NI 43-101 PRELIMINARY ECONOMIC ASSESSMENT TECHNICAL REPORT ON THE EMPIRE STATE MINES, GOUVERNEUR, NEW YORK, USA

Prepared for:

ST. LAWRENCE ZINC COMPANY, LLC.
A wholly owned subsidiary of TITAN MINING CORPORATION
408 Sylvia Lake Road
Gouverneur, NY, 13642, USA

Prepared by:

JDS ENERGY & MINING INC.
Suite 900, 999 W Hastings St
Vancouver, BC V6C 2W2, Canada

Qualified Persons	Company
Garett Macdonald, P. Eng.	JDS Energy & Mining Inc.
Mike Makarenko, P. Eng.	JDS Energy & Mining Inc.
Indi Gopinathan, P. Eng.	JDS Energy & Mining Inc.
Mike Creek, PE	JDS Energy & Mining Inc.
Allan Reeves, P. Geo.	Tuun Consulting Inc.
Robert Raponi, P. Eng.	TR Raponi Consulting 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 St. Lawrence Zinc Company, LLC. 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.

St. Lawrence Zinc Company, LLC 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

Contents

Tables and Figures

List of Appendices

1 Executive Summary

1.1 Introduction

St. Lawrence Zinc Company, LLC (SLZ) is an indirect, wholly owned subsidiary of Titan Mining (US) Corporation (Titan). SLZ owns the Balmat No. 4 Zinc Mine (the Mine) which is now known as Empire State Mines (ESM). ESM is located in the Balmat-Edwards mining district in northern New York State, near Gouverneur and is 25 miles (mi) south of the Port of Ogdensburg. SLZ commissioned JDS Energy & Mining Inc. (JDS) to complete a Preliminary Economic Assessment (PEA) for the potential re-opening of the mine.

The last Technical Report prepared for the mine was produced in 2005 by Hudbay Minerals Inc. (Hudbay), from which time the mine went into production for three years and has since been on care and maintenance.

This Technical Report summarizes the results of the 2017 PEA study and was prepared following the guidelines of NI 43-101.

All currency in this report is United States dollars (US$), unless stated otherwise. Imperial and metric units are used and defined as required.

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 the PEA will be realized.

1.2 Project Description

ESM is a past producer with over 100 years of history, and has been in a state of care and maintenance since 2008. The mine is fully developed with shaft access and mobile equipment on-site. Existing surface facilities at the mine include a maintenance shop, offices, mine dry, primary crusher, mine ventilation fans, 12,000-ton (t) covered concentrate storage building, rail siding, warehouse and storage buildings. The mine and its facilities have been maintained to good standards during this period of care and maintenance.

Mineralization is hosted within an Upper Marble rock unit, comprised of metamorphosed and complexly folded (silicified) marbles. The mineralization is located primarily in hinges of large fold structures.

The mine has historically used a combination of selective longhole stoping, modified or stepped room and pillar and mechanized cut and fill as mining methods. The PEA envisions that rehabilitation, development, and production are planned to take place approximately 3,100 ft below surface. An underground crusher is in place and is capable of feeding a surface flotation concentrator with name plate capacity of 5,000 tons per day (t/d). PEA mine production is planned to start at 800 t/d and grow to 1,800 t/d with an average 1,465 t/d of mill feed over the 8- year mine life.

Tailings will be placed in the existing permitted 260-acre conventional impoundment. The Tailings Management Facility (TMF) is categorized as a low-risk dam by the New York State Bureau of Flood Protection & Dam Safety.

The ultimate capacity of the 260-acre foot print has been estimated at 20 million tons (Mt), with immediate capacity of 2.7 Mt, before further embankment construction will be needed.

1.3 Location, Access and Ownership

ESM is located approximately 1.3 mi southwest of Fowler, New York State, in St. Lawrence County. SLZ owns a total of 2,699 acres of fee simple surface and mineral rights in three towns in St. Lawrence County. The majority of the property consists of the 1,754 acres in the town of Fowler where the ESM, mill and tailings disposal facility are located. Nine parcels totalling 703 acres are owned in the town of Edwards, which includes the Edwards mine. The remainder of the fee ownership covers the Pierrepont mine which is located on four owned parcels totalling 242 acres.

1.4 History, Exploration and Drilling

The Balmat-Edwards district consists of four mines. Edwards produced from 1915 to 1980, Balmat from 1930 to 2008, Pierrepont from 1982 to 2001 and Hyatt from 1974 to 1998 on an intermittent basis. The Balmat mine operated continuously from 1930 to 2001 when production ceased due to depressed zinc metal prices. Production resumed in 2006 until Hudbay Minerals placed the Balmat mine on care and maintenance in the third quarter of 2008 in response to depressed metal prices. Since that time all typical care and maintenance tasks have been performed. The mine remains dewatered and is readily accessible and the mill is in good condition.

The Balmat mine (now ESM) has produced a total of 30.7 Mt grading 8.6% zinc. A history of mine ownership is listed in Table 1.1.

Table 1.1: Balmat (now ESM) Ownership History

Date	Company
1930	St. Joe Minerals Corporation, purchased by Fluor Corporation in 1981.
1987	Zinc Corporation of America
2003	OntZinc (renamed Hudbay Minerals in December 2004)
2015	Star Mountain Resources Inc.
2017	Titan Mining (US) Corporation

Source: SLZ (2017)

1.5 Geology and Mineralization

The mine's mineral resources are in seven mineralized zones, known as Mud Pond, Mahler, New Fold, NE Fowler, Davis, Sylvia Lake and Cal Marble, between 1,400 feet (ft) and 5,500 ft below surface. The zones are aerially scattered and all zones except NE Fowler and Cal Marble are connected by existing development to the shaft. The zones are up to 50 ft thick but average 8 ft and dip between 20º and 35º, with local variations from 10 to 90º. The elongated mineralized zones are up to 500 ft wide and in the order of 6,000 ft long. The mineralized zones while generally continuous, display considerable geometrical variability.

The Balmat deposits are classified as Sedex in origin, forming initially in a marine sequence of carbonates and evaporates. They were deeply buried, metamorphosed to amphibolite grade and strongly deformed during the late Precambrian Grenville Orogen.. Historical mining and diamond drilling have shown that the geometry and continuity of the mineralized zones is consistent.

1.6 Metallurgical Testing and Mineral Processing

A test program was undertaken in 2005 to confirm the processing requirements of selected mineralized material zones from the ESM mine. These mineralized material zones were selected based on projected tonnage, mineralized material type, and sample availability. The results were used to confirm concentrate grades and recoveries for the re-start of operations in 2005.

Flotation tests were completed under the guidance of Fred Vargas, the metallurgical consultant who developed the pHLOTEC flotation process in use at ESM since 1984.

No additional metallurgical testing will be undertaken for the current re-opening. The mineralized zones to be mined are a continuation of the mineralization mined from 2005 to 2008.

The 2005 metallurgical test results and operational results from 2006 to 2008 support a zinc recovery of 96% and a zinc concentrate grade of 56% for the re-start of operations.

1.7 Mineral Resource Estimates

1.7.1 Drill Hole Database

The drilling database consists of historical drilling totalling 4,317 holes. A total of 633 of these holes were used for the Resource Estimate contained in this report. The majority of these holes were drilled during the most recent mining campaign by Hudbay Minerals Inc. from 2006 to 2009. All other holes were either distal exploration holes or holes defining the historic underground workings not relevant to this study.

Table 1.2: Drill Holes Used in Resource Estimate

Year	No. of Holes	Drilled Footage
Pre-2000	142	126,407
2000	33	23,384
2001	12	3,539
2004	5	3,143
2005	98	47,312
2006	120	42,446
2007	77	31,028
2008	140	36,931
2009	6	3,567
TOTAL	**633**	**317,758**

Source: Tuun (2017)

1.7.2 Geologic Model

ESM geologists provided 12 key domains which are constrained by the well-documented geologic horizons described in Sections 7.4 and 7.5 of this report. The mineralized zones are identified in Table 1.3.

