Dewatering – A 6 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 ball regrind mill in closed circuit, reducing Zn rougher concentrate to a P_{80} of 50 µm
- Zn Concentrate Dewatering – An 11 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%
- Py Flotation – Rougher flotation to produce a concentrate that will be sent to a 11 m diameter thickener followed by filtration and then paste mixing
- Ba Flotation – Rougher and cleaner flotation to produce a saleable Ba concentrate
- Ba Concentrate Dewatering – A 14 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 dewatering and filtering that will be trucked to the filtered tailings facility or mixed with Py concentrate as paste for deposition underground.

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 based on Palmer deposit 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	**3,500**	2018 mine plan
Process Plant Availability	%	**92**	Industry Standard
Process Plant Throughput	t/h	**159**	Engineering Calculation
LOM Average Cu Head Grade	%	**0.81**	2019 mine plan
LOM Average Zn Head Grade	%	**4.24**	2019 mine plan
LOM Average Ba Head Grade	%	**13.3**	2019 mine plan
LOM Average Au Head Grade	g/t	**0.33**	2019 mine plan
LOM Average Ag Head Grade	g/t	**49.6**	2019 mine plan
Overall Cu Recovery – Main/AG	%	**88.9/78.9**	SGS (2018): 14063-002 LCT1
Cu Concentrate Grade	% Cu	**24.5**	SGS (2018): 14063-002 LCT1
Au Recovery	%Au	**49.5**	SGS (2018): 14063-002 LCT1
Ag Recovery	%Ag	**70.8**	SGS (2018): 14063-002 LCT1
Overall Zn Recovery – Main/AG	%	**93.1/88.1**	SGS (2018): 14063-002 LCT1
Zn Concentrate Grade	% Zn	**61.3**	SGS (2018): 14063-002 LCT1
Au Recovery	%Au	**20.1**	SGS (2018): 14063-002 LCT1
Ag Recovery	%Ag	**20.1**	SGS (2018): 14063-002 LCT1
Overall Ba Recovery	%	**91.1**	SGS (2018): 14063-002 LCT1-CF08
Ba Concentrate Grade	% Ba	**52.3**	SGS (2018): 14063-002 LCT1-CF08
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	**2,500**	Design Consideration
Stockpile Capacity (live)	h	**24**	Engineering Calculation
Grinding			
SMC – Comps 1 and 3A	Mia – kWh/t	**9.7**	SGS (2018): 14063-003
	Axb	**89.6**	SGS (2018): 14063-003
Bond Ball Mill Work Index	kWh/t	**6.9**	SGS (2018): 14063-003
Primary Grinding Mill Type	-	**SAG Mill**	Industry Standard for primary grinding to target transfer size
Mill Diameter	m	**4.9**	Vendor Recommended
Mill Length	m	**2.7**	Vendor Recommended
Installed Power	kW	**895**	Vendor Recommended

Criteria	Unit	Nominal Value	Source
Circuit Configuration	-	**Open**	Design Consideration
Primary Grinding Transfer Size (T_{80})	µm	**800**	Design Consideration
Secondary Grinding Mill Type	-	**Ball Mill**	Selected to achieve target product size
Mill Diameter	m	**3.4**	Vendor Recommended
Mill Length	m	**5.5**	Vendor Recommended
Installed Power	kW	**1,119**	Vendor Recommended
Circuit Configuration	-	**Reverse Closed**	Industry Standard
Circulating Load	%	**300**	Industry Standard
Final Product Target Size (P_{80})	µm	**72**	SGS (2018): 14063-002
Flotation			
Rougher Flotation Time Scale-up	-	**2.0**	Industry Standard
Cleaner Flotation Time Scale-up	-	**4.0**	Industry Standard
Cu Rougher Flotation			
Laboratory Retention Time	min	**9**	SGS (2018): 14063-002 LCT1
Design Retention Time	min	**18**	Engineering Calculation based on 2.0x scale-up factor
Number of Rougher Flotation Cells	#	**5**	Design
Rougher Flotation Cell Size	m^3	**30**	Designed to achieve retention time
Cu Regrind Circuit			
Rougher Concentrate Mass Pull	%	**24**	SGS (2018): 14063-002 LCT1
Regrind Mill Type	-	**Ball Mill**	Industry Standard
Final Product Target Size (P_{80})	µm	**35**	SGS (2018): 14063-002 LCT1
Cu 3-Stage Cleaner Flotation			
Laboratory Retention Time	min	**8 / 4 / 5**	SGS (2018): 14063-002 LCT1
Design Retention Time	min	**32 / 16 / 20**	Engineering Calculation based on 4.0x scale-up factor
Number of Cleaner Flotation Cells	#	**7 / 4 / 3**	Designed to achieve retention time
Cleaner Flotation Cell Sizes	m^3	**10 / 5 / 5**	Designed to achieve retention time
Zn Rougher Flotation			
Laboratory Retention Time	min	**7**	SGS (2018): 14063-002 LCT1
Design Retention Time	min	**14**	Engineering Calculation based on 2.0x scale-up factor
Number of Rougher Flotation Cells	#	**5**	Design
Rougher Flotation Cell Size	m^3	**20**	Designed to achieve retention time
Zn Regrind Circuit			
Rougher Concentrate Mass Pull	%	**25**	SGS (2018): 14063-002 LCT1
Regrind Mill Type	-	**Ball Mill**	Industry Standard
Final Product Size (P_{80})	µm	**50**	SGS (2018): 14063-002 LCT1

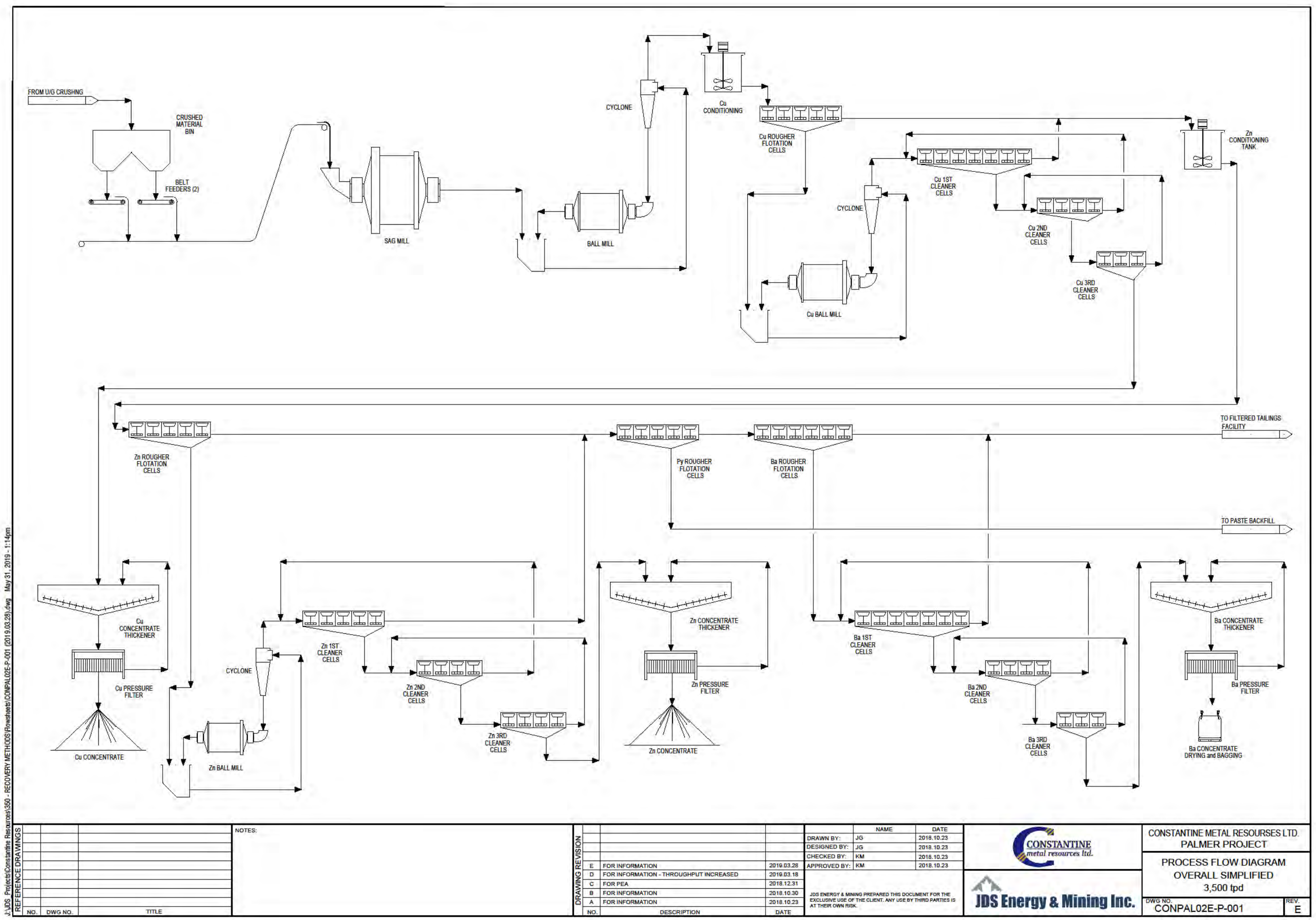
Criteria	Unit	Nominal Value	Source
Zn 3-Stage Cleaner Flotation			
Laboratory Retention Time	min	**7 / 4 / 4**	Base Met (2018): BL0236 LCT-45
Design Retention Time	min	**28 / 16 / 16**	Engineering Calculation based on 4.0x scale-up factor
Number of Cleaner Flotation Cells	#	**5 / 4 / 4**	Designed to achieve retention time
Cleaner Flotation Cell Size	m^3	**20 / 10 / 5**	Designed to achieve retention time
Py Rougher Flotation			
Laboratory Retention Time	min	**9**	SGS (2018): 14063-002 LCT1-CF08
Design Retention Time	min	**18**	Engineering Calculation based on 2.0x scale-up factor
Number of Rougher Flotation Cells	#	**5**	Design
Rougher Flotation Cell Size	m^3	**30**	Designed to achieve retention time
Ba Rougher Flotation			
Laboratory Retention Time	min	**5**	SGS (2018): 14063-002 LCT1-CF08
Design Retention Time	min	**10**	Engineering Calculation based on 2.0x scale-up factor
Number of Rougher Flotation Cells	#	**6**	Design
Rougher Flotation Cell Size	m^3	**10**	Designed to achieve retention time
Ba 3-Stage Cleaner Flotation			
Laboratory Retention Time	min	**7 / 4 / 4**	SGS (2018): 14063-002 LCT1-CF08
Design Retention Time	min	**28 / 16 / 16**	Engineering Calculation based on 4.0x scale-up factor
Number of Cleaner Flotation Cells	#	**7 / 4 / 4**	Designed to achieve retention time
Cleaner Flotation Cell Size	m^3	**10 / 10 / 10**	Designed to achieve retention time
Concentrate Dewatering			
Thickener Type	-	**High Rate**	Industry Standard
Cu Thickener Loading Rate	$t/h/m^2$	**0.26**	Design Consideration
Cu Thickener Diameter	m	**6**	Vendor Recommended
Zn Thickener Loading Rate	$t/h/m^2$	**0.27**	Design Consideration
Zn Thickener Diameter	m	**11**	Vendor Recommended
Ba Thickener Loading Rate	$t/h/m^2$	**0.27**	Design Consideration
Ba Thickener Diameter	m	**14**	Vendor Recommended
Filtration Type	-	**Pressure**	Industry Standard
Final Cu, Zn and Ba Concentrate Moisture Content (Ba will be dried to 1% and bagged for shipment)	%	**8**	Design Consideration

Source: JDS (2019)

17.3 Plant Description

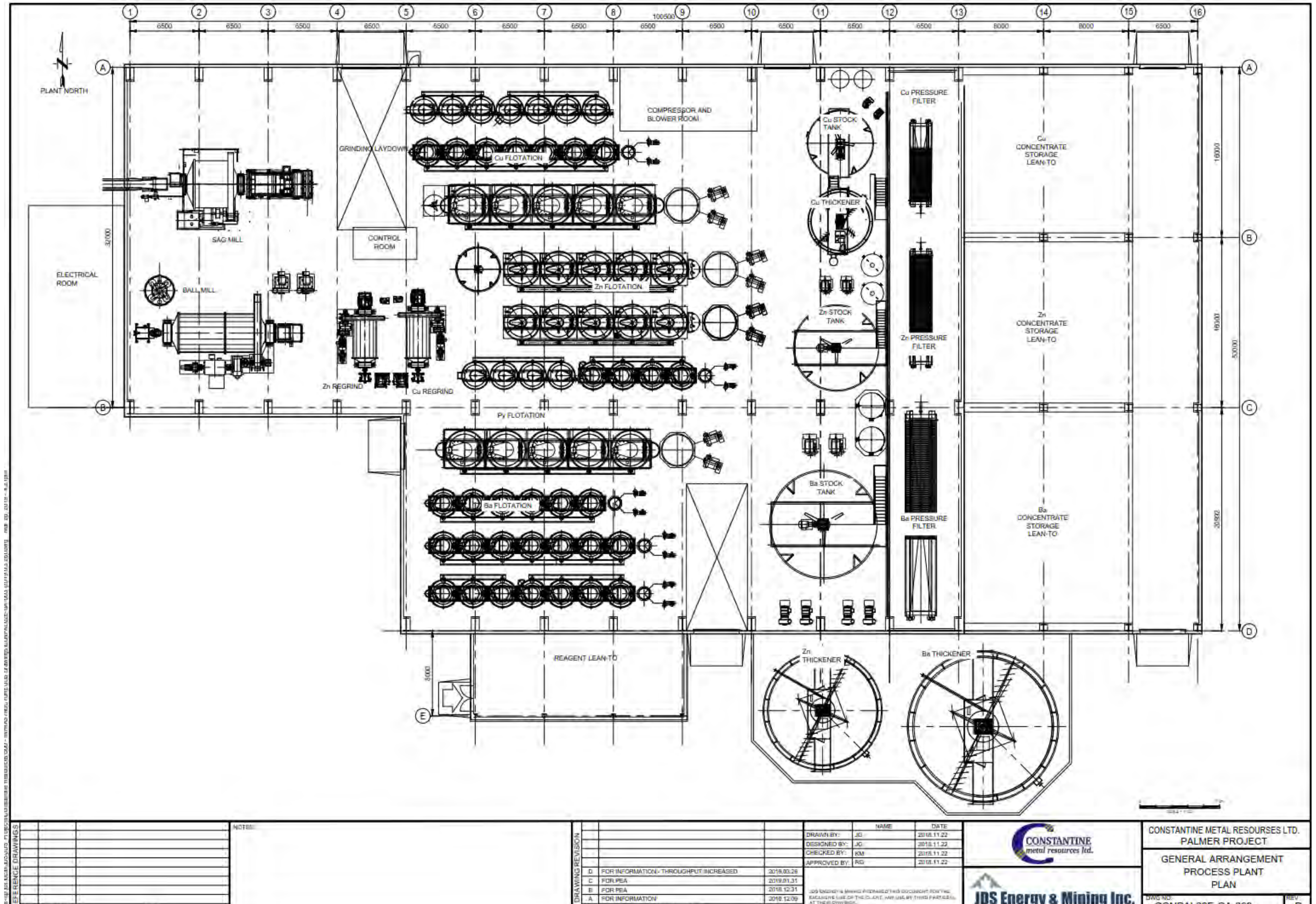
The process flowsheet and plant layout are shown in Figure 17-1 and Figure 17-2.

Figure 17-1: Overall Process Flowsheet



Source: JDS (2019)

Figure 17-2: Process Plant Layout



Source: JDS (2019)

17.4 Process Plant Description

17.4.1 Crushing

The crushing plant will be located underground and will include a 30” x 42” jaw crusher with an installed power of 110 kW. The jaw crusher will process 194 t/h and with a closed side setting (CSS) of 125 mm will produce a final product P_{80} of approximately 110 mm.

17.4.2 Grinding

The grinding circuit will consist of a 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 has been designed to process a nominal throughput of 159 t/h (fresh feed) and produce a final product P_{80} of 72 µm.

Product from the crushing circuit will be conveyed from a storage bin to a 4.9 m diameter by 2.7 m long SAG mill with an installed power of 895 kW motor. A belt-scale on the feed conveyor will monitor the 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 at a transfer size (T_{80}) of approximately 800 µm will flow into the cyclone feed pump box.

Product from the SAG mill screen discharge will combine with the ball mill discharge before being pumped up to a cluster of hydrocyclones for size classification. The coarse underflow will flow by gravity to the secondary ball mill, 3.4 m diameter by 5.5 m long ball mill with an installed power of 1,119 kW, for additional grinding. The fine cyclone overflow, at a final product P_{80} of 72 µm, will report to the Cu Conditioning Tank. The hydrocyclones have been designed for a 300% circulating load.

17.4.3 Flotation

17.4.3.1 Copper Circuit

Cyclone overflow will flow by gravity to a Cu conditioning tank, which will provide 2 minutes of conditioning time prior to Cu flotation. Frother methyl isobutyl carbinol (MIBC), sulfide collector sodium xanthate (PAX), Zn depressant sodium cyanide (NaCN), pH modifier soda ash and zinc sulfate ($ZnSO_4$) will be added to the copper circuit. The slurry will then gravitate to the rougher flotation circuit, which consists of five 30 m^3 flotation tanks cells operating in series.

Cu rougher concentrate will be collected in a common launder and pumped to the regrind circuit. The rougher concentrate will feed the regrind mill with the cyclone underflow at a density of 65% solids. The cyclone overflow will have a target grind size of P_{80} of 35 µm and report to the first cleaner flotation circuit.

Regrind product and the Cu second cleaner tailings will feed seven 10 m^3 Cu first cleaner tank cells. The Cu first cleaner concentrate will be collected in a common launder and fed to the Cu second cleaner flotation circuit. The Cu first cleaner tailings will combine with the Cu rougher tailings and be pumped to the Zn conditioning tank.

The Cu first cleaner concentrate will combine with the Cu third cleaner tailings and flow into the first of four 5 m^3 Cu second cleaner flotation tank cells. The Cu second cleaner concentrate will be collected in a common launder and pumped to the third cleaner flotation cells, while the Cu second cleaner tailings will flow back to the Cu first cleaner flotation feed box.

The Cu second cleaner concentrate will flow into the first of three 5 m^3 Cu third cleaner flotation tank cells. The Cu third cleaner concentrate will be collected in a common launder and pumped to the Cu concentrate thickener, while the Cu third cleaner tailings will flow back to the Cu second cleaner flotation feed box.

Cu concentrate from the third cleaners will report to a 6 m diameter Cu thickener. The thickener overflow will be sent to the process water tank. Thickened Cu concentrate will be pumped to an 8-hour stock tank that feeds a pressure filter for further dewatering. Cu final concentrate, at approximately 8% moisture, will be loaded into trucks for transportation to the port for shipment to overseas smelters.

17.4.3.2 Zinc Circuit

Tailings from the rougher and first cleaner copper flotation circuits will feed a Zn conditioning tank, which will provide 3 minutes of conditioning time prior to Zn flotation. Frother methyl isobutyl carbinol (MIBC), PAX, pH modifier lime and copper sulfate ($CuSO_4$) will be added to the zinc circuit. The slurry will then gravitate to the rougher flotation circuit, which consists of five 20 m^3 flotation tanks cells operating in series.

Cu rougher concentrate will be collected in a common launder and pumped to the regrind circuit. The rougher concentrate will feed the regrind mill with the cyclone underflow at a density of 65%. The cyclone overflow will have a target grind size of P_{80} of 50 µm and report to the 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 Py rougher circuit.

The Zn first cleaner concentrate will combine with the Zn third cleaner tailings and flow into the first of four 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 four 5 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 third cleaners will report to an 11 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 into trucks.

17.4.3.3 Pyrite Circuit

Tailings from the rougher and first cleaner zinc flotation circuits will be pumped to the pyrite rougher flotation which will consists of five 30 m^3 flotation tanks cells. Frother methyl isobutyl carbinol (MIBC) and PAX will be added to aid in flotation. The rougher concentrate, approximately 24% mass pull, will be thickened in an 11 m diameter thickener and then filtered. The filtered concentrate, potentially PAG material, will be processed as paste for deposition underground. The rougher tailings will feed the Ba flotation circuit.

17.4.3.4 Barite Circuit

Pyrite rougher tailings will feed the Ba rougher flotation circuit, which consists of six 10 m^3 flotation tanks cells operating in series. Frother methyl isobutyl carbinol (MIBC), Fuel Oil, Lime and Aero 845 will be added to the rougher circuit.

Rougher concentrate and the Ba second cleaner tailings will feed seven 10 m^3 Ba first cleaner tank cells. The Ba first cleaner concentrate will be collected in a common launder and fed to the Ba second cleaner flotation circuit. The Ba first cleaner tailings will combine with the Ba rougher tailings and be pumped to the dewatering circuit for processing as filtered tailings.

The Ba first cleaner concentrate will combine with the Ba third cleaner tailings and flow into the first of four 10 m^3 Ba second cleaner flotation tank cells. The Ba second cleaner concentrate will be collected in a common launder and pumped to the third cleaner flotation cells, while the Ba second cleaner tailings will flow back to the Ba first cleaner flotation feed box.

The Ba second cleaner concentrate will flow into the first of four 10 m^3 Ba third cleaner flotation tank cells. The Ba third cleaner concentrate will be collected in a common launder and pumped to the Ba concentrate thickener, while the Ba third cleaner tailings will flow back to the Ba second cleaner flotation feed box.

Ba concentrate from the third cleaners will report to a 14 m diameter Ba thickener. The thickener overflow will be sent to the process water tank. Thickened Ba concentrate will be pumped to a pressure filter for further dewatering. Ba filtered concentrate will be fed into a rotary dryer to reduce the moisture content to approximately 1% moisture, bagged and transported to Haines, Alaska.

17.4.4 Tailings Management Facility

Ba rougher tailings and Ba first cleaner tailings will combine in the final tailings pump box and be pumped to the paste backfill building. The slurry will be dewatered in a 14 m diameter thickener and then filtered. The filtered tailings will be either trucked to the filtered tailings facility or processed as paste and pumped underground. The thickener underflow and filtrate water will be reclaimed as process make-up water in the plant.

17.4.5 Reagents Handling and Storage

Reagents added to the 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 (kg/d)
Lime ($Ca(OH)_2$)	2 tonne bags (dry)	10,525
Copper Sulfate ($CuSO_4$)	1 tonne bags (dry)	1,505
Soda Ash (Na_2CO_3)	1 tonne bags (dry)	1,435
Potassium Amyl Xanthate (PAX)	1 tonne bags (dry)	1,141
AERO 845	1 tonne bags (dry)	998
SAG Mill Grinding Media – 125 mm chrome steel	1 tonne bags	876
Ball Mill Grinding Media – 75 mm chrome steel	1 tonne bags	799
Zinc Sulfate ($ZnSO_4$)	1 tonne bags (dry)	630
Methyl Isobutyl Carbinol (MIBC)	1 tonne totes (liquid)	497
Regrind Mill Grinding Media – 25 mm	1 tonne bags	277
Sodium Silicate (Na_2SiO_3)	1 tonne bags (dry)	245
Antiscalant	1 tonne totes (liquid), or 50 kg barrels	105
Fuel Oil	50 kg barrels (liquid)	96
Sodium Cyanide (NaCN)	25 kg bags (dry)	70
Flocculant	25 kg bags (dry)	53

Source: JDS (2019)

17.4.6 Plant Air

The primary consumers of compressed air are: the primary crushing plant and the pressure 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 from the thickener overflows and pressure filter filtrate. 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 Cu, Zn, Ba and Fe. Samples may also be analyzed for C, SiO_2, S, and SO_4. The concentrates will be tested for Cu, Zn, Ba, Au, Ag, 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 by titration or X-ray florescence.

Two main tests will be performed, water quality and acid rock drainage (ARD) potential. Water samples will be analyzed for sulfates, ammonia, nitrates, nitrites, cyanide, thiocyanide, pH, and hardness. One sample will be collected for ADR testing. 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 infrastructure is designed to support the operation of a 3,500 t/d mine and processing plant, operating on a 24 hour per day, seven day per week basis. The project envisions the upgrading or construction of the following key infrastructure items:

- Crushed product bin, and mill feed conveyor
- Process plant and re-agent storage warehouse
- Liquid natural gas (LNG) fuel power generating plant and LNG storage facility
- On-site power distribution with overhead power lines
- Tailings filtering and paste backfill plant
- Tailings management facility / waste rock storage facility (TMF/WRSF)
- Temporary mine rock stockpile (TMRS)
- Water treatment plant (WTP)
- Administration and mine dry building
- Warehouse
- 120,000 L of on-site fuel storage and distribution
- Industrial waste management facilities such as the incinerator
- Site water management facilities

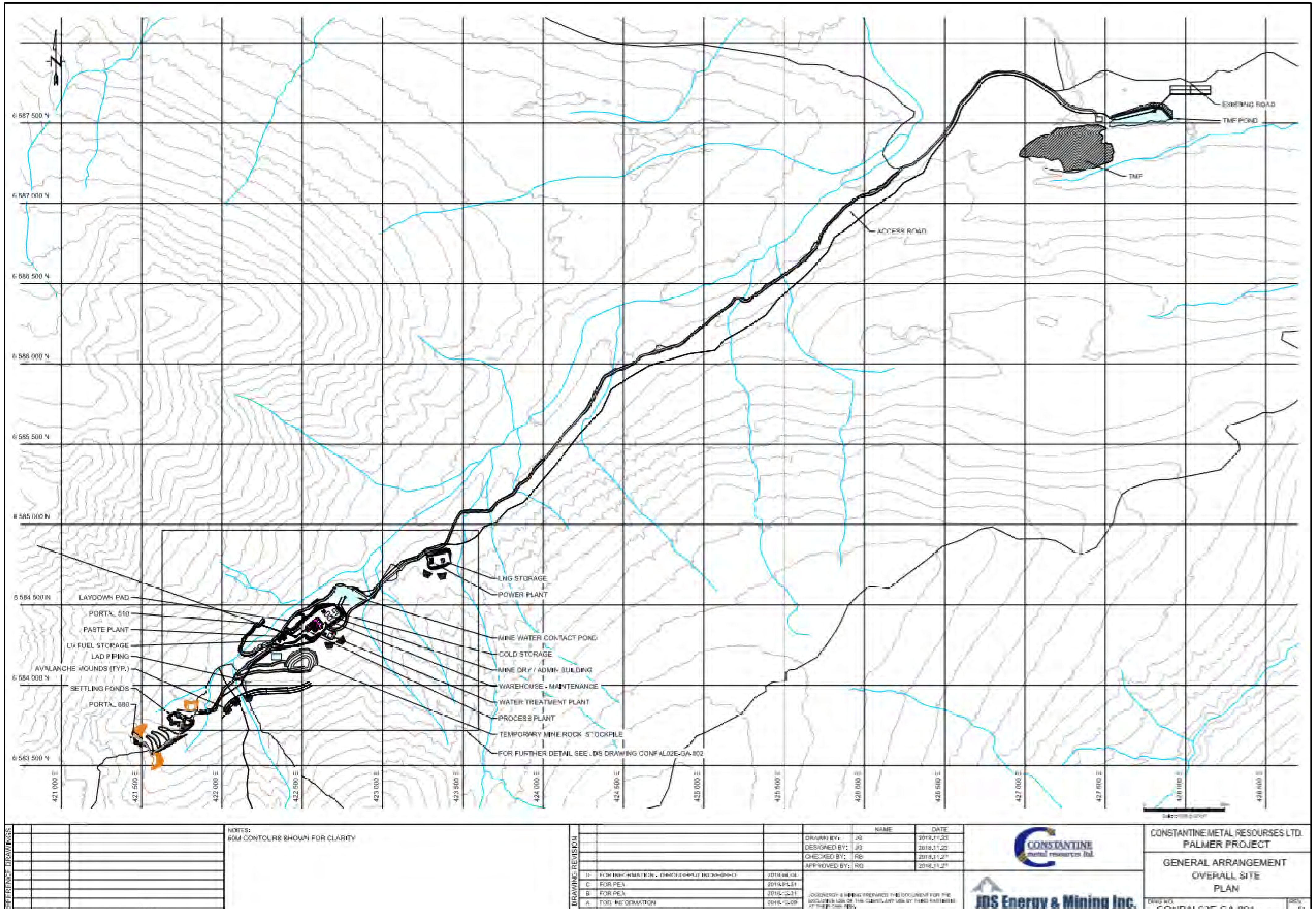
18.1 General Site Layout

The overall project site layout site is shown in Figure 18-1.

The proposed site layout has been configured for optimal construction access and operational efficiency. Primary buildings have been located away from avalanche areas, and in the locations identified by Constantine. The proposed TMF/WRSF has been strategically located to avoid mapped wetlands, take advantage of potentially preferable foundation conditions and maximize storage capacity.

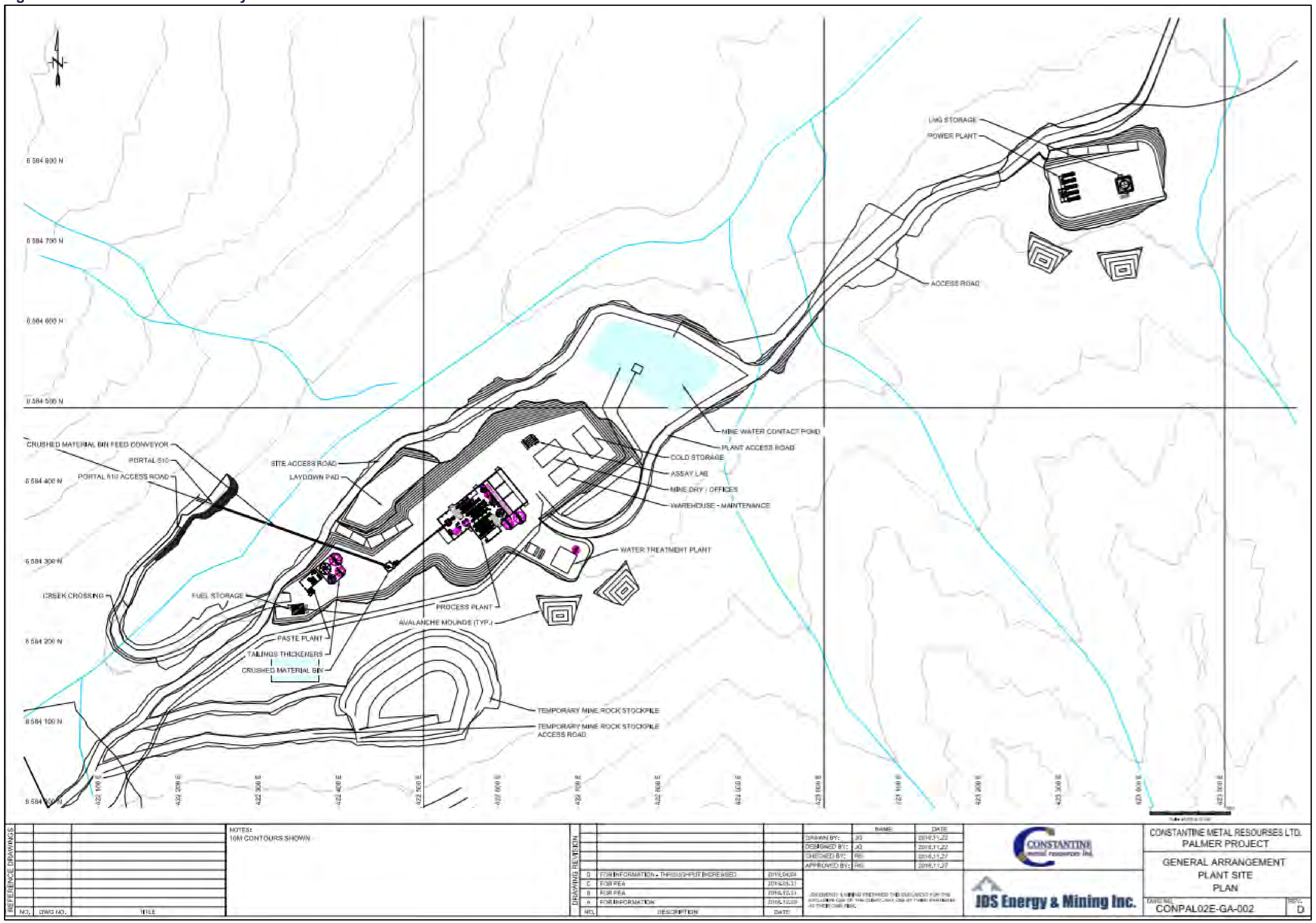
The site infrastructure layout and plant location are shown in Figure 18-2.

Figure 18-1: Overall Site Layout



Source: JDS (2019)

Figure 18-2: Site Infrastructure Layout



Source: JDS (2019)

18.2 Roads

18.2.1 Access Roads

Access to the Palmer Project will be via Hwy from Haines Alaska, approximately 40 km north to the start of an existing gravel access road. From the turnoff, the existing gravel site access road to the project site is approximately 20 km. Vehicles are likely to be a combination of buses, personal vehicles (cars, pick-up trucks), tractor-trailers, and equipment.

Improvements to the existing road will be constructed as necessary including proper gradients, widening and general cut / fill operations. The road upgrade is approximately 10 km in length.

18.2.2 Haul and Service Roads

Haul roads are planned to be upgrades of the existing site exploration access road, and new roads constructed for transporting mined material and waste to their designated destinations. The roads will connect the mine portals, plant site, mine rock stockpile and TMF/WRSF for the transport of mineralized mine rock and waste. Mine haul roads are planned to be constructed to accommodate 35-tonne trucks.

18.3 Power Supply and Distribution

18.3.1 Power Generation

Power necessary to support the Project operation will be supplied by on-site generator sets. A single power plant set up comprising five natural gas-fired reciprocating engine generator sets (gensets) in a N+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 4.16 kV.