Table 1.3: Mineralized Zones

Mineral Zone	Zone Code
1. Davis	10
2. Cal Marble	20
3. Cal Upper	21
4. Sylvia Lake	30
5. Mud Pond Main	40
6. Mud Pond Apron	41
7. Mud Pond Quartz Diopside	43
8. Mahler Main	50
9. Mahler White Dolomite	51
10. Mahler Quartz Diopside	52
11. NE Fowler	60
12. New Fold	70

Source: Tuun (2017)

Decades of face-mapping were used to develop the wireframes in 2009. These wireframes had been constructed along vertical cross-sections. That methodology was updated in February - March 2017 by re-interpretation and adjustment of polylines to ‘snap’ to drill hole intercepts. The revised 2017 mineralized zone wireframes were used for this resource estimation.

1.7.3 Block Model

A 3D block model was created using Geovia GEMS to represent the lithological and structural characteristics specific to ESM. This model was used as a framework for the grade model, which relied on statistical analysis of the sample data and a detailed understanding of the geology to produce a robust estimate of the resource.

The GEMS model of 15 ft x 15 ft x 15 ft was subsequently sub-blocked to 2.5 ft x 2.5 ft x 2.5 ft in Maptek Vulcan^{TM} software for mine planning exercises.

Table 1.4: Block Model Parameters

Origin	Block Dimension (ft)	# of Blocks
12,750 E	15	630
7,425 N	15	745
-925 El (max)	15	200
Rotation	-25^{o}	

Source: Tuun (2017)

Block model grades were estimated in three passes using the Inverse Distance Squared (IDS) method. Models for the Nearest Neighbour (NN) and the Mean Value of Composites Used (MVCU) were also created. The NN and MVCU block models were used for comparative and validation purposes. The grade models were visually validated by comparing the blocks estimated by the various techniques with actual drill hole composite data on both section and in plan view.

In order to determine the quantities of material satisfying “reasonable prospects for economic extraction”, The Qualified Person (QP) assumed a minimum mining cut-off grade of 6.0% Zinc, representing an approximate operating cost of $70/t, a zinc price of $1.00/lb and 96% recovery.

The QP is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political issues that may adversely affect the mineral resources presented in this report.

The QP considers that the blocks with grades above the cut-off grade satisfy the criteria for “reasonable prospects for economic extraction” and can be reported as a Mineral Resource. Mineral resources for each of the mineralized zones at ESM are summarized in Tables 1.5 and 1.6.

Table 1.5 outlines the Mineral Resource estimate effective as of April 6th, 2017, at the selected cut-off zinc grade of 6.0%.

Table 1.5: Empire State Mines Mineral Resource Estimate

Cut-Off (% Zinc)	Measured		Indicated		M&I		Inferred	
	tons	% Zinc	tons	% Zinc	tons	% Zinc	tons	% Zinc
>10%	543,000	16.15	840,600	16.27	1,383,600	16.22	1,499,200	16.02
>9%	617,500	15.34	962,500	15.42	1,580,000	15.39	1,772,600	15.01
>8%	696,100	14.57	1,080,000	14.67	1,776,100	14.63	1,970,400	14.36
>7%	770,200	13.89	1,200,500	13.96	1,970,700	13.93	2,100,600	13.94
>6%	850,100	13.19	1,307,900	13.35	2,158,000	13.29	2,276,000	13.37
>5%	932,800	12.51	1,416,700	12.76	2,349,500	12.66	2,393,400	12.98
>4%	1,004,900	11.94	1,524,400	12.18	2,529,300	12.08	2,887,100	11.88
>3%	1,074,300	11.39	1,612,400	11.70	2,686,700	11.58	2,824,300	11.60

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
2. The UG mining economics used operating costs of $70.00/t, and a commodity price of $1.00/pound at 96% recovery.
3. Mineral resources are reported 'in-situ' using a 'new development' cut-off grade of 6% Zn to determine 'reasonable prospects for eventual economic extraction'.
4. Mineral resources are also reported 'in-situ' using an incremental cut-off grade of 2% Zn to determine 'reasonable prospects for eventual economic extraction'.
5. Tonnages are reported to the nearest 100 tons, and grades are rounded to the nearest two decimal places.
6. Rounding as required by reporting guidelines may result in apparent summation differences between tons, and grade.

Source: Tuun (2017)

Table 1.6: Mineral Resources by Zone at 6.0% Zn Cut-Off Grade

Mineralized Zone	MEASURED		INDICATED		M&I		INFERRED	
	tons	% Zinc	tons	% Zinc	tons	% Zinc	tons	% Zinc
Davis	400	6.24	600	8.53	1,000	7.61	200	8.10
Cal Marble	0	0.00	35,600	9.58	35,600	9.58	440,200	11.46
Sylvia Lake	44,500	10.77	47,300	10.62	91,800	10.69	38,200	12.86
Mud Pond	231,400	10.38	148,700	11.48	380,100	10.81	369,300	10.80
Mud Pond Apron	43,400	11.98	115,800	10.34	159,200	10.79	23,600	9.18
Mud Pond QD	61,900	9.37	9,400	8.43	71,300	9.25	0	0.00
Mahler Main	311,800	15.14	590,900	15.11	902,700	15.12	329,100	12.76
Mahler WD	82,100	18.75	80,300	17.97	162,400	18.36	180,700	20.95
Mahler QD	6,600	15.85	29,700	11.21	36,300	12.05	6,800	9.92
NE Fowler	0	0.00	0	0.00	0	0.00	348,500	14.61
New Fold	68,000	12.75	249,600	11.72	317,600	11.94	539,400	13.97
Total	**850,100**	**13.19**	**1,307,900**	**13.35**	**2,158,000**	**13.29**	**2,276,000**	**13.37**

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
2. The UG mining economics used operating costs of US$70.00/ton, and a commodity price of US$1.00/pound at 96% recovery.
3. Mineral resources are reported 'in-situ' using a 'new development' cut-off grade of 6% Zn to determine 'reasonable prospects for eventual economic extraction'.
4. Tonnages are reported to the nearest 100 tons, and grades are rounded to the nearest two decimal places.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tons, and grade.

Source: Tuun, 2017

Mineral resources were estimated in conformity with CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. Mineral resources are not mineral reserves and have no demonstrated economic viability. This PEA does not support an estimate of mineral reserves, since a Pre-Feasibility Study (PFS) or Feasibility Study (FS) is required for reporting of Mineral Reserve estimates. This report is based on mine plan tonnage (mine plan tons and/or mill feed).

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

1.8 Mining

The ESM deposit is proposed to be mined using four underground mining methods, based on the geometry and the grade of the mineralized zones:

- Longhole stoping (LH) for mining blocks dipping steeper than 45°, which represents about 50% of the mine plan tonnage. This is the preferred mining method from a productivity and operating cost perspective;
- Mechanized Cut and Fill (C&F), for mining blocks with dips of less than 45° and zones not amenable to LH stoping, is more selective and represents about 7% of the mine plan tonnage;
- Modified or Stepped Room and Pillar (RP), for mining blocks with dips of less than 45° and grades, do not warrant the application of a fill to permit multiple panel extraction, representing 11% of the mine plan tonnage;
- Sub-level drift slashing and pillar slashing (SLS) for mining blocks which require lateral extension from the sub-level drift to either accommodate long hole drills to drill LH stopes, or to recuperate remnant pillars left between rooms in the existing workings, representing 19% of the mine plan tonnage; and
- The remaining 13% of the mine plan tonnage comes from sub-level drives, access, and stope cross-cut development.

Un-cemented rock fill will be used as backfill to maximize mining recovery. Where availability of fill material is not present, structural pillars will be left within the mineralization. Approximately 8% of the mineralization targeted for extraction will be left behind as pillars.

The deposit will be accessed from the existing No. 4 shaft and level development, which is extensive. On level, access ramps will be driven at maximum grade of 15% at a 15 ft x 17.5 ft profile to accommodate 40-ton haul trucks.

Level spacing is variable up to a maximum of 70 ft. Mineralized zone development will be driven using a 13 ft x 13 ft profile.

The mine requires drift rehabilitation and utility refurbishments, as well as mobile equipment servicing. These activities commenced in March of 2017 with completion scheduled for end of July.

The initial mine design was based on basic assumptions to generate lower limits for cut-off grades (COG) for the planned mining methods. A value of 6.0% Zn was determined as the COG for mining. These COG's were used to design initial mining shapes. An incremental COG of 2.0% Zn was applied to mined development material which covers costs for processing and administration only.

The PEA mine plan focusses on accessing and mining higher operating margin material early in the mine life. As such, the plan commences with the mining of Mahler, Mud Pond, and New Fold, followed by Cal Marble, Davis, and NE Fowler. The mine production rate is targeted to maximize utilization of existing equipment while maintaining ventilation limits. Production rates start at 800 t/d and grow to 1,800 t/d with an average 1,465 t/d over the life of mine.

Mining recovery and dilution factors were applied to each mining shape based on the mining method used. The PEA production plan for the ESM mine is summarized in Table 1.7.

Table 1.7: Mine Production Schedule

	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mine Plan Tonnage	kt	276	639	657	657	657	657	553	183	**4,278**
Production Rate	tpd	756	1,751	1,800	1,795	1,800	1,800	1,514	499	**1,630**
Zn Grade	%	9.5	8.2	10.9	10.1	9.3	10.5	6.5	5.9	**9.2**
Zn Tons	kt	26	52	72	66	61	69	36	11	**394**
Lateral Development (Excl. Rehab)	ft (‘000)	12	17	27	23	19	23	38	5	**165**
Vertical Development	ft	96	79	330	370	93	146	92	56	**1,261**
Waste Fill	ft^3 (‘000)	2.7	6.5	6.1	4.2	4.4	3.4	4.0	1.1	**32.4**

Note: totals may not add due to rounding.
Source: JDS (2017)

Approximately 50% of the mine plan tons are Inferred. 80% of the mine plan in the first two years is sourced from Measured and Indicated zones.

1.9 Recovery Methods

Mineralized material mined from the ESM deposits will be processed at the existing concentrator that was commissioned in 1970 and last shut down in 2008. The existing plant flotation circuit consists of a lead flotation circuit followed by zinc flotation. Lead grades for the mill feed material will be less than 1%, and as such, a lead concentrate will not be produced.

The concentrator flowsheet includes crushing, grinding, sequential lead and zinc flotation circuits, concentrate dewatering circuits, and loadout facilities.

The zinc flotation circuit consists of rougher flotation followed by scavenger flotation. The scavenger concentrate returns to the head of the rougher circuit. Rougher concentrate undergoes two stages of cleaner flotation. Cleaner tailings are returned to the previous stage of flotation in the traditional manner.

The nameplate capacity of the concentrator is 5,000 t/d. Throughout the history of the ESM operation, the capacity of the concentrator has exceeded that of the mines. The traditional operating strategy has been to operate the concentrator at its rated hourly throughput of 200 to 220 t/h, but for only as many hours as necessary to suit mine production. In the last full year of production (2008), the concentrator was operated at 25% of the total available hours in the year.

Similar to past operations at ESM, mine production rates will not be able to sustain the fulltime use of the concentrator. A single 10-hour shift will operate four days per week to process mill feed at an equivalent operating rate of 5,000 t/d.

All major circuits in the ESM concentrator have been reviewed to ensure they are suitable to process the design throughput. The concentrator will require minimal work to be placed back into operation.

1.10 Infrastructure

Access to the ESM facility is by existing paved state, town and site roads. All access to the mine/mill facility as well as concentrate haulage from the facility is by paved public roads and/or an existing CSX rail short line. The existing facilities at ESM mine are well established and will generally meet the requirements of the planned operations.

The ESM mine site is located adjacent to State Highway 812, approximately 1.5 mi from the junction with State Highway 58. A mile long stretch of Sylvia Lake Road currently handles traffic to and from the site, including truck haulage of concentrate to the Port of Ogdensburg should overseas shipping be used. Road maintenance is carried out by the Town and State Government Department of Highways.

There are currently two entries from Sylvia Lake Road providing access to the site. The main entry gives access to the parking lot and the access to the office complex, and the tailings line entry is the waste truck haulage route to the tailings impoundment.

The ESM No. 4 Mine surface infrastructure includes 15 buildings, most of which were constructed in 1969 to 1970, including and not limited to:

- Office complex;
- Maintenance and warehouses;
- Maintenance vehicle storage, boiler room, and change rooms;
- Headframe & hoist house;
- Concentrator & concentrate storage;
- Maintenance shop;
- Storage facilities for timber, tires, electrical, pine oil, warehouse, and miscellaneous; and
- Three pump houses for lake water, booster station, and fuel and oil.

Power to site is fed by line from Niagara Mohawk's substation at Battle Hill-ESM #5 circuit. On-site power is distributed to the plant and mine. SLZ owns two portable generators for emergency use. One is a 125 kVA portable used for general 480 V / 220 V / 110 V applications. The other is a 100 kVA portable generator which will run the No. 2 emergency egress hoist.

Mill process and cooling water (non-potable) for the site will be pumped from the Sylvia Lake pump house to two 100,000 gallon (gal.) each concrete deluge tanks near the concentrate storage building/rail loadout shed. Water will be pumped from the reservoir tanks to the concentrator. Mine water will be pumped from the mill basement sump down the 4" shaft water line to the various mine levels.

The tailings disposal facility covers 260 acres approximately 4,000 ft north of the mill. Water from tailings flows through a series of retention ponds before discharge into Turnpike Creek. Discharge is regulated by NYSDEC under permit NY0001791.

The mineralized materials and waste rock from the development and operation of the mine is non-acid-generating due to the alkaline nature of the host rock. The designated surface pads were designed such that any run-off will drain to the concentrator pond. The capacity of this stockpile area is sufficient for the tonnages in the contained mine schedule

1.11 Environment and Permitting

The mine has licenses and permits for air, water withdrawal, mining, water discharge, explosives storage and use, petroleum and chemical storage, radiological equipment, and other miscellaneous licenses and permits. There are no additional permits or licenses required prior to returning the property to production.

During mine operations prior to the 2006 re-start, discharge limits for Fe and Zn were exceeded. To avoid such exceedance in the future, a new water treatment plant, satisfactory to the New York State Department of Environmental Conservation (NYSDEC), has been constructed and is in operation today.

There are no Notices of Violation outstanding for the mine site on any environmental matter. In 2003, a $1.663 M cash deposit reclamation bond was put in place for five years for site reclamation of 432 acres at mine closure. Remaining SLZ liabilities include reclaiming the tailings impoundment area, the mine site area, capping of underground openings to surface, and re-vegetating the area to blend in with the surroundings. Closure costs for the mine and associated facilities and severance pay have been assumed to be paid by the sale of the mine assets at closure.

1.12 Operating and Capital Cost Estimates

Estimated life of mine capital costs total $69.2 M, consisting of the following distinct stages:

- **Initial Capital Costs** – includes all pre-production costs to replace, repair and upgrade the infrastructure and resource to support the mine plan production. Initial capital costs total $10.7 M and are expended over a 5-month refurbishment and commissioning period;
- **Sustaining Capital Costs** – includes all costs related to the capital development and acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations.

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

Table 1.8 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q1 2017 US dollars with no escalation.

Table 1.8: Summary of Capital Cost Estimate

Area	Pre-Production (M$)	Production (M$)	LOM (M$)
Mining	5.3	40.4	45.7
Mineral Processing	1.1	0.7	1.8
Tailings Management	0	4.7	4.7
Infrastructure	0.8	0	0.8
Indirect Costs Incl. EPCM	0.4	0.2	0.5
Owners Costs	0.1	0.1	0.2
Closure Costs	0	11.9	11.9
Salvage Value	0	-4.0	-4.0
Subtotal Pre-Contingency	**7.6**	**53.9**	**61.6**
Contingency	1.0	4.6	5.6
Subtotal	**8.6**	**58.5**	**67.2**
Capitalized Operating Cost	7.6	0	7.6
Revenue Credit	-5.5	0	-5.5
Total	**10.7**	**58.5**	**69.2**

Source: JDS (2017)

Table 1.9: Summary of Site Operating Cost Estimate

Site Operating Costs	Unit Cost ($/t milled)	Unit Cost ($/lb Zn payable)	LOM Cost (M$)
Mining	42.27	0.28	180.9
Processing	8.89	0.06	38.0
G&A	9.60	0.06	41.1
Total	**60.77**	**0.40**	**260.0**

Source: JDS (2017), RT (2017)

Table 1.10: Main OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.04
Average power consumption	MW	5.1
Overall power consumption (all facilities)	kWh/t processed	8.3
Diesel cost (delivered)	$/gallon	1.89
LOM average manpower (including contractors, excluding corporate)	Employees	151

Source: JDS (2017), SLZ (2017)

1.13 Economic Analysis

1.13.1 Main Assumptions

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 to approximate the true investment value. It must be noted that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations to represent an indicative value of the after-tax cash flows of the project.