The power plant and switchgear rooms will be modular units. They will be packaged in walk-in, sound-attenuated enclosures that are constructed, assembled and tested prior to shipment to site. They will be installed with interconnecting all weather walkways as a complete self-contained facility at site.

An LNG storage facility will be installed ate the power plant site with capacity for approximately 1 week of operation. The fuel storage facility will include a vaporizer plant to gasify the LNG before use. The fuel will be sourced from existing liquefaction plants in Canada and use tractor trailers for highway transport to the project site, via Skagway, then barge to Haines.

18.3.2 On-site Power Distribution

On site power will be distributed from the generation station to the plant and portals at 4.16 kV on overhead lines. The process plant and site infrastructure facilities will have localized substations to step down voltage from 4160 V as required.

18.4 Process Plant Facility

The process plant facility will consist of a crushed mineralized rock storage bin, feed conveyor, mill processing plant, and tailings filter plant. Refer to Section 17 for the full process plant description.

The mineralized rock storage bin will be store 24 hrs of material, 3,500 t. The bin will have two reclaim feeders, 1 operating and one standby, transferring material to the plant feed conveyor. The plant feed conveyor will be 265m long from the mineralized rock bin feeders transfer to the SAG mill feed.

The process plant building, as shown in Figure 17-2, will be approximately contain the grinding circuits, flotation circuits, and concentrate filtration equipment. It will also contain the concentrate load out stations for the copper, zinc and barite concentrates. Thickeners and large tanks will be outside along the building, within a concrete containment area. Pumps and piping associated with the tanks and thickeners will be inside the building or enclosed within the insulated thickener skirting.

A filtration plant will be located next to the paste plant, which will dewater the tailings and pyrite process streams. The pyrite and tailings stock tanks and thickeners will be outside within and concrete containment area. The filtration plant building will contain the filtration equipment, pumps and auxiliary equipment, and filtration discharge. Filtered pyrite tails will be kept within a contained area before reclaiming to use as paste backfill. Filtered tails material will be stockpiled outside and reclaimed for either paste backfill or loaded onto haul truck for delivery to the TMF/WRSF. Filtered tailings and pyrite material handling will be by site services mobile equipment and operators for transfer to the paste plant or the tailings facility.

18.5 Ancillary Facilities

18.5.1 Warehouse and Maintenance Shop Building

A separate building will be on the plant site for warehouse storage and the plant and surface facilities maintenance. It will have concrete foundation and concrete slab floor, lighting, heating and ventilation.

18.5.2 Cold Storage Warehouse

A sprung structure building will be erected for storage of parts and bulk materials requiring protection from the elements. It will be unheated with gravel floor, and concrete block foundation.

18.5.3 Assay Laboratory

An assay laboratory will be located in a separate building or attached to the main office / warehouse. This facility will serve the plant's assay, environmental, metallurgical requirements and underground grade control needs. The laboratory will consist of pre-fabricated modules and ancillary equipment, such as drying ovens, dust and fume control, and heating equipment.

18.5.4 Mine Dry and Office Complex

A separate two storey pre-engineered building will be on the plant site for operations office and mine dry facility. The lower level will serve as the personnel dry for the plant and mine operations, and the upper level will serve as the site administration and technical office facility.

The mine dry facility will service construction and operations staff during the life of the project. It will be set up with the following, with separate areas for male and female staff:

- Change areas with lockers and hanging baskets
- Showers and washroom facilities

- Operations line-up area

18.5.5 Fuel Storage

On-site diesel fuel storage will be installed on site with approximately one-week supply capacity. Two 40,000 L tanks will be installed within a lined containment berm, in addition to the existing tank already in use for exploration. 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.

18.6 Development Waste Rock Management

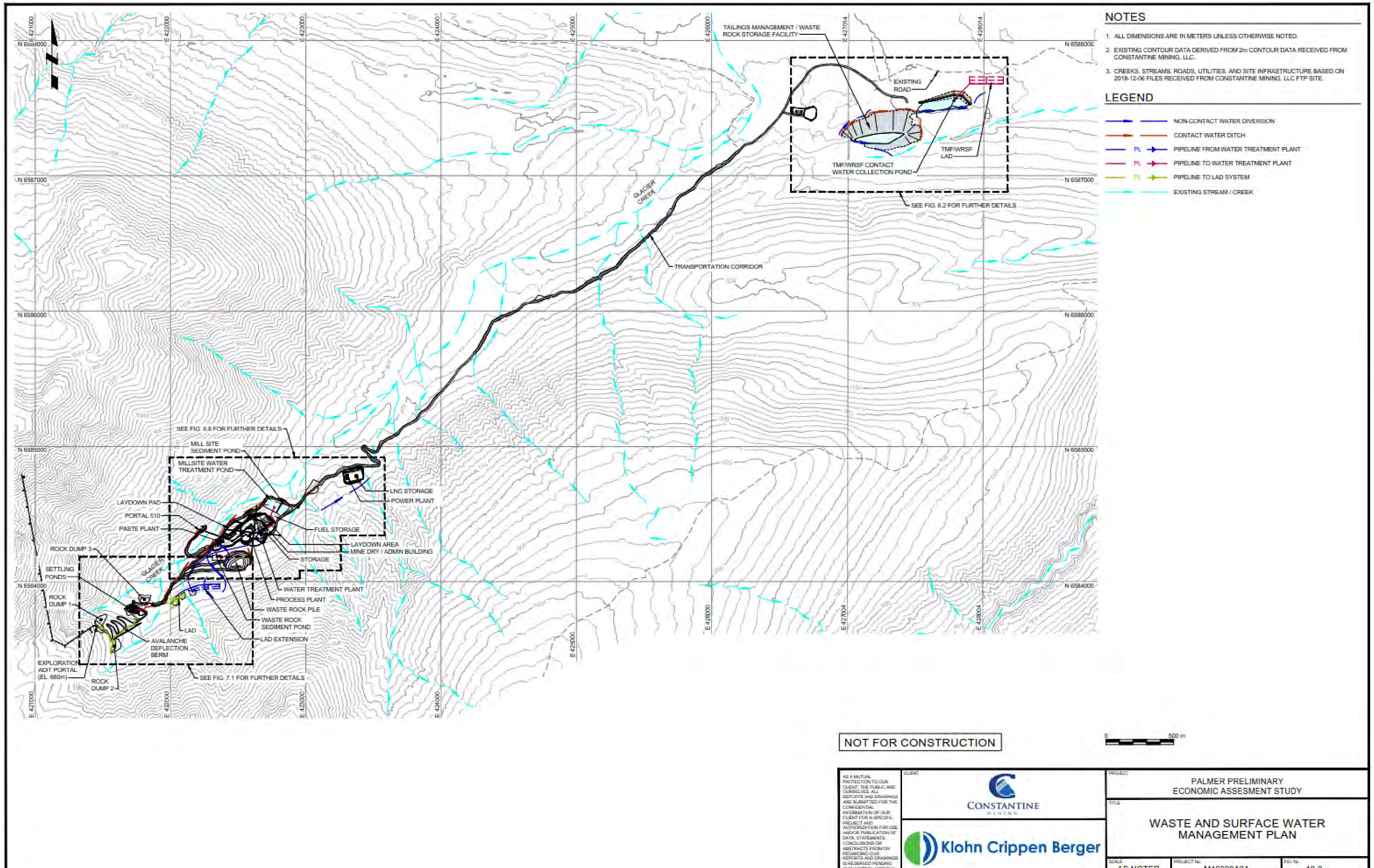
Potentially acid generating (PAG) waste rock from underground workings will primarily be disposed of as backfill in the underground workings. During the initial years, some PAG waste rock will be temporarily stored, until adequate space is available underground, in a segregated lined PAG rock storage / ore stockpile facility (capacity 400 kt) located on-surface at the processing plant. Runoff from the temporary facility will be treated by a Water Treatment Plant (WTP) located in the process plant.

The majority (93%) of non-potentially acid generating (NPAG) waste rock from development drifting will be disposed of on-surface and used for construction of protective outer berms around the perimeter of the TMF/WRSF. Based on geochemical testing, for the PEA it was assumed that neutral leaching will not occur from the NPAG waste rock and this material can be placed in the TMF/WRSF.

18.7 Tailings and Waste Rock Facilities

Design of a non-sulfide tailings management and NPAG waste rock storage facility (TMF/WRSF), along with the associated surface water management features, were developed by Klohn Crippen Berger Ltd. (KCB). The TMF/WRSF is located approximately 6 km northeast of the Conveyor Portal (see Figure 18-3).

Figure 18-3: Waste and Surface Water Management Plan



Source: KCB (2019)

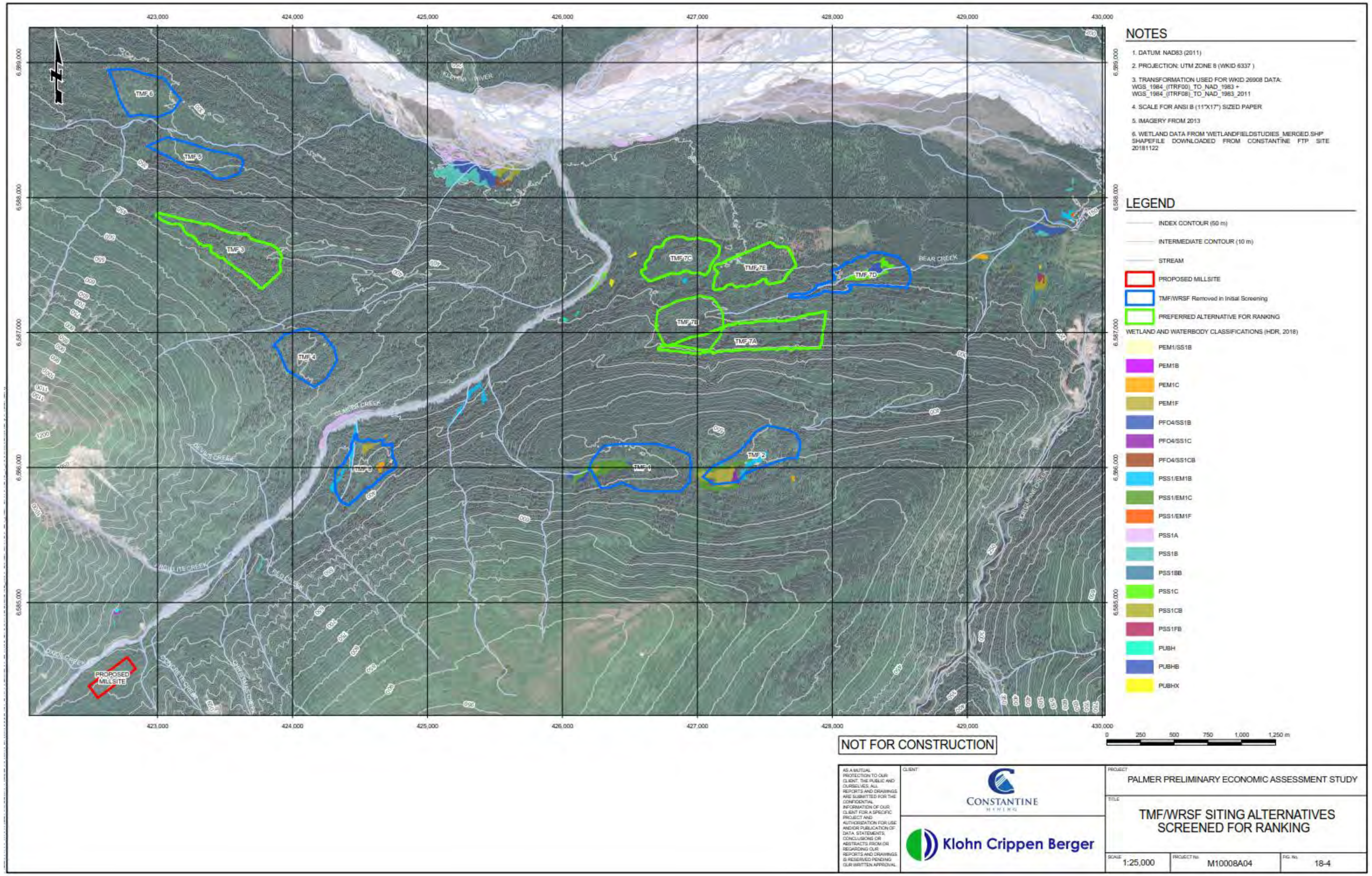
18.7.1 Site Selection

Conceptual designs for tailings and waste rock management alternatives were developed by KCB for twelve candidate sites. These siting alternatives along with potential tailings disposal methods were assessed using a screening approach to review siting and disposal alternatives against design criteria requirements. Filtered tailings with waste rock placed as outer containment berms in the same lined facility was selected as the preferred waste management method.

The five siting alternatives that passed initial screening were ranked using a semi-quantitative Multi-Criteria Analysis (MCA) method to score and select the preferred site. The ranking criteria used included: permitting (land tenure, wetlands); social (visibility from highway); environmental (presence of wetlands and watercourses, footprint size); water management (catchment area); geotechnical foundation conditions (based on assumed interpretation of sites); constructability (% structural fill based on topography); closure; total capital; and operational costs (based on haul distance, ratio of structural to non-structural zones and potential need for ground improvement).

The ranked merit scores developed within the MCA indicated that a TMF/WRSF site in the TMF-7C area is the preferred site (see Figure 18-4). An additional design optimization process was completed to develop a TMF/WRSF design that completely avoids water courses and an unconnected wetland adjacent to the area.

Figure 18-4: TMF/WRSF Siting Alternatives Screened for Ranking



Source: KCB (2019)

18.7.2 Design Basis

A flotation process that produces a desulfurized tailings stream and a pyrite tailings stream will be used in the plant. Approximately 78% of the tailings production will be used for paste backfill. The pyrite tailings stream will be only used for backfill. The desulfurized tailings are classified as NPAG and do not have neutral leaching characteristics based on geochemical testing performed to date. Initial geochemical laboratory testing on tailings samples included acid base accounting (ABA), inductively coupled plasma (ICP), whole rock characterization, shake flask and quantitative x-ray diffraction (XRD). Only desulfurized tailings will be stored on surface in the TMF/WRSF.

Life of mine total tailings production requiring storage on surface will be approximately 1,830,000 tonnes along with 1,110,000 tonnes of NPAG waste rock also stored on surface. The following summarizes the design basis for the TMF/WRSF:

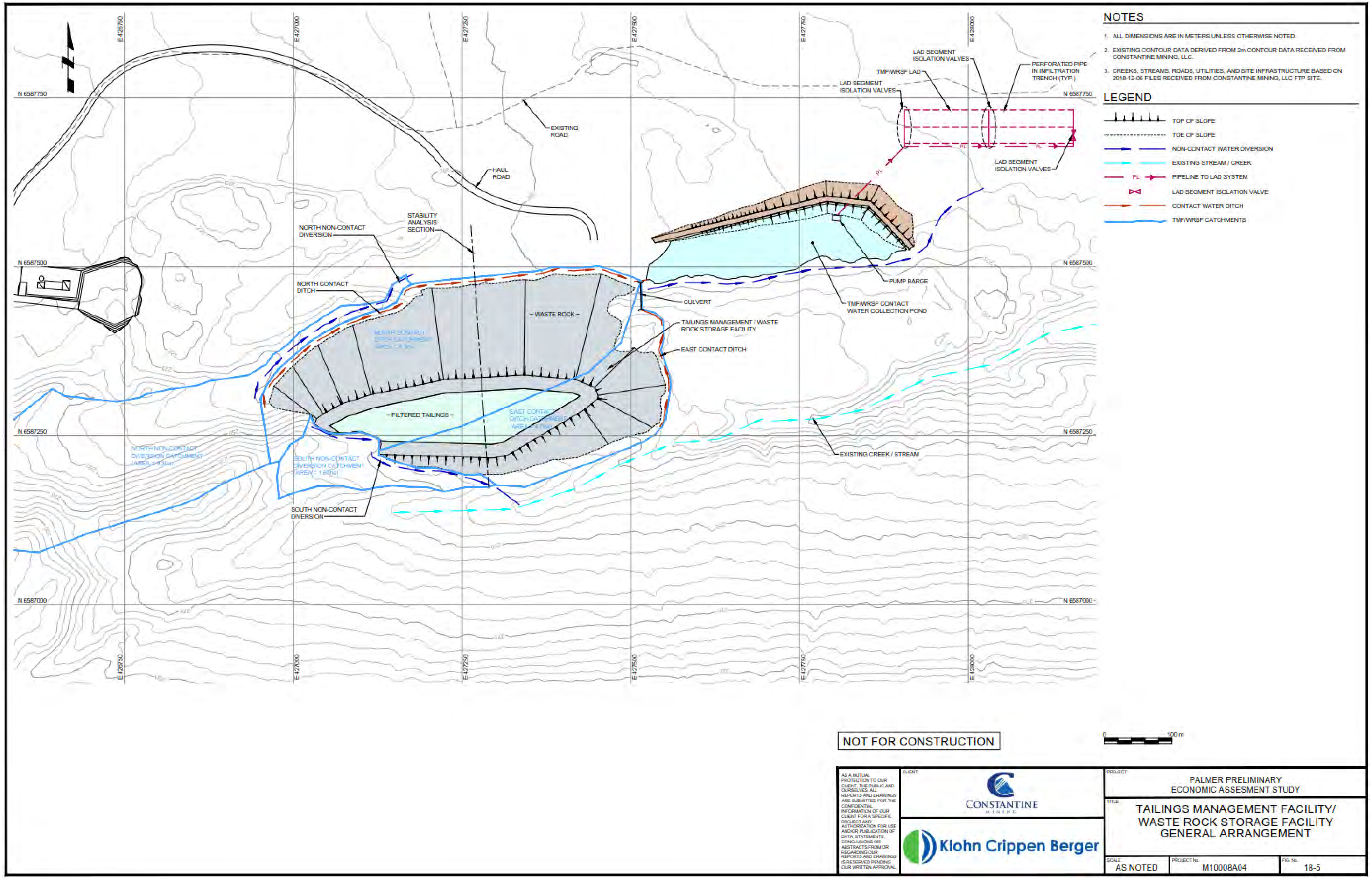
- The TMF/WRSF will have capacity to store the tailings and NPAG waste rock that is not used in backfill operations.
- The TMF/WRSF will not be constructed over areas that may be considered “jurisdictional” wetlands by the USACE, or over “intermittent streams”, as identified in mapping by HDR (2018).
- 1,830,096 tonnes of non-sulfide tailings will be stored in the TMF/WRSF. The assumed average dry density of the tailings is approximately 1.7 t/m3, resulting in a volume of 1,101,884 m3.
- 1,108,065 tonnes of NPAG waste rock will be stored in the TMF/WRSF. The assumed dry density of the waste rock is 2.1 t/m3, resulting in a volume of 540,520 m3.
- Tailings filter presses located at the mill site will be used to dewater tailings and prepare them for transport to the TMF/WRSF or for use in backfill.
- A tailings storage shed located at the mill site will provide capacity for approximately 5 days of tailings production destined for the TMF/WRSF in the event of an operational upset or inclement weather that prevents transportation and placement of tailings at the TMF/WRSF.
- The base of the TMF/WRSF will be lined with an engineered low-permeability layer to limit migration of tailings and waste rock-impacted seepage into the groundwater. At closure, the TMF/WRSF will be covered with a low-permeability closure cover and protected from erosion.
- The TMF/WRSF will meet applicable regulatory and guideline stability requirements. The maximum design earthquake (MDE) is defined for the PEA as the 1 in 2,475-year return period event (with PGA = 0.7 g, based on USGS mapping).
- Contact water from the TMF/WRSF will be collected in ditches and a pond and either treated or discharged, depending on water quality.
- Non-contact water will be diverted around disturbed areas where practical.
- The environmental design flood (EDF) (1 in 200-yr return period event) is defined as the flood event that must be stored and routed through a treatment facility. The inflow design flood (IDF) is defined as the 1 in 1,000-yr return period event.

18.7.3 Design Features

Key features of the TMF/WRSF during start-up and operations are shown on Figure 18-5 and include:

- Riprap-lined diversion ditches upstream of the pile to convey non-contact water around the pile.
- A waste rock and filtered tailings pile that includes both structural and non-structural zones. Waste rock is assumed to be placed in structural zones due to its coarser gradation, which makes it less susceptible to over-wetting and greater frictional strength compared to filtered tailings.
- Underdrain system comprised of a network of rockfill underdrains installed in the native foundation materials that reduce uplift pressure on the low-permeability layer during construction. Water collected in the underdrains will be discharged to the environment.
- Low-permeability layer (assumed to be a geomembrane liner) placed over the prepared foundation of the TMF/WRSF to prevent migration of tailings/waste rock impacted water into the groundwater. The liner is underlain by a granular bedding layer placed over the foundation to provide access for liner installation equipment and a smooth surface for liner deployment. The requirements for the low-permeability layer will be reviewed in future design stages once the geochemical risk of the waste rock and tailings is better understood.
- Service layer comprised of granular fill placed on top of the low-permeability layer to allow construction equipment to access the TMF/WRSF without damaging the low-permeability layer. A potential optimization to consider is use of tailings for the service layer, which may aid to reduce seepage rate through liner defects.
- Above liner drainage system comprised of rockfill finger drains and/or blanket drains in the footprint of the waste rock structural zones to provide pile toe area drainage. Water collected in this drainage system will report to the collection pond.
    - Modeling of tailings-liner-drain systems by KCB in conjunction with laboratory testing at Queen's University (Joshi and McLeod 2018) has shown that in contrast to typical landfill design, liners under tailings perform better without above liner drainage, since tailings more effectively seal liner defects if there is no above liner drainage present. For this reason, the above liner drainage system is not extended under the tailings portions of the pile.
- Geomembrane or concrete-cloth lined collection ditches downstream of the pile to collect pile runoff that report to the collection pond.
- A Contact Water Collection Pond (CWCP) located downstream of the pile to collect contact water conveyed by contact water ditches and the above liner drainage system. Water collected in the pond can be treated and/or discharged, depending on water quality. For the PEA it is assumed that the water is discharged to a land application disposal (LAD) downstream of the CWCP.
- Geochemical analysis of tailings samples indicate that the tailings are NPAG and of low risk for neutral leaching; therefore, treatment of contact water is assumed not to be required. However, the tailings, waste rock and associated water management facilities have been designed to allow for collection of contact water should treatment be necessary.
- A truck wash is assumed to not be required at the TMF/WRSF based on geochemical characterization of the tailings and waste rock. As the geochemical characterization of the materials placed in the TMF/WRSF is advanced, this assumption should be revisited.

Figure 18-5: TMF/WRSF General Layout



Source: KCB (2019)

18.7.4 Construction Methods

Placing tailings and waste rock in the same facility, and utilizing waste rock in exterior structural zones, has several desirable features. These include:

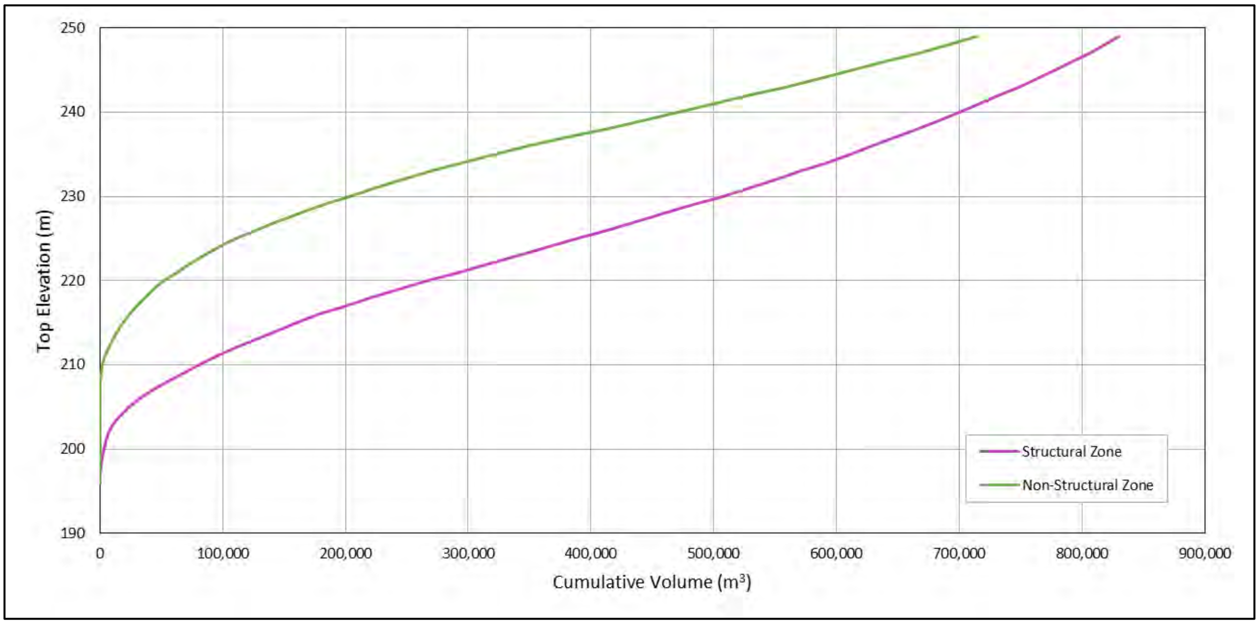
- The higher hydraulic conductivity of the outer berms of waste rock results in well-drained perimeter slopes which improves stability.
- Waste rock is more erosion resistant than tailings and, when placed at the closure slope angle, allows for the reclamation cover to be progressively extended during operations. Depending on the gradation of the waste rock, a filter zone(s) may be required between the filtered tailings and waste rock to prevent tailings migration.
- Presence of the higher strength waste rock in structural zones reduces the requirements for tailings compaction, which allows tailings placement under a wider range of climatic conditions. Note that tailings may also be used for structural zones, as is the case at the Greens Creek facility (Condon and Lear 2006), provided weather conditions are amenable to achieving sufficient compaction to preclude liquefaction.
- The availability of waste rock allows for construction of waste rock roads on the tailings surfaces, which improves trafficability and facilitates tailings placement.
- The production rate of waste rock will vary throughout the mine life with a greater production rate in the initial development years and a lower production rate towards the end of the mine life. Waste rock berms can be placed in outer sections of the pile in advance of tailings placement to reduce the need for stockpiling and double handling and may be utilized to construct a waste rock toe starter berm that can facilitate initial placement of tailings.

18.7.5 Layout and Staging

The three-dimensional modeling software, Muck 3D (Minebridge 2017), was used to develop the TMF/WRSF layout (see Figure 18-5) based on the volume requirements and other design criteria. Key observations from the modeling include:

- The maximum toe-to-crest height of the pile is approximately 50 m.
- The maximum thickness of the tailings is approximately 30 m.
- For modelling purposes, the pile was assumed to be constructed in horizontal lifts. Stage-storage curves based on horizontal raising for the structural and non-structural zones based are shown on Figure 18-6. The structural zone of compacted tailings is approximately 55% of the total pile volume.

Figure 18-6: TMF/WRSF Storage Curves



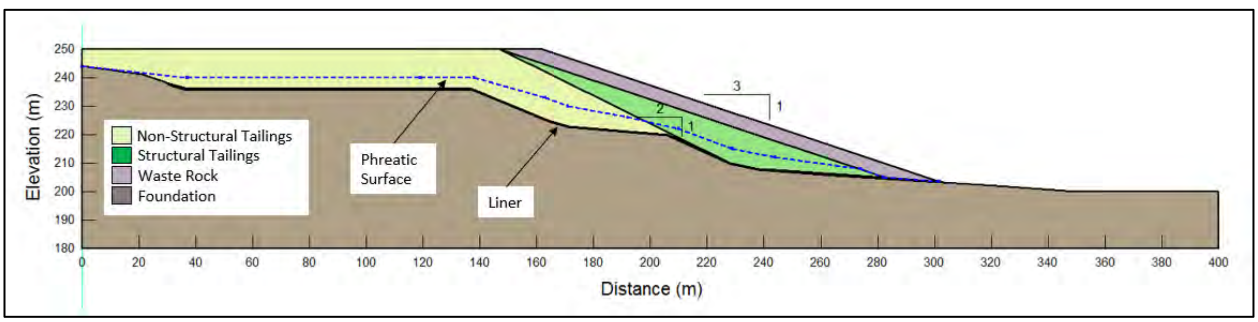
Source: KCB (2019)

18.7.6 Pile Stability

A section through the highest portion of the TMF/WRSF pile was selected as the critical stability section as shown on Figure 18-7. Stability of the pile was assessed using the limit-equilibrium modeling software, Slope/W (GeoStudio 2018) for the following scenarios:

- Static: effective friction angles applied to both waste rock and tailings; no seismic loading.
- Liquefied/post cyclic: effective friction angle applied to waste rock; liquefied/residual undrained shear strength applied to tailings in non-structural zones; no seismic loading.
- Pseudo-static: 80% of the effective friction angle applied to tailings and waste rock; horizonal seismic coefficient equal to 50% of the peak ground acceleration (PGA) (0.7 g).

Figure 18-7: TMF/WRSF Cross Section Showing Critical Stability Section



Source: KCB (2019)

The target limit-equilibrium Factors of Safety (FOS) are summarized in Table 18-1 along with the calculated FOS. Key conclusions from the analysis are summarized below:

- The critical slip surface for scenarios analyzed was along the geomembrane liner interface.
- Stability criteria were achieved for the static and post-cyclic/liquefaction scenarios assuming a structural zone constructed from waste rock, with the dimensions and slope angles as shown on Figure 18-7.
- The analysis assumes that the foundation will be non-liquefiable by either static or seismic loading. The pile was strategically sited on the glacial moraine terraces as these materials are less likely to be potentially liquefiable than the alluvial sediments on the lowlands closer to the Klehini River. If potentially liquefiable materials are identified in future site investigations, mitigations such as foundation improvement techniques, shallowing of outer slopes or installation of a foundation shear key may be required.

Table 18-1: Target and Calculated Factors of Safety for TMF/WRSF

Scenario	Target Factor of Safety	Calculated Factor of Safety
Static	1.5	1.6
Post-Earthquake	1.2	1.4
Pseudo-Static	1.0, or deformation analysis required	<1.0 (refer to footnote)[1]

Source: KCB (2019)

18.7.7 Water Balance

The CWCP has a peak monthly excess of water of 85 L/s. The highest monthly average flow is the 28 L/s freshet flow in April. The annual average excess of water is 6 L/s. The LAD was designed to discharge the peak monthly excess with some contingency (100 L/s).

18.7.8 Closure Plan

The TMF/WRSF will be constructed in a phased approach to allow progressive covering and reclamation. Once mine production is complete the TMF/WRSF will be entirely covered with a low permeability cover, an erosion protective layer and revegetated. Upon closure there will be no exposed tailings and contact-water runoff will cease. The period of post-closure TMF/WRSF water management will depend primarily on pile seepage water volume and quality. Long-term water treatment or monitoring is not expected to be required.

[1] Pseudo-static analyses are not intended to simulate limit equilibrium conditions but, rather, are considered to provide a preliminary seismic deformation screening analysis. Indeed, the various methods of pseudo-static analyses published in the literature are often associated with a particular magnitude of seismic deformation which is purported to not be exceeded if the seismic coefficient and material strength parameters are selected in accordance with the particular method and the specified minimum pseudo-static FOS is met. A pseudo-static FOS below criterion does not indicate failure but, rather, indicates that the seismic deformations could exceed those implied by the particular method used. In that case, a more rigorous seismic deformation analyses should be conducted. This will be assessed for the Project in future design stages.

18.8 Water Management

Using site climate data, longer term climate databases and climate models conducted a hydrology assessment was conducted and compiled storm runoff and snow accumulation as inputs to a site wide monthly water balance used to inform facility design. Water management facilities are shown on Figure 18-3.

The site wide water balance includes recharge-based estimates of sustained flows from the Exploration Portal and Conveyor Portal and effects of precipitation (i.e., runoff from TMF/WRSF facility, direct precipitation on the associated pond, mill site runoff, process reclaim and excess waters and underground drill water).