The results of the economic analysis are shown in Table 1.11.

This PEA is preliminary in nature and includes the use of Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the results of the PEA will be realized.

Sensitivities to metal prices, operating cost estimate (OPEX), and capital cost estimate (CAPEX) were conducted by adjusting each variable up and down 20% independently of each other. As with most metal mining projects, the project is most sensitive to metal prices.

Table 1.11: Economic Assumptions

Item	Unit	Value
NPV Discount Rate	%	8
Federal Income Tax Rate	%	35
State Income Tax Rate	%	4.9
Capital Cost Allowance Rate	%	Per New York State schedule
Capital Cost Allowance Term	Years	7
Depletion Charge	%	Lesser of: 50% of Taxable Income Before Depletion or 22% of EBITDA* less Royalties
Capital Contingency (Overall)	%	10

*Earnings before interest, tax, depreciation and amortization

Source: JDS (2017)

Table 1.12: Net Smelter Return Assumptions

Off-site Costs and Payables	Unit	Estimated Value
Payables	%	85.0
Treatment Charges	$US/dT	150
Losses and Penalties	$US/dT	15.0
Transport, Marketing, Insurance, etc.	$US/dT	85
Royalties	%NSR	0.3

Source: JDS (2017)

1.13.2 Results

Table 1.13 below outlines the pre- and post-tax economic results at a 0% and 8% discount rate.

Table 1.13: Economic Results

Parameter	Unit	Pre-tax Results	After-tax Results
$NPV_{0\%}$	M$	295	206
$NPV_{8\%}$	M$	216	150
Internal Rate of Return (IRR)	%	153	121
Payback period	Production years	1.2	1.3

Source: JDS (2017)

1.13.3 Sensitivities

Sensitivity analyses were performed using metal prices, mill head grade, CAPEX and OPEX as variables. The value of each variable was changed plus and minus 20% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.14.

Table 1.14: Sensitivities Analyses

Variable	Pre-tax NPV @ 8% (M$)			Post-tax NPV @ 8% (M$)		
	-20% Variance	0% Variance	20% Variance	-20% Variance	0% Variance	20% Variance
Price	98	216	335	65	150	232
CAPEX	227	216	205	161	150	139
OPEX	253	216	179	176	150	122
Grade	108	216	324	75	150	223

Source: JDS (2017)

1.14 Project Development

As of March 2017, ESM is undergoing mine rehabilitation activities, including shaft utility refurbishments, installation of new hoist cable, and reconditioning of the mobile equipment fleet and underground ground support installations.

A two month refurbishment period will take place upon completion of project financing, followed by initial mine production.

1.15 Conclusions

It is the conclusion of the QPs that the PEA summarized in this Technical Report contains adequate detail and information to support the positive economic result. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the project.

Approximately 50% of the mineralization within the PEA mine plan is classified as Measured and Indicated with the remainder in the Inferred classification.

1.15.1 Risks

The most significant potential risks associated with the project are commodity prices, uncontrolled dilution, mineral recovery, operating and sustaining capital cost escalation, ventilation limitations, Inferred resource confidence, and unforeseen schedule delays.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active management. The first two years of production focuses on Measured and Indicated mineralized zones to mitigate risk within the payback period of the project. Eighty percent (80%) of the mineralization mined in the first two years is classified as Measured and Indicated.

1.15.2 Opportunities

Mine production may be limited by the ability to ventilate the underground workings. As such, there is opportunity in investigating alternate hauling methods that reduce or eliminate the diesel particulates produced from traditional diesel powered truck haulage. Railveyor and electric motor technology has become a viable source for underground haulage which does not rely on diesel engines and may provide the ability to meet and/or beat the estimated production rate proposed in this report.

Dilution is important to manage in any mining operation, and especially so in narrow resources. The implementation of grade control geologists on shift with electronic survey and mapping software is an opportunity to better control the excavations and follow the mineralization.

The dark mineralization hosted within a light dolomitic rock may lend itself to optical sorting technology, which could provide an increase to mill feed head grade while simultaneously providing a source of crushed waste rock for cemented and un-cemented backfill. A sorted mill feed may additionally permit a lower mine cut-off grade which could increase the mineral resources with the PEA mine plan without additional exploration.

The resource potential has not been fully defined, and as such there is opportunity for resource expansion. The mine historically operated with little definition drilling in comparison to greenfield exploration properties. Much reliance was placed on the ability to follow the resource through mine development for the replacement of reserves. Additional exploration drilling may yield high returns in the discovery and upgrade of additional mineralized resources.

Opportunities may exist to improve the mill feed grades by detailing level designs and identifying pillar locations upon completion of geotechnical analysis. Detailed production schedules integrating backfill schedules may provide opportunities to reduce the volume of structural pillars currently planned to be left within the mineralized resource.

1.15.3 Recommendations

The items shown in Table 1.15 are recommended for the ESM to improve confidence and performance of the PEA mine plan and economics.

Table 1.155: Project Recommendations and Cost

Item	Cost ($)
Infill drilling	1,000,000
Surface and underground exploration drilling	4,300,000
3D lithology Model	50,000
Digitize maps and survey plans	150,000
Updated mine survey	150,000
Geotechnical review	30,000
Sorting test work and integration study	100,000
Alternate haulage investigation (Railveyor)	45,000
Total Estimate	**5,825,000**

Source: JDS (2017)

2 Introduction

2.1 Basis of Technical Report

This Preliminary Economic Assessment (PEA) Technical Report was compiled by JDS Energy & Mining Inc. (JDS) for St. Lawrence Zinc Company, LLC (SLZ) a wholly owned subsidiary of Titan Mining Corporation (Titan). The purpose of this study is to provide a Mineral Resource estimate with mine plan and economics for SLZ's Empire State Mines (ESM) operation.

The structure and content of this report uses NI 43-101 guidelines.

2.2 Scope of Work

The following companies contributed to this Technical Report and provided Qualified Person (QP) sign-off for their respective sections:

JDS Energy & Mining Inc. (JDS):

- Overall PEA lead;
- Introduction, project description and history;
- Mine engineering;
- Infrastructure;
- Environment, socio-economics and permitting;
- Tailings management;
- Water management;
- Cost estimation;
- Project execution plan;
- Economic analysis; and
- Conclusions, risks and opportunities.

Tuun Consulting Inc. (Tuun):

- Mineral Resource estimate;
- Deposit type;
- Geology;
- Drilling;
- Exploration;
- Sample Preparation, analyses and security; and
- Data verification.

TR Raponi Consulting Ltd. (TR)

- Metallurgical test work analyses;

- Processing methods; and
- Process Cost estimations.

2.3 Qualifications and Responsibilities

The QPs preparing this Technical Report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics (Table 2.1).

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

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions. The QPs are responsible for specific sections as follows:

Table 2.1: QP Responsibilities

QP	Company	QP Responsibility/Role	Report Section(s)
Garett Macdonald, P. Eng.	JDS Energy & Mining Inc.	Overall Responsibility, Costs And Economics	1 to 3, 21, 22, 25 to 29
Mike Makarenko, P. Eng.	JDS Energy & Mining Inc.	Mineral Resource Estimate, Mining	15, 16
Mike Creek, PE.	JDS Energy & Mining Inc.	Environment & Infrastructure	4 to 6, 18, 20
Allan Reeves, P. Geo.	Tuun Consulting Inc.	Geology & Mineral Resource Estimate	7 to 12, 14, 24
Robert Raponi, P. Eng.	TR Raponi Consulting Ltd.	Process & Metallurgy	13, 17
Indi Gopinathan, P. Eng.	JDS Energy & Mining Inc.	Economics & Market Studies	19, 23

Source: JDS (2017)

2.4 Site Visit

QP site visits were conducted as per Table 2.2.