18.8.1 Surface Water Management

The primary objectives of the surface management are to:

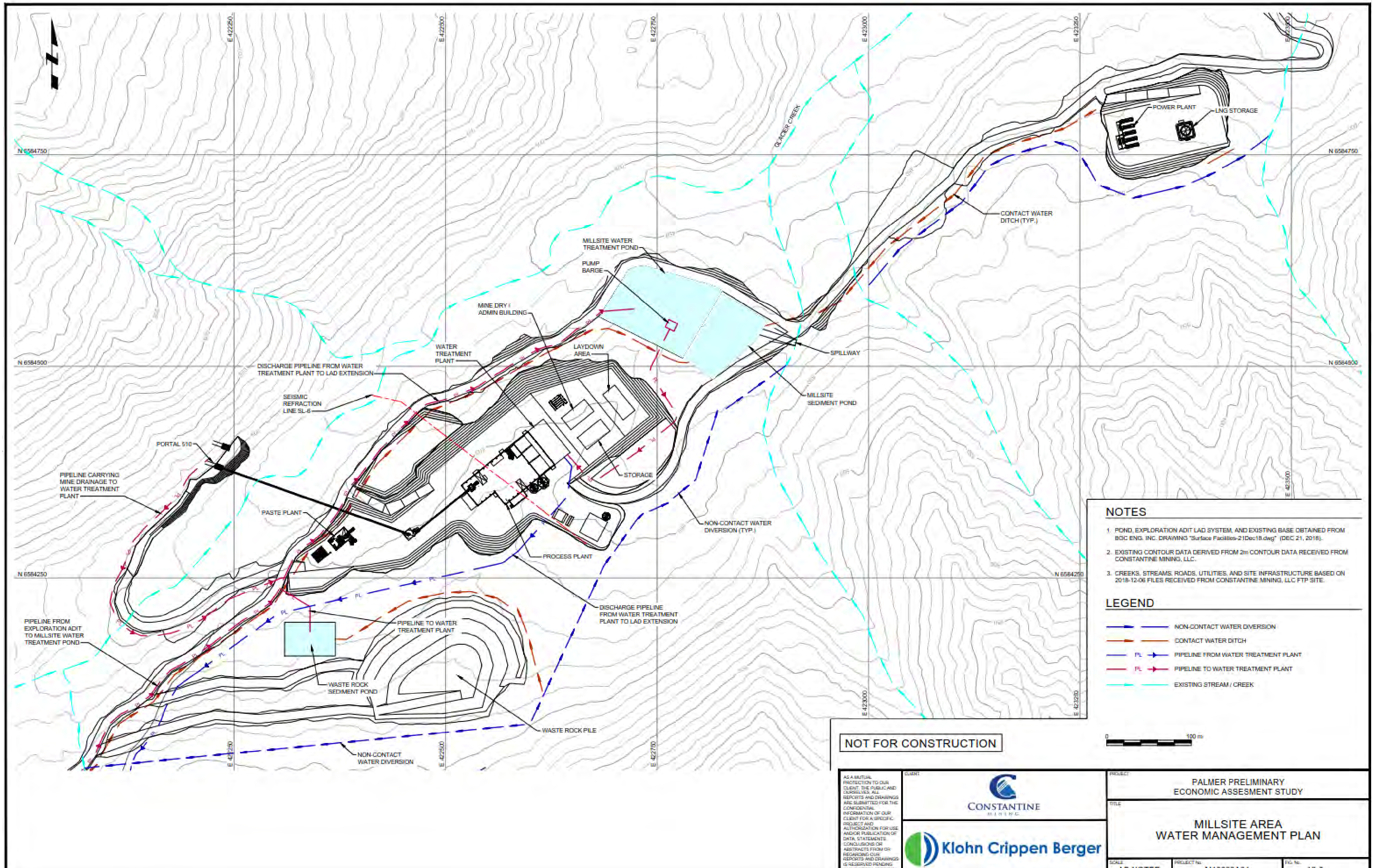
- Minimize short and long-term geochemical and water quality risks;
- Divert as much non-contact water around disturbed areas as practical to reduce contact water volumes, limit erosion and ponding/flooding; and maintain flow in natural watercourses; and
- Collect contact water that may contain high concentration of total suspended solids (TSS) for settlement and/or other contaminants for treatment if required.

Key features of the site-wide surface water management include are summarized below. Surface water management features associated with the TMF/WRSF are described in Section 18.7:

- A WTP located at the mill site with associated collection ponds: the Millsite Water Treatment Pond (MWTP) and Millsite Sediment Pond (MSP).
- A diversion ditch located upslope of the mill site that diverts non-contact water from the upstream catchment to natural water courses.
- Contact water ditches that surround the mill site and report to associated collection ponds.
- Ditches currently in place that flank the Access Road / Transportation Corridor.

A general arrangement, highlighting the locations of most of these water management features is shown on Figure 18-3. Mill site surface water management features are shown on Figure 18-8.

Figure 18-8: Mill Site Area Water Management Plan



Source: KCB (2019)

18.8.2 Management of Underground Water

Exploration Portal

Hydrogeological studies and flow estimates for the Exploration Portal outflows were completed by Tundra Consulting (Tundra 2018). This study was based on hydraulic conductivity measurements from packer testing, water levels and analytical calculations based on a conceptual hydrogeology model. An analytical estimate for the first 1.25 km section of adit was completed.

Outflows estimated by Tundra were used by BGC (BGC 2018) as a basis to design sediment ponds for outflows from the Exploration Portal. For conservativism, a sustained flow rate of 32 L/s from the entire Exploration Portal was assumed by Constantine as a basis for this work. For sizing of pumping and ponds, a flow rate of 190 L/s was assumed to account for potential higher hydraulic conductivities in the final sections of the adit and provide a factor of safety for design. Pond storage capacity was designed by BGC to contain the 100-year return period 24-hr storm from inflow to the surface catchment of the pond. The emergency spillway was designed for the 190 L/s design flow rate.

Outflow from the ponds will be routed via a gravity pipeline to a LAD with peak LAD capacity of 50 L/s (BGC 2018). KCB water management designs added a pipeline between the Exploration Adit Settling Pond (EASP) and the Mill Site Water Treatment Pond (MWTP), which can be used during periods when inflow rate to the EASP exceeds LAD capacity (see Figure 18-9). This pipeline can also be used to route drainage from the lined temporary PAG rock storage pad (located next to the EASP) to the MWTP if treatment of pad runoff is required.

Conveyor Portal

The geometry of the mine workings will include a lower production portal for the ore transport conveyor and spiral ramps providing access to SW and RW zone stopes situated both above and below the exploration adit. At the PEA level, it is appropriate to estimate portal outflows from the development of the workings using a water balance approximation based on recharge from total annual precipitation applied to appropriate infiltration areas. This approach allows the estimation of an upper bound of sustainable steady state flows.

Conservatively assuming a high annual infiltration fraction of 0.5 of total precipitation (fractured alpine terrain with little or no fines in soil), the estimated sustained outflow from the mine workings is 15 L/s. To allow for potential uncertainties the flow estimate was doubled (to 30L/s) which is comparable to the flow rate used for design of the Exploration Portal LAD (BGC 2018).

Given that the estimated flow from the workings are comparable to the flow used in the design of the LAD system, these additional flows will be treated by a comparable LAD section in an extension to the LAD system (see Figure 18-9).

18.8.3 Water Treatment

Flows from the Conveyor Portal, wastewater from the mill, and runoff from the temporary PAG waste rock storage pile are assumed to require treatment. Treatment will be provided by a WTP located at the mill site, with design details developed by JDS. It is planned to be a high-density sludge (HDS) treatment system to remove metals, with allowances for micro filtration, clarification, and dewatering of the resultant sludge. The design of the WTP allows for the following:

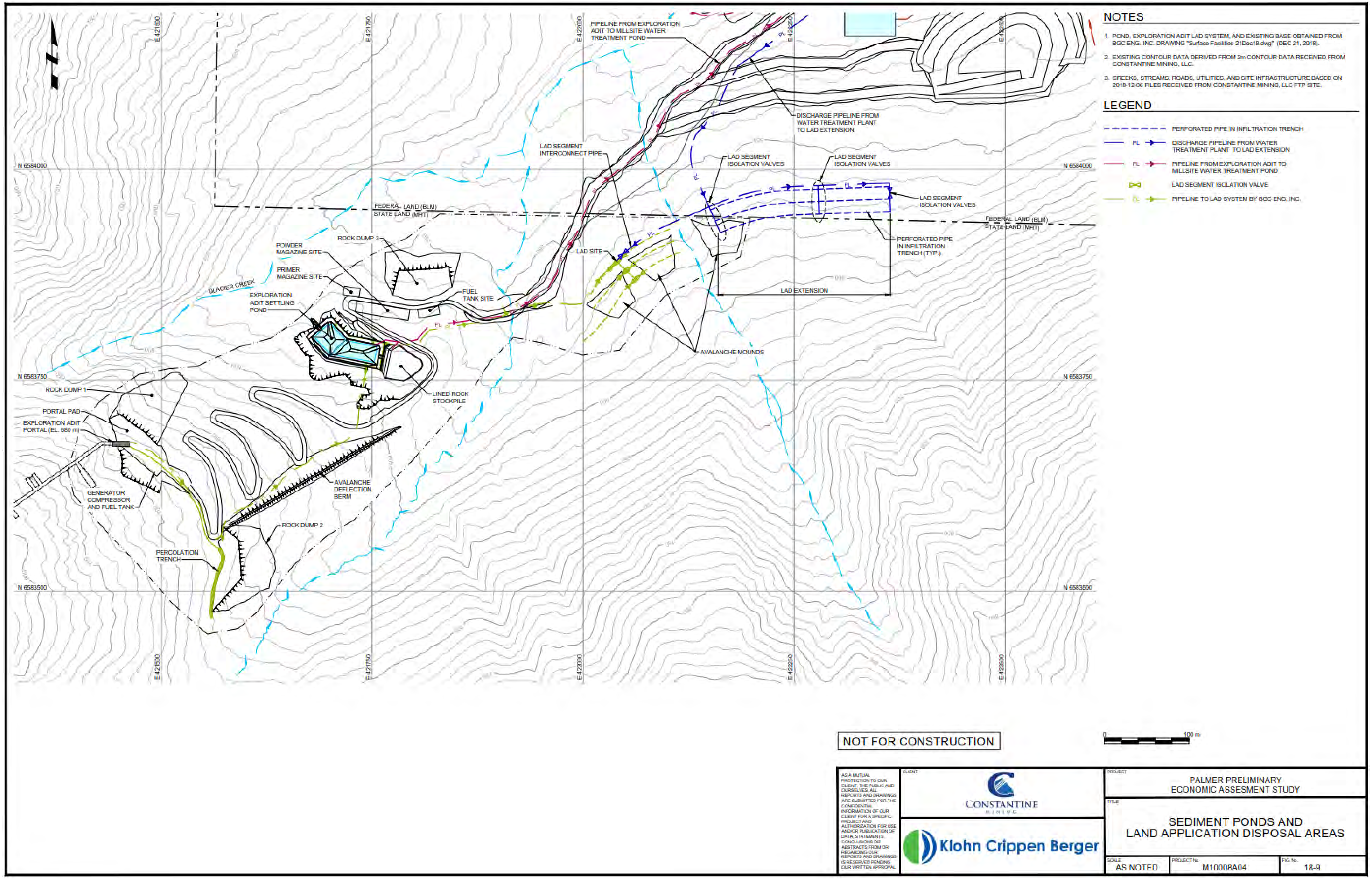
- Multimedia filtration (MMF)

- Chemical and pH treatment to promote metals precipitation
- Clarification
- Sludge dewatering of clarifier underflow and MMF backwash
- Ammonia removal
- pH control prior to release to environment

The WTP monthly water balance has a peak monthly excess of water of 67 L/s. The highest monthly average flow is the 40 L/s freshet flow in April. The annual average excess of water is 32 L/s.

To provide capacity above the peak flows, the WTP is assumed to require a peak capacity of 70 L/s. Discharge from the WTP is directed to an extension of the LAD as shown on Figure 18-9. The LAD facilities situated above the mill site were initially designed by BGC and subsequently expanded by KCB to increase capacity for discharge of excess project water.

Figure 18-9: Sediment Ponds and Land Application Disposal Areas



Source: KCB (2019)

18.9 Temporary Mine Rock Stockpile

JDS has provided a preliminary design for a temporary mine rock stockpile (TMRS) that will be set up near the plant site to store PAG mine rock temporarily. Mined mineralized rock and waste will both be stored at this location before being reclaimed for processing or use underground as backfill. The facility will be constructed and operated to store the materials separately and avoid contact between them.

18.9.1 Design Features

The facility will be designed and constructed to accommodate approximately 40,000 m^3 of mineralized rock and 160,000 m^3 of PAG mine development rock, for a total of 200,000 m^3 capacity. It includes a High-Density Polyethylene (HDPE) geomembrane liner lain atop a gravel cushion to contain contact material and water. Specific features of the design include:

- Foundation drainage system
- Fully lined area to minimize seepage losses
- PAG material water collection drain system and ditch conveyance to the MWTP

18.9.1.1 Foundation Drain

A foundation drain will be installed below the geomembrane liner to collect groundwater flows and seepage. Collected water will drain to a contact water conveyance ditch.

The foundation drain comprises interconnected perforated pipes surrounded by drain gravel. The foundation drain will be constructed using processed material generated from local borrow sources and will be constructed beneath the TMRS liner bedding layer.

18.9.1.2 HDPE Liner

The entire area, including the upstream face of the storage pads and the downstream collection ditches of the PAG storage area, will be lined with 80-mil HDPE geomembrane. The liner system incorporates a prepared subgrade comprising processed bedding material, which is expected to be fine-grained material with no rocks that can damage the HDPE liner. The HDPE geomembrane is effectively impermeable, with seepage only possible through defects that may occur during fabrication and/or installation.

18.9.1.3 Rock Storage Underdrain

A rock storage area underdrain will be installed above the geomembrane to collect contact water and convey it to the contact water conveyance ditch. The conveyance ditch will drain to the MWTP.

The underdrain will be constructed using processed material from local borrow sources. The underdrain includes perforated drainpipes within a free draining surround. A 300 mm thick layer of filter material will be placed around the drainpipes to assist in providing drainage and protecting the HDPE geomembrane liner.

18.10 Off-site Facilities

18.10.1 Accommodations

The operation personnel will be largely will be based on personnel living in or nearby Haines. Transportation via bussing will be provided for employees from Haines to site during operations.

A temporary camp will be rented and set up to accommodate construction workers, for a period of approximately 16 months. The camp will be set up off site nearby within 15 km of the project site.

18.10.2 Administration Offices

The general administration offices will be located in Haines in a rented or leased facility. The administration, purchasing, and clerical support staff will work in the Haines office, and report to site when needed.

19 Market Studies and Contracts

19.1 Market Studies

No market studies have been completed for the project at this time. Except for barite, all commodities considered in this Study are regularly sold commercially on vast international markets. As the Palmer concentrates are clean, that is relatively free of contaminants, competitive treatment charges are anticipated with relatively easy sales to open markets.

19.2 Contracts

Constantine has an agreement with Dowa that includes the right of first refusal on the zinc concentrate.

The following is extracted from the February 1st, 2013 agreement:

3. DOWA'S RIGHTS TO ZINC AND COPPER CONCENTRATES AND RECOVER

(a) Dowa will have the following rights which will survive any termination of this Agreement:

(i) if Dowa makes or funds Earn-in Expenditures totalling $5,500,000 but not exceeding $11,000,000, Dowa will have a right of first offer to purchase 25% of any zinc concentrate produced from Operations and

(ii) if Dowa makes or funds Earn-in Expenditures totalling $11,000,000 but not exceeding $22,000,000:

(A) Dowa will have the right to purchase 50% of any zinc concentrates produced from Operations, subject to arm's length commercial terms and freight charges, and this right to purchase will replace the right of first offer set forth in Section 3(a)(i) above and

(B) Dowa will have the right to recover 50% of its Earn-in Expenditures from Net Profits after pay back at a rate of 5% of Net Profits interest per year, as further described in Schedule E

and any subsequent transfer of rights under the Mineral Lease Agreement will be made subject to the foregoing rights of Dowa.

(b) As of and from the Operative Date and as long as Dowa maintains a 49% Participating Interest, Dowa will have the right to purchase 100% of any zinc concentrates and up to 50% of any copper concentrates produced from the Property.

(c) After the Operative Date, if Dowa's Participating Interest decreases to below 49%, its off-take right to zinc concentrates and copper concentrates shall be reduced in a manner proportionate to the reduction in its Participating Interest, provided that that in no event will Dowa's off-take right to zinc concentrates be reduced to less than 50% of the zinc concentrates produced from the Property.

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 copper, zinc, silver and gold prices are shown in Figure 19-1, Figure 19-2, Figure 19-3 and Figure 19-4. Historical exchange rate trends are plotted in Figure 19-5.

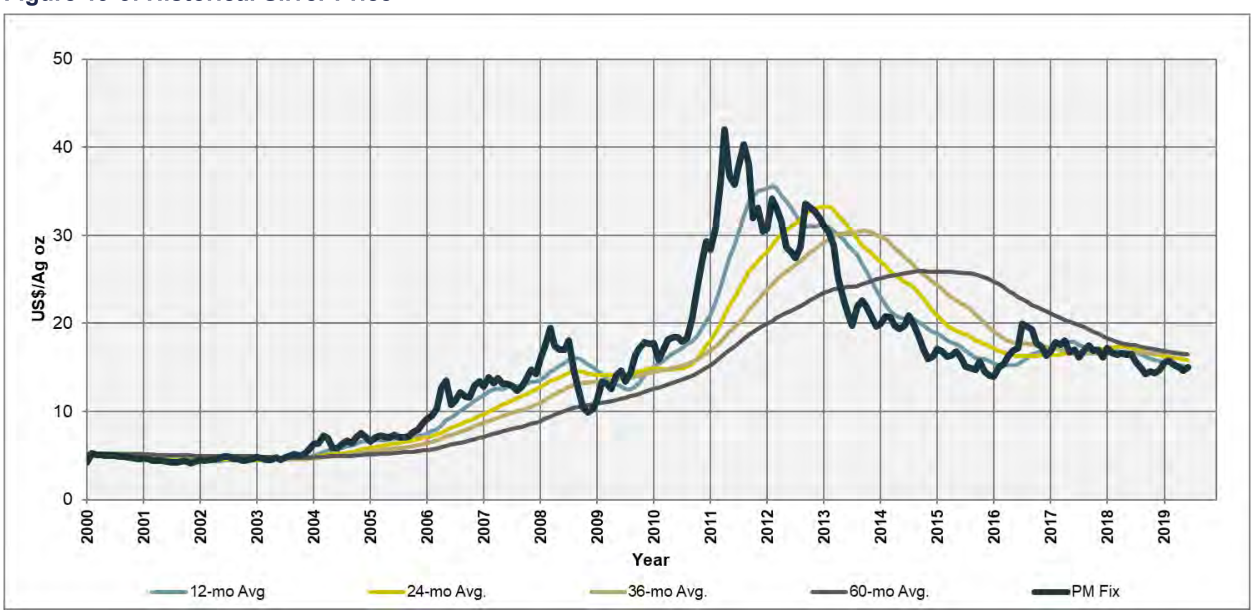
Figure 19-1: Historical Copper Price

Source: London Metals Exchange (2019)

Figure 19-2: Historical Zinc Price

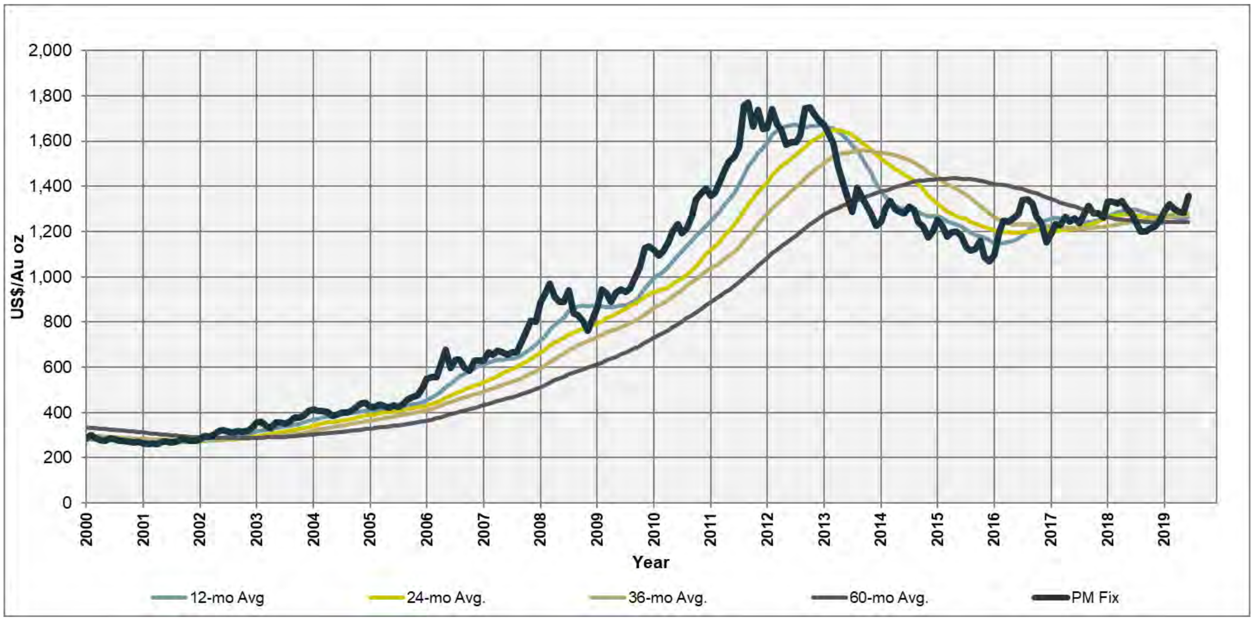
Source: London Metals Exchange (2019)

Figure 19-3: Historical Silver Price



Source: Kitco (2019)

Figure 19-4: Historical Gold Price



Source: Kitco (2019)

Figure 19-5: Historical US$:C$ F/X Rates

Source: Bank of Canada (2019)

The copper, zinc, silver and gold 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 April 15th, 2019. These parameters are in line with other recently released comparable Technical Reports.

The Barite price used in this PEA study was selected based on an average price of competitive wholesale prices of Barite Concentrate.

A sensitivity analysis on the metal prices was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Table 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
Copper Price	US$/lb	2.82
Zinc Price	US$/lb	1.22
Silver Price	US$/oz	16.26
Gold Price	US$/oz	1,296.37
Barite Price	US$/tonne	220
Exchange Rate	US$:C$	0.75

Source: JDS (2019)

20 Environmental Studies, Permitting and Social or Community Impacts

This section summarizes the existing environmental information for the Palmer Project area, describes the major mine permits that may be required to develop the Project into a mine, and describes social and community considerations for the Project.

20.1 Environmental Studies

The Palmer Project area includes lands within Glacier Creek valley within the Klehini River drainage.

Environmental baseline monitoring has been conducted in the area starting in 2008 with formal reporting starting in 2013. The primary objectives of the baseline monitoring are to characterize the natural environment and identify reference locations for comparison throughout the Project life to assess impacts.

To date, a moderate amount of baseline environmental data collection has occurred in the area including surface water quality sampling, surface hydrology monitoring, wetlands mapping, stream flow monitoring, aquatic life surveys, avian and mammal habitat surveys, cultural resource surveys, hydrogeology studies, meteorological monitoring, and acid base accounting studies. The existing data are summarized in Sections 20.1.1 to 20.1.9.

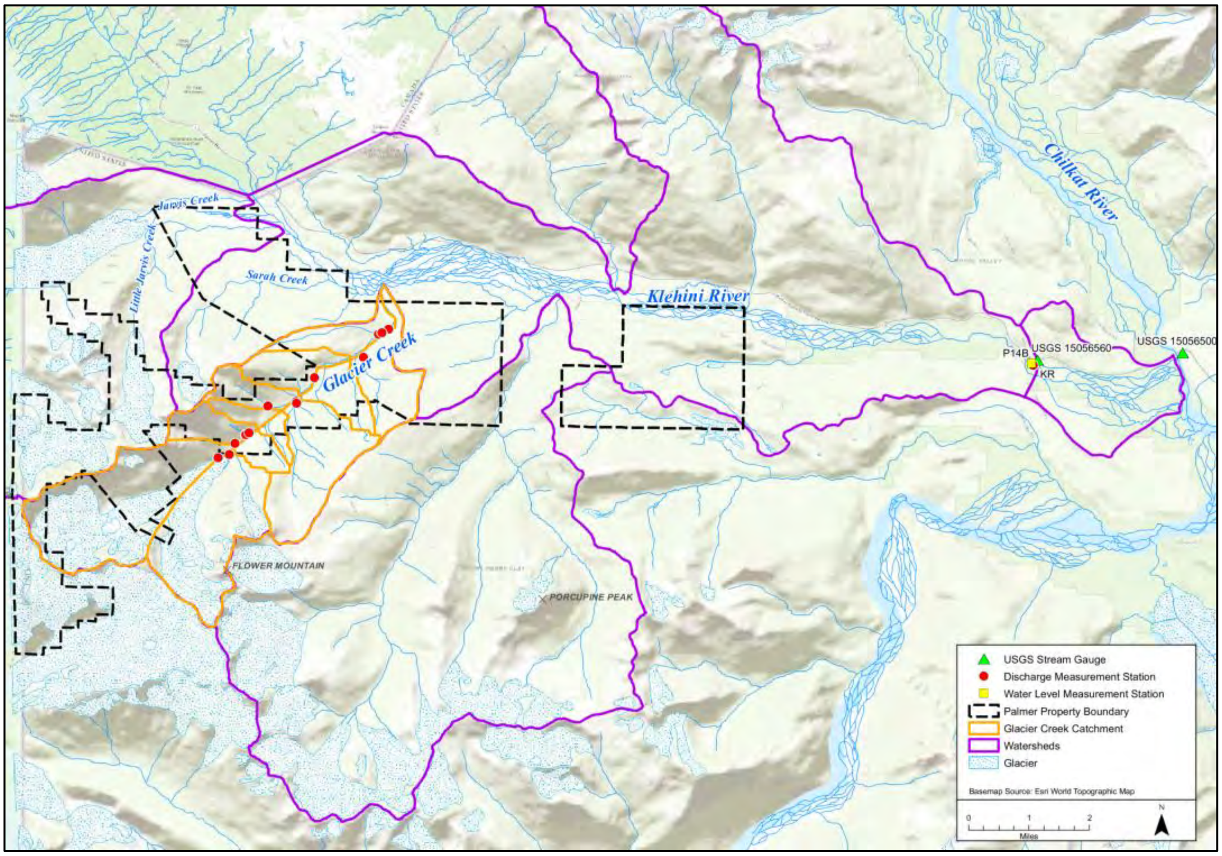
20.1.1 Surface Hydrology

Constantine has been monitoring surface water flows (aka hydrology) at 11 sites in Glacier Creek since 2014. These monitoring sites and two USGS stream gauge sites are illustrated on Figure 20-1.

The Palmer Project is centered within the Glacier Creek watershed that drains into the Klehini River. The Klehini River flows into the Chilkat River approximately 14 miles downstream of confluence of Glacier Creek with the Klehini River. The Klehini River follows a large braided channel that runs from west to east along the northern property boundary. Within the larger Palmer property, the names of the stream systems are Jarvis Creek, Little Jarvis Creek, Sarah Creek (also informally called Pump Valley Creek), and Glacier Creek. Outside the project area water levels are measured at two USGS stations and Constantine's station P14B and water flow is measured at Constantine station KR (Figure 20-2).

Glacier Creek and its tributaries provide the primary drainage within the Palmer project boundary, ultimately flowing into the Klehini River. Individual flow measurements on Glacier Creek from 2015 and 2016 ranged from 41 ft^3/s to 471 ft^3/s, roughly 10% to 20% of the Klehini River discharge volume.

Figure 20-1: Surface Water Flow Monitoring Stations



Source: Constantine (2019)

The highest discharge measurements in the Glacier Creek drainage were collected in the month of June, while the lowest discharge was measured in October, like the regional patterns observed in the Klehini River hydrograph.

Integral Consulting Inc. evaluated regional (Chilkat River and Klehini River) and localized (Glacier Creek and tributaries) surface water hydrology patterns based on stream discharge and meteorological data (Integral, 2016). The following paragraphs are excerpted from their report. Surface water in the region exhibits seasonal and diurnal patterns due to the influences of snowmelt, rainfall events, and ambient air temperature.

Integral interprets the hydrograph data to recognize two seasonal patterns. The first pattern illustrates the dominant character of the Klehini River watershed, whereby snowmelt runoff drives annual peak stream discharge in the spring. The onset of peak flow occurs quickly in the late spring to early summer months, generally late April through June. The onset of peak flows also occurs when average daily air temperatures routinely exceed freezing and when total monthly precipitation is lowest. The combined effect of above-freezing temperatures and low precipitation is reflected in the sharply rising limb of the discharge hydrograph. During the period following peak flow (the declining limb), beginning in the early summer months until early fall, stream discharge gradually declines as the snowpack recedes and average daily air temperatures approach freezing. Klehini River discharge remains near base flow throughout the winter and early spring months, typically from November to April. During this time, individual precipitation events produce short-lived increases in stream discharge above base flow.

A second seasonal pattern can be seen in the Klehini River hydrograph for most years, when river discharge shows a secondary peak in late fall. The late fall peak discharge events occur when total monthly precipitation begins to increase and before average daily temperatures dip below freezing. The increase, and subsequent decline in late fall discharge, occur over a much shorter period of time than the first peak discharge cycle because individual precipitation events, rather than snowmelt runoff, provide runoff for the late fall peak flow.

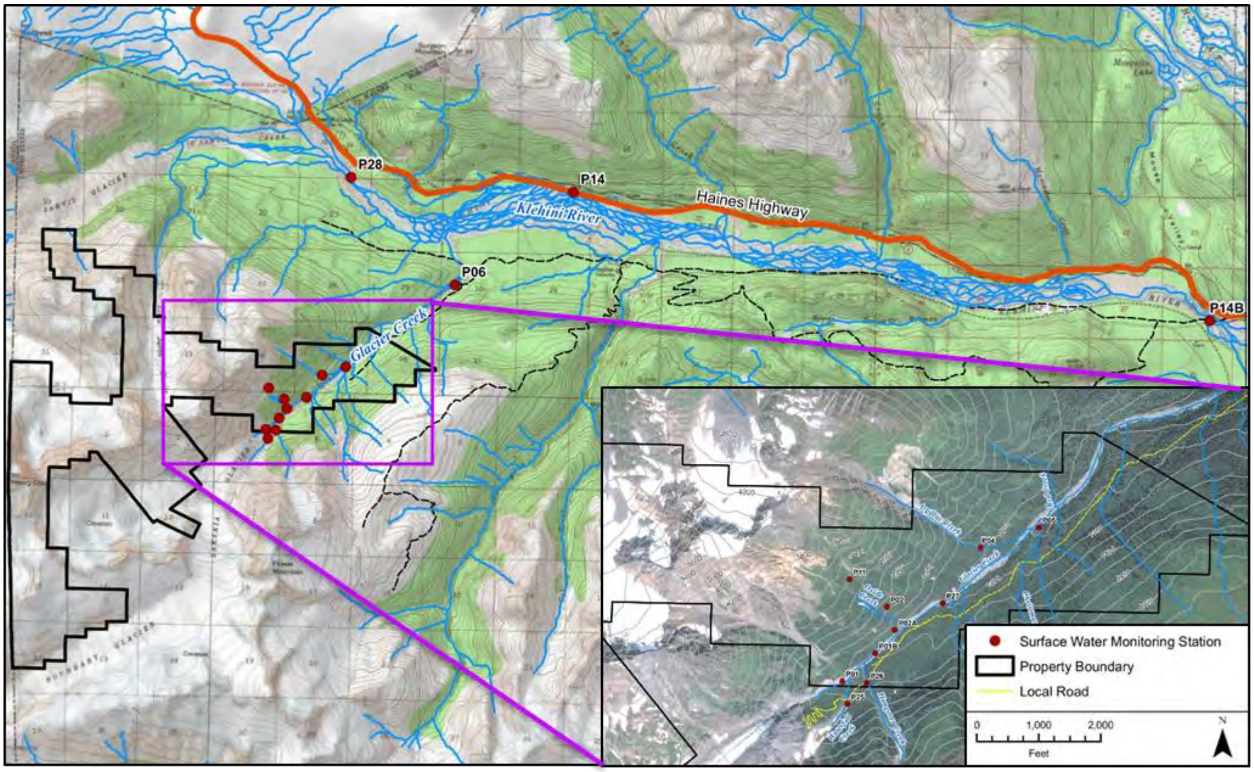
Stream discharge measurements taken at Glacier Creek downstream stations P6 and LG and midstream station GC show seasonal changes like the Klehini River hydrograph. The higher discharge measured at Glacier Creek stations in summer months corresponds to periods of peak flows driven by snowmelt, but little rainfall. The relatively lower discharge measured in October 2016 corresponds to the decreasing limb of the Klehini River hydrograph as temperatures cool, snowpack is diminished, and the influence of snowmelt decreases.

Integral (2016) also recognized diurnal trends. During the warmer spring, summer, and fall months, when average air temperature remains above freezing, diurnal discharge patterns are apparent for the Klehini River. Hourly Klehini River stage and ambient air temperature measured from. For example, during the period May 20th to 27th, 2015, during a period of peak stage and when there was no measurable precipitation, daily maximum air temperatures occurred in the late afternoon, while daily maximum stage was observed from late night to early morning. This diurnal pattern suggests gradually increasing rates of snowmelt runoff through the daylight hours in response to increasing air temperature followed by a lag in the peak stage (Integral 2016). Peak stage and streamflow discharge lag peak temperature because of increased travel times required for runoff to move through tributaries upstream from the Klehini River gage station. Average daily temperatures are below freezing from December through March. Clear diurnal patterns are not apparent during these colder winter months.

20.1.2 Surface Water Quality

Constantine has been monitoring surface water quality by collecting samples from up to 27 stations since 2008. Those stations are illustrated in Figure 20-2. Constantine's consultant, Integral Consulting, prepared a summary water quality memorandum for the period 2008 – 2016. Surface water samples are collected from flowing water (away from eddies and interferences). Unfiltered surface water samples are collected as grabs (not composited) from the stream using native-water rinsed non-reactive collection containers. Filtered surface water samples are collected using a peristaltic pump with a 0.45-μm filter placed in-line at the tubing outlet to filter samples immediately before the water was discharged into the sample bottle. Alternatively, if an in-line filter and peristaltic pump was not available or practical for the sample collection, samples were filtered manually through a 0.45-μm syringe filter. Immediately after filling, the sample containers were capped, labeled, and placed inside a cooler. Detailed sample collection, handling, and analysis information is presented in the Palmer project Quality Assurance Protection Plan (QAPP).

Figure 20-2: Surface Water Quality Sample Location Map



Source: Constantine (2019)

Measurements of conductivity, dissolved oxygen, pH, temperature, and turbidity were collected in the field at all sampling stations in accordance with the QAPP. A YSI 556 multi-probe was used for dissolved oxygen, pH, temperature, and conductivity measurements. A LaMotte 2020 turbidity meter was used for turbidity measurements. Measurements were recorded when the field instrumentation readings stabilized. Field instruments were maintained, cleaned, and calibrated with standard reference solutions per the manufacturer's specifications. Field instrument standard operating procedures (SOPs), manuals, and calibration log forms are presented in the Palmer project QAPP. Solids settling samples were collected in the field and analyzed as described within the project QAPP.