Table 2.2: QP Site Visits

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Garett Macdonald, P.Eng.	JDS	Feb. 20, 2017	Ryan Schermerhron, SLZ	Inspection of site facilities, concentrate shipment systems, and record keepings.
Mike Creek, PE.	JDS	Feb. 16, 17, 2017	Ryan Schermerhorn, SLZ	Inspection of tailings and surface facilities,
Mike Makarenko, P. Eng.	JDS	Feb. 20-23, 2017	Jamie Hance, SLZ	Inspection of UG mine and infrastructure, mill, tails, and surface facilities.
Allan Reeves, P. Geo.	Tuun	Feb. 20-23, 2017	Jamie Hance, SLZ	Inspection of UG workings, production zones, core shack, assay certificates, and record keepings.
Robert Raponi, P. Eng.	TR	Feb. 20, 2017	Ryan Schermerhorn, SLZ	Inspection of mill facility, assay lab, and record keepings.
Indi Gopinathan, P. Eng.	JDS	Feb. 20, 2017	Ryan Schermerhron, SLZ	Inspection of site facilities, concentrate shipment systems, and record keepings.

Source: JDS (2017)

2.5 Units, Currency and Rounding

The units of measure used in this report are as per the Imperial system unless otherwise noted.

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

Frequently used abbreviations and acronyms can be found in Section 29. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.6 Sources of Information

This report is based on information collected by JDS during site visits performed between January 27, 2017 and February 25, 2017, and on additional information provided by SLZ throughout the course of JDS's investigations. Other information was obtained from the public domain. JDS has no reason to doubt the reliability of the information provided by SLZ. This Technical Report is based on the following sources of information:

- Discussions with SLZ personnel, including;
  - Ryan Schermerhorn, SLZ mill superintendent;
  - Jamie Hance, SLZ mine Forman;
  - Mike Porter, SLZ mine safety;
  - Bob Baderman, SLZ mine survey;

    - John Jonson, SLZ mine geologist;
- Discussions with independent consultants to SLZ;
    - Brett Armstrong, independent geologist;
    - Kim Tyler P. Geo, independent geologist; and
    - Mark Odell, independent mine engineer.
- Technical reports, memos, and internal studies prepared for and by the ESM operation;
- Engineering work in infrastructure, processing, site services, mine design, and ventilation analysis has been completed by recent mine owners Hudbay Minerals Inc. and Star Mountain Resources Inc., on which JDS has relied for various details and site specifications.
- Inspection of the ESM area, including outcrop and drill core;
- Review of exploration data collected by SLZ; and
- Additional information from public domain sources.

The QPs have taken reasonable measures to confirm the information provided by others and take responsibility for the information. The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

3 Reliance on Other Experts

Not applicable.

4 Property Description and Location

4.1 Location

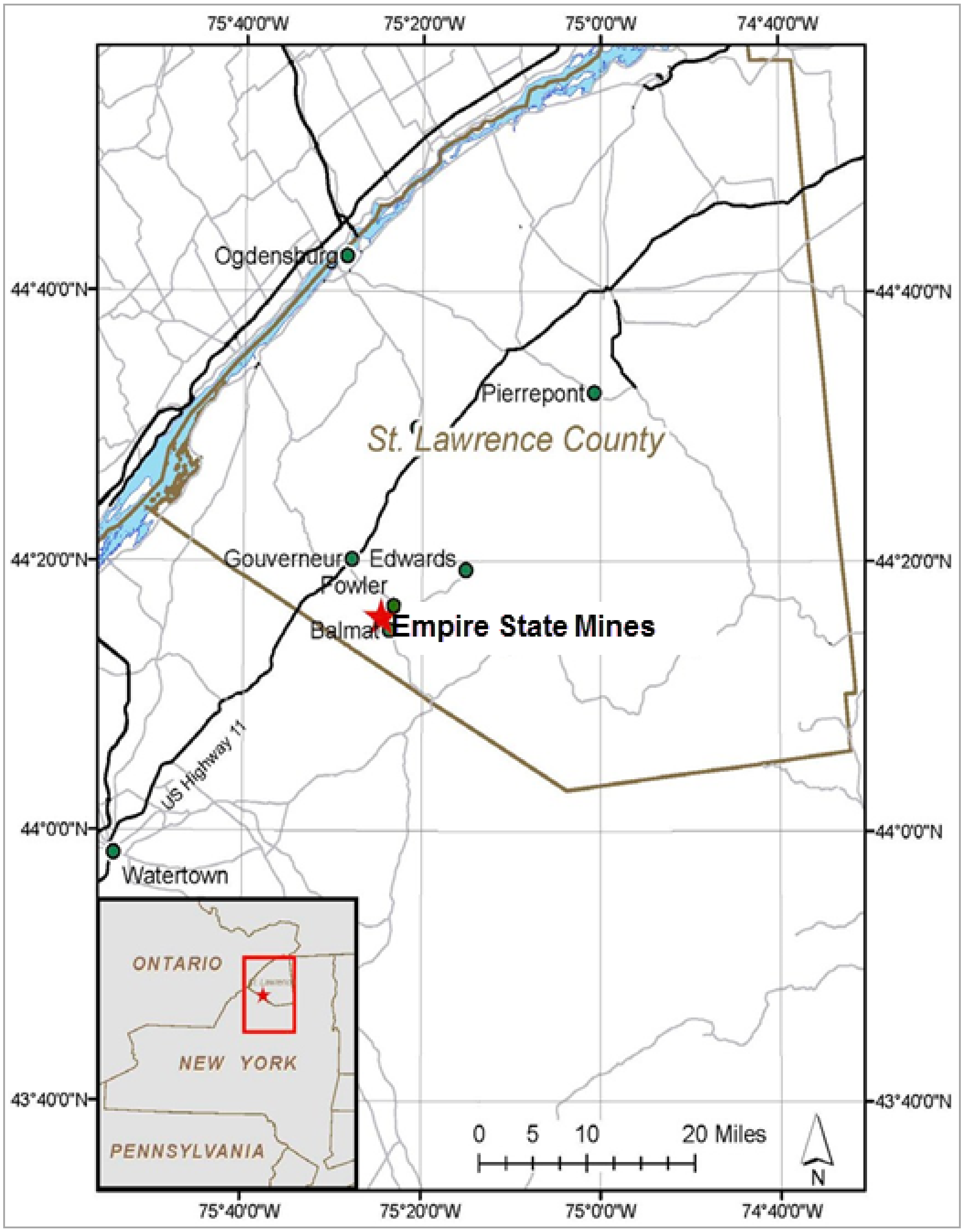
The ESM mine is located 7 miles (mi) southeast of Gouverneur, New York at 44°14'51" N latitude, 75°23'50" W longitude, and 710' ASL. The site is 38 mi via State Road #812 from the St. Lawrence Seaway at Ogdensburg, NY (Figure 4.1).

The town of Gouverneur is located 90 mi from Ottawa, Canada, and is 100 mi northeast of Syracuse, New York.

Figure 4.1: Regional Project Location

Source: JDS (2017)

Figure 4.2: Local Project Location



Source: SLZ (2017)

4.2 Mineral Tenure

The 2,699 acres of surface rights owned by SLZ are divided among the Fowler, Edwards and Pierrepont townships, containing, respectively 1,754, 703 and 242 acres. There are 51,428 acres of mineral rights located in St. Lawrence and Franklin Counties that are comprised of multiple individual parcels in selected areas in and around the mines.

The acquisition also includes transference of 29,054-acres of leased and optioned mineral rights in portions of the Balmat, Hyatt, and Pierrepont mine areas as well as areas of interest for exploration purposes.

Leases have an initial 20-year term, renewable for an additional 20 years, and are subject to a 4% net smelter return (NSR) royalty. One primary lease holding and five smaller leases are included in the ESM mine land package that covers 20% of the mineral rights of the major area of the Mahler resource. Three leases are held in the area around the Hyatt mine and 11 leases are held in the Pierrepont mine area, covering 515 and 1,049 acres respectively. Leases comprising 300 acres are also held in the Emeryville and Talcville exploration areas.

Optioned mineral rights have a renewable 5-year initial term. Option payments amount to US$ 4 per acre per annum.