The following parameters are typically measured in baseline surface water samples:

- Field parameters – Field measurements of general water quality characteristics, including conductivity, dissolved oxygen, oxygen-reduction potential (ORP), pH, temperature, and turbidity, were taken at all sampling stations, in accordance with the QAPP (Integral 2015). A YSI 556 multi-probe was used for dissolved oxygen, pH, temperature, and conductivity measurements. A LaMotte 2020 turbidity meter was used for turbidity measurements. If these instruments were not available, equivalent instruments that meet method requirements were used and calibrated per the manufacturer instructions.
- Conventional parameters – Conventional analyses performed by the ALS laboratory included acidity, hardness as $CaCO_3$, total dissolved solids (TDS), and total suspended solids (TSS).

- Solids Settling– Solids settling results are presented with conventional parameters. The solids settling samples were collected in the field and analyzed following Standard Method 2540F (SM 1997). The solids settling analysis was performed as soon as possible following the sampling event. Solids settling concentrations were determined using the Imhoff cone volumetric technique to measure the mass of solids that settle from 1 L of water in a 60-minute period. The solids settling measurement SOPs are presented in the Palmer project QAPP (Integral 2015).
- Cations/anions – Major cations and anions typically analyzed included alkalinity as CaCO3, bromide, chloride, fluoride, nitrate, combined nitrate/nitrite, sulfate, and ammonia.
- Total/Dissolved Metals – Thirty-three metals were analyzed for both the total and dissolved fraction.

For all metals except cadmium, the laboratory reports and/or validator-assigned concentration detection limits are below the Alaska water quality standards. This indicates that the analytical methods used meet the Data Quality Objectives (DQO) outlined in the project QAPP are appropriate, and that the baseline data set is acceptable for comparison to aquatic life standards. For dissolved cadmium, four samples were undetected by the laboratory at a detection limit of 0.25 µg/L: samples P8 and P14 for the August 2009 sampling event, and samples P1 and P7 for the June 2011 sample event. The hardness- based chronic aquatic life standard for these four samples ranged from 0.18 µg/L to 0.25 µg/L. Typical detection limits for cadmium for this project have ranged from 0.005 µg/L to 0.1 µg/L; detection limits greater than the aquatic life standards have not been observed in subsequent sampling events (Integral, 2016).

Background surface water concentrations for metals are compared with the freshwater aquatic life criteria in Table 20-1.

As summarized by Integral (2018), the surface waters in the project area generally exhibit high-quality water. Some natural exceedances of aquatic life water quality standards for metals were observed during one or more sampling events including: dissolved cadmium and zinc in Oxide Creek (P2), total selenium in Argillite Creek (P4), and total aluminum and iron at multiple stations. For the individual surface water samples that exceeded aquatic life standards, these relatively higher concentrations are representative of the background water quality conditions of each location. Groundwater seepage near the P2A location (see inset, Figure 20-2) may be a source of dissolved cadmium and zinc to the naturally elevated concentrations observed for the Oxide Creek (P2) drainage. Total aluminum and iron concentrations show a clear positive correlation with Total Suspended Solids (TSS) and a pattern of higher concentration in upstream locations relative to the paired downstream locations, which suggests that solids associated with glacial and snowpack melt add particle-bound aluminum and iron to the streams at their headwaters. Although the Oxide Creek and Argillite Creek tributaries to Glacier Creek exhibited naturally elevated concentrations of cadmium, selenium, and zinc, these parameters did not exceed aquatic life standards in Glacier Creek (i.e. P27) (Integral 2016).

Integral (2018) observed a large variability in the concentrations of many water quality parameters between locations and at different times of year. Differences in local geology and mineralization, as well as the variable proportion of glacial melt/surface runoff and base flow comprising streamflow, are expected to influence water quality and drive variations in conventional, major ion, and metal concentrations between sampling locations. Larger, glacier-fed streams (the Klehini River, the Jarvis and Little Jarvis Creek system, and Glacier Creek) tend to carry higher amounts of suspended solids during periods of snowmelt (late spring through summer) and during precipitation events. Smaller tributaries generally have lower suspended solid loads, clearer waters, and lower flow volume; water chemistry in these streams may be more heavily influenced by groundwater and local geology.

Now that Constantine has characterized the baseline water quality in the broader Palmer Project area, the Company now plans to reduce the total number of surface water quality sample sites. Reductions in the scope of environmental baseline monitoring are common for advanced exploration projects following collection of sufficient data to characterize an area somewhat larger than the anticipated footprint of the Project. Constantine will continue monitoring at sites P1 and P27 in upper and mid-Glacier Creek, respectively. These sites are the most relevant sites for detecting any significant change in water quality, over time that may be concomitant with any proposed underground exploration activities in the upper Glacier Creek area.

Table 20-1: Comparison of Surface Water Quality to Freshwater Aquatic Life Criteria for Metals

			Water Measurements						Chronic Aquatic Life Standard Screen						Acute Aquatic Life Standard Screen					
Parameter	Basis	Units	Sample Count	Detection Frequency	Minimum Detected Value	Maximum Detected Value	Minimum Detection Limit	Maximum Detection Limit	Exceedance Flag	Count of Exceedances	Exceedance Frequency	Minimum Screening Level	Maximum Screening Level	Exceedance Locations	Exceedance Flag	Count of Exceedances	Exceedance Frequency	Minimum Screening Level	Maximum Screening Level	Exceedance Locations
Surface Water																				
Aluminum[a]	Total	mg/L	131	95%	0.003	57.8	0.0024	0.0059	Yes	61	47%	0.087	0.75	P1; P2; P2A; P4; P6; P7; P8; P9; P10, P12; P14; P14B; P15	Yes	59	45%	0.75	0.75	P1; P2; P2A; P4; P6; P7; P8; P9, P10; P14; P14B; P15
Arsenic	Dissolved	µg/L	130	13%	0.11	0.31	0.1	0.5	–	0	0%	150	150	–	–	0	0%	340	340	–
Cadmium	Dissolved	µg/L	130	53%	0.0054	0.863	0.005	0.25	Yes	12	9%	0.13	0.69	P2A; P11	–	0	0%	0.80	8.6	–
Chromium III	Dissolved	µg/L	130	8%	0.1	3.84	0.1	2.5	–	0	0%	34	251	–	–	0	0%	262	1932	–
Copper	Dissolved	µg/L	130	38%	0.1	7.4	0.1	2.5	–	0	0%	4.0	32	–	–	0	0%	5.5	54.7	–
Iron[b]	Total	µg/L	131	88%	11	107,000[b]	10	30	Yes	63	48%	1,000	1,000	P1; P2; P2A; P4; P6; P7; P8; P9; P10; P14; P14B, P15; P15B	–	–	–	–	–	–
Lead	Dissolved	µg/L	130	8%	0.051	0.775	0.05	0.25	–	0	0%	0.68	12.2	–	–	0	0%	22.7	312	–
Mercury	Dissolved	µg/L	130	1%	0.025	0.025	0.005	0.01	–	0	0%	0.77	0.77	–	–	0	0%	1.4	1.4	–
Nickel	Dissolved	µg/L	130	23%	0.52	7.16	0.5	2.5	–	0	0%	23.3	184	–	–	0	0%	210	1653	–
Selenium	Total	µg/L	131	72%	0.1	9.42	1	5	Yes	8	6%	5.0	5.0	P4	–	–	–	–	–	–
Silver	Dissolved	µg/L	130	1%	0.019	0.019	0.01	0.05	–	–	–	–	–	–	–	0	0%	0.63	42	–
Zinc	Dissolved	µg/L	130	42%	1.1	1810	1	15	Yes	6	5%	52.9	418	P2	Yes	6	5%	52	414	P2A

Notes:

Table includes water samples collected from September 2008 through October 2016.

Table includes normal samples only (does not include field replicates).

The dissolved sample collected at Station P16 (Bear Creek) on 8/4/2009 was excluded due to the use of an unconventional sample collection container.

– = indicates that screening value was not available or that a value was not calculated.

[a] Where the pH is greater than or equal to 7.0 and the hardness is greater than or equal to 50 ppm as $CaCO_3$, the chronic aluminum standard will then be equal to the acute aluminum standard, 750 µg/L as total recoverable aluminum.

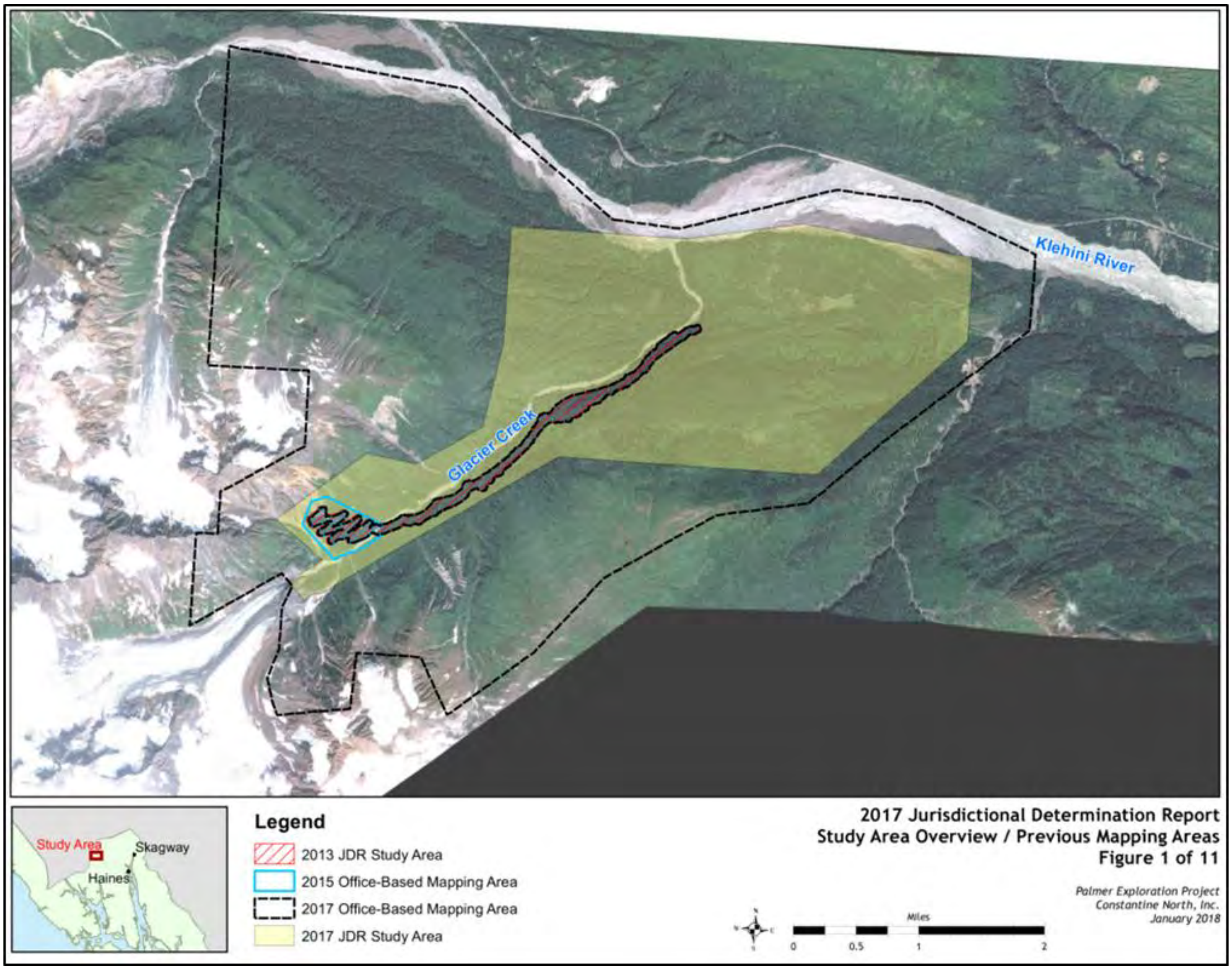
[b] Total (unfiltered) iron concentrations of 107,000 µg/L were reported for stations P1 and P7 for the June 2011 sampling event. These results have been confirmed by the laboratory, however, the results are suspect because they are much higher than any other iron concentrations measured at any station, and because the results are identical for the two stations. The next-highest total iron value reported was 58,900 µg/L from station P7 in September 2014.

Source: Integral (2016)

20.1.3 Wetlands Surveys

Constantine engaged consultant HDR, Inc. to perform wetlands delineation work in 2013 including mapping approximately 233 acres of land along a corridor for the proposed Glacier Creek access road. That segment of the road was constructed in 2016 to 2018. In 2014 and 2017, HDR also completed an office-based wetland mapping effort that focused on an area comprising 12,800 acres. In 2018, HDR prepared a wetlands jurisdictional determination report encompassing certain areas included in the previous studies and new areas surrounding them for a total 2017 study area of 4,580 acres (HDR, 2018). The wetlands work performed to date covers all the areas that are part of Constantine's proposed underground exploration program and those areas that would host the potential mine facilities contemplated in this PEA (Figure 20-3).

Figure 20-3: Wetlands Study Coverage Area



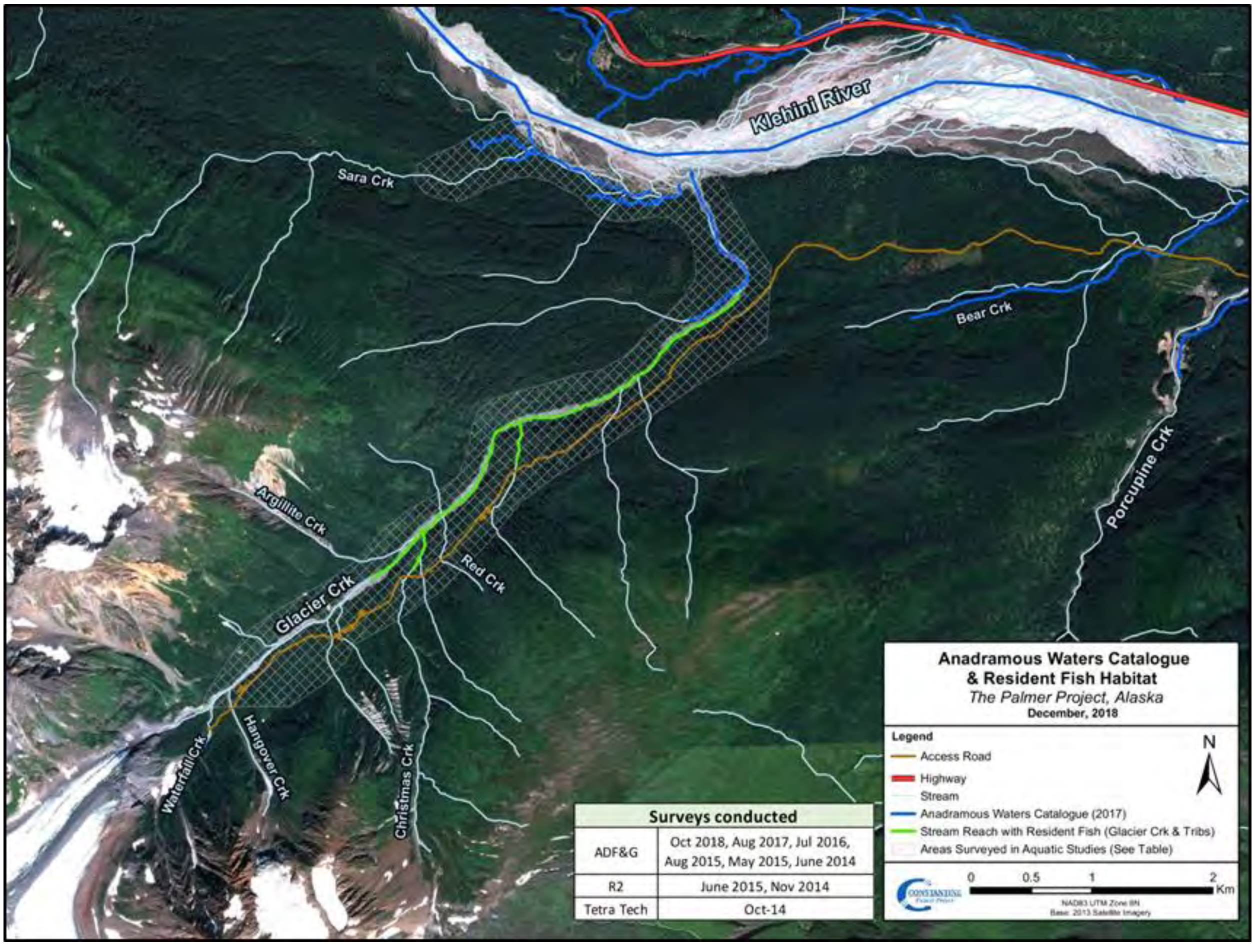
Source: HDR (2018)

20.1.4 Aquatic Life Surveys

Constantine initiated aquatic surveys in the Palmer project area using consultant Tetra Tech in 2013. Tetra Tech performed fish presence surveys in tributaries along the southeast side of Glacier Creek as part of planning for the Glacier Creek access road. The road has since been constructed along Glacier Creek on Federal Bureau of Land Management (BLM) lands and Mental Health Trust (MHT) lands. The final portion of the road, terminating at the portal site, is under construction at the time of this writing, as authorized by the MHT Plan of Operations Approval in April 2018. No species of salmon were recorded during sampling efforts on Glacier Creek or any of the 15 tributaries to Glacier Creek that were surveyed (Tetra Tech, 2013).

In 2014, the Alaska Department of Fish and Game (ADF&G) -Habitat Division performed fish studies along the proposed Glacier Creek access road alignment. They identified 23 drainages that cross the road alignment, including ephemeral and perennial streams, none of which were documented to contain anadromous fish. They did identify Dolly Varden trout in three of these streams but did not identify any fish at or above the road alignment. Because of that work, ADF&G determined that fish habitat permits were not required for any of the proposed stream crossings (ADFG, 2014) along the proposed road alignment. In addition, ADF&G made a formal submission to modify the Alaska Anadromous Fish Catalog, by moving the upstream extent of Coho salmon presence downstream on Glacier Creek, to an unnamed stream below the washed-out bridge on Porcupine Road. Figure 20-4 illustrates the extent of resident fish and the modifications to the catalog for anadromous fish along lower Glacier Creek.

Figure 20-4: Aquatic Life Survey Area



Source: Constantine (2018)

ADF&G made trips to the site again in 2015, 2016, 2017 and 2018 for the purposes of furthering their survey work. ADF&G published reports titled Glacier Creek Aquatic Studies for their work in 2016 and 2017 that are available online at: http://www.adfg.alaska.gov/index.cfm?adfg=habitat_publications.main

20.1.5 Hydrogeology Monitoring, Testing and Modeling

Constantine has performed several hydrogeology tests and computer modeling to estimate the seepage water inflow rates into a proposed underground exploration ramp as discussed below. In addition, it has been monitoring seasonal fluctuations in the groundwater levels (water table) in several holes.

20.1.5.1 Hydrogeology Testing

Beginning in 2016, Constantine performed a series of hydrogeology tests in drill holes including isolated interval tests (packer tests) and flow / shut-in tests.

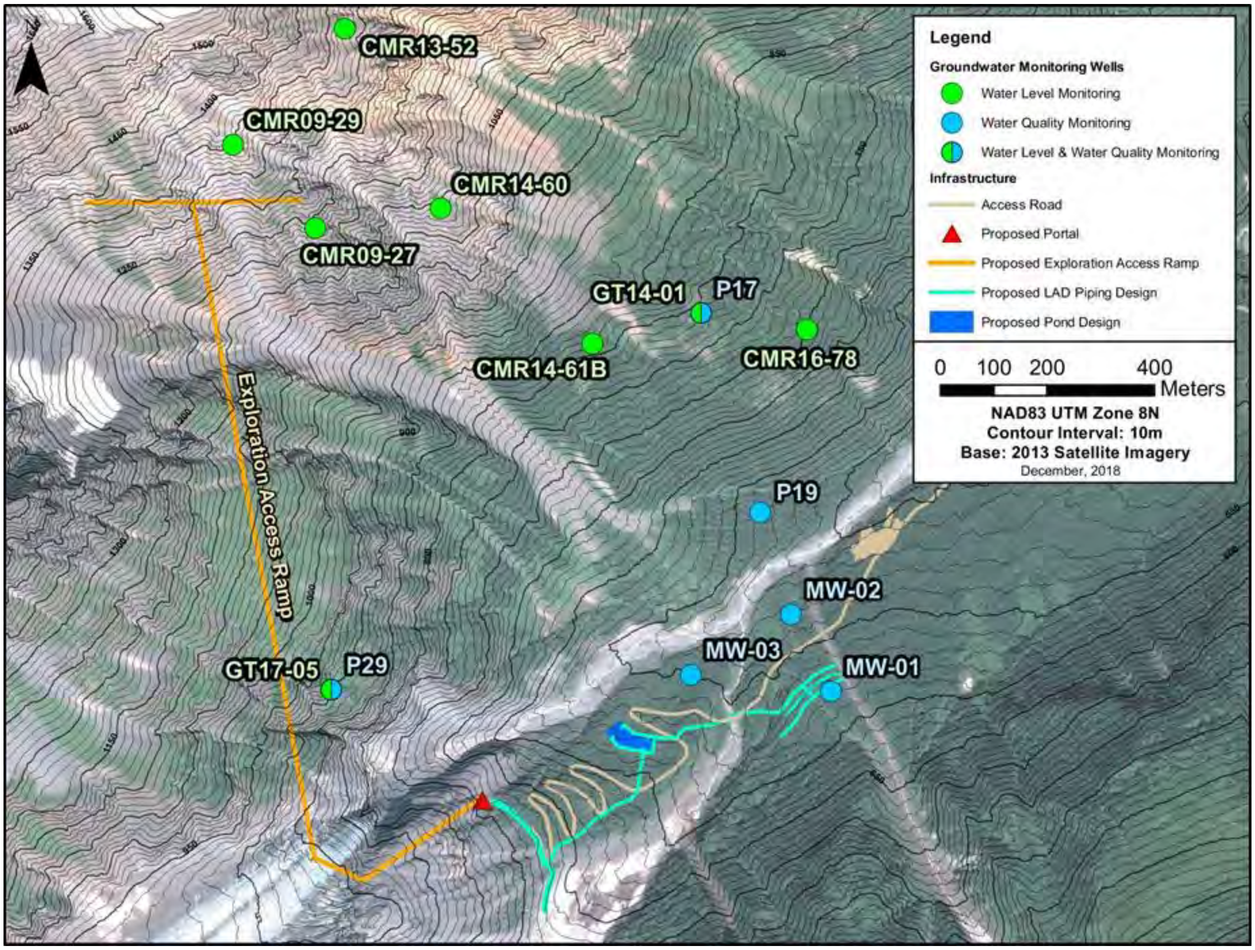
Based on the results of the 2016 program, Tundra (2017) concluded that groundwater flow is largely vertical, as a function of the steep character of bedrock faulting. Groundwater flow along the faults varies seasonally. The work also suggested that the reservoir capacity of the faults is low and that after an initial inflow of underground seepage water from these structures, they should drain down relatively quickly, and base flow rates may be relatively low. The 2016 program was focused on a proposed exploration ramp alignment option on the north side of Glacier Creek Valley that has since been dropped from further consideration due to geotechnical considerations but the general characterizations of the rock mass and controls on hydrogeology remain valid.

In 2017, the hydrogeology program re-focused on a different proposed exploration ramp alignment near the terminus of Saksaia Glacier on the south side of Glacier Creek. That program included 19 packer tests and one 52-day flow/shut-in test. The program is described by Tundra Consulting (2018). The packer tests were conducted in three drillholes located near the alignment of the proposed underground ramp and provide high-quality hydrogeology data for the Jasper Mountain Basalt. Three additional packer tests were performed in a fourth drillhole with one test each in the Hanging Wall Basalt, the SW mineralized zone, and in the footwall schist. Hydraulic conductivity (K) values from the packer tests ranged from 5.51×10^{-6} meters per day (m/d) to 7.10×10^{-1} m/d for the Jasper Mountain Basalt and a single value of 6.35×10^{-1} m/d for the Hanging Wall Basalt. Data analysis indicates that the Jasper Mountain Basalt can be subdivided into two hydraulic units; a shallow unit (less than 110 m below the ground surface) that has an average K of 0.102 m/d, and the remainder of the Jasper Mountain Basalt which has a K of 4.34×10^{-4} m/d.

20.1.5.2 Groundwater Level Monitoring

Constantine has measured groundwater levels in ten drillholes (Figure 20-5) using pressure transducers and continues to do so in eight of these as listed in Table 20-2. The original wells have more than a three-year water level record.

Figure 20-5: Groundwater Monitoring Location Map



Source: Constantine (2018)

Table 20-2: List of Groundwater Level Monitoring Wells

Hole ID	Pad	Elevation (mamsl)	Azimuth	Dip	Monitoring Start	Monitoring End	Period (yr)	Status
CMR09-27	Long	1194	337	-48	7-Oct-14	na	3.3	Active
CMR09-29	JP	1358	340	-53	7-Oct-14	na	3.3	Active
CMR13-52	Stryker	1323	153	-72	13-Jul-15	na	2.6	Active
CMR14-58	Green	1258	342	-60	6-Oct-14	1-Jun-15	0.7	Dropped
CMR14-60	Marmot	1096	317	-68	7-Oct-14	na	3.3	Active
CMR14-61B	Brazil	820	0	-50	6-Oct-14	na	3.3	Active
CMR15-76	Pocket	585	0	-46	30-Jun-16	12-Sep-16	0.2	Dropped
CMR16-78	Taz	701	359	-51	23-Aug-16	na	1.4	Active
GT14-01	U6	793	280	-5	na	na	0.0	Active
GT17-05	Hari	898	334	-15	7-Sep-17	na	0.4	Active

Source: Constantine (2018)

The groundwater level monitoring well data are evaluated on a two-year cycle. The last full evaluation was in 2017 (Tundra 2018). Findings from water level monitoring include:

- The piezometric surface (upper surface of groundwater) is irregular, but generally parallels the ground surface. It is deepest at high elevations and closer to the surface at lower elevations and on steep hillsides.
- The groundwater levels show a pronounced seasonality with high and variable water levels in the summer, a steady drop in water levels starting in early winter, very low levels in late winter, and rapidly rising levels in the spring. These water levels correspond to recharge patterns including; unrestricted summer recharge, freeze-up and snow accumulation in the early winter, and rapid snow melt in the spring.
- During the summer, the water levels in the wells have broadly correlative highs and lows that only generally correspond to recharge. Summer water-level patterns correspond poorly between wells in detail, however, suggesting that multiple factors control observed water levels in the summer including recharge, structure, location (dominantly elevation), proximity to glaciers and permanent snowfields, and well construction.
- The seasonal pattern seen in the monitoring well hydrographs group by elevation.
- Wells located at higher elevations show an extreme seasonal range with over 37 m of drawdown in the winter, and high and variable summer water levels. This pattern suggests filling and draining of fracture systems.
- Mid-elevation wells show moderate seasonal variation and small variation in the summer water levels. The moderately low-elevation wells also appear to group and have a different pattern than the other wells, but the period of record is too short to draw inferences currently.

These data suggest that Constantine should anticipate higher seepage inflows underground during spring thaw, potentially during intervals of high rainfall, but that seepage should be significantly lower during the winter when recharge rates are lowest.

20.1.5.3 Groundwater Modeling

Tundra (2018) used a transient analytical flow model to estimate natural groundwater inflows into the proposed exploration ramp. The method incrementally estimates flow as the ramp is advanced. The analysis is most sensitive to K and pressure head above the ramp (i.e., saturated thickness above the ramp) and is less sensitive to the storage capacity of the rocks and the ramp radius. Sufficient data are available to estimate inflows for the first 1,250 m of the ramp (Jasper Mountain Basalt) which would take almost a year to excavate. The estimated flow for this portion of the ramp would peak at approximately 200 gpm and settle at a sustained rate of approximately 160 gpm during the first year of ramp development. It should be noted that short-term higher flow rates will likely occur from faults and fracture zones. Insufficient data are currently available to perform a flow estimation for the remainder of the ramp. However, based on the hydrogeological model and the available data for the Hanging Wall Basalt, Tundra (2018) suggests that a high flow rate is expected for the remainder of the ramp.

Constantine should monitor the quantity of underground seepage water inflows underground as part of the ongoing monitoring and this data will be used to inform future mine planning.

20.1.6 Cultural Resources Data

Archeological surveys have been performed within areas of potential disturbance by exploration activities in 2014 and 2017 by consultant Northern Land Use Research Alaska (NLURA). NLURA refers to these areas as Areas of Potential Affect (APE). No cultural sites or artifacts were identified in the 2014 and 2017 pedestrian surveys. In both surveys NLURA concluded that “no cultural resources were encountered, and the likelihood of buried (unknown) cultural features is low, NLURA recommends a finding of no historic properties affected (36 CFR 800.4(d)(1)). In our opinion, no further fieldwork is required in advance for this proposed project.” These surveys were performed under the permit authority of the Office of History and Archaeology/State Historic Preservation Office. Additional surveys may be required to cover areas contemplated for future mine development including areas slated for siting of the major mine facilities in this PEA. There are five historic sites within approximately five miles of the project area, two of which are listed on the National Register of Historic Places (NRHP) and another site (SKG-00026) is considered eligible for inclusion on the NRHP. Four of these sites relate to activities associated with the Klondike gold rush (turn of the 20th century), the construction of the Haines-Fairbanks pipeline (1950’s), and WWII era road construction. The fifth is a historic campsite that, "from time immemorial been an important place for the Natives of Klukwan. It was the highest campsite to which the Natives could get by canoe" located at confluence of Porcupine Creek with Klehini River (NLURA, 2015).

20.1.7 Wildlife, Endangered Species, Migratory Birds, and Bald and Golden Eagle Protection

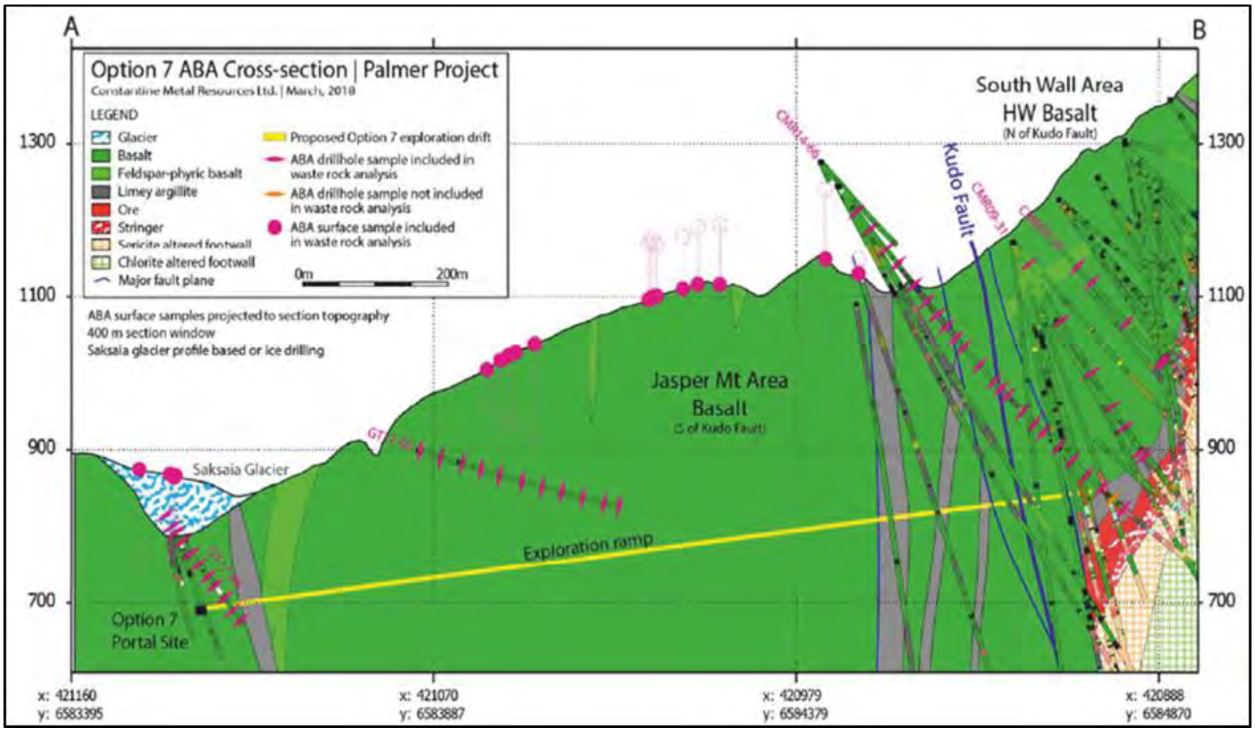
Wildlife and Terrestrial studies were initiated by consultant Hemmera in 2014 and performed seasonally through 2018. Wildlife habitat mapping and assessment for suitability for wildlife species of interest was done and resident species of interest were identified. Bird surveys (songbirds and birds of prey) were also completed. Mountain Goat populations in the project area are seasonally surveyed via fixed-wing aircraft (since 2014). Incidental wildlife observations are reported by Palmer Project employees using digital geo-referenced reporting forms. Constantine personnel have identified the following wildlife in the area: black and brown bear, mountain goat, coyote, wolf, red fox, moose, Steller’s Jay, Rock Ptarmigan, Belted Kingfisher, Golden Eagle, Red-tailed Hawk, Hoary Marmot and ground squirrels. The data for these surveys is in the form of trip reports and summaries for the goat surveys, a baseline summary report and a digital file of incidental wildlife observations. Constantine intends to continue with seasonal goat surveys. Constantine also has developed an invasive species management plan for the BLM (developed for the first portion of the Glacier Creek access road which is on BLM land) and the entire project will benefit from its implementation regardless of land ownership. Golden eagles have been observed in Glacier Creek and their nests have been identified but are outside the area of current activities. As part of permitting any future mining project, the USD&F will be consulted with regard to protection of migratory birds and Golden and Bald eagles

20.1.8 Acid Base Accounting Data

Constantine initiated acid base accounting studies in 2014. The purpose of the program was to characterize the various rock types at Palmer in terms of their capacity to generate acid rock drainage and/or metals leaching (ARD/ML) into the environment if they were subjected to the surface weathering environment. The study was expanded in 2017 to include additional core and surface outcrop samples that are representative of the rock types that Constantine anticipates intersecting with the proposed exploration ramp. It is important to understand that much of this work was directed specifically at rocks that would be encountered during an underground exploration program and did not fully consider the character of waste rock and tailings that would be encountered under a future mining scenario. Additional studies will be required to expand the characterization of wastes.

To date, 101 rock samples have been collected from drill core and surface outcrops that are interpreted as representative of the rock units that the proposed exploration ramp would pass through (see Figure 20-6) ARD/ML study work was performed by pHase Geochemistry and the final report was completed in 2018 and included an assessment of 101 samples that are representative of the lithologies that are going to be intersected by the proposed underground ramp.

Figure 20-6: ABA Results by Rock Type for Planned Exploration Ramp Representative Sample



Source: Constantine (2018)

The ARD/ML studies included acid base accounting (ABA), which includes geochemical analyses that help define the constituents of the rock samples and their relative abilities to generate and/or neutralize acid in the weathering environment. ABA analyses were followed with several kinetic tests (humidity cell tests) that mimic the weathering environment and the resultant leachate is analyzed for mobilized metals and pH. A third aspect of the ARD/ML study is the determination of the mineralogy of the samples. Finally, four (4) barrel tests, which are a form of long-term kinetic tests, were initiated at the Palmer site in 2017.

Acid base accounting (ABA) results for the 101 rock samples are summarized in Table 20-3. Buffering capacity, or neutralizing potential (modified NP), of the 101 samples is generally high with a range of 6 to 651 kg CaCo3/t. The limey argillites had the highest modified NP. Nearly all the neutralizing potential (NP) is contributed by carbonate minerals as indicated by a strong correlation between modified NP and carbonate NP. Trace amounts of iron carbonate were visually logged and a minor amount of ankerite was identified in the limey argillite. Iron carbonate minerals are not net neutralizing under oxygenated conditions.

However, calcite is predominant in those samples and contributes to the overall neutralizing potential. Screening criteria as provided by the Mine Environment Neutral Discharge Program (MEND, 2009) guidelines and the Global Acid Rock Drainage (GARD) Guide (INAP, 2009) were adopted in this assessment whereby a sample is considered:

- Potentially acid generating (PAG) if acid neutralization potential ratios (NPR) are < 1,
- Non-potentially acid generating (non-PAG) if NPR is > 2, and
- Uncertain (UC) if NPR is between 1 and 2.

Table 20-3: ABA Results by Rock Type for Exploration Ramp Representative Samples

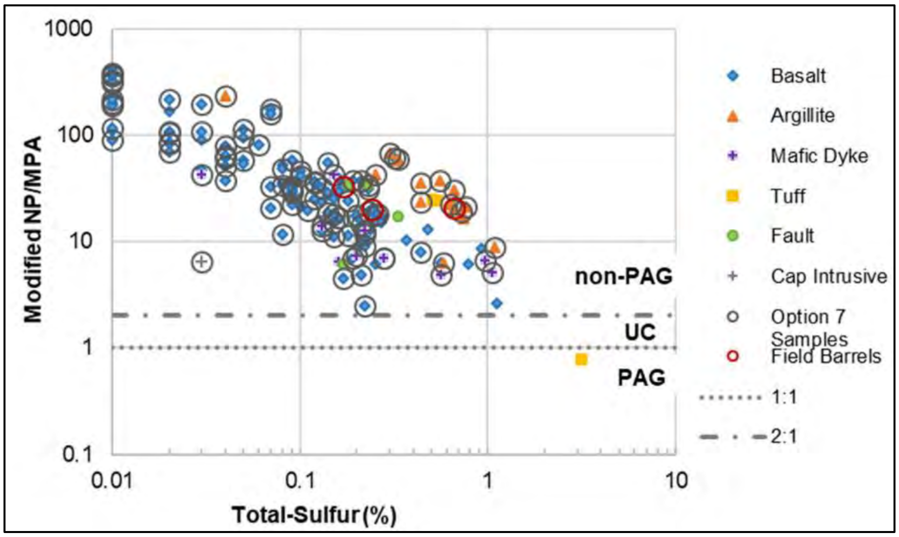
Rock Type	Statistic	Paste pH	Total S	Sulfate S	Sulfide S	MPA	Modified NP	CO_3 NP	NNP	NPR
			wt. %			kg$CaCO_3$/t				
All Rock (n = 101)	Min	7.5	0.01	0.01	0.01	0.3	6	4	5	2.5
	Median	8.8	0.13	0.01	0.12	4	100	89	96	33
	Max	9.8	1.09	0.19	1.05	34	651	647	634	381
Jasper Mtn Basalt (n=38)	Min	8.1	0.01	0.01	0.01	0.3	17	4	10	2.5
	Median	8.8	0.12	0.01	0.11	4	93	78	88	31
	Max	9.2	0.32	0.19	0.26	10	617	622	607	219
Limey Argillite (n = 14)	Min	7.5	0.04	0.01	0.03	1	114	110	96	6.3
	Median	8.6	0.57	0.02	0.55	18	435	457	414	27
	Max	8.9	1.09	0.04	1.05	34	651	647	634	235
HW Basalt (n=37)	Min	8.0	0.01	0.01	0.01	0.3	28	13	28	7.9
	Median	8.8	0.05	0.01	0.04	2	91	82	89	80
	Max	9.7	0.44	0.03	0.41	14	381	381	379	381
Mafic Dyke (n = 8)	Min	8.2	0.13	0.01	0.12	4	46	37	40	4.9
	Median	9.0	0.28	0.01	0.28	9	74	62	61	7.0
	Max	9.8	1.06	0.01	1.05	33	201	211	196	43
Gabbro (n = 2)	Min	8.8	0.03	0.01	0.03	1	40	26	39	13
	Max	9.0	0.22	0.01	0.21	7	88	74	81	43
Fault (n = 1)		8.4	0.23	0.01	0.23	7	245	237	238	34
Cap Intrusive (n=1)		8.9	0.03	0.02	0.01	1	6	4	5	6

Source: pHase (2018)

The ABA data indicate that the lithologies that Constantine will intercept in the proposed exploration ramp can all be classified as non-PAG, with acid neutralization potential (NPR) ratios ranging from 2.5 to 381 with a median of 3, an indication of excess neutralizing capacity in the samples.

As shown in Figure 20-7, the ABA results indicate that all the samples classify as non-PAG (pHase, 2018).

Figure 20-7: Classification of Samples Representative of UG Exploration Drift Development Rock



Source: pHase (2018)

20.1.9 ASTM Phase I Environmental Site Assessment

Stantec Consulting Services, Inc. ("Stantec") completed an ASTM Phase I Environmental Site Assessment (ESA) on November 11th, 2015. The ESA was conducted to identify if there are historical and/or current, potential on-site and/or nearby, off-site environmental concerns that might pose possible impact to the Project. Stantec performed the ESA in conformance with the scope and limitations of ASTM Practice E1527-13.

In summary, Stantec did not identify any on-site or off-site Recognized Environmental Concerns that might impact the Project detrimentally (Stantec, 2015). Stantec's analysis of data and information reviewed as part of the ESA, as well as visual observations during the site visit, did not identify any indications of stockpiled, waste rock or similar, prospect-related material, nor released hazardous or petroleum product constituents, in any quantity deemed of potential environmental concern.

20.1.10 Additional Baseline Data requirements

For the current stage of the Palmer exploration program, Constantine has initiated a robust environmental baseline program and is addressing the key needs for current permitting and for some aspects of future mine permitting. As described above Constantine will likely need to expand some aspects of the baseline work as future development plans are better defined. This would include a broader ARD/ML program to characterize waste rock and tailings waste streams, and further evaluation of the potential need to treat and release mine drainage water.

20.2 Permitting

20.2.1 Exploration Permits

Exploration permit-types are dictated by land ownership which is varied in the Project area. Constantine performs mineral exploration at the Palmer Project under State of Alaska, Mental Health Trust Land Office, and Federal (BLM) permits as described below.

20.2.1.1 Surface Exploration Program

Constantine has been performing surface exploration consisting of surface mapping and sampling, geophysics and diamond drilling on State, Mental Health Land Trust (Trust) and Federal land since 2008. Constantine controls the mineral rights under these surface estates through their ownership of state mining claims (State manages both surface and mineral estates), federal mining claims (Federal government manages the surface and mineral estates) and through an Upland Mining Lease with the Trust where the Trust owns the mineral estate.

The Federal Bureau of Land Management (BLM) authorizes surface exploration activities where the federal government owns the surface estate (federal mining claims). The State of Alaska authorizes exploration activities where the state owns the surface estate (state mining claims and Trust split estate where Trust only owns the mineral estate) and the Mental Health Land Trust authorizes surface exploration where the Trust owns the surface estate.

The BLM authorized surface exploration activities on federal mining claims through a Record of Decision (ROD) approving the Plan of Operations on August 18th, 2016 (Case file AA-094088). The BLM also authorized a modification of the Plan of Operations through another ROD on September 21st, 2017. These ROD's approve all the surface exploration activities (mostly core drilling) planned for the federal claims for the next few years. A lawsuit was filed in 2017 by SEACC et al. against the BLM for their 2016 and 2017 Palmer Project Plan of Operations approvals, claiming the BLM failed to consider the future impacts of mine development before approving the exploration plans. Other parties in the suit are Lynn Canal Conservation, Rivers without Borders, and the Chilkat Indian Village of Klukwan. In March 2019, the lawsuit was decided in favor of the BLM, upholding their NEPA analysis and approval of exploration work, however an appeal was filed by SEACC et al. and is pending. Of note, the access road construction authorized under BLM's 2016 and 2017 Plan of Operations approvals is now complete and the ongoing litigation does not impede exploration activities.

Exploration on State of Alaska mining claims and Mental Health Trust lands where the Trust only owns the mineral estate (State retains surface ownership) is authorized under State of Alaska Miscellaneous Land Use Permit - APMA #5690, which was renewed in 2019 for another five years with a current expiration date of December 31st, 2023.

On Mental Health Trust lands where the Trust owns the surface and subsurface estates, surface and underground exploration activities are authorized under Upland Mining Lease #9100759 issued by the Trust. The lease has an effective date of September 1st, 2014 and a three-year term with two extensions of three years (for a total of nine years). The lease can be further extended indefinitely as long as the project is being advanced. Under the terms of the lease Constantine is required to submit a Plan of Operations to the Trust annually for approval. On lands where the Trust only owns the mineral estate, and the State of Alaska retains ownership of the surface estate, then the Trust Plan of Operations Approval would only approve underground activities and the State Miscellaneous Land Use Permit would authorize the surface activities.

Surface exploration activities require use of some surface water for drilling. Constantine is in possession of several Temporary Water Use Authorizations (TWUA) issued by Alaska Department of Natural Resources' (ADNR) Water Section. Constantine currently has authorization for designated water source locations under two temporary water use authorizations - TWUA F2014-102 and F2015-080 to support potential drilling activities at various locations across the property. TWUA F2014-102 expired on December 31st, 2018; F2015-080 expires on July 13th, 2020. The TWUA's contain 27 conditions that must be complied with, including conditions designed to protect water quality and aquatic resources.

20.2.1.2 Advanced Underground Exploration Program

Constantine is permitting an advanced underground program in two Phases. Phase I includes development of the access road and other surface civil construction in preparation of facility construction which will follow. All these activities are on Alaska Mental Health Trust Authority (MHT) lands. The Phase I Plan of Operations was approved by the MHT in April 2017. The associated Reclamation Plan and was approved by ADNR in May 2017.

The Phase II Plan of Operations and associated Reclamation Plan were submitted to the MHT and ADNR in December 2018 and approvals are anticipated by May 2019. Constantine also submitted the engineered design drawings for a water management and disposal system to ADEC in December 2018 for their review and approval. The Phase II Plan of Operations will approve construction of facilities, developing the underground exploration ramp and executing the underground exploration drilling program.

The schedule would have Constantine receiving all Phase II exploration approvals by May 2019 and initiating surface facility construction then and collaring the portal shortly thereafter. The proposed underground exploration ramp will take a bit over one year to develop.

20.2.2 Major Mine Permits

The following discussion identifies the major permits and approvals that will likely be required for the Palmer Project to be developed into an operating mine.

The types of major mine permits required by this project are largely driven by the underlying landownership; regulatory authorities vary depending on land ownership. The Palmer Project area includes:

- Federal land administered by the BLM - controlled by Constantine through federal mining claims.
- State of Alaska land administered by the Alaska Department of Natural Resources – controlled by Constantine through state mining claims.
- State of Alaska land administered by the Alaska Mental Health Trust Authority – controlled by Constantine through mining with the Trust.

The present facility layout shows that the 540 Conveyor and the 540 Access Portals, the mill and power plant facilities will be located on BLM lands while the tails dewatering plant and tailings management facility will be located on State of Alaska land. The 680 Portal will be located on MHT land.

A list of likely major mine permits required to develop the mine is included in Table 20-4.

Table 20-4: Major Mine Permits That May Be Required for the Palmer Project

Agency	Authorization
State of Alaska	
ADNR	Plan of Operations Approval (including Reclamation Plan) for Activities on State Land
	Water Rights Permit to Appropriate Water
MHT	Plan of Operations Approval for Activities on MHT lands
ADEC	APDES Water Discharge Permit
	Alaska Multi-Sector General Permit (MSGP) for Storm Water
	Section 401 Water Quality Certification of the Clean Water Act (CWA) Section 404 Permit
	Integrated Waste Management Permit
	Air Quality Emission Control – Construction Permit
	Air Quality Emission Control – Title V Operating Permit
Federal Government	
EPA	Spill Prevention, Control, and Countermeasure (SPCC) Plan (fuel transport and storage)
USACE	CWA Section 404 Dredge and Fill Permit – Note only if placement in Waters of the US
BLM	Plan of Operations Approval (including Reclamation Plan)

Source: Jack DiMarchi (2019)

Under the scenario contemplated in this technical report it is likely that Constantine would develop a single Plan of Operations for the entire mining project and that this could be used by the MHT, ADNR, ADEC and BLM to authorize the activities that occur on their respective lands or the activities they are authorized to manage across land ownership lines.

The BLM would be authorizing all surface activities on their lands including reclamation of the land. ADNR would be authorizing all activities on their land including reclamation of their land and MHT and BLM lands. The MHT would be approving all activities on their land. ADEC would be authorizing all waste management activities (i.e. waste rock and tailings management), any point source water discharges, reclamation of these waste management facilities and any long-term water treatment plans, regardless of land ownership.

The U.S. Army Corps of Engineers (USACE) would require a Clean Water Act Section 404 permit for dredging and filling activities in Water of the United States (WOTUS) including jurisdictional wetlands. To minimize environmental impacts the PEA mine design deliberately located facilities to avoid WOTUS. A 404 permit may not be required.

The USACE Section 404 permitting action and the BLM's decision to approve the Plan of Operations would require both agencies to comply with the National Environmental Policies Act (NEPA) and, for a project of this magnitude, the development of an environmental impact statement (EIS). The BLM would likely be the lead federal agency for the NEPA process. The NEPA process will require an assessment of direct, indirect and cumulative impacts of the Palmer Project and the identification of project alternatives, and include consultation and coordination with additional federal agencies, such as the USF&W and National Marine Fisheries Service, and with the State Historic Preservation Office and Tribal Governments under Section 106 of the Historical and Cultural Resources Protection Act.

As part of the Section 404 permitting process, the Palmer Project will have to meet USACE wetlands guidelines to avoid, minimize and mitigate impacts to wetlands. The USACE may require Constantine to develop a compensatory wetlands mitigation plan for mitigating unavoidable wetlands impacts. The USACE is required to identify the least environmental damaging practicable alternative (LEDPA) for the project.

The overall timeline required for permitting would be largely driven by the time required for the NEPA process, which is triggered by the submission of the 404-permit application to the USACE or the Plan of Operations submittal to the BLM. The timeline includes the development and publication of a draft and final EIS and ends with a Record of Decision, and 404-permit issuance. In Alaska, the EIS and other state and federal permitting processes are generally coordinated so that permitting and environmental reviews occurs in parallel. The NEPA process could require between 1.5 to 3 years to complete and could potentially take longer.

“Major” permits generally define critical permitting path. Additional “minor” permits are also required.

20.3 Social or Community Considerations

The nearest communities to the Project are Klukwan and Haines. Klukwan is a Chilkat Indian Village located approximately 15 road miles southeast of the Project and Haines is located 35 road miles southeast of the project. These two communities are likely to see the biggest effects from any future mine development at Palmer. Constantine has recognized the importance of community considerations since beginning exploration in 2006. Dedicated community relations staff and consultants have been engaged and a local office was opened in 2015. The Company plans to continue efforts in 2019 to further strengthen stakeholder communications and relations, with the goal of maximizing mutual benefits and finding solutions to any concerns as the project design advances.

A new mine at Palmer would bring some social and economic changes to both communities. The Palmer Project has the potential to significantly improve work opportunities for both communities. A new mine could generate approximately 260 direct full-time jobs plus the indirect jobs in the support and services industries. Regional engagement by Constantine has encountered a strong desire for the economic benefits that come with mining projects.

Constantine’s exploration activities have already had an economic impact on the communities. Constantine has hired locally since 2006 for the seasonal jobs including field and camp support, geological and environmental technicians, drill helpers, and road construction. These local employees comprise more than 50% of the Palmer Project workforce. Through wages and purchase of supplies Constantine estimates it contributed more than US$1.7 million directly to local Haines economy in 2018 from its exploration program. Annual economic input has been maintained at similar levels since 2013.

Constantine participates in local economic development organizations and job fairs to maximize business and employment opportunities. As part of their ongoing efforts, the Company conducts regular stakeholder meetings, maintains community outreach materials, hosts project site tours, attends and supports local programs and events, supports local hire and procurement, and participates in local community organizations.

The BLM conducted an Environmental Assessment of proposed exploration activities in 2015 (amended in 2017). The Assessment included public comment, government-to-government consultation with local tribes, and an analysis of potential effects of project activities on key social and environmental factors. The 2015 and 2017 Environmental Assessments both resulted in a 'Finding of No Significant Impacts' (FONSI). A NEPA analysis for a mine will examine larger impacts than those for an exploration program.

In 2018, Constantine donated over US$12,000 to support local community and educational programs, events, charities and service organizations. It also provided comprehensive project site tours for approximately 50 community members including local community leaders and elected officials, Tribal staff and Council members, area users and residents, environmental NGO members, and students. Donations and site tours have been maintained at similar levels since 2013.

In 2017, Rain Forest Data generated a memo called The Economic Impact of the Mining Industry in Haines Alaska, 2016. They conclude that the seventh largest wage providing industry in the Haines community is the mining sector, accounting for 8% of all workforce earnings in the Haines Borough. In 2016, mine workers in Haines directly earned $3.4 million. Mining represents 4% of all Southeast Alaska wages – which means mining is twice as important in Haines as an overall driver of the economy as it is for the region as a whole.

These Haines mining jobs are commuter miners working at the Greens Creek and Kensington mines (located 80 and 32 miles south of Haines, respectively), road builders, quarry workers and placer miners, as well as employees on the Palmer exploration program.

In general terms, rural Alaska residents are often concerned about potential mining impacts to wildlife and fish for those projects within their traditional use areas. Constantine acknowledged these concerns and is taking substantive steps to address them during the current exploration stage of the Project.

Local community concerns will also be formally recognized during the development of the Project Environmental Impact Statement (EIS). Early in the EIS process, the lead federal permitting agency will hold scoping meetings to hear and record the concerns of the local communities so that the more significant of these concerns can be addressed during the development of the EIS. In addition, the lead federal agency would have government-to-government consultations with the Tribal Councils in each of the villages, as part of the EIS process, to discuss the project and hear Council concerns.

20.4 Mine Reclamation and Closure

20.4.1 Reclamation and Closure Plan

Mine reclamation and closure are largely driven by state regulations (11 AAC 86.150, 11 AAC 97.100-910, and 18 AAC 70) and statutes (AS 27.19) and Federal Regulations Title 43, Subtitle B, Chapter 11, Subpart C, Part 3800. A detailed reclamation plan will be submitted to the state and federal agencies for review and approval in the future, during the formal mine permitting process. The approval process for the plan varies somewhat depending on the land status for any mine. Because the Palmer Project is likely to have facilities on a combination of federal, state and MHT lands, it is likely that the Reclamation Plan will be submitted as part of the Plan of Operations and may be subject to review and approval from more than one agency.

20.4.2 Reclamation and Closure Financial Assurance

There are two State of Alaska agencies and one federal agency that require financial assurance in conjunction with approval and issuance of large mine permits.

The ADNR requires a reclamation plan be submitted prior to mine development and requires financial assurance, typically prior to construction, to assure reclamation of the site. The ADEC requires financial assurance both during and after operations, and to cover short and long-term water treatment if necessary, as well as reclamation costs, monitoring, and maintenance needs. The State requires that the financial assurance amount also include the property holding costs for a one-year period.

The BLM has its own requirements for reclamation financial assurance which is generally aligned with the State's needs and a single reclamation plan and reclamation cost estimate is generally prepared in a manner that it meets the regulatory requirements of all agencies, eliminating the need for multiple financial assurance instruments. There is precedence for the BLM and State to rely on a single financial assurance to meet the requirements of both agencies.

The final financial assurance amount will be calculated through the process of reviewing and approving the Palmer Project reclamation plan during the formal permitting process. In general, the approach is to combine the reclamation costs, post-closure monitoring costs and the long-term annual water treatment costs (if any) into a financial amount that includes deriving the NPV of the long-term costs and combining that with the reclamation cost.

Constantine may satisfy the state financial assurance requirement by providing any of the following:

1. A surety bond;
2. A letter of credit;
3. A certificate of deposit;
4. A corporate guarantee that meets the financial tests set in regulation by the ADNR commissioner;
5. Payments and deposits into the trust fund established in AS 37.14.800; or
6. For the TSF or ADEC-related obligation - any other form of financial assurance that meets the financial test or other conditions set in regulation by the ADNR or ADEC commissioners and the BLM.

The adequacy of the reclamation plan, and the amount of the financial assurance, are reviewed by the state and federal agencies at a minimum of every five years and may be reviewed whenever there is a significant change to the mine operations, or other costs that could affect the reclamation plan costs.

Since no reclamation plan was developed for this PEA, no reclamation cost estimate was calculated. However, it is not unreasonable to make a comparison with the Greens Creek Mine located near Juneau Alaska about 97 miles (156 km) south of the Palmer Project. Greens Creek is an underground Pb-Zn-Ag-Au mine that mines at a rate of approximately 3,000 t/d, utilizes flotation, produces and ships concentrate and disposes of its tailings in a dry stack facility. Unlike Palmer, the mine has a recognized need for long-term water treatment. The 2014 cost estimate for the Greens Creek mine reclamation, excluding long-term water treatment, is $75.2 M. In the absence of a reclamation cost estimate specific to the Palmer project the 2014, estimated costs for the Greens Creek mine is reasonable basis to use to estimate closure costs for the Palmer PEA.

For the purposes of this PEA, the Greens Creek direct cost items have been summarized by area and scaled according to the projected TSF and surface building footprints. The summary of the closure estimate is shown in Table 20-5. Indirect costs have been derived by scaling the direct costs using the factors shown in the table.

Table 20-5: Closure Cost Estimate for the Palmer Project (Based on Creek Closure Costs)

Closure Component	Greens Creek 2014	Inflation 2019	Greens Creek 2019	Scaling Criteria	GC	Palmer	Ratio	Palmer 2019 ($K)
Direct Costs								
Earthworks and Recontouring	19,988	6.1%	21,211	Area of Drystack	0.25	0.13	52%	11,030
Revegitation and stabilization	222	6.1%	236	Area of Drystack	0.25	0.13	52%	123
Water Treatment	22,059	6.1%	23,409	no LT H2O treatment	100%	5%	5.0%	1,170
Equipment and Facility Removal	3,846	6.1%	4,081	Area of Main Buildings	8,029	5,286	78%	3,176
Monitoring	5,637	6.1%	5,982	no LT H2O treatment	100%	20%	20.0%	1,196
Construction Management and Support	351	6.1%	372	Factored	100%	100%	100%	372
Closure Planning	16,328	6.1%	17,327	Factored	100%	25%	25.0%	4,332
Total Direct Costs	**68,431**		**72,619**					**21,400**
Indirect Costs								
Enigneering Design and Construction	1,882	6.1%	1,997	% of indirects	2.75%	8.0%		1,712
Contingency	8,896	6.1%	9,440	% of indirects	13.0%	15.0%		3,210
Performance Bond	2,053	6.1%	2,179	% of indirects	3.0%	3.0%		642
Contractor Profit	10,607	6.1%	11,256	% of indirects	15.5%	10.0%		2,140
Contract Administrator	4,790	6.1%	5,083	% of indirects	7.0%	8.0%		1,712
Liability Insurance				% of indirects	0.0%	1.0%		214
Total Indirects	**28,228**		**29,956**			45.0%		**9,416**
Total Closure Costs	**96,659**	**6.1%**	**102,575**					**30,815**

Source: JDS (2019)

21 Capital and Operating Cost Estimates

21.1 Capital Costs

21.1.1 Capital Cost Summary

Table 21-1 presents the capital estimate summary for initial, sustaining, and closure capital cost with no escalation.

Table 21-1: Capital Cost Summary

Area	Pre-production ($M)	Sustaining ($M)	Closure ($M)	Total ($M)
Mining	55	108		163
Site Development	12	1		13
Mineral Processing	75	3		78
Tailings Management	2	3		5
On-Site Infrastructure	34	1		35
Off-Site Infrastructure	0	0		0
Project Indirects	26	0		26
EPCM	32	0		32
Owner Costs	8	0		8
Closure	0		31	31
Salvage Value	0		-6	-6
Subtotal	245	115	25	385
Contingency	33	0		33
Total CAPEX	278	115	25	418

Source: JDS (2019)

All capital cost are in 2019 US Dollars.

LOM project capital costs consist of the following distinct components:

- Pre-production Capital Development – includes all costs to develop the property to production, a mining and milling rate of approximately 2,712 tpd in Year 1. Initial capital costs are expensed over a 24-month pre-production construction and commissioning period
- Sustaining Capital Cost – 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 are expended in operating years 1 through 11 and
- Closure Cost - includes all costs related to the closure, reclamation, and ongoing monitoring of the mine, post operations. Closure costs are incurred in Year 12.

The capital cost estimate was compiled using a combination of quotations, database costs, and scaling factors.

21.1.2 Basis of Estimate

21.1.2.1 Key Estimate Parameters

Estimate Class: The capital cost estimates are considered Class 4 estimates (-20% / +30%). The overall project's definition is estimated to be 10%.

- Estimate Base Date: The base date of the estimate is April 2019. 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
- Currency: All capital costs are expressed in 2019 US Dollars ($US). Estimates in Canadian Dollars ($CAD) were converted to US Dollars at a rate of $1.00 CAD : $0.75 USD.

21.1.2.2 Key Assumptions

The following assumptions were made during the development of the capital estimates:

- Underground mine development activities will be performed by the Owner's work force
- All surface construction (civil, structural, architectural, mechanical, piping, electrical, and instrumentation) will be performed by contractors.

21.1.3 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.

21.1.4 Underground Mine CAPEX

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases, and by comparison to similar mines in Alaska and Western Canada. Table 21-2 summarizes the combined capital cost estimates for the two mines.

Table 21-2: Mining CAPEX Summary

Description	Unit	Initial	Sustaining	Total
UG Mobile Equipment Lease	US$M	5.5	50.7	56.2
UG Mobile Equipment Rebuilds	US$M		0.2	0.2
Fixed Equipment	US$M	21.9	21.1	43.0
Capital Lateral Development	US$M	15.0	29.2	44.1
Capital Vertical Development	US$M	0.8	6.4	7.2
Capital Period OPEX	US$M	12.3		12.3
Total	**US$M**	**55.4**	**107.6**	**163.1**

Source: JDS (2019)

21.1.4.1 Mobile Equipment and Replacement

Underground mining equipment quantities and costs were determined through buildup of the mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. Mobile equipment for the mine will be leased as required. Costs for the mobile equipment lease are a combination of the initial down payment and the lease payments. Leasing terms were based on the assumption of 20% down payment and an APR of 8% for 96 months. Equipment replacements and rebuilds have been calculated based on manufactures recommendations.

21.1.4.2 Fixed Equipment

Budgetary quotes or database cost were used for major infrastructure components including the underground crusher, ventilation fans, heaters, pumps, paste plant, conveyor system, and communications system. The cost of construction and/or installation of the various fixed equipment has been included.

21.1.4.3 Lateral and Vertical Capital Development

Lateral Development includes all main haulage ramps and accesses to ventilation and other raises. A total of 7,747 m of lateral and 190 m of vertical development will be driven during the pre-production period. An additional 10,961 m of lateral and 1,774 m of vertical development will be driven as sustaining capital during the operating years. Approximately 51% of all lateral development in the mine has been capitalized, the remaining 49% is captured in the operating cost.

All vertical development has been capitalized.

21.1.4.4 Capitalized Pre-production Operating Costs

Capitalized operating costs are defined as mine operating expenses (operating development, mine maintenance, and mine general cost) incurred prior to and during commissioning and cease at the commencement of commercial operations and generation of project revenues begins. Once commissioning is complete and the processing begins, these costs transition to operating expenses. During pre-production, 120,000 tonnes of mineralized rock is mined and stockpiled until it is processed in Year 1.

The bases of these cost are described in Section 22, Operating Costs, as they are estimate in the manner. Capitalized product cost is included in the asset value of the mine development and are depreciated over the LOM.

21.1.5 Processing Capital Costs

The processing plant capital costs are provided in Table 21-3.

Table 21-3: Process Plant CAPEX

Processing Plant CAPEX	Unit	Initial	Sustaining	Total
Crusher Material Bins & Reclaim	**$M**			
Crushed Material Bin	$M	3.9	0.2	4.1
Mill Feed Conveyor	$M	0.5	0.0	0.6
Grinding				
Grinding Circuit	$M	11.1	0.5	11.6
Copper Circuit	**$M**			
Copper Rougher Flotation	$M	1.8	0.1	1.9
Copper Regrind	$M	1.2	0.1	1.2
Copper Cleaner Flotation		3.0	0.1	3.1
Copper Concentrate	$M	1.7	0.1	1.8
Zinc Circuit	**$M**			
Zinc Flotation	$M	1.7	0.1	1.8
Zinc Regrind	$M	1.1	0.0	1.2
Zinc Cleaner Flotation	$M	3.3	0.1	3.4
Zinc Concentrate	$M	2.2	0.1	2.3
Pyrite Circuit	**$M**			
Pyrite Flotation	$M	1.6	0.1	1.7
Pyrite Filter	$M	3.0	0.1	3.1
Barite Circuit	**$M**			
Barite Flotation	$M	4.7	0.2	4.9
Barite Concentrate	$M	5.4	0.2	5.7
Process Tailings	$M	6.9	-	6.9
Reagents	$M	1.3	-	1.3
Plant Utilities, Building, & General	**$M**			
Plant Building	$M	14.0	0.6	14.6
Plant Water Systems	$M	1.0	0.0	1.0
Plant Air Systems	$M	0.9	0.0	0.9
Assay Lab	$M	1.7	0.1	1.7

Processing Plant CAPEX	Unit	Initial	Sustaining	Total
Plant Control Systems	$M	1.2	0.1	1.2
Electrical Services	$M	1.5	0.1	1.6
Total	**$M**	**74.7**	**2.7**	**77.4**

Source: JDS (2019)

21.1.6 Surface Construction Costs

Surface construction costs include site development, mineral processing plant, LNG power generators and storage, TMF/WRSF, WTP and off-site infrastructure. These cost estimates are primarily based on database or recently quoted costs, with factors applied for minor cost elements. Table 21-4 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Table 21-4: Surface Construction Basis of Estimate

Commodity	Basis
Contractor Labour Rates	Database values based on similar remote northern projects
Bulk Earthworks, Including On-Site Roads	Estimate volumes from preliminary site layout model Database unit rates for bulk excavation and fill, geomembrane liner installation, civil construction works, surface drainage and site water management
Concrete	Quantities developed based on building and equipment sizes, and geotechnical recommendations provided by KCB. Database unit rates from recent similar projects
Structural Steel	Quantities developed based on equipment sizes and cross checked against similar projects Database unit rates
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
Ancillary Buildings & Warehouses	Database costs from recent projects for the mine dry, administration offices, maintenance and warehouse building, cold storage structure
Mechanical Equipment	A combination of quoted costs and database costs from recent quotations on similar projects 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 (2019)

21.1.6.1 Surface Construction Sustaining Capital

Sustaining capital costs are included in the estimate for yearly construction of the TMF/WRSF. The balance of the facility is expanded yearly throughout the LOM to provide storage capacity based on the mine plan material balance.

Allowances are provided in the initial years for additional avalanche protection construction, using NPAG waste rock from the mining operation.

The sustaining capital cost estimate also includes allowances for the processing plant, power generation and on-site infrastructure for major equipment overhauls, minor capital projects and upgrades.