A list of leases with expiration dates are provided in Table 4.1. Several lease and option agreements have expired; however, the Company continues to make payments to the relevant rights holders and expects to commence negotiations for new lease agreements with respect to the expired leases and options in 2017. The Company will consult with its legal advisors prior to any activities in areas covered by expired leases or options, as it cannot be assumed that the lease agreement or option agreement will extend beyond the expiration date despite acceptance of payment by leasors or option grantors , as the case may be. The current resource and subsequent planned mining areas and possible extensions to those zones are not situated on lands covered by the expired leases or option agreements.

Table 4.1: Lease List with Expiration Dates Organized by Area

Empire State Mine Area

Current Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
Karen and Brooke E. Bishop Lease (1.19 Ac)	Lease	6/15/2037	6/15/2018	1.19	20 years: renewable for additional 20 years	4%	Lease payment with escalator schedule
Davis (Robert and Peggy) Lease (0.5 Ac)	Lease	26/05/2030	26/05/2017	0.5	20 years: renewable for additional 20 years	4%	Lease payment with escalator schedule
Davis (Stanley and Carol) Lease (14.4 Ac)	Lease	6/12/2026	6/12/2017	12.28+2.12	20 years: renewable for additional 20 years	4%	
Hull Lease	Lease	30/04/2017	30/04/2017	20	20 years: renewable for additional 20 years	4%	**RENEWED** 30/04/2017
Manning Lease	Lease	1/10/2027	1/10/2017	0.65	20 years: renewable for additional 20 years	4%	
Timothy J. Sweeney (Lease)	Lease	16/07/2030	16/07/2017	1.91	20 years: renewable for additional 20 years	4%	Lease payment with escalator schedule
Brian Tripp Lease (90Ac)	Lease	22/03/2021	22/03/2017	90	20 years: renewable for additional 20 years	4%	
Brian Tripp Lease (0.79Ac)	Lease	6/12/2026	6/12/2017	0.79	20 years: renewable for additional 20 years	4%	
Robert G., Sr. and Phyllis J. Tripp Lease (19 Ac)	Lease	6/12/2026	6/12/2017	19	20 years: renewable for additional 20 years	4%	

Table 4.1: Lease List with Expiration Dates (continued)

Warriner Lease	Lease	18/01/2031	18/01/2017	80.82	20 years: renewable	4%	
Whitman Lease	Lease	10/2/2018	10/2/2017	30	20 years: renewable for additional 20 years	4%	
Yerdon Lease	Lease	10/7/2027	7/7/2017	0.3	20 years: renewable for additional 20 years	4%	
Zira Lease	Lease	27/07/2027	25/07/2017	0.93	20 years: renewable for additional 20 years	4%	

Expired Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
St. Lawrence Ore Lease	Lease	25/01/2010	25/01/2017	135	20 years: NOT renewable	4%	Expired 1/25/2010 however minimum annual payment was made on time

Hyatt Mine Area

Expired Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes

Table 4.1: Lease List with Expiration Dates (continued)

Cole Lease	Lease	20/06/2000	20/06/2017	94	20 years: renewable for additional 20 years	4%	Expired 20/06/2000 $200 - this payment is redistributed to heirs below
Kelly Freeman Lease	Lease	2/5/2015	2/5/2017	310	20 years: renewable for additional 20 years	4%	Expired 2/5/2015
Jenne Lease	Lease	7/4/2000	7/4/2017	111	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year

Pierrepont Mine Area

Current Options and Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
Edwards Lease	Lease	17/06/2023	17/06/2017	96	20 years: renewable for additional 20 years	4%	
Alan Latimer Lease	Lease	7/7/2023	7/7/2017	20	20 years: renewable for additional 20 years	4%	
Walter Planty Option (64.39 Ac)	Option	19/11/2018	19/11/2017	64.39	5-year option	0%	
Wells Lease	Lease	10/1/2029	16/04/2017	178	40 years: NOT renewable	4% zinc; 5% lead	Lease payment date 4/16 (changed from

Table 4.1: Lease List with Expiration Dates (continued)

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
							7/23) used for all Wells leases taken directly from original index file cards

Expired Options and Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
Barkley Lease	Lease	30/07/1999	00/01/1900	78	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year
Barrigar Lease	Lease	24/07/1999	7/7/2017	280	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year
Caswell Lease	Lease	5/11/2002	5/11/2017	98	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on

Table 4.1: Lease List with Expiration Dates (continued)

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
							time each year
Hutchinson Lease	Lease	1/10/2002	1/10/2017	37	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year
Stiles Lease	Lease	27/09/2002	27/09/2017	32	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year
Thivierge Lease	Lease	27/08/2002	27/08/2017	66	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year
Van Brocklin Lease	Lease	27/07/2002	27/07/2017	100	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year.

Table 4.1: Lease List with Expiration Dates (continued)

Exploration Areas

Current Options and Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
Gilbert Lease	Lease	22/03/2031	22/03/2017	96.4*	20 years: renewable	4%	The lease portion of the agreement was signed – with escalator
Gouverneur Talc Co Lease	Lease	28/06/2030	None	2,500	20 years	4%	Renewed for an additional 20 years 6/28/2010-06/28/30
James Morrill Lease	Lease	8/9/2029	8/9/2017	464	20 years: renewable for additional 20 years	4%	
Stanley Morrill Lease	Lease	8/9/2029	8/9/2017	266.22	20 years: renewable for additional 20 years	4%	
St. Lawrence County Option	Option	11/3/2024	20/04/2017	85.5 & 30	5-year option	4%	Option payment with escalator schedule
Emery W ebb Lease	Lease	22/09/2029	22/09/2017	181.46	20 year: renewable for additional 20 years	4%	

Expired Options and Leases

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
Aleta Billings	Option	4/6/2015	4/6/2017	157.5	5-year option	4%	Expired

Table 4.1: Lease List with Expiration Dates (continued)

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
Heirs Options							4/6/2015 Option payment with escalator schedule
Aleta Billings Heirs Options	Option	25/06/2015	25/06/2017		5-year option	4%	Expired 25/06/2015 Option payment with escalator schedule
Aleta Billings Heirs Options	Option	15/07/2015	15/07/2017		5-year option	4%	Expired 15/07/2015 Option payment with escalator schedule
Bogardus Options	Option	2/9/2015	2/9/2017	162.2	5-year option	4%	Expired Option payment with escalator schedule
Bogardus Options	Option	8/9/2015	8/9/2017		5-year option	4%	Expired 8/9/2015 Option payment with escalator schedule
Brown Lease	Lease	11/8/1999	11/8/2017	165	20 years: renewable for additional 20 years	4%	First 20-year term has expired; however lease is renewable for 20 years and payments have been made on time each year
Cromwell Heir Option	Option	16/06/2016	16/06/2017	369	5-year option	4%	Expired 16/06/2016

Table 4.1: Lease List with Expiration Dates (continued)

Name	Type	Expiration Date	Payment Anniversary	Acres	Term	NSR	Notes
							Option payment with escalator schedule
Cromwell Heir Option	Option	21/10/2016	21/10/2017		5-year option	4%	Option payment with escalator schedule
Lawrence Emrich Heirs Options	Option	17/08/2015	17/08/2017	229.04	5-year option	4%	Expired 17/08/2015 Option payment with escalator schedule
Gilbert Option	Option	3/3/2016	3/3/2017	0*	5-year option	4%	Expired 3/3/2016 Option with escalator
Lansing-Dodge Option	Option	15/09/2015	15/09/2017	~ 22,000	5-year option	4%	Expired 15/09/2015
Steven A. Sullivan Option	Option	28/10/2012	28/10/2017	158.8 (98.45 [60.00+38.45] + 60.35)	3-year option	4%	Expired 28/10/2012
Marjory Tyler Option	Option	2/12/2015	2/12/2017	183	5-year option	4%	Expired 2/12/2015 Option payment with escalator schedule
Webb Option	Option	26/07/2015	26/07/2017	46	5-year option	4%	Expired 26/07/2015 Option payment with escalator schedule

Source: SLZ (2017)

Table 4.1: Lease List with Expiration Dates (continued)

* The area covered by the Gilbert Lease is 96.4 acres and is subject to agreement with 2 heirs of the original lessor. One heir has signed the lease referred to above as the Gilbert Lease in respect of the other heir the option has expired and is shown above as the Gilbert Option. The total acreage for the Gilbert Lease and Gilbert Option is a total of 96.4 acres which have been reflected in the Gilbert Lease for the purposes of this table

Land surface rights for the purpose of construction of buildings and for other purposes are purchased from landowners; SLZ owns the surface rights to lands where the surface facilities of the ESM mine, concentrator and tailings impoundment are located. In New York State, mineral rights were part of the surface right title granted to the original owner, and are deeded in real property transactions (real property). Mineral rights may be reserved during property transactions or they may be transferred (severed) at the time of a real property transfer. Such reservations often date back to the early 1800's. Mineral rights may or may not be subject to property taxes depending on the town taxing authority. The interest in mineral rights for a particular parcel is commonly divided. For example, in the town of Fowler, it is common to have one party own 4/5 (80%) of the mineral rights, and have a second party own the remaining 1/5 (20%) interest (Hudbay, 2009).