21.1.7 Indirect Costs

Indirect costs are those that not directly accountable to a specific cost object for services, supplies and labour associated with supporting construction activities. Table 21-5 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-5: Indirect Costs Basis of Estimate

Commodity	Basis
Heavy Equipment	A build of up heavy cranes and equipment for the surface construction, based on scheduled construction duration and database rates.
Contractor Field Indirect Costs	Factor (15.0%) of construction labour 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 • Diesel construction power • Contractor mobilization
Freight & Logistics	Factor (7%) for freight and logistics related to the materials and equipment required for the crushing plant, mineral processing plant, on-site and off-site infrastructure. Mining equipment has a freight factor (7%) based on the first five years of lease payments. Mine consumables have a freight factor of 2%.
Start-up and Commissioning	Includes first fills for 4 months of and wear parts, and 1-month supply of re-agents, contractor assistance factored (.25% of direct costs), and vendor representatives.
Vendor Representatives	An allowance of hours, equipment and expenses for the provision of vendor services for commissioning equipment, based on recent similar project.
Capital Spares	Factor (5%) of direct equipment costs for spare parts
Detailed Engineering & Procurement	Factor (10%) 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, contract administration and the start-up and commissioning in year 1.
	Database unit (hourly) rates

Source: JDS (2019)

21.1.8 Owners Costs

Owner's costs are included within the operating costs during production, but during the construction period these items are included in the initial capital costs and are 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 Processing: Costs for 3 months of Owner's process plant labour, 1 month of process plant power, 2 months of wear parts and 1 month of re-agents for the initial start-up of the processing plant.

21.1.9 Closure Cost Estimate

Closure Costs have been estimated at 30.8 $M for all closure activities, including:

- Removal of all surface infrastructure and buildings
- Closure and capping of the TMF/WRSF
- Re-vegetation and seeding allowance, and
- Ongoing site monitoring.

These costs were estimated based on scaling the existing bonding requirements of Greens Creek Mine by proportionate building sizes, TMF/WRSF footprint, etc. See section 20 for details on the closure cost estimate.

21.1.10 Cost Contingency

An overall contingency of 13.3% was applied to the initial capital costs of the project. LOM project contingency amounts to $32.2M, or approximately 11.8% of initial capital costs. The overall contingency is a blend of separate factors that were applied different areas as follows:

- Mining Development – 20%
- Underground infrastructure – 5%
- Process Plant, Site Infrastructure and Project Indirect Costs – 15%
- Civil Works and Tailings Management – 15% and
- Indirect and Owners Costs – 15%.

A growth allowance of 15% was applied to civil earthworks quantities prior to the application of the contingency, to reflect uncertainties in design material take offs.

21.1.11 Infrastructure Capital Costs

The infrastructure capital cost is provided in Table 21-6.

Table 21-6: Infrastructure CAPEX

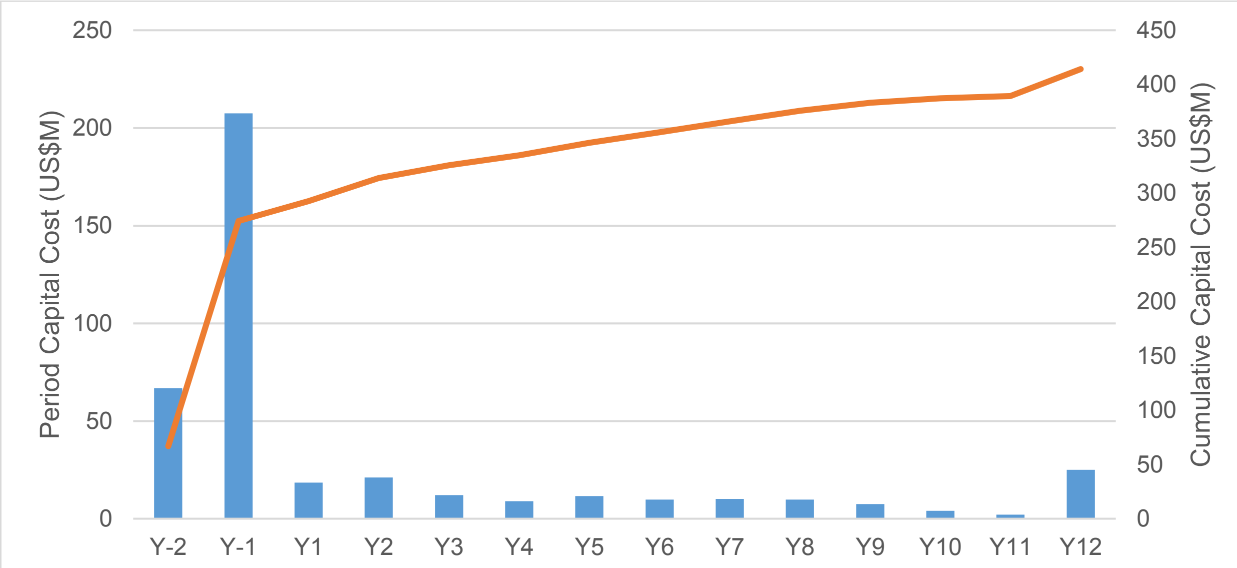
Infrastructure CAPEX	Unit	Initial	Sustaining	Total
Tailings Management Facility				
Foundation & Borrow Development	$M	0.2	0.4	0.6
Liner System	$M	1.4	2.7	4.1
Tailings LAD	$M	0.4	0.0	0.4
Power Supply & Distribution				
Power Generation	$M	13.9	0.7	14.6
LNG Fuel Storage	$M	1.8	0.0	1.8
On-Site Power Distribution	$M	0.4	0.0	0.4
Waste Management	$M	0.5		0.5
Ancillary Buildings				
Mine Dry/Office Building	$M	2.5	0.1	2.6
Surface Maintenance Shop	$M	0.3	0.0	0.3
Emergency Response Facility	$M	0.6	0.0	0.6
Mine/Plant Warehouse	$M	0.6	0.0	0.6
Cold Storage Warehouse	$M	0.4	0.0	0.4
Surface Mobile Equipment	$M	4.5	0.5	5.0
Bulk Fuel Storage & Distribution	$M	0.4		0.4
IT & Communications	$M	1.8		1.8
Fire Detection and Safety Systems				
Site Fire Safety System	$M	0.7		0.7
Water Treatment Plant	**$M**			
Water Treatment Plant	$M	5.6		5.6
Temp Construction Camp Facility Prep	$M	0.3		0.3
Total	$M	36.34	4.4	40.8

Source: JDS (2019)

21.1.12 Capital Cost Profile

All capital costs for the project have been scheduled according to development and production needs in order to support the economic cash flow model. Figure 21-1 presents an annual LOM capital cost profile.

Figure 21-1: Capital Cost Distribution by Project Year



Source: JDS (2019)

21.2 Operating Cost Estimate

21.2.1 Operating Cost Summary

The LOM costs are summarized in Table 21-7.

Table 21-7: LOM Total Operating Cost Estimates

Description	Total Estimate ($M)	Average Unit Cost ($/t)
UG Mining	362.7	29.06
Processing	210.0	16.83
G&A	103.3	8.28
Total Operating Costs	**676.1**	**54.17**

Source: JDS (2019)

The operating cost (OPEX) estimates are based on combination of experiential judgment, reference to similar operating 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.

In addition to the OPEX, the sustaining cost, as detailed in Section 21, averages $11.22/tonne over the LOM. The total operating cost and sustaining cost totals $40.32/tonne over the LOM.

21.2.2 Mine Operating Costs

21.2.2.1 Underground Mine Operating Cost

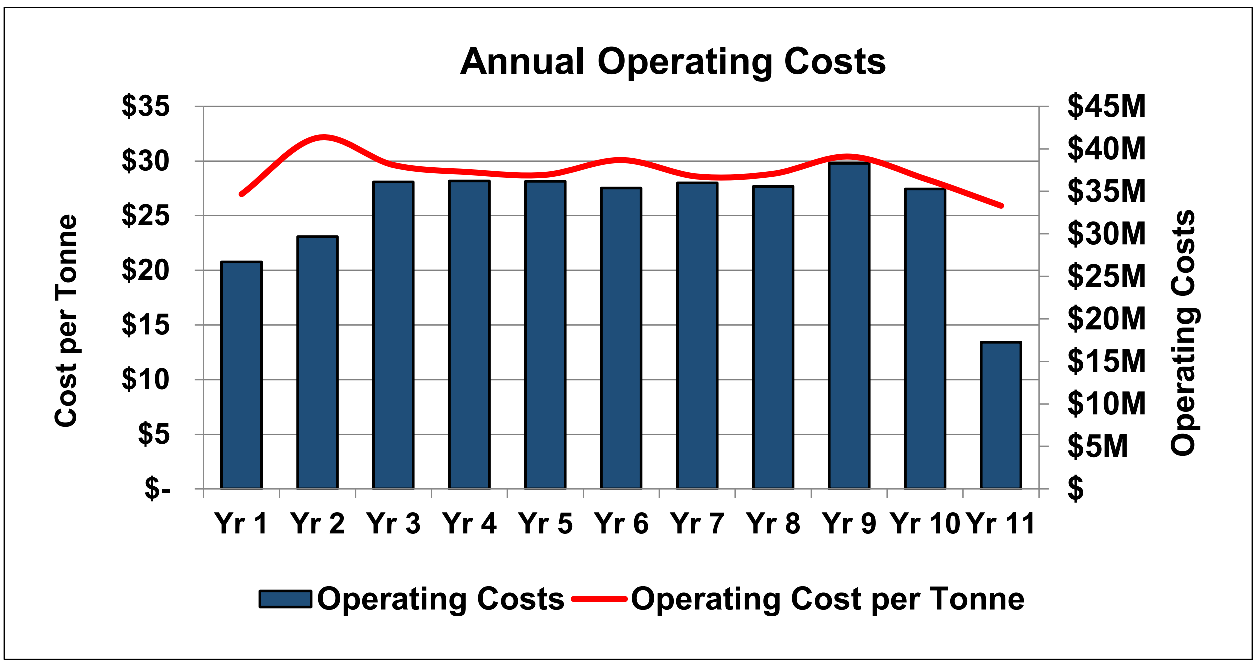
The total UG mine OPEX per tonne mined is broken out by the cost centers in Table 21-8 and shown graphically by year in Figure 21-2.

Table 21-8: Overall Mining OPEX

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Lateral Waste Development	16.1	1.29
Production	180.4	14.45
Backfill	67.6	5.42
UG Crusher and Conveyor	9.6	0.77
Mine Maintenance	39.6	3.17
Mine General	49.3	3.95
Total	**362.7**	**29.06**

Source: JDS (2019)

Figure 21-2: Annual Operating Cost



Source: JDS (2019)

Operating costs for non-capital lateral development include the following types:

- Footwall drives
- Sumps
- Orepass Drive
- Ventilation Drive
- Exploration Drive
- Mechanic Shop
- Longhole Access
- Slashing
- Remuck
- Transverse Access
- Cut and Fill Access and
- Cut and Fill Drive.

These total 41,772 m over the LOM. In addition, a total of 10,961 m of ramp will be driven of the LOM, which is included as sustaining capital in Section 21.

Table 21-9: OPEX Lateral Waste Development

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Labour	2.40	0.19
Fuel	0.73	0.06
Equipment	1.95	0.16
Power	0.26	0.02
Consumables	7.23	0.58
Explosives	3.57	0.29
Total	**16.12**	**1.29**

Source: JDS (2019)

Production operating costs are directly associated with the extraction of the minable resource, including lateral development in mineralized material and longhole stoping, these are summarized in Table 21-10.

Table 21-10: Production OPEX

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Labour	97.4	7.80
Fuel	16.0	1.28
Equipment	30.5	2.44
Power	10.7	0.85
Consumables	10.2	0.82
Explosives	15.6	1.25
Total	**180.4**	**14.45**

Source: JDS (2019)

Backfill operating costs are associated with the manufacturing, distribution and placement of sulfide paste, de-sulfide paste and unconsolidated PAG rock in the mine. The costs associated with stope preparation, backfill production, transport and placement are shown in Table 21-11.

Table 21-11: Backfill OPEX

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Labour	3.3	0.27
Fuel	0.03	0.00
Equipment	0.05	0.00
Power	8.5	0.68
Cement	34.2	2.74
Parts, Other Consumables, Bulkheads	21.6	1.73
Total	**67.6**	**5.42**

Source: JDS (2019)

All costs associated with the operation, maintenance, and consumable materials for the UG Crusher and Conveyor OPEX have been summarized in Table 21-12.

Table 21-12: UG Crusher and Conveyor OPEX

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Labour	4.7	0.38
Power	1.8	0.15
Crusher and Conveyor Maintenance	3.1	0.25
Total	**9.6**	**0.77**

Source: JDS (2019)

Mine Maintenance OPEX includes all costs associated with labour and general shop consumables required to maintain the mobile fleet, costs are summarized in Table 21-13.

Table 21-13: Mine Maintenance OPEX

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Labour	38.6	3.09
Shop Consumables	1.0	0.08
Total	**39.6**	**3.17**

Source: JDS (2019)

General mine expenses include pumping, ventilation, heating, compressed air, definition drilling, and supervisory and technical support are summarized in Table 21-14.

Table 21-14: General Mine OPEX

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
Power	7.9	0.64
Fuel	2.1	0.16
Equipment	4.2	0.33
Definition Drilling	2.4	0.19
Mine Air Heating	0.7	0.05
Technical Services Labour	27.5	2.20
Technical Services Supplies	2.8	0.22
Misc. Supplies/PPE	1.9	0.15
Total	**49.4**	**3.95**

Source: JDS (2019)

21.2.3 Process Operating Costs

Process operating costs were estimated to include all lead and zinc recovery steps required to produce saleable concentrates. The process plant was designed to process 3,500 t/d at a 92% availability. Labour rates and benefit packages were based on industry information compiled by JDS. Power costs were calculated from the total installed power assuming $0.15/kWh. Liner pricing and Vendor recommended spare parts for one year of operation were used to estimate mill 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 21-15.

Table 21-15: Breakdown of Process Operating Costs

Process Operating Costs (LOM Total)	$M/a	$/t processed
Labour	5.4	4.40
Power	4.3	3.56
Maintenance and Consumables	9.6	7.86
Total Processing OPEX	**19.3**	**15.82**

Source: JDS (2019)

21.2.4 General and Administration Costs

General and administrative costs comprise the following categories:

- Administration, site services, power plant, and water treatment plant labour
- Haulage tailings to the TSF
- On-site items as such as, health and safety, environmental, human resources, insurance, legal, external consulting, IT, communications and office supplies, site service equipment operation and maintenance and
- Employee travel via bussing from Haines.

Table 21-16 summarizes the annual G&A operating costs.

Table 21-16: G&A OPEX Estimate by Area

Parameter	LOM ($M)	$/t processed
G&A Labour	57.2	4.58
G&A Items - On-site	39.4	3.16
Satellite Office and Off-Site Warehousing	1.6	0.13
Employee Travel	5.0	0.41
Total Operating Cost – G&A	**103.2**	**8.28**

Source: JDS (2019)

22 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 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 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 the zinc metal price, copper metal price, all metal prices, head grade, operating costs and capital costs to determine each item's relative importance as a project value driver.

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 2019 US dollars). The economic analysis has been prepared on a constant dollar base, without allowances for inflation or escalation.

22.1 Assumptions

The model excludes all exploration 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.). This includes all costs associated with the upcoming 2019 exploration program. As such, the 680 portal, exploration drift, 848 diamond drill drift and all supporting services are assumed to be existing at the start of this PEA construction. Costs for these exploration elements have been specifically excluded from the PEA estimates even though they have not been occurred.

Table 22-1 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 April 15th, 2019. These have been compared to the spot prices at the close of London Metal Exchange on April 15th, 2019. 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 22-1: Metal Price and Exchange Rates Used in Economic Analysis

Parameter	Unit	Base Price Value	Spot Price Value
Copper Price	US$/lb	2.82	2.93
Zinc Price	US$/lb	1.22	1.36
Silver Price	US$/oz	16.26	15.00
Gold Price	US$/oz	1,296	1289
Barite Price	US$/tonne	220	220
Exchange Rate	US$:C$	0.75	0.75

Source: JDS (2019)

Other economic factors include the following:

- Discount rate of 7%;
- Closure cost of $30.8 M (pre-contingency);
- Nominal 2019 US dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- Working capital calculated as three months of operating costs (mining, processing, and G&A) in Year 1;
- Results are presented on a 100% ownership basis; and
- No management fees or financing costs (equity fund-raising was assumed).

22.2 Processing and Concentrate Terms

Mine revenue is derived from the sale of copper concentrate and zinc concentrate into the international marketplace. Barite concentrate will be sold to bulk barite suppliers to the North American oil rig service industry. The concentrate is in a ‘drill-mud’ ready powder form and does not require any additional processing.

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 22-2 outlines the recoveries, payable terms, treatment charges and transportation costs used in the economic analysis.

Table 22-2: Concentrate Terms

Assumptions and Inputs	Unit	Value
Copper Concentrate		
Metal Recovery to Concentrate	% Cu	88.5%
	% Zn	4.8%
	% Ag	70.8%
	% Au	49.5%
Cu Concentrate Grade Produced	% Cu	25%
Moisture Content	%	8%
Minimum Deduction	%Cu/tonne	1%
	g/t Ag	31.10
	g/t Au	1.24
Metal Payable	% Cu	97%
	% Ag	90%
	% Au	90%
Cu Treatment Charge	US$/dmt conc	86
Cu Refining Charge	US$/lb	0.086
Ag Refining Charge	US$/oz	0.75
Au Refining Charge	US$/oz	6
Zn Penalties per % over 4%	US$/tonne	2
Cu Concentrate Transport Cost	US$/wmt conc	91
Zinc Concentrate		
Metal Recovery to Concentrate	% Zn	91.5%
	% Cu	6.30%
	% Ag	20.10%
	% Au	20.10%
Zn Concentrate Grade Produced	% Zn	61%
Moisture Content	%	8%
Minimum Deduction	% Zn/tonne	8%
	g/t Ag	93.3
	g/t Au	0.311
Metal Payable	% Zn	85%
	% Ag	70%
	% Au	70%
Zn Treatment Charge	US$/dmt conc	150
Ag Refining Charge	US$/oz	0.75
Au Refining Charge	US$/oz	6
Zn Concentrate Transport Cost	US$/wmt conc	91
Barite Concentrate		
Metal Recovery to Concentrate	% Barite	91%
Barium Concentrate Grade	% Ba	52%
Barite Concentrate Density	SG	4.44

Assumptions and Inputs	Unit	Value
Moisture Content	%	1%
Barite Treatment Charge	US$/t conc	-
Barite Concentrate Transport Cost	US$/tonne	132

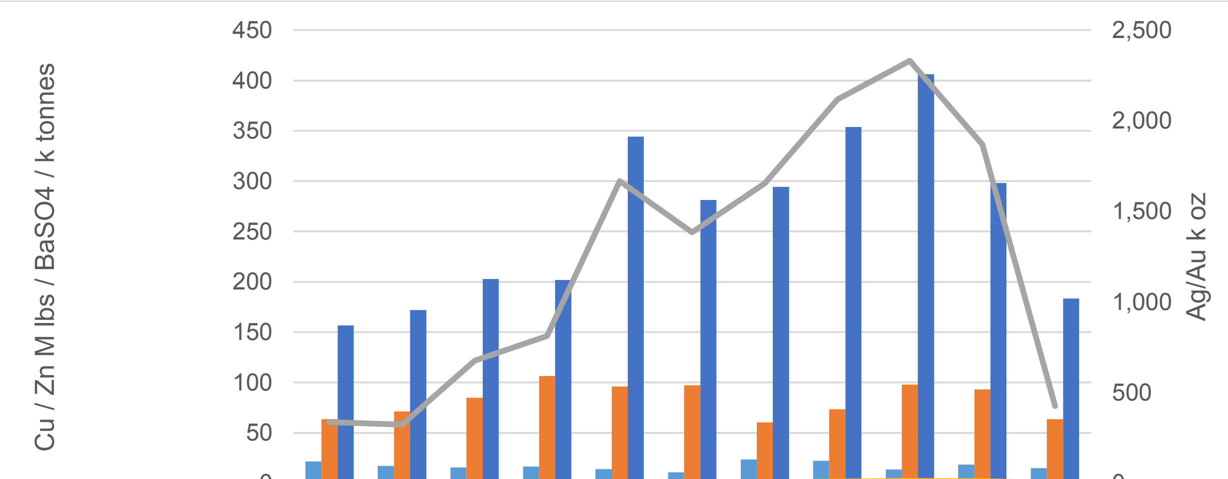
Source: JDS (2019)

During the LOM, the following metals are produced:

- 210 Mlbs of copper,
- 1,124 Mlbs of zinc,
- 18,077 koz of silver,
- 91 koz of gold, and
- 2,894 kt of barite,

Figure 22-1 shows a breakdown of the payable copper, zinc, silver, gold and barite recovered during the mine life. The distribution of revenues by commodity is shown in Figure 22-2. As can be seen, Zinc is the most significant commodity, accounting for 43% of sales.

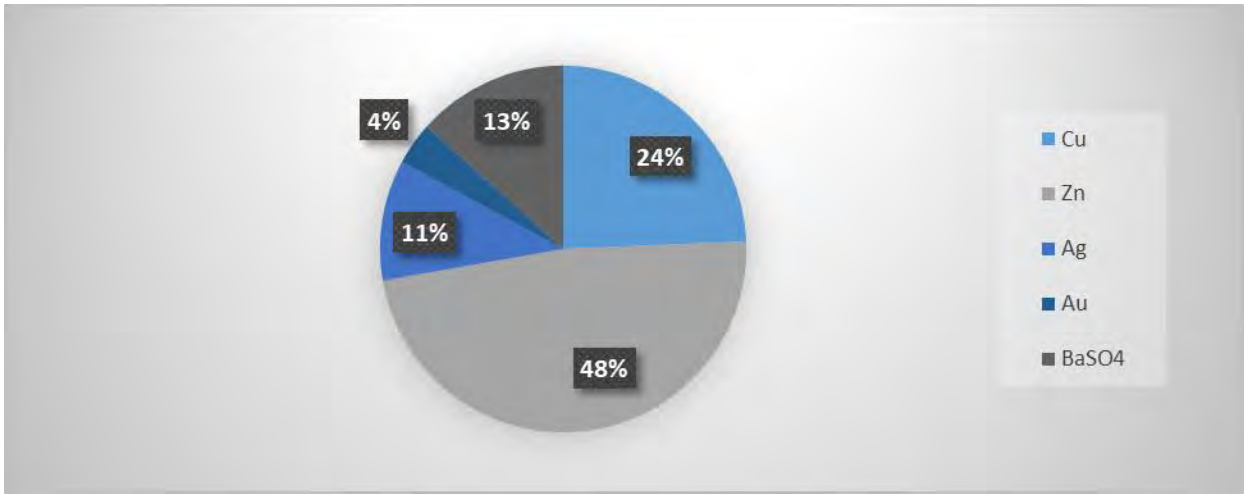
Figure 22-1: Payable Metal Production by Year



	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Payable Cu M lbs	22	17	15	16	14	11	24	22	14	18	15
Payable Zn M lbs	64	71	85	107	96	97	60	73	98	93	63
Payable BaSO4 k t	157	172	203	202	344	281	294	354	406	298	183
Payable Ag k oz	337	322	676	813	1,667	1,384	1,654	2,118	2,331	1,868	426
Payable Au k oz	3	3	3	3	4	4	4	8	12	10	4

Source: JDS (2019)

Figure 22-2: NSR Revenue Distribution by Commodity



Source: JDS (2019)

Although Barite sales account for a significant portion of the total revenue, it must be considered that the sales margin is much lower due to the higher proportional off-site transportation costs.

Table 22-3 presents the NSR value of the project broken down by metal. This takes into account all treatment charges, refining charges, transportation costs and penalties. This does not include any royalties payable.

Table 22-3: NSR Value

Commodity	NSR ($M)	% of Total
Cu	457.4	24%
Zn	900.2	48%
Ag	199.7	11%
Au	71.1	4%
$BaSO_4$	250.8	13%
Total	**1,879.2**	**100%**

Source: JDS (2019)

22.3 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, as detailed in Section 21.

Table 22-4 presents the capital estimate summary for initial, sustaining, and closure capital costs in 2019 US dollars with no escalation.

Table 22-4: Capital Cost Summary

Area	Pre-production ($M)	Sustaining ($M)	Closure ($M)	Total ($M)
Mining	55	108		163
Site Development	12	1		13
Mineral Processing	75	3		77
Tailings Management	2	3		5
On-Site Infrastructure	34	1		35
Off-Site Infrastructure	0	0		0
Project Indirects	26	0		26
EPCM	32	0		32
Owner Costs	8	0		8
Closure	0		31	31
Salvage Value	0		-6	-6
Subtotal	245	115	25	385
Contingency	33	0		33
Total CAPEX	**278**	**115**	**25**	**418**

Source: JDS (2019)

22.4 Summary of Operating Cost Estimates

The Operating Costs are summarized in Table 22-5.

Table 22-5: LOM Total Operating Cost Estimate

Description	Total Estimate ($M)	Average Unit Cost ($/t)
UG Mining	362.7	29.06
Processing	210.0	16.83
G&A	103.3	8.28
Total Operating Costs	**676.0**	**54.17**

Source: JDS (2019)

22.5 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. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- 21% federal income tax rate;
- 9.4% Alaska tax rate;
- 7.0% State Mining License tax rate; and
- Alaska State Mineral Exploration Expense Credit.

Total taxes for the project amount to $157.87 M over the LOM.

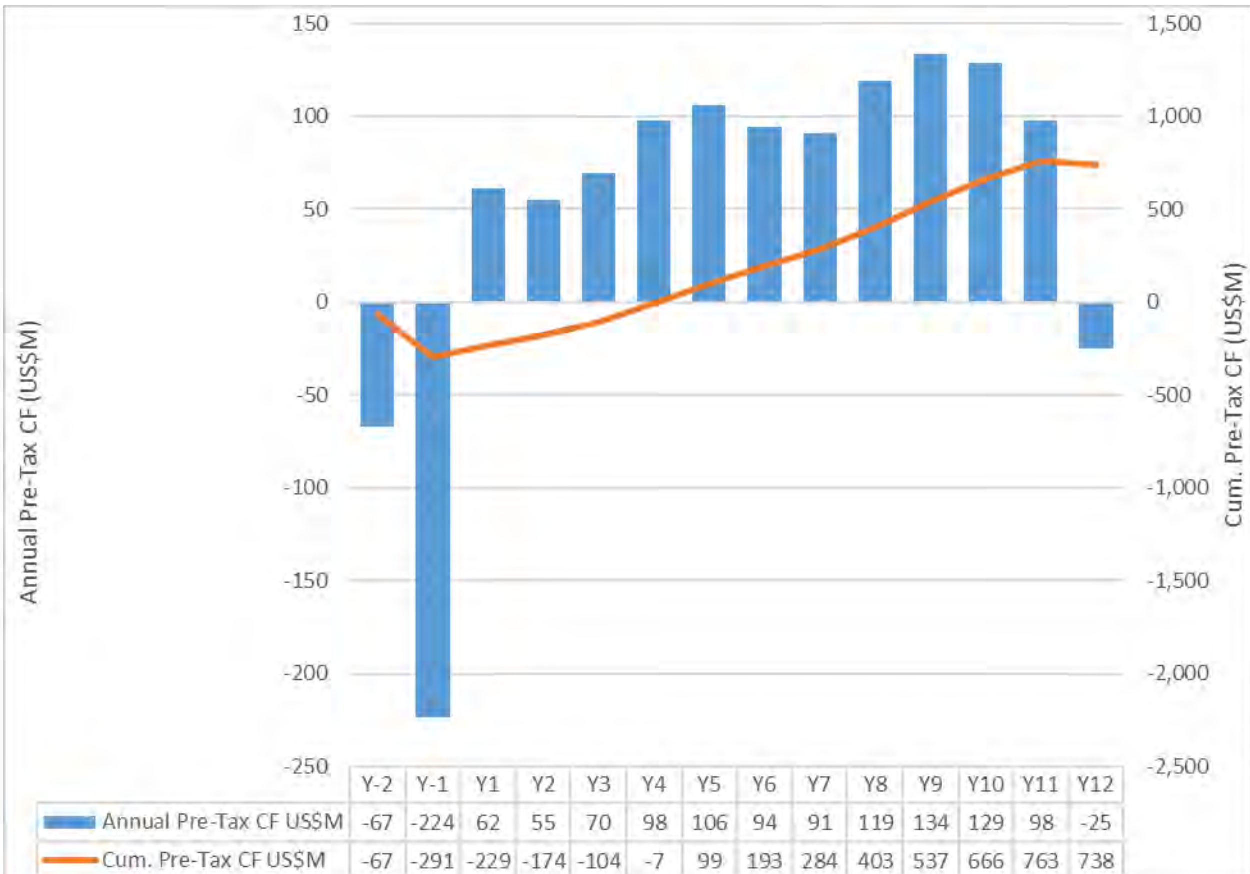
22.6 Economic Results

At this preliminary stage, the project has a pre-tax IRR of 24.1% and a net present value of $354M, at a discount rate of 7%, using the metal prices described in Section 19.

The project has an after-tax IRR of 21.1% and a net present value of $266 M, at a discount rate of 7%.

Figure 22-3 shows the projected pre-tax cash flows, and Table 22-6 summarizes the economic results of the Palmer Project.

Figure 22-3: Annual Pre-Tax Cash Flow



Source: JDS (2019)

Table 22-6: Summary of Results

Parameter	Unit	Base Value Price	Spot Value Price
Capital Cost			
Pre-production Capital	US$M	245	245
Pre-production Contingency	US$M	33	33
Total pre-production Capital	US$M	278	278
Sustaining and Closure Capital	US$M	115	115
Sustaining and Closure Contingency	US$M	115	115
Total Sustaining and Closure Capital	US$M	418	418
Total Capital Costs	US$M	418	418
Cash Flows			
Working Capital	US$M	13	13
Pre-Tax Cash Flow	US$M	738	861
	US$/a	69	80
Taxes	US$M	158	187
Post-Tax Cash Flow	US$M	581	675
	US$/a	54	63
Economic Result			
Pre-Tax $NPV_{7\%}$	US$M	354	433
Pre-Tax IRR	%	24%	28%
Pre-Tax Payback	years	3.1	2.6
Post-Tax NPV7%	US$M	266	328
Post-Tax IRR	%	21%	24%
Post-Tax Payback	years	3.3	2.9

Source: JDS (2019)

22.7 Sensitivity Analysis

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 -20% to +20%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM.

The analysis revealed that the project is most sensitive to overall metal prices, followed by zinc metal price specifically, operating costs, head grade and capital costs. Of those elements tested, the Project showed the least sensitivity to copper metal price. Table 22-7 and Figure 22-4 show the results of the sensitivity tests.

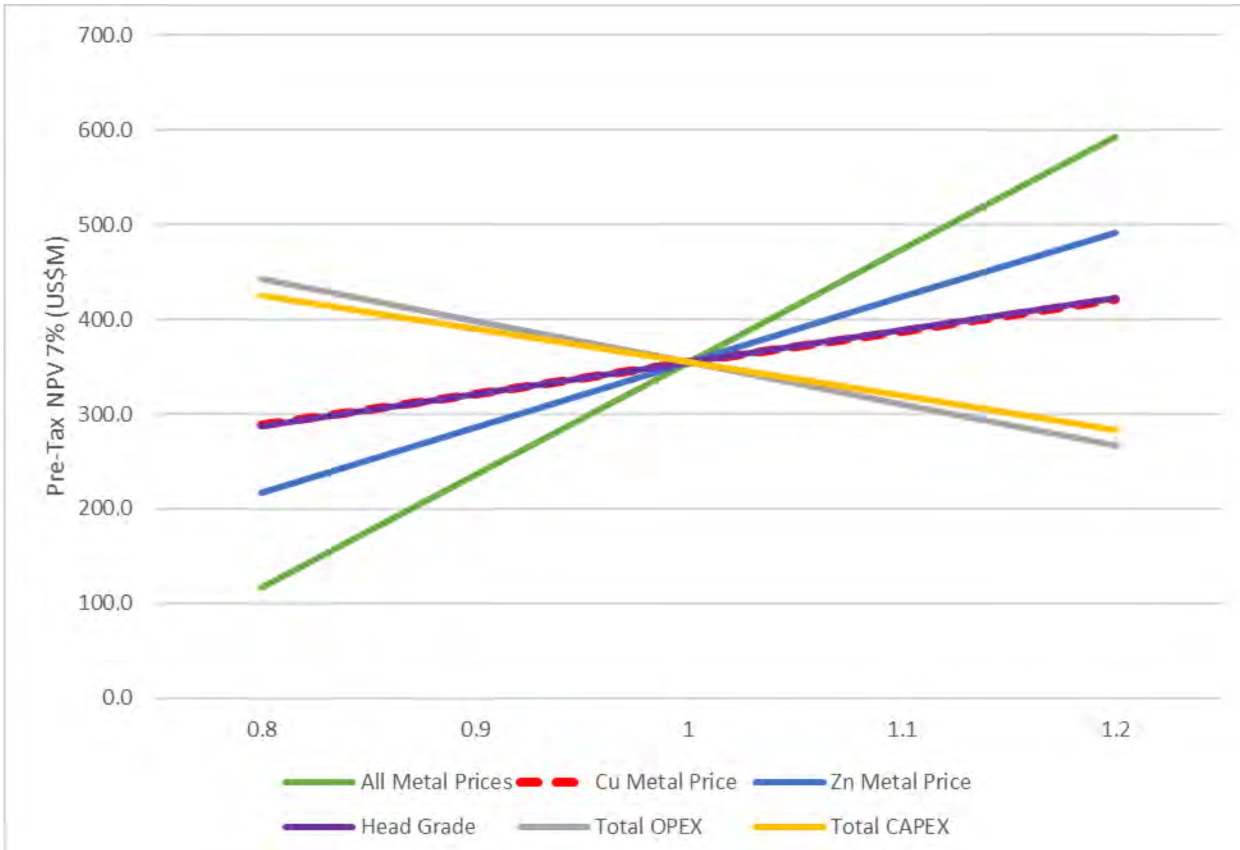
The economic cash flow model for the project over the LOM is illustrated in Table 22-8.