Table 4.1: Mineral Tenure Information

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2014 Taxes US$
119.001-1-8	Pierrepont	80.4				816.57
119.001-1-10	Pierrepont	102.1				1036.82
119.001-1-11	Pierrepont	0.52				3.39
119.001-1-12	Pierrepont	59.3				703.9
119.001-1-18./1	Pierrepont		1.4			84.71
174.004-3-2	Edwards	0.85				64.01
174.004-4-2	Edwards	10.37				265.19
174.004-4-1	Edwards	1.35				115.82
175.003-3-1.1	Edwards	71.6				822.96
175.003-3-19.1	Edwards	3.4				158.49
175.002-1-5.1	Edwards	370.2				3553.96
175.002-1-33	Edwards	161.7			322	1648.97
175.002-1-34.1	Edwards	72.2			330	829.04
175.002-1-32.1	Edwards	11.7			720	277.37
175.002-1-34./1	Edwards		74		720	216.41
1.044-18	Edwards		100		720	213.36
175.002-1-25./1	Edwards		92.2		314	201.17
175.001-1-4./1	Edwards		165		720	216.41
175.002-1-5./1	Edwards		1044		314	798.56
175.003-1-1./2	Edwards		72		720	201.17
175.003-1-1./4	Edwards		18.8		720	201.17
175.003-3-1.1/1	Edwards		70		323	630.94
175.003-3-1.1/4	Edwards			Electrical	323	1767.83
175.003-3-10./1	Edwards		115		330	201.17
175.003-3-13./2	Edwards		53.1		330	201.17
175.004-1-3./1	Edwards		58		720	201.17
175.004-1-6./1	Edwards		20		720	201.17
175.004-1-7./1	Edwards		63.8		720	201.17
175.004-1-11./1	Edwards		97.4		720	323.08

Table 4.2: Mineral Tenure Information (continued)

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2014 Taxes US$
175.004-1-14./2	Edwards		62		720	201.17
187.002-2-1./1	Edwards		30		720	201.17
187.002-2-1./2	Edwards		80.9		720	201.17
188.001-1-15./2	Edwards		25		720	201.17
188.001-1-15./3	Edwards		169.1		720	201.17
188.001-1-17./1	Edwards		65.6		720	201.17
188.001-1-27./1	Edwards		73.8		720	201.17
188.002-1-2./1	Edwards		36		720	201.17
174.004-1-18	Fowler	89.3	89.3		720	679.92
187.001-1-5	Fowler	2.5			720	194.73
187.001-1-21.2	Fowler	44.49			720	403.1
186.004-1-44	Fowler	705.3			720	2266.39
186.004-1-33.11	Fowler	86.5			720	2298.79
186.004-1-31	Fowler	61.6			720	2096.43
187.003-1-2	Fowler	82.3			720	389.46
187.003-1-1	Fowler	1.6			720	7822.09
187.069-1-38	Fowler	0.7			720	2932.26
187.003-1-4.11	Fowler	63.8			720	3049.43
187.003-1-4.121	Fowler	124.7			720	681.58
187.003-2-1.1	Fowler	45.2			322	389.46
199.001-2-52	Fowler	445			314	2266.39
186.002-1-14.11/3	Fowler		146.6		720	19.46
186.002-1-14.11/4	Fowler		144		720	19.46
187.003-1-3./1	Fowler		0.01		720	194.73
187.003-1-4.11/2	Fowler			shaft 4	311 w	93829.03
187.003-1-4.11/3	Fowler		0.01		323	19547.72
187.003-1-4.11/5	Fowler			shop	720	7819.09
187.003-1-4.11/7	Fowler			electric	720	39095.43
187.003-1-4.11/9	Fowler			buildings	720	73812.17
187.003-1-4.11/10	Fowler			warehouse	720	117286.3
187.003-1-4.11/11	Fowler			paint, oil	720	4378.68
187.003-1-4.11/12	Fowler			timber storage	720	4691.45
187.003-1-4.11/13	Fowler			service hoist	720	39095.43
187.003-1-4.11/14	Fowler			large hoist	720	54733.62
187.003-1-4.11/15	Fowler			hoist house	720	46445.36

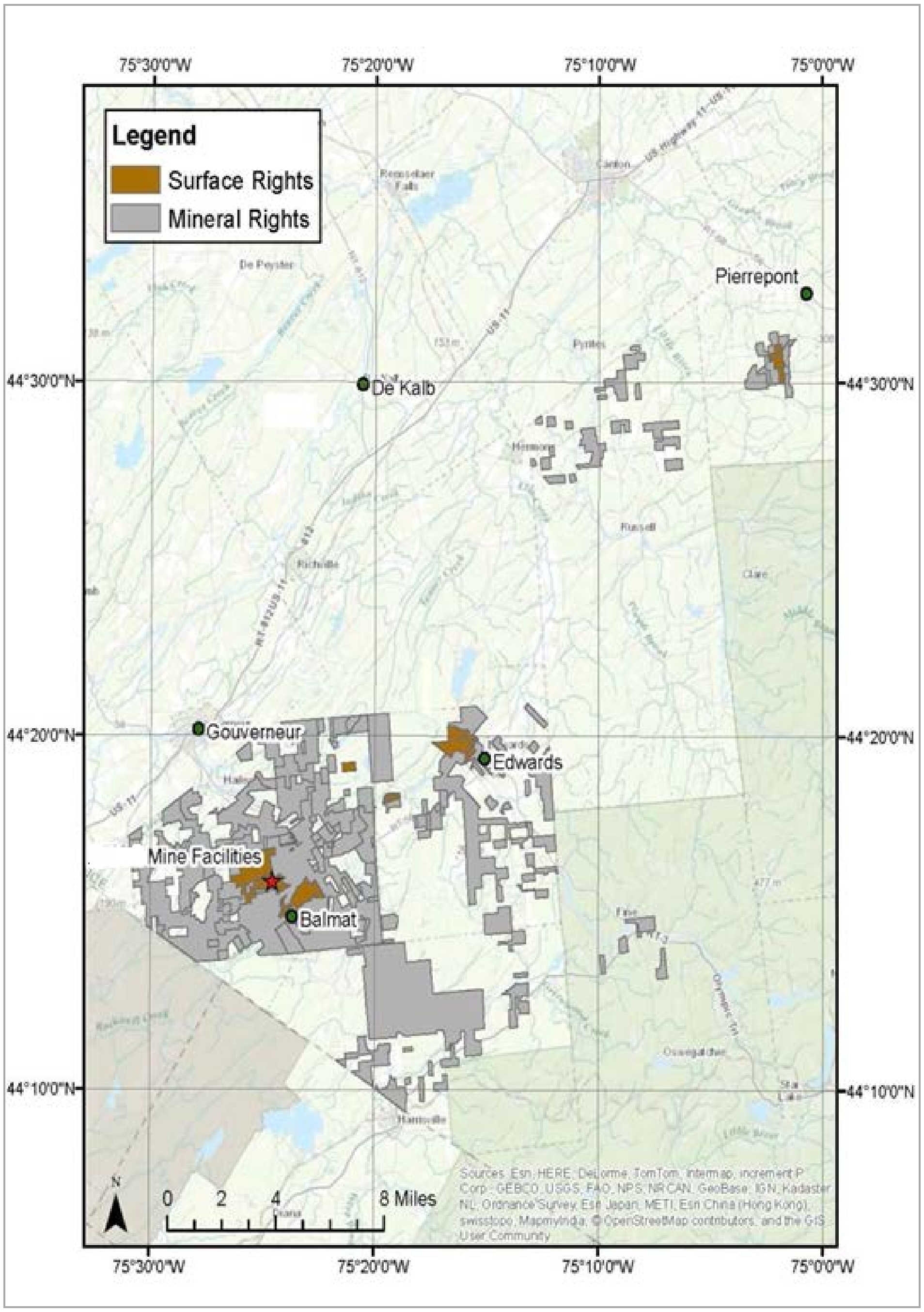
Table 4.2: Mineral Tenure Information (continued)