Table 22-7: Sensitivity Results (Pre-Tax NPV7%)

Parameter	-20%	-10%	Base	+10%	+20%
Zinc Metal Price	217	286	354	423	492
Copper Metal Price	288	321	354	388	421
Metal Prices	117	236	354	473	592
Head Grade	287	321	354	388	423
OPEX	442	398	354	311	267
CAPEX	425	390	354	319	284

Source: JDS (2019)

Figure 22-4: Sensitivity, Pre-Tax NPV @ 7% Discount Rate



As the project is located in the United States, the Canada/US dollar exchange rate is nearly inconsequential and was not tested.
Source: JDS (2019)

Table 22-8: Economic Cash Flow Model

Constantine Metal Resources Ltd.
Palmer Project, Alaska, US
2019 PEA - Economic Model

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	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
METAL PRICES & F/X RATE																
Cu	US$/lb	**2.82**	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82	2.82
Zn	US$/lb	**1.22**	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22
Ag	US$/oz	**16.26**	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26	16.26
Au	US$/oz	**1,296**	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296
BaSO4	US$/tonne	**220.00**	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00	220.00
F/X	USD:CAD	**0.75**	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
PRODUCTION SCHEDULE																
UNDERGROUND - MAIN ZONE																
Resource Mined	ktonnes	**8,351**			990	874	899	848	772	637	720	695	668	686	564	
Cu	%	**1.15%**			1.16%	1.02%	0.88%	0.99%	0.92%	0.83%	1.69%	1.64%	0.99%	1.34%	1.36%	
Zn	%	**4.36%**			3.68%	4.50%	4.08%	4.84%	5.07%	4.93%	2.23%	3.16%	5.30%	5.05%	5.69%	
Ag	g/t	**24.33**			18.46	17.79	23.22	29.15	29.38	29.35	13.76	21.57	33.06	26.55	30.59	
Au	g/t	**0.26**			0.28	0.28	0.19	0.20	0.20	0.21	0.16	0.27	0.41	0.35	0.42	
Ba	%	**11.63%**			9.08%	10.99%	9.19%	10.36%	13.78%	13.79%	6.43%	10.26%	16.14%	13.90%	17.77%	
Mining Rate	*ktpd*	***2.1***			*2.7*	*2.4*	*2.5*	*2.3*	*2.1*	*1.7*	*2.0*	*1.9*	*1.8*	*1.9*	*1.5*	
Contained Metal																
Cu	ktonnes	**96**			11.4	9.0	7.9	8.4	7.1	5.3	12.2	11.4	6.6	9.2	7.7	
Zn	ktonnes	**364**			36.4	39.4	36.6	41.0	39.1	31.4	16.1	22.0	35.4	34.7	32.1	
Ag	kg	**203,177**			18,279.5	15,552.5	20,866.5	24,709.6	22,676.0	18,682.7	9,906.5	14,984.3	22,076.3	18,201.2	17,241.8	
Au	kg	**2,191**			272.4	244.0	168.0	166.1	153.6	133.0	116.6	188.3	272.4	237.9	239.0	
Ba	ktonnes	**971**			89.9	96.1	82.5	87.8	106.3	87.8	46.3	71.3	107.7	95.3	100.2	
UNDERGROUND - AG ZONE																
Resource Mined	ktonnes	**4,130**				49	320	402	488	540	540	540	592	558	102	
Cu	%	**0.11%**				0.18%	0.10%	0.11%	0.08%	0.10%	0.08%	0.11%	0.12%	0.13%	0.21%	
Zn	%	**3.99%**				3.16%	3.95%	5.27%	3.43%	4.78%	3.64%	3.94%	3.70%	3.54%	4.47%	
Ag	g/t	**100.56**				32.70	42.00	40.75	106.54	82.44	115.34	141.88	133.77	115.67	47.37	
Au	g/t	**0.46**				0.14	0.20	0.19	0.33	0.29	0.40	0.66	0.75	0.70	0.24	
Ba	%	**16.71%**				5.26%	10.57%	6.99%	18.73%	13.67%	22.72%	24.41%	21.18%	13.58%	4.94%	
Mining Rate	*ktpd*	***1.4***				*0.1*	*0.9*	*1.1*	*1.3*	*1.5*	*1.5*	*1.5*	*1.6*	*1.5*	*0.3*	
Contained Metal																
Cu	ktonnes	**4**				0.1	0.3	0.4	0.4	0.5	0.4	0.6	0.7	0.7	0.2	
Zn	ktonnes	**165**				1.6	12.6	21.2	16.7	25.8	19.6	21.3	21.9	19.7	4.5	
Ag	kg	**415,361**				1,608.8	13,440.6	16,368.9	51,937.8	44,518.7	62,284.7	76,614.1	79,223.5	64,551.6	4,811.8	
Au	kg	**1,894**				6.8	63.0	75.1	159.9	158.8	218.2	358.0	442.4	387.9	24.2	
Ba	ktonnes	**690**				2.6	33.8	28.1	91.3	73.8	122.7	131.8	125.4	75.8	5.0	
TOTAL																
Resource Mined	ktonnes	**12,481**			990	924	1,219	1,249	1,259	1,177	1,260	1,235	1,260	1,244	665	
Cu	%	**0.81%**			1.16%	0.98%	0.67%	0.70%	0.60%	0.49%	1.00%	0.97%	0.58%	0.80%	1.18%	
Zn	%	**4.24%**			3.68%	4.43%	4.04%	4.98%	4.43%	4.86%	2.83%	3.50%	4.55%	4.37%	5.50%	
Ag	g/t	**49.56**			18.46	18.58	28.15	32.88	59.25	53.72	57.29	74.18	80.40	66.54	33.15	
Au	g/t	**0.33**			0.28	0.27	0.19	0.19	0.25	0.25	0.27	0.44	0.57	0.50	0.40	
Ba	%	**13.31%**			9.08%	10.68%	9.55%	9.27%	15.69%	13.73%	13.41%	16.45%	18.50%	13.76%	15.81%	
Mining Rate	*ktpd*	***3.2***			*2.7*	*2.5*	*3.3*	*3.4*	*3.4*	*3.2*	*3.5*	*3.4*	*3.5*	*3.4*	*1.8*	
Contained Metal																
Cu	ktonnes	**100**			11.4	9.0	8.2	8.8	7.5	5.8	12.6	12.0	7.3	9.9	7.9	
Zn	ktonnes	**529**			36.4	40.9	49.2	62.2	55.8	57.2	35.7	43.3	57.3	54.4	36.6	
Ag	kg	**618,537**			18,279.5	17,161.3	34,307.2	41,078.5	74,613.7	63,201.4	72,191.2	91,598.4	101,299.9	82,752.8	22,053.6	
Au	kg	**4,086**			272.4	250.7	231.1	241.3	313.5	291.8	334.8	546.3	714.9	625.8	263.2	
Ba	ktonnes	**1,662**			89.9	98.7	116.4	115.8	197.6	161.6	169.0	203.1	233.2	171.1	105.2	

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	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
MILL SCHEDULE																
MAIN ZONE																
Resource Milled	ktonnes	**8,351**			**990**	**874**	**899**	**848**	**772**	**637**	**720**	**695**	**668**	**686**	**564**	
Cu	%	**1.15%**			1.16%	1.02%	0.88%	0.99%	0.92%	0.83%	1.69%	1.64%	0.99%	1.34%	1.36%	
Zn	%	**4.36%**			3.68%	4.50%	4.08%	4.84%	5.07%	4.93%	2.23%	3.16%	5.30%	5.05%	5.69%	
Ag	g/t	**24.33**			18.46	17.79	23.22	29.15	29.38	29.35	13.76	21.57	33.06	26.55	30.59	
Au	g/t	**0.26**			0.28	0.28	0.19	0.20	0.20	0.21	0.16	0.27	0.41	0.35	0.42	
Ba	%	**11.63%**			9.08%	10.99%	9.19%	10.36%	13.78%	13.79%	6.43%	10.26%	16.14%	13.90%	17.77%	
Milling Rate	*ktpd*	***2.1***			*2.7*	*2.4*	*2.5*	*2.3*	*2.1*	*1.7*	*2.0*	*1.9*	*1.8*	*1.9*	*1.5*	
Contained Metal																
Cu	ktonnes	**96**			11.4	9.0	7.9	8.4	7.1	5.3	12.2	11.4	6.6	9.2	7.7	
Zn	ktonnes	**364**			36.4	39.4	36.6	41.0	39.1	31.4	16.1	22.0	35.4	34.7	32.1	
Ag	kg	**203,177**			18,279.5	15,552.5	20,866.5	24,709.6	22,676.0	18,682.7	9,906.5	14,984.3	22,076.3	18,201.2	17,241.8	
Au	kg	**2,191**			272.4	244.0	168.0	166.1	153.6	133.0	116.6	188.3	272.4	237.9	239.0	
Ba	ktonnes	**971**			89.9	96.1	82.5	87.8	106.3	87.8	46.3	71.3	107.7	95.3	100.2	
AG ZONE																
Resource Milled	ktonnes	**4,130**				**49**	**320**	**402**	**488**	**540**	**540**	**540**	**592**	**558**	**102**	
Cu	%	**0.11%**				0.18%	0.10%	0.11%	0.08%	0.10%	0.08%	0.11%	0.12%	0.13%	0.21%	
Zn	%	**3.99%**				3.16%	3.95%	5.27%	3.43%	4.78%	3.64%	3.94%	3.70%	3.54%	4.47%	
Ag	g/t	**100.56**				32.70	42.00	40.75	106.54	82.44	115.34	141.88	133.77	115.67	47.37	
Au	g/t	**0.46**				0.14	0.20	0.19	0.33	0.29	0.40	0.66	0.75	0.70	0.24	
Ba	%	**16.71%**				5.26%	10.57%	6.99%	18.73%	13.67%	22.72%	24.41%	21.18%	13.58%	4.94%	
Milling Rate	*ktpd*	***1.4***				*0.1*	*0.9*	*1.1*	*1.3*	*1.5*	*1.5*	*1.5*	*1.6*	*1.5*	*0.3*	
Contained Metal																
Cu	ktonnes	**4**				0.1	0.3	0.4	0.4	0.5	0.4	0.6	0.7	0.7	0.2	
Zn	ktonnes	**165**				1.6	12.6	21.2	16.7	25.8	19.6	21.3	21.9	19.7	4.5	
Ag	kg	**415,361**				1,608.8	13,440.6	16,368.9	51,937.8	44,518.7	62,284.7	76,614.1	79,223.5	64,551.6	4,811.8	
Au	kg	**1,894**				6.8	63.0	75.1	159.9	158.8	218.2	358.0	442.4	387.9	24.2	
Ba	ktonnes	**690**				2.6	33.8	28.1	91.3	73.8	122.7	131.8	125.4	75.8	5.0	
TOTAL																
Resource Milled	ktonnes	**12,481**			**990**	**924**	**1,219**	**1,249**	**1,259**	**1,177**	**1,260**	**1,235**	**1,260**	**1,244**	**665**	
Cu	%	**0.81%**			1.16%	0.98%	0.67%	0.70%	0.60%	0.49%	1.00%	0.97%	0.58%	0.80%	1.18%	
Zn	%	**4.24%**			3.68%	4.43%	4.04%	4.98%	4.43%	4.86%	2.83%	3.50%	4.55%	4.37%	5.50%	
Ag	g/t	**49.56**			18.46	18.58	28.15	32.88	59.25	53.72	57.29	74.18	80.40	66.54	33.15	
Au	g/t	**0.33**			0.28	0.27	0.19	0.19	0.25	0.25	0.27	0.44	0.57	0.50	0.40	
Ba	%	**13.31%**			9.08%	10.68%	9.55%	9.27%	15.69%	13.73%	13.41%	16.45%	18.50%	13.76%	15.81%	
Milling Rate	*ktpd*	***3.2***			*2.7*	*2.5*	*3.3*	*3.4*	*3.4*	*3.2*	*3.5*	*3.4*	*3.5*	*3.4*	*1.8*	
Contained Metal																
Cu	ktonnes	**100**			11.4	9.0	8.2	8.8	7.5	5.8	12.6	12.0	7.3	9.9	7.9	
Zn	ktonnes	**529**			36.4	40.9	49.2	62.2	55.8	57.2	35.7	43.3	57.3	54.4	36.6	
Ag	kg	**618,537**			18,279.5	17,161.3	34,307.2	41,078.5	74,613.7	63,201.4	72,191.2	91,598.4	101,299.9	82,752.8	22,053.6	
Au	kg	**4,086**			272.4	250.7	231.1	241.3	313.5	291.8	334.8	546.3	714.9	625.8	263.2	
Ba	ktonnes	**1,662**			89.9	98.7	116.4	115.8	197.6	161.6	169.0	203.1	233.2	171.1	105.2	

Cu CONCENTRATE																
Recovery to Cu Concentrate	% Cu	**88.5%**			88.9%	88.8%	88.5%	88.4%	88.4%	88.0%	88.5%	88.4%	88.0%	88.1%	88.6%	
	% Zn	**4.8%**			4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	
	% Ag	**70.8%**			70.8%	70.8%	70.8%	70.8%	70.8%	70.8%	70.8%	70.8%	70.8%	70.8%	70.8%	
	% Au	**49.5%**			49.5%	49.5%	49.5%	49.5%	49.5%	49.5%	49.5%	49.5%	49.5%	49.5%	49.5%	

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	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Metal in Cu Concentrate	Cu ktonnes	**89**	-	-	10	8	7	8	7	5	11	11	6	9	7	-
	Cu Mlbs	**196**	-	-	22	18	16	17	15	11	25	23	14	19	15	-
	Zn ktonnes	**25**	-	-	2	2	2	3	3	3	2	2	3	3	2	-
	Zn Mlbs	**56**	-	-	4	4	5	7	6	6	4	5	6	6	4	-
	Ag kg	**437,925**	-	-	12,942	12,150	24,289	29,084	52,827	44,747	51,111	64,852	71,720	58,589	15,614	-
	Ag koz	**14,080**	-	-	416	391	781	935	1,698	1,439	1,643	2,085	2,306	1,884	502	-
	Au kg	**2,022**	-	-	135	124	114	119	155	144	166	270	354	310	130	-
	Au koz	**65**	-	-	4	4	4	4	5	5	5	9	11	10	4	-
Pull Factor		**34**	-	-	24	28	41	39	47	56	28	29	48	35	23	-
Cu Concentrate Grade	% Cu	**24.5%**			24.5%	24.5%	24.5%	24.5%	24.5%	24.5%	24.5%	24.5%	24.5%	24.5%	24.5%	
	% Zn	**7.0%**			4.2%	6.0%	8.0%	9.4%	9.9%	13.2%	3.8%	4.8%	10.5%	7.3%	6.2%	
	g/t Ag	**1,207.13**	-	-	311.81	370.82	817.56	915.34	1,952.27	2,145.15	1,121.61	1,503.35	2,725.37	1,644.68	548.64	-
	Ag oz	**39**	-	-	10.0	11.9	26.3	29.4	62.8	69.0	36.1	48.3	87.6	52.9	17.6	-
	g/t Au	**5.57**	-	-	3.25	3.79	3.85	3.76	5.73	6.92	3.64	6.27	13.45	8.70	4.58	-
	Au oz	**0.2**	-	-	0.1	0.1	0.1	0.1	0.2	0.2	0.1	0.2	0.4	0.3	0.1	-
Cu Concentrate Produced	dmt	**362,781**	-	-	41,506	32,766	29,710	31,774	27,059	20,859	45,570	43,138	26,316	35,623	28,459	-
	wmt	**394,327**	-	-	45,115	35,615	32,293	34,537	29,412	22,673	49,532	46,889	28,604	38,721	30,934	-
Moisture Content	%	**8%**			8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	
Minimum Deduction	Units	**1%**			1%	1%	1%	1%	1%	1%	1%	1%	1%	1%	1%	
Cu Payable	% Payable	**97%**			97%	97%	97%	97%	97%	97%	97%	97%	97%	97%	97%	
Cu Payable based on Min. deduction	%	**24%**			24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	
Cu Payable based on %	%	**24%**			24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	
Payable Cu in Cu Concentrate	ktonnes	**85**	-	-	10	8	7	7	6	5	11	10	6	8	7	-
	Mlbs	**188**	-	-	22	17	15	16	14	11	24	22	14	18	15	-
Revenues Cu in Cu Concentrate	**US$M**	**529.8**	**-**	**-**	**60.6**	**47.9**	**43.4**	**46.4**	**39.5**	**30.5**	**66.5**	**63.0**	**38.4**	**52.0**	**41.6**	**-**
Minimum Deduction	Units															
Zn Payable	% Payable															
Zn Payable based on Min. deduction	%	**7%**			4%	6%	8%	9%	10%	13%	4%	5%	10%	7%	6%	
Zn Payable based on %	%															
Payable Zn in Cu Concentrate	ktonnes		-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mlbs		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Revenues Zn in Cu Concentrate	**US$M**		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Min. Deduction Ag in Cu Conc	g/t Ag	**31.10**	-	-	31.10	31.10	31.10	31.10	31.10	31.10	31.10	31.10	31.10	31.10	31.10	-
Ag Payable	% Payable	**90%**			90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	
Payable Ag in Cu Concentrate	Ag koz	**12,345**			337	322	676	813	1504	1276	1438	1838	2052	1663	426	
Revenues Ag in Cu Concentrate	**US$M**	**200.8**	**-**	**-**	**5.5**	**5.2**	**11.0**	**13.2**	**24.5**	**20.8**	**23.4**	**29.9**	**33.4**	**27.0**	**6.9**	**-**
Min. Deduction Au in Cu Conc	g/t Au	**1.24**	-	-	1.24	1.24	1.24	1.24	1.24	1.24	1.24	1.24	1.24	1.24	1.24	-
Au Payable	% Payable	**90%**			90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	
Payable Au in Cu Concentrate	Ag koz	**46**	-	-	2	2	2	2	4	3	3	6	9	8	3	-
Revenues Au in Cu Concentrate	**US$M**	**59.0**	**-**	**-**	**3.1**	**3.1**	**2.9**	**3.0**	**4.6**	**4.4**	**4.1**	**8.1**	**12.0**	**10.0**	**3.6**	**-**
Cu Treatment Charge	US$/dmt conc	**86.00**	-	-	86.00	86.00	86.00	86.00	86.00	86.00	86.00	86.00	86.00	86.00	86.00	-
	US$M	**31.2**	-	-	3.6	2.8	2.6	2.7	2.3	1.8	3.9	3.7	2.3	3.1	2.4	-
Cu Refining Charge	US$/lb	**0.09**	-	-	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	-
	US$M	**16.2**	-	-	1.8	1.5	1.3	1.4	1.2	0.9	2.0	1.9	1.2	1.6	1.3	-
Ag Refining Charge	US$/Ag oz	**0.75**	-	-	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	-
	US$M	**9.3**	-	-	0.3	0.2	0.5	0.6	1.1	1.0	1.1	1.4	1.5	1.2	0.3	-
Au Refining Charge	US$/Au oz	**6.00**	-	-	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	-
	US$M	**0.3**	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	-
Total Treatment + Refining Charges	**US$M**	**56.9**	**-**	**-**	**5.7**	**4.5**	**4.4**	**4.8**	**4.7**	**3.7**	**7.0**	**7.0**	**5.0**	**5.9**	**4.1**	**-**
Zn Penalties	US$/tonne per %	**2.00**	-	-	2.00	2.00	2.00	2.00	2.00	2.00	-	2.00	2.00	2.00	2.00	-
	% over max	**3.0**	-	-	0.21	2.00	3.96	5.40	5.90	9.16	-	0.81	6.45	3.33	2.17	-
	US$/tonne	**6.1**	-	-	0.42	3.99	7.91	10.79	11.80	18.32	-	1.63	12.91	6.66	4.34	-
	US$M	**2.2**	-	-	0.02	0.13	0.24	0.34	0.32	0.38	-	0.07	0.34	0.24	0.12	-
Cu Concentrate Transport Cost	US$/wmt conc	**91.00**	-	-	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	-
	US$M	**35.9**	-	-	4.1	3.2	2.9	3.1	2.7	2.1	4.5	4.3	2.6	3.5	2.8	-
NSR Cu in Cu Concentrate	US$M	**482.4**	-	-	55.2	43.6	39.5	42.3	36.0	27.7	60.6	57.4	35.0	47.4	37.8	-
NSR Zn in Cu Concentrate	US$M	**-2.2**	-	-	- 0	- 0	- 0	- 0	- 0	- 0	-	- 0	- 0	- 0	- 0	-
NSR Ag in Cu Concentrate	US$M	**191.5**	-	-	5.2	5.0	10.5	12.6	23.3	19.8	22.3	28.5	31.8	25.8	6.6	-
NSR Au in Zn Concentrate	US$M	**58.7**	-	-	3.1	3.1	2.9	3.0	4.5	4.4	4.1	8.1	12.0	9.9	3.5	-
Total Cu Concentrate NSR	**US$M**	**694.6**	**-**	**-**	**59.4**	**48.3**	**49.7**	**54.4**	**60.9**	**49.5**	**82.5**	**89.6**	**75.9**	**79.3**	**45.1**	**-**

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	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Zn CONCENTRATE																
Recovery to Zn Concentrate	% Zn	**91.5%**			93.1%	92.9%	91.8%	91.4%	91.6%	90.8%	90.3%	90.6%	91.2%	91.3%	92.5%	
	% Cu	**6.3%**			6.3%	6.3%	6.3%	6.3%	6.3%	6.3%	6.3%	6.3%	6.3%	6.3%	6.3%	
	% Ag	**20.1%**			20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	
	% Au	**20.1%**			20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	20.1%	
Metal in Zn Concentrate	Zn ktonnes	**484**	-	-	34	38	45	57	51	52	32	39	52	50	34	-
	Zn Mlbs	**1,068**	-	-	75	84	100	125	113	115	71	86	115	109	75	-
	Cu ktonnes	**6**	-	-	1	1	1	1	0	0	1	1	0	1	0	-
	Cu Mlbs	**14**	-	-	2	1	1	1	1	1	2	2	1	1	1	-
	Ag kg	**124,326**	-	-	3,674	3,449	6,896	8,257	14,997	12,703	14,510	18,411	20,361	16,633	4,433	-
	Ag koz	**3,997**	-	-	118	111	222	265	482	408	467	592	655	535	143	-
	Au kg	**821**	-	-	55	50	46	48	63	59	67	110	144	126	53	-
	Au koz	**26**	-	-	2	2	1	2	2	2	2	4	5	4	2	-
Pull Factor		**16**	-	-	18	15	17	13	15	14	24	19	15	15	12	-
Zn Concentrate Grade	% Zn	**61.3%**			61.3%	61.3%	61.3%	61.3%	61.3%	61.3%	61.3%	61.3%	61.3%	61.3%	61.3%	
	% Cu	**0.80%**			1.3%	0.9%	0.7%	0.6%	0.6%	0.4%	1.5%	1.2%	0.5%	0.8%	0.9%	
	g/t Ag	**157.37**	-	-	66.45	55.60	93.48	89.02	179.80	149.90	275.75	287.87	238.85	205.37	80.29	-
	Ag oz	**5**	-	-	2	2	3	3	6	5	9	9	8	7	3	-
	g/t Au	**1.04**	-	-	0.99	0.81	0.63	0.52	0.76	0.69	1.28	1.72	1.69	1.55	0.96	-
	Au oz	**0.03**	-	-	0.03	0.03	0.02	0.02	0.02	0.02	0.04	0.06	0.05	0.05	0.03	-
Zn Concentrate Produced	dmt	**790,035**	-	-	55,295	62,044	73,766	92,749	83,410	84,748	52,622	63,956	85,247	80,992	55,207	-
	wmt	**858,734**	-	-	60,103	67,439	80,181	100,814	90,663	92,117	57,198	69,517	92,660	88,034	60,008	-
Moisture Content	%	**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%	
Zn Payable	% Payable	**85%**			85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	
Zn Payable based on Min. deduction	%	**53%**			53%	53%	53%	53%	53%	53%	53%	53%	53%	53%	53%	
Zn Payable based on %	%	**52%**			52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	
Payable Zn in Zn Concentrate	ktonnes	**412**	-	-	29	32	38	48	43	44	27	33	44	42	29	-
	Mlbs	**908**	-	-	64	71	85	107	96	97	60	73	98	93	63	-
Revenues Zn in Zn Concentrate	**US$M**	**1,111**			**78**	**87**	**104**	**130**	**117**	**119**	**74**	**90**	**120**	**114**	**78**	
Minimum Deduction	Units															
Cu Payable	% Payable															
Cu Payable based on Min. deduction	%	**1%**			1%	1%	1%	1%	1%	0%	2%	1%	1%	1%	1%	
Cu Payable based on %	%															
Payable Cu in Zn Concentrate	ktonnes		-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mlbs		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Revenues Cu in Zn Concentrate	**US$M**		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Min. Deduction Ag in Zn Conc	g/t Ag	**93.30**	-	-	93.30	93.30	93.30	93.30	93.30	93.30	93.30	93.30	93.30	93.30	93.30	-
Ag Payable	% Payable	**70%**			70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	
Payable Ag in Zn Concentrate	Ag koz	**1,250**	-	-	-	-	0	-	162	108	216	280	279	204	-	-
Revenues Ag in Zn Concentrate	**US$M**	**20.3**	**-**	**-**	**-**	**-**	**0.0**	**-**	**2.6**	**1.8**	**3.5**	**4.6**	**4.5**	**3.3**	**-**	**-**
Min. Deduction Au in Zn Conc	g/t Au	**0.31**	-	-	0.31	0.31	0.31	0.31	0.31	0.31	0.31	0.31	0.31	0.31	0.31	-
Au Payable	% Payable	**70%**			70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	
Payable Au in Zn Concentrate	Au koz	**13**	-	-	1	1	1	0	1	1	1	2	3	2	1	-
Revenues Au in Zn Concentrate	**US$M**	**16.8**	**-**	**-**	**1.1**	**0.9**	**0.7**	**0.6**	**1.1**	**0.9**	**1.5**	**2.6**	**3.4**	**2.9**	**1.0**	**-**
Zn Treatment Charge	US$/dmt conc	**172.00**	-	-	172.00	172.00	172.00	172.00	172.00	172.00	172.00	172.00	172.00	172.00	172.00	-
	US$M	**135.9**	-	-	9.5	10.7	12.7	16.0	14.3	14.6	9.1	11.0	14.7	13.9	9.5	-
Ag Refining Charge	US$/Ag oz	**0.50**	-	-	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	-
	US$M	**0.6**	-	-	-	-	0.0	-	0.1	0.1	0.1	0.1	0.1	0.1	-	-
Au Refining Charge	US$/Au oz	**6.00**	-	-	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	-
	US$M	**0.1**	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-
Total Treatment + Refining Charges	**US$M**	**136.6**	**-**	**-**	**9.5**	**10.7**	**12.7**	**16.0**	**14.4**	**14.6**	**9.2**	**11.2**	**14.8**	**14.0**	**9.5**	**-**

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Constantine Metal Resources Ltd.
Palmer Project, Alaska, US
2019 PEA - Economic Model

	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Zn Concentrate Transport Cost	US$/wmt conc	91.00	-	-	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	-
	US$M	78.1	-	-	5.5	6.1	7.3	9.2	8.3	8.4	5.2	6.3	8.4	8.0	5.5	-
NSR Zn in Zn Concentrate	US$M	975.5	-	-	68.28	76.6	91.1	114.5	103.0	104.6	65.0	79.0	105.3	100.0	68.2	-
NSR Cu in Zn Concentrate	US$M		-	-	-	-	-	-	-	-	-	-	-	-	-	-
NSR Ag in Zn Concentrate	US$M	19.7	-	-	-	-	0.0	-	2.6	1.7	3.4	4.4	4.4	3.2	-	-
NSR Au in Zn Concentrate	US$M	16.7	-	-	1.1	0.9	0.7	0.6	1.1	0.9	1.5	2.6	3.4	2.9	1.0	-
Total Zn Concentrate NSR	**US$M**	933.8	-	-	63.9	71.4	84.5	105.9	98.4	98.9	64.7	79.7	104.6	98.1	63.7	-
BARITE CONCENTRATE																
Recovery to Barite Concentrate	% Barite	91.1%			91.1%	91.1%	91.1%	91.1%	91.1%	91.1%	91.1%	91.1%	91.1%	91.1%	91.1%	
Metal in Barite Concentrate	Ba ktonnes	1,514	-	-	82	90	106	106	180	147	154	185	212	156	96	-
	Ba Mlbs	3,337	-	-	181	198	234	233	397	325	339	408	468	344	211	-
Barium Concentrate Grade	% Ba	52%			52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	
Barite Concentrate Produced	dmt	2,894,203			156,636	171,843	202,706	201,796	344,233	281,451	294,405	353,747	406,140	298,003	183,243	-
	wmt	2,923,438	-	-	158,219	173,578	204,754	203,834	347,710	284,294	297,379	357,321	410,242	301,013	185,094	-
Moisture Content	%	1%			1%	1%	1%	1%	1%	1%	1%	1%	1%	1%	1%	
Revenues Barite Concentrate	**US$M**	636.7	-	-	34	38	45	44	76	62	65	78	89	66	40	-
Barite Treatment Charge	US$/dmt conc		-	-	-	-	-	-	-	-	-	-	-	-	-	-
	US$M		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Barite Concentrate Transport Cost	US$/wmt	132.00	-	-	132.00	132.00	132.00	132.00	132.00	132.00	132.00	132.00	132.00	132.00	132.00	-
	US$M	385.9	-	-	20.9	22.9	27.0	26.9	45.9	37.5	39.3	47.2	54.2	39.7	24.4	-
Total Barite Concentrate NSR	**US$M**	250.8	-	-	13.58	14.9	17.6	17.5	29.8	24.4	25.5	30.7	35.2	25.8	15.9	-
ROYALTIES																
NSR Royalty paid to third parties	US$M	47.0	-	-	3.4	3.4	3.8	4.4	4.7	4.3	4.3	5.0	5.4	5.1	3.1	-
Total NSR	US$M	1,832.2	-	-	133.5	131.2	148.0	173.3	184.3	168.5	168.3	195.0	210.3	198.2	121.6	-
	US$/tonne	146.8	-	-	134.81	142.08	121.43	138.73	146.40	143.20	133.60	157.90	166.91	159.37	182.75	-
OPEX																
UG Mining	US$M	362.7			26.7	29.7	36.1	36.2	36.2	35.4	36.0	35.6	38.3	35.3	17.2	-
	US$/tonne mined	29.06			26.96	32.12	29.63	28.98	28.74	30.09	28.58	28.83	30.41	28.37	25.90	
Processing	US$M	210.0	-	-	16.7	15.5	20.5	21.0	21.2	19.8	21.2	20.8	21.2	20.9	11.2	-
	US$/tonne milled	16.83	-		16.83	16.83	16.83	16.83	16.83	16.83	16.83	16.83	16.83	16.83	16.83	-
G&A	US$M	103.3	-	-	10.0	9.6	9.8	9.6	9.6	9.5	9.8	9.6	9.5	9.4	6.8	-
	US$/tonne milled	8.28			10.12	10.42	8.05	7.70	7.60	8.10	7.77	7.74	7.55	7.59	10.29	
Total OPEX	**US$M**	676.1	-	-	53.4	54.8	66.4	66.9	66.9	64.7	67.0	65.9	69.0	65.6	35.3	-
	US$/tonne milled	54.17	-	-	53.92	59.37	54.51	53.51	53.16	55.01	53.18	53.40	54.79	52.78	53.02	-
Net Operating Income	US$M	1,156.2	-	-	80.1	76.4	81.6	106.5	117.4	103.8	101.3	129.0	141.3	132.6	86.3	-
	US$/tonne	92.63	-	-	80.90	82.70	66.93	85.22	93.23	88.19	80.41	104.49	112.13	106.58	129.73	-

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Constantine Metal Resources Ltd.
Palmer Project, Alaska, US
2019 PEA - Economic Model

	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
CAPEX																
Mining	US$M	**163.1**	14.3	41.1	18.1	20.7	11.2	7.5	10.1	9.2	9.4	8.3	7.2	3.8	2.0	-
Site Development	US$M	**12.7**	10.5	1.7	0.1	0.0	0.0	-	0.4	-	-	-	-	-	-	-
Mineral Processing	US$M	**77.3**	10.0	64.6	-	-	0.5	0.9	-	-	0.5	0.9	-	-	-	-
Tailings Mangement	US$M	**5.1**	0.5	1.5	0.3	0.4	0.4	0.3	0.3	0.3	0.3	0.3	0.3	0.2	-	-
On-Site Infrastructure	US$M	**35.4**	5.9	28.1	-	-	-	0.2	0.7	0.2	-	0.2	-	-	-	-
Off-Site Infrastructure	US$M	**0.3**	0.3	0.1	-	-	-	-	-	-	-	-	-	-	-	-
Project Indirects	US$M	**26.4**	4.1	22.3	-	-	-	-	-	-	-	-	-	-	-	-
EPCM	US$M	**32.2**	17.7	14.5	-	-	-	-	-	-	-	-	-	-	-	-
Owner Costs	US$M	**7.6**	0.5	7.0	-	-	-	-	-	-	-	-	-	-	-	-
Closure	US$M	**30.8**														30.8
Salvage Value	US$M	**-5.9**														-5.9
Subtotal	US$M	**385.0**	63.8	181.0	18.5	21.1	12.0	8.9	11.6	9.7	10.1	9.7	7.5	4.0	2.0	24.9
Contingency	US$M	**32.7**	3.3	29.4	-	-	-	-	-	-	-	-	-	-	-	-
Total CAPEX	**US$M**	**417.7**	**67.1**	**210.4**	**18.5**	**21.1**	**12.0**	**8.9**	**11.6**	**9.7**	**10.1**	**9.7**	**7.5**	**4.0**	**2.0**	**24.9**
	US$/tonne	**33.46**	**-**	**-**	**18.7**	**22.8**	**9.9**	**7.1**	**9.2**	**8.2**	**8.0**	**7.9**	**6.0**	**3.2**	**3.1**	**-**
Pre-Production	**US$M**	**277.5**	**67.1**	**210.4**												
Sustaining	**US$M**	**140.1**			**18.5**	**21.1**	**12.0**	**8.9**	**11.6**	**9.7**	**10.1**	**9.7**	**7.5**	**4.0**	**2.0**	**24.9**
		11.23														
Working Capital	US$M			13.3											-13.3	
Net Pre-Tax Cash Flow	**US$M**	**738.5**	**-67.1**	**-223.8**	**61.6**	**55.3**	**69.5**	**97.6**	**105.8**	**94.1**	**91.2**	**119.3**	**133.7**	**128.6**	**97.6**	**-24.9**
Cumulative Net Pre-Tax Cash Flow	**US$M**		**-67.1**	**-290.9**	**-229.3**	**-174.0**	**-104.5**	**-6.9**	**98.9**	**193.0**	**284.2**	**403.5**	**537.2**	**665.8**	**763.4**	**738.5**
Taxes	**US$M**	**157.7**	**-**	**-**	**0.9**	**4.4**	**6.0**	**11.2**	**13.7**	**9.8**	**11.0**	**21.3**	**30.5**	**29.7**	**19.4**	**-**
Net Post-Tax Cash Flow	**US$M**	**580.7**	**-67.1**	**-223.8**	**60.7**	**50.9**	**63.6**	**86.4**	**92.1**	**84.2**	**80.2**	**98.1**	**103.3**	**98.9**	**78.2**	**-24.9**
Cumulative Net Post-Tax Cash Flow	**US$M**		**-67.1**	**-290.9**	**-230.2**	**-179.3**	**-115.7**	**-29.3**	**62.8**	**147.1**	**227.3**	**325.3**	**428.6**	**527.5**	**605.7**	**580.7**

ECONOMIC INDICATORS		
Pre-Tax Results		
Pre-Tax NPV @ 0%	**US$M**	**738.5**
Pre-Tax NPV @ 7%	**US$M**	**354.4**
IRR	**%**	**24%**
Payback	**Years**	**3.07**
Post-Tax Results		
Post-Tax NPV	**US$M**	**580.7**
Post-Tax NPV	**US$M**	**266.1**
IRR	**%**	**21.1%**
Payback	**Years**	**3.32**

Source: JDS (2019)

23 Adjacent Properties

There are no properties of merit in the immediate area of the Palmer Exploration Project. However, there are two VMS deposits in the greater district with some similarities to Palmer.

23.1 Greens Creek Ag-Zn-Pb-Au VMS Deposit, Admiralty Island, Alaska, USA

The Greens Creek Mine, owned by the Hecla Greens Creek Mining Company, is located 175 km southeast of the Palmer Project on Admiralty Island, Alaska. It is one of the largest and lowest-cost primary silver mines in the world. In 2017, Greens Creek produced 8.4 million ounces of silver at a cash cost, after by-product credits, per silver ounce of US$0.71 (a non-GAAP measure), and 50,854 ounces of gold (Table 23-1). Production in 2018 is expected to be 7.5 to 8.0 million silver ounces at a cash cost, after by-product credits of US$0.50 an ounce.

Table 23-1: Greens Creek Mine – Annual Production

Production (years ended December 31st)	2014	2015	2016	2017
Silver (ounces)	7,826,341	8,452,153	9,253,543	8,351,882
Gold (ounces)	58,753	60,566	53,912	50,854
Lead (tons)	20,151	21,617	20,596	17,996
Zinc (tons)	59,810	61,934	57,729	52,547
Cash cost per ounce of silver, after by-product credits, ($/oz) (1)	$2.89	$3.91	$3.84	$0.71

Source: https://www.hecla-mining.com/greens-creek/

The Greens Creek deposit is a polymetallic, stratiform, massive sulfide deposit located within the Admiralty sub-terrane of the Alexander Terrane (similar to Palmer). The host rock consists of predominantly marine sedimentary, and mafic to ultramafic volcanic and plutonic rocks, which have been subjected to multiple periods of deformation. These deformational episodes have imposed multiple folding of the mineralized bodies to create a complex geometry. Mineralization occurs discontinuously along the contact between a structural hanging wall of quartz mica carbonate phyllites, and a structural footwall of graphitic and calcareous argillite.

Ore lithologies fall into two broad groups: massive ores with over 50% sulfides and white ores with less than 50% sulfides. The massive ores are further subdivided as either being base-metal or pyrite dominant. Massive ores vary greatly in precious-metal grade from uneconomic to bonanza Au (>.5 opt) and Ag (>100 opt). White ores are subdivided into three groups by the dominant gangue mineralogy white carbonate, white siliceous, and white baritic ore. These ores tend to be base-metal poor and precious-metal rich. Major sulfide minerals are pyrite, sphalerite, galena, and tetrahedrite/tennantite.

Greens Creek is an underground mine which produces approximately 2,100 to 2,300 tons of ore per day. The primary mining methods are cut and fill and long hole stoping.

Information with respect to proven and probable ore reserves, measured, and inferred resources is set forth below in Table 23-2. (As of December 31st, 2017, unless otherwise noted).

Table 23-2: Greens Creek Mine - Proven and Probable Reserves

Measured and Inferred Resources (as of December 31, 2017)									
	Tons (000)	Silver (oz/ton)	Gold (oz/ton)	Lead (%)	Zinc (%)	Silver (000 oz)	Gold (000 oz)	Lead (Tons)	Zinc (Tons)
Proven Reserves (1,2)	7	12.2	0.09	2.4	6.1	89	1	170	440
Probable Reserves (1,2)	7,543	11.9	0.10	3.0	8.1	90,130	725	224,880	614,390
Proven and Probable Reserves	7,550	11.9	0.10	3.0	8.1	90,219	725	225,050	614,840
Measured Resources (3)	341	9.1	0.09	2.4	8.3	3,086	30	8,090	28,420
Indicated Resources (3)	2,464	11.4	0.09	2.9	7.6	28,211	229	72,120	187,060
M&I Resources	2,805	11.2	0.09	2.9	7.7	31,296	259	80,210	215,480
Inferred Resources (3)	2,708	12.1	0.08	2.7	6.9	32,711	222	73,350	185,66

Source: https://www.hecla-mining.com/greens-creek/

23.2 Windy Craggy Cu-Co-Au VMS Deposit, BC, Canada

The 297 million tonne Windy Craggy deposit, located 60 km northwest of the Palmer Project, is the world's fourth largest VMS deposit by size, and tops the list as the largest of the copper-rich (Besshi type) category of VMS deposits. It is situated in the Alsek-Tatshenshini River area of the St. Elias Mountains, within the confines of the Tatshenshini-Alsek Provincial Park, designated World Heritage Site by UNESCO.

Windy Craggy lies within the allochthonous Alexander terrane (similar to Palmer) which comprises a thick succession of complexly deformed Proterozoic to Permian basinal and platformal carbonate and clastic rocks with a subordinate volcanic component. These rocks have been subject to relatively low-grade metamorphism and are unconformably overlain by a Late Triassic succession of calcareous turbidites and a mafic volcanic suite which host the Windy Craggy deposit.

Continuous massive sulfide mineralization is developed over a minimum strike length of 1600 m, at least 600 m vertical extent, and greater than 200 m in width. It appears to consist of two discrete sulfide bodies, the North and South Sulfide Bodies, each with a variably developed stockwork/stringer zone. The tabular to lenticular, concordant North Sulfide Body is about 120-150 m thick by 500 m in diameter. The body is elongated in a WNW direction and dips moderately to steeply to the NNE. The South Sulfide Body is more deformed, is lensoidal and plunges steeply to the SE, extending to the SE as a series of 15 to 60 m wide massive sulfide lenses. The massive sulfides and enclosing hosts have been subjected to two phases of deformation producing isoclinal and open folds respectively. The main faults close to the deposit strike are steeply dipping and strike NW, subparallel to the strike of the host rocks.

Measured geological reserves at the Windy Craggy deposit have been quoted at:

- 297 Mt @ 1.38% Cu, 0.2 g/t Au, 3.83 g/t Ag and 0.069% Co using a 0.5% Cu cut-off, to
- 198 Mt @ 1.75% Cu using a 1.0% Cu cut-off, or
- 139 Mt @ 1.96% Cu using a 1.5% Cu cut-off.

This summary is based on the more detailed online British Columbia Geological Survey MINFILE record summarizing this deposit.

http://minfile.gov.bc.ca/Summary.aspx?minfilno=114P++002

24 Other Relevant Data and Information

The authors are not aware of any information that is relevant to this Project that has not been disclosed in the Study.

25 Interpretations and Conclusions

25.1 Risks

There are several risks associated with the Project that should be considered. Some are generic and shared by nearly all mining projects, including:

- Sensitivity to metal pricing (as discussed in Section 22.6)
- Cost escalation
- Permitting difficulties, costs, and delays
- Efficiency of construction management: a project can be properly estimated and improperly constructed, resulting in significant construction cost overruns.

There are also several site-specific risks that are identified and discussed in detail in the following sections.

It should be noted that the project is not sensitive to any exchange rates, as all revenues and nearly all capital and operating cost items were sourced in $US rather than converted from other currencies.

25.1.1 Avalanche

Glacier Creek valley is subject to snow avalanches between October and June with the most active periods between November and April, owing to high snowfall and steep terrain. Constantine has been studying the local avalanche cycles since 2010 in order to understand and mitigate avalanche risk. The results of that monitoring program suggest that the access road to the 680 Exploration Portal site is subject to periodic avalanches that could restrict access both during periods of high avalanche danger and during snow clearing operations after avalanches. The monitoring program has also informed design and placement of proposed mine infrastructure.

The mill site, backfill plant and LNG power plant have been located in areas mapped as low hazard zones. Metal shed coverings were assumed for both the 680 Exploration Portal and the 510 Conveyor Portal for the protection of workers and equipment. Secondary egress and/or ventilation adits for both mines were located in low risk rock outcrop areas. 300 Kt of NPAG rock in the development schedule was assumed to be used for construction activities during the pre-production period, including the construction of avalanche deflection berms as required.

An Operational Avalanche Safety Plan will be required during winter operation, which will include site-specific weather and avalanche forecasting, road closures and artificial triggering and cleanup. No operational downtime or special equipment other than the usual snow removal fleet was included for avalanche control measures, as the mines can be accessed from separate portals. It is assumed that mine entry via the 510 Conveyor Portal will be possible during periods of time when the road to the 680 Exploration Portal is blocked from the clean-up activities after intentionally triggered avalanches.

25.1.2 510 Conveyor Portal Construction

The 510 Conveyor Portal is determined by the location of the processing plant and the centroid of the orebody. The talus slope it is collared into is less than ideal for the construction of a mine adit and could prove to be more costly and time-consuming than currently considered in the PEA. This was mitigated by using a ground penetrating radar analysis, supported by one visual outcrop, to locate the portal in the minimum depth of talus based on existing information.

25.1.3 AG Metallurgical Response

The test work completed on the AG Zone deposit is limited to mineralogy. Recoveries for the deposit were based on a comparison of the AG Zone deposit mineralogy to the Palmer deposit and existing flotation test work performed on Palmer deposit mineralization. The copper and zinc recoveries were discounted for the AG Zone deposit because the copper is lower grade and present as tennantite rather than chalcopyrite. The actual recovery of copper in the AG Zone deposit will only be confirmed through testwork. This impacts the recovery of copper, but also the loss of zinc to the copper concentrate. This risk is further complicated by the high lead grade in the AG Zone deposit, as compared to the Palmer deposit, which may require a combined lead-copper flotation or separate lead flotation circuit.

25.1.4 Site Surface Geotechnical Conditions

No site-specific subsurface information is currently available in the vicinity of the mill site or TMF/WRSF except for a single line of geophysical surveying near the mill site.

Characterization of foundations is a key step during design to identify potential critical conditions. In the PEA design, foundation conditions are largely assumed based on surface observations. Key design considerations for foundation characterization include:

- Potential presence of weak layers that could govern stability of the TMF/WRSF, pond embankments and other structures on site;
- Potential presence of soft or weak layers that could result in differential or excess settlement of building foundations;
- Potential for foundation strength loss or liquefaction (static or dynamic) which could govern stability of all structures; and
- Hydrogeologic conditions, particularly beneath the TMF/WRSF pile and collection ponds (i.e. influencing uplift of liner during construction).

The processing plant is located on deep till. Rafting foundations have been assumed for the larger equipment and adequate allowances have been included in the design and costs estimate to allow for this requirement.

25.1.5 Water Management

The risks associated with water management on site are as follows:

- Basic assumptions were made for surface and underground water flows based on preliminary drilling and hydro-geologic information.
- Flows from the AG Zone workings are not yet defined and have not been addressed in the water balance. The impact of AG Zone inflows on water management facility design and water treatment requirements are not known.

- Little information is available on the hydraulic conductivity of deeper sections of the ore deposits. Consequently, estimates of potential inflows are restricted to approximate assessments of recharge based sustained average flows and little information is available on peak flow rates.
- The underground water collection and treatment strategy includes identification of water requiring treatment and separation from water that does not. Inability to differentiate would increase the amount of water requiring treatment and associated cost.
- Geochemical test work to date indicates runoff from tailings will be relatively benign and operational controls will be in place for geochemical monitoring and management to confirm this. Therefore, the PEA design includes discharge of contact water collected at the CWCP after settlement to the TMF/WRSF LAD without additional treatment either during operations or upon closure. Additional tailings and waste rock characterization and more detailed operational planning will be required in future design stages to confirm indications of initial test work. Future results may indicate that water treatment at the TMF/WRSF may be necessary.

25.1.6 Seismicity

Stability analysis indicates that under maximum design earthquake (MDE) loadings, seismic displacements of the TMF/WRSF are possible. Deformations could impact pile integrity, pile infrastructure (roads, pipelines, etc.) and liner and drainage systems. Other site structures may also be impacted by seismic loading.

25.1.7 Geochemical Management (Waste Materials)

Geochemical testing to date on tailings is limited to initial pilot plant samples, and geochemical characterization of all waste rock has not yet been carried out. The limited testing represents uncertainty related to geochemical management. However, preliminary results from tailings testing indicate that the mill circuit is successful at removing the majority of sulfides from the tailings and the material is relatively benign. Pyrite tailings will be stored underground. A portion of PAG waste rock will be temporarily stored at the mill site before being placed underground later in operations.

25.1.8 TMF/WRSF Dust Management

Wind-blown tailings could impact and exceed air quality standards if areas of the TMF/WRSF pile are left unmitigated. Although the TMF/WRSF design includes progressive reclamation and waste rock armoring of pile slopes, filtered tailings can be susceptible to dusting if left exposed. Temporary dust management alternatives prior to placement of a reclamation cover include: synthetic dust suppressants, wind fences and temporary sand and gravel erosion protection layers.

25.1.9 TMF/WRSF Post-Closure

Long-term closure goals for the TMF/WRSF include providing a stable pile and limiting long-term risk to the downstream environment (safety, water and air quality). The PEA design addresses these goals by eliminating ponded water on the TMF/WRSF pile surfaces, routing water around the pile and progressively covering the pile with a low-permeability cover to limit water and oxygen ingress. Pile drain down water will be greatly reduced upon closure due to the completion of the low-permeability cover. Long-term risks to water quality are not fully defined by short-term geochemical testing and the need for additional long-term measures is uncertain. Differential settlement of tailings has the potential to modify drainage paths resulting in formation of localized ponds.

25.2 Opportunities

The Project has several opportunities to improve the current results that should be investigated further as part of the ongoing development of the Project.

25.2.1 Expansion and Definition of Resources

The simplest and most obvious opportunity is to continue to explore the property to expand and better define the resources. The current level of drilling is sufficient to define a resource but has certainly not closed off either deposit and there is ample opportunity to either or both. An expansion of mineral resources would allow the mine plan to contemplate either a higher mining rate, longer mine life, or some combination of both. Increased resource definition through tighter drill spacing would allow the resource to form the basis of a reserve for a PFS or FS.

25.2.2 Sale of Pyrite

The current study treats pyrite as a waste product that must be returned underground as paste backfill because of its acid-generating potential. Pyrite is a saleable commodity, however, as it is used to generate sulfuric acid. As the pyrite is already ground, floated, and filtered there would be no incremental processing costs associated with making the product saleable. If a contract could be arranged with an adequate margin to ship the product to a smelter, this could be a significant opportunity.

There will be 3.0 Mt of pyritic tails generated over the LOM. This total exceeds the quantity of material (NPAG rock and de-sulfide tails) sent to the TSF. As such, it is possible that pyrite sales could result in the elimination of a TSF for surface waste storage. As such, it may even be beneficial to the Project to sell the pyrite at a loss.

25.2.3 AG Zone Deposit Metallurgy

While the limited testwork for the AG Zone deposit has been identified as a risk in Section 25.1.3, it must also be considered an opportunity. AG Zone deposit recoveries were estimated by applying a 10% deduction recovery to the copper (from 89% to 79%) and a 5% to the zinc from the AG Zone deposit (from 93% to 88%). Should testwork prove these deductions to be conservative, the economics of the AG Zone deposit will be improved. Similarly, the increased presence of lead may impose a complication on the copper recovery, but it may also provide an additional revenue stream from the sale of a lead concentrate or a combined copper-lead concentrate.

25.2.4 Good Used Equipment

The purchase and employment of good used equipment is always a consideration to be made to reduce capital costs. In general, this should not be considered for the mobile mining, as the savings do not warrant the reduction of availability, making production targets hard to achieve. However, much of the major equipment list for the processing plant, backfill plant, the surface fleet, and power generation and distribution could be comprised of used units should suitable unit be available at the time of procurement.

Realistic assumptions and expectations should be applied to the purchase of used equipment, as there are usually significant costs for rebuilding, retrofitting, or modernizing componentry.

25.2.5 Conveyor for AG Zone Deposit

Haulage costs for the AG Zone deposit are significant due to the long haulage distance. A trade-off study should be performed to determine if a second conveyor should be installed to replace the truck haulage. This would require that a second crusher installation be installed for the AG Zone deposit to size rock for the conveyor.

26 Recommendations

26.1 General

The primary finding of this PEA is that the Project has the potential to be economic based on the data acquired to-date and the mine model that was produced for this evaluation. Accordingly, it is recommended that exploration continue on the Project and that the company advance its permits and baseline environmental data collection with the ultimate goal of constructing the mine and putting it into production.

To this end, the Authors believe continued delineation drilling is warranted and has the potential expand the known resource and increase its confidence. Once this is done, it would be worthwhile to replace this PEA with a pre-feasibility study (PFS) or feasibility study (FS) that includes a mining reserve based on a mine model applied to the expanded resource.

Given the mountainous terrain, this drilling can only be effectively achieved from underground development. There is also significant potential to discover additional mineralized zones within the greater Palmer Property. Accordingly, the planned exploration adit, drift, and underground drill program is recommended.

26.2 Recommended Work Plan – Exploration Program

The following activities are recommended as part of the next exploration program, currently anticipated for summer 2019:

- Review the option of a lateral underground exploration adit to provide access to the mineral resource area for delineation drilling, hydrological and geotechnical studies, and metallurgical testing. This may be more cost effective for drilling on close-spaced centers for conversion from Inferred to Indicated mineral resource categories than drilling from surface and would also facilitate year-round drilling, which is currently impractical during the winter months.
- Conduct geotechnical, hydrogeological, engineering, environmental, avalanche risk studies and permitting work to aid in the assessment of a conceptual underground exploration development.
- Prepare and submit a Plan of Operations permit application in support of the conceptual underground exploration program.
- Conduct 30,000 m of resource-scale definition and exploration drilling on 100 m and 50 m nominally spaced centers to test the limits of the SW, RW and AG mineralized zones. Priority drill areas would include the on-strike and down-dip extensions of the Palmer deposit¸ with emphasis on the potential 200 m down-dropped faulted offset of the SWZII-III-EM Zone, as well as, the expansion of the AG Zone deposit along strike and at depth, and the conversion of Inferred to Indicated mineral resources.
- Conduct 10,000 m of exploration drilling to test existing regional exploration targets.
- Develop of new regional exploration targets within the Federal and State mining claims, and within the greater Mental Health Trust Lands that are under lease.
- Update the mineral resource estimate to form the basis of a new PFS or FS Study.

26.3 Recommended Work Plan – Pre-feasibility or Feasibility Study

The logical next step for the project is to produce a pre-feasibility study (PFS) or Feasibility Study (FS). This effort would require the following field programs and engineering evaluations:

- Complete geotechnical characterization program for the PEA underground mine and infrastructure including geotechnical core drilling and oriented core and/or televiewer;
- Investigation of the talus slope to finalize the location of the 510 Conveyor Portal.
- Ongoing metallurgical testwork to confirm the flotation characteristics of the AG Zone deposit through lock-cycle test.
- Ongoing metallurgical testwork on blended samples of Palmer deposit and AG Zone deposit mineralization, matching the production forecast to simulate the predicted scheduled mill feed.
- A new avalanche hazard analysis should be performed now that site facilities have been generally located to identify specific concerns and countermeasures that should be applied to the next level of study. This would include design and location of avalanche deflection berms to protect the site roads, industrial complex, water treatment ponds and the two portals.
- The 510 Portal location should be finalized, and a detailed engineering design should be completed for a snow cover shed and avalanche deflection structure(s).
- Hydrogeological studies of AG Zone to assess flow rates and treatment requirements will be required. Potential measures to reduce inflows during operation such as construction of bulkheads between operational and depleted zones should be investigated.
- Continue to advance understanding of geochemistry and waste management strategies. Include TMF/WRSF water treatment as a contingency in project costing estimates if required. 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.
- It is recommended that a site investigation program to characterize the foundations and potential borrow materials be conducted. A detailed site investigation plan has not been prepared as part of the PEA, but would likely include the following:
    - Drilling to perform penetration tests (correlated to soil density), visually describe the soils, collect samples for laboratory testing, measure in-situ hydrogeologic properties of the soils and install geotechnical instrumentation (inclinometers and vibrating wire piezometers) or groundwater monitoring wells. Shear wave velocity measurements could be collected in standpipe piezometers to inform SSHA and liquefaction triggering assessments.
    - Cone penetration testing (CPT) to provide continuous soil classification that can be used in part of liquefaction triggering assessments. Shear wave velocities may also be measured.
    - Geophysics to infill data between drill holes and CPT soundings.
    - Test pits to characterize borrow sources.

- Mitigation measures for foundation improvement could be developed (weight drop or other foundation densification, stone columns, etc.), or if impractical, re-siting of the mill site, TMF/WRSF, CWCP and other structures could be required as a result of the investigation findings.
- Site-specific seismic hazard assessment (SSHA) to characterize seismic hazard to assess if MDE adopted is appropriate. Perform deformation analysis to quantify the magnitude of seismic deformations. If required, revise designs to reduce deformation to acceptable levels. Conduct a site-specific seismic hazard assessment to establish seismic loadings for structures and site facilities. Perform stability assessments to confirm designs comply with design criteria for static and seismic stability.
- Develop temporary storage contingencies outside of the TMF/WRSF for periods when pyrite tailings cannot be placed immediately underground. Options could include use of the lined temporary PAG rock / ore stockpile adjacent to the mill, a storage shed or a separate, contained area or section of the TMF/WRSF.
- Install instruments to establish baseline water quality and hydrogeological conditions.
- Optimize site-wide water balance to evaluate interactions between surface water storages and how the water management system will perform under prolonged dry or wet conditions, flood events, at different stages of the mine life, and following an operational upset of the WTP.
- Conduct additional testing on tailings and waste rock to define geotechnical properties. The next design stages for the TMF/WRSF should also anticipate settling and provide long term positive drainage to prevent ponding.
- Conduct additional testing on tailings and waste rock to further define geochemical properties. Assess tailings effluent quality to inform water management designs and water treatment needs.
- Conduct wind tunnel trials to assess dusting potential and appropriate mitigation strategies for the filtered tailings.
- Complete a constructability review, including material requirements and preparation of an execution plan, for the TMF/WRSF design to define risks to cost, schedule and identify areas for potential optimization.
- Undertake an operations review of the TMF/WRSF to assess whether adequate flexibility and management of risk have been incorporated into the design.
- Review the effect of tailings consolidation and differential settlement between structural and non-structural zones on closure cover design,
- Undertake a Failure, Modes, and Effects Analysis (FMEA) of the TMF/WRSF and other project components specific to technical assessments of risks, consequence, design resilience and potential operational failure modes. Results of the FMEA can be used (1) to provide guidance for instrumentation monitoring during operations; (2) to establish a surveillance program; and (3) to screen out failure modes that can be effectively managed by the Observational Method during operations.

The following trade-off evaluations should be performed as part of the next phase of work on this Project:

- The construction of a loading terminal in Haines to replace the Skagway facility.
- The optimal end-product form and packaging of the barite to maximize the sales margins.
- The best processing option for the AG Zone deposit should be evaluated, including preparation of a lead concentrate, suppression of the lead in the floatation circuits to maintain the value of the copper concentrate, or the generation of a cooper-lead concentrate.
- The economic viability of selling pyrite for the production of sulfuric acid off-site instead of returning it to the mine as backfill. Environmental baseline studies to include water quality sampling, species of interest studies, environmental rock geochemistry studies, and meteorological data collection.
- Continued ABA testwork to chemically characterize the TSF.
- Ongoing engagement with community, local stakeholders and governments with continued local hiring practices.

26.4 Recommended Budget

A proposed budget of US$30.0 million for the above recommendations is shown below in Table 26-1.

Table 26-1: Proposed Budget

Component	Estimated Cost ($M)	Comment
Resource and Exploration Drilling (Surface and Underground - All-in Cost with assays, helicopter, salaries, supplies)	**8.0**	Conversion of inferred to indicated & measured resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	**0.5**	Comminution, DMS, flotation optimization, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Underground Development	**18.0**	Access for underground drilling and possible bulk sample. Based on actual quotes.
Geochemistry	**0.4**	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	**0.6**	Site investigation drilling, sampling and lab testing.
Geotechnical, Hydrology & Hydrogeology	**0.8**	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	**1.1**	PFS-level mine, infrastructure and process design, cost estimation, scheduling & economic analysis.
Environment	**0.8**	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology.
Total	**30.0**	**Excludes corporate overheads and future permitting activities.**

Source: JDS (2019)

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28 Qualified Person Certificates

CERTIFICATE OF QUALIFIED PERSON

I, Richard Goodwin, P. Eng., do hereby certify that:

1. This certificate applies to the technical report (Report) entitled “NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA” prepared for Constantine Metals Resources Ltd. with an effective date of 3 June 2019.
2. I am a Principal of JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings St., Vancouver, B.C. V6C 2W2, Canada.
3. I am a Registered Professional Mining Engineer in good standing with APEGBC (BC), Engineers Yukon, and NAPEG (NWT). I am a graduate of the University of B.C. with a Bachelor of Applied Science degree in Mining Engineering, 1984. I have practiced my profession continuously since 1984. Relevant experience includes project engineering, project management, study management, operations management, and executive management for mineral related properties, mines and companies.
4. 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.
5. I completed a personal inspection of the Palmer project site in September 2018.
6. I am responsible for Sections 1.1, 1.2, 1.5, 1.7, 1.9 to 1.14, 2 to 6, 16 (except 16.2), 18 (except 18.7 and 18.8), 19 to 22, and 24 to 27 of this Technical Report.
7. I am independent of the issuer, Constantine Metal Resources Ltd., as defined in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the Palmer Project.
9. I have read the NI 43-101 and confirm that the sections of the Report for which I am responsible, have been prepared in compliance of NI 43-101 and Form 43-101F1.
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: June 3, 2019
Signing Date: July 18, 2019

*[Original signed and sealed) “**Richard Goodwin**, P. Eng.”]*

Richard Goodwin, P. Eng.
Principal, JDS Energy & Mining Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Kelly McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the technical report (Report) entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" prepared for Constantine Metals Resources Ltd. with an effective date of 3 June 2019;
2. I am currently employed as Process Engineer 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 12 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, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
5. I have not visited the Palmer Project site;
6. I am responsible for Sections 13 and 17 of this Technical Report;
7. I am independent of the Issuer Constantine Metal Resources Ltd., 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. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1; and
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.

Effective Date: June 3, 2019
Signing Date: July 18, 2019

(Original signed and sealed) "Kelly McLeod, P. Eng."

Kelly McLeod, P. Eng

CERTIFICATE OF QUALIFIED PERSON

I, Michael Levy, P. Eng., do hereby certify that:

1. This certificate applies to the technical report (Report) entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" prepared for Constantine Metals Resources Ltd. with an effective date of 3 June 2019;
2. I am currently employed as Geotechnical Engineering Manager with JDS Energy & Mining Inc. with an office at 1120 Washington Avenue, Suite 200, Golden, Colorado, 80401;
3. I am a Professional Civil Engineer registered with the Association of Professional Engineers Yukon (P.Eng. #2692) and Colorado (P.E. #40268). I am a current member of the Society for Mining, Metallurgy & Exploration (SME) and Metallurgical Engineers (SME) 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 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;
5. I have not visited the project site;
6. I am responsible for section 16.2 of the Technical Report;
7. I am independent of the Issuer Constantine Metal Resources Ltd., and related companies as defined in Section 1.5 of NI 43-101;
8. I have had no prior involvement with the property that is the subject of this Technical Report;
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 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.

Effective Date: June 3, 2019
Signing Date: July 18, 2019

[Original signed and sealed) "Michael Levy, P. Eng."]

Michael Levy, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

I, James N. Gray, P. Geo., do hereby certify that:

1. This certificate applies to the technical report (Report) entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" prepared for Constantine Metals Resources Ltd. with an effective date of 3 June 2019;
2. I am a consulting geologist with Advantage Geoservices Limited, residing at 1051 Bullmoose Trail, Osoyoos, BC, Canada V0H 1V6.
3. I graduated from the University of Waterloo in 1985 where I obtained a B.Sc. in Geology. I have practiced my profession continuously since 1985. My experience includes resource estimation work at operating mines as well as base and precious metal projects in North and South America, Europe, Asia and Africa. I am a Professional Geoscientist, registered and in good standing with the Engineers & Geoscientists of British Columbia (#27022).
4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with professional associations (as deemed in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not personally inspected the Palmer Exploration Project site.
6. I am responsible for Sections 1.3, 1.6, 7 to 12, 14 and 23 of this technical report entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" with an effective date of 20 June 2019.
7. As a qualified person, I am independent of Constantine Metal Resources Ltd. as defined in Section 1.5 of NI 43-101.
8. My prior involvement with the Palmer Project consists of the completion of resource updates with effective dates of May 11, 2015, September 27, 2018 and January 31, 2019.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form; and
10. As of the report date, and 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.

Effective Date: June 3, 2019
Signing Date: July 18, 2019

[Original signed and sealed) "James N. Gray, P.Geo"]

James N. Gray, P. Geo
Advantage Geoservices Limited

CERTIFICATE OF QUALIFIED PERSON

I, John J. DiMarchi, CPG, do hereby certify that:

1. This certificate applies to the technical report (Report) entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" prepared for Constantine Metals Resources Ltd. with an effective date of 20 June 2019.
2. I am a Principal/Owner of Core Geoscience LLC, with an office at 5319 NE 62nd St., Seattle, WA, USA.
3. I am a Certified Professional Geologist (#9217) in good standing with AIPG (US) and a Registered Professional Geologist (#403) in Alaska. I am a graduate of Colorado State University with a Bachelor of Science degree in Geology, 1978. I have practiced my profession continuously since 1979. Relevant experience includes project permitting, project management and study management for mineral properties, mines, operating companies and regulatory agencies.
4. 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.
5. I last completed a personal inspection of the Palmer project site in September 2018.
6. I am responsible for Sections 1.10 and 20 of this Technical Report.
7. I am independent of the issuer, CMR.V, as defined in Section 1.5 of NI 43-101.
8. I have had prior permitting involvement with the Palmer Project.
9. I have read NI 43-101 and confirm that the sections of the Report for which I am responsible, have been prepared in compliance of NI 43-101 and Form 43-101F1.
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: June 3, 2019
Signing Date: July 18, 2019

[Original signed and sealed) "John J. DiMarchi, CPG"]

John J. DiMarchi, CPG
Principal/Owner
Core Geoscience LLC

CERTIFICATE OF QUALIFIED PERSON

I, Jim Casey, P.E., P.Eng. do hereby certify that:

1. This certificate applies to the technical report (Report) entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" with an effective date of 3 June 2019.
2. I am a Geological Engineer of KCB Consultants Ltd. with an office at 2 N. Central Avenue, Phoenix, AZ, 85004, United States. KCB Consultants Ltd. is the US subsidiary of Klohn Crippen Berger Ltd.
3. I am a Registered Professional Civil Engineer in good standing in the states of Alaska, Arizona, New Mexico and Washington, and the province of British Columbia. I am a graduate of the University of British Columbia with a Bachelor of Applied Science in Geological Engineering, 2009. I have practiced in my profession continuously since 2010. Relevant experience includes geotechnical site investigation, construction quality control and quality assurance and design of tailings storage facilities.
4. 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.
5. I completed a one-day visit to the Palmer project site in September 2018.
6. I am responsible for Sections 1.9.1 to 1.9.3, 18.7 and 18.8 of this technical report entitled "NI 43-101 TECHNICAL REPORT PALMER PROJECT, ALASKA, USA" with an effective date of 3 June 2019.
7. I am independent of the issuer, Constantine Metals Ltd., as defined in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the Palmer Project.
9. I have read the NI 43-101 and confirm that the sections of the Report for which I am responsible, have been prepared in compliance of NI 43-101.
10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: June 3, 2019

Signing Date: July 18, 2019

[Original signed and sealed) "Jim Casey, P.E., P.Eng"]

Jim Casey, P.E., P. Eng.

29 Appendices

Appendix I – 1979-2017 DDH HEADER INFORMATION

Appendix II – 2018 DDH HEADER INFORMATION

Appendix III – 1979-2018 SIGNIFICANT DDH ASSAY HIGHLIGHTS – PALMER DEPOSIT

Appendix IV – 1979-2018 SIGNIFICANT DDH ASSAY HIGHLIGHTS – AG ZONE DEPOSIT

Appendix V – PALMER PROJECT GEOLOGICAL SECTIONS