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2014 Taxes US$
187.003-1-4.11/17	Fowler			railroad #4	720	11728.62
187.003-1-4.11/18	Fowler			mill	720	82768.05
187.003-1-4.11/20	Fowler			storage buildings	720	15638.19
187.003-1-4.11/21	Fowler			storage	720	19547.72
199.001-2-43.1/2	Fowler			pipe shop 2	720	537.48
Owned Fee Parcels		2699	2967			674425

Source: St. Lawrence County Government (2017)

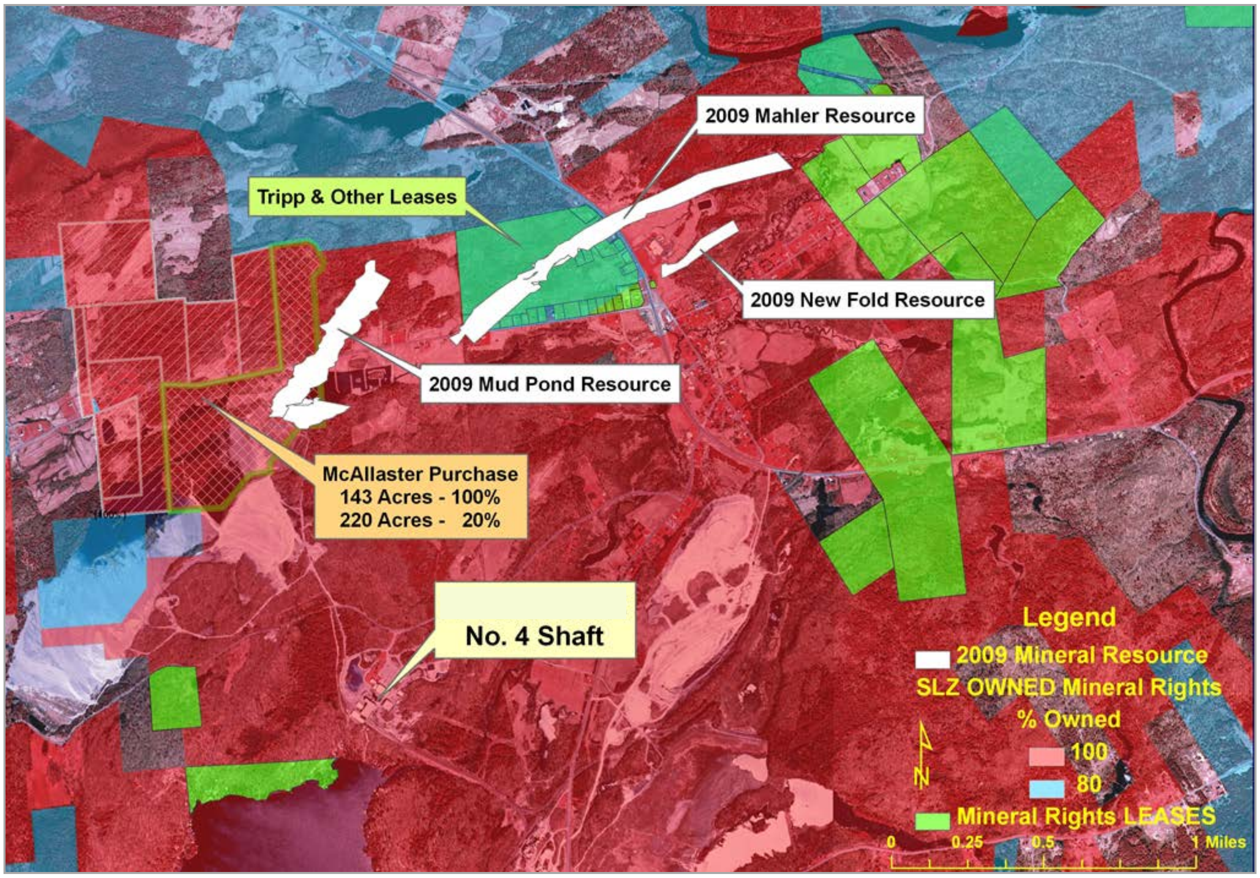
All property listed in Table 4.2 matches the St. Lawrence County 2016 tax rolls and are fully paid and current as of March 1, 2017. The 2016 values are approximately the same and consistent with those listed for 2014; the above tax payment amounts have not been updated for this report due to the County's time and resource constraints.

Figure 4.3: Mineral Tenure Map



Source: SLZ (2017)

Figure 4.4: Mineral Tenure Map



Source: SLZ (2017)

4.3 Mining Rights

Real property in New York State was original granted to the original owner to include both surface and mineral rights. However, mineral rights can subsequently be reserved or sold (severed) separately. SLZ controls both surface and mineral rights for the project area. Land not owned by the Company is either leased or lease optioned from property owners.

4.4 Project Agreements

Mineral rights may be acquired from the owner by lease, or option or purchase. Leases may be renewable and also may be subject to the payment of royalties to the land owner. Average royalties for ESM mineral production are estimated to average 0.3% over the life of the mine (Hudbay, 2010).

4.5 Environmental Liabilities and Considerations

Mining permits and permits for water release to the environment are granted and administered by the New York State Department of Environmental Conservation (NYSDEC). NYSDEC has accepted the reclamation completed at four of the sites and released them from the permit requirements. Some minor monitoring may be required. The NYSDEC has reviewed the reclamation at the satellite properties also acquired with the Balmat purchase, Hyatt mine tailings and mine sites and the Pierrepont mine site, and has released the reclamation bonds posted for these areas. No further work is required.

Reclamation plans approved by the NYSDEC are in place for ESM No. 4 Mine and the ESM No. 2 shaft area (which is still in use as an alternate exit route and ventilation shaft for ESM No. 4 Mine) and are the ongoing responsibility of SLZ. ESM No. 4 mine and mine tailings reclamation is assured with a $1,662,870 certificate of deposit.

The mining activity in the Balmat region has not created any known long term liabilities, beyond those described in Section 20 of this report, as a result of the long operating history at the various operations. The mineralization in the region is typically hosted in an alkaline host rock which has no tendency to generate acid mine drainage and mobilize metals in surface and ground waters. Minor excursions above compliance levels have been historically corrected by additions of sodium sulphate or lime upstream from the water holding ponds.

4.6 Permit Requirements

According to the Hudbay Minerals (Hudbay) Annual Information Filing (AIF) 2008, the extraction of minerals in New York State is governed by the New York State Mined Land Reclamation Law and the rules and regulations adopted thereunder (Hudbay, 2008). A Mined Land Reclamation Permit must be obtained from the Division of Mineral Resources within the New York State Department of Environmental Conservation in order to extract minerals from lands within the state. Such permits are issued for annual terms of up to five years and may be renewed upon application. Permit holders must submit annually to the DEC a fee based upon the total acreage covered by the permit, up to a maximum of $8,000 per year.

To the extent known, all permits required to operate the ESM mine are active and in place. Additionally, there are not any other significant factors or risks that may affect access, title or the right or ability to perform work on the ESM properties.

Major environmental permits required for operation of the ESM No. 4 Mine are listed in Table 4.3.

Table 4.2: Environmental Permits for Operation of No. 4 Mine

Permit Type	Permit	Permit Number	Expiration
Air	Registration to Operate a Zinc Mining and Milling Complex (amended)	6-4038-00024/02001	9/30/2024
Water	SPDES Water Discharge Permit	NY0001791	5/31/2019
Water	Water Withdrawal Permit	6-4038-00024/02001	5/31/2019
Mining	Mining Permit	6-4038-00024/00006	8/1/2020
Storage	NYDEC Chemical Bulk Storage	CBS#6-000122	10/1/2017
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770	9/26/2018
Radiation	Certificate for Density Gauge	44023174	9/15/2018

*SPDES = State Pollutant Discharge Elimination System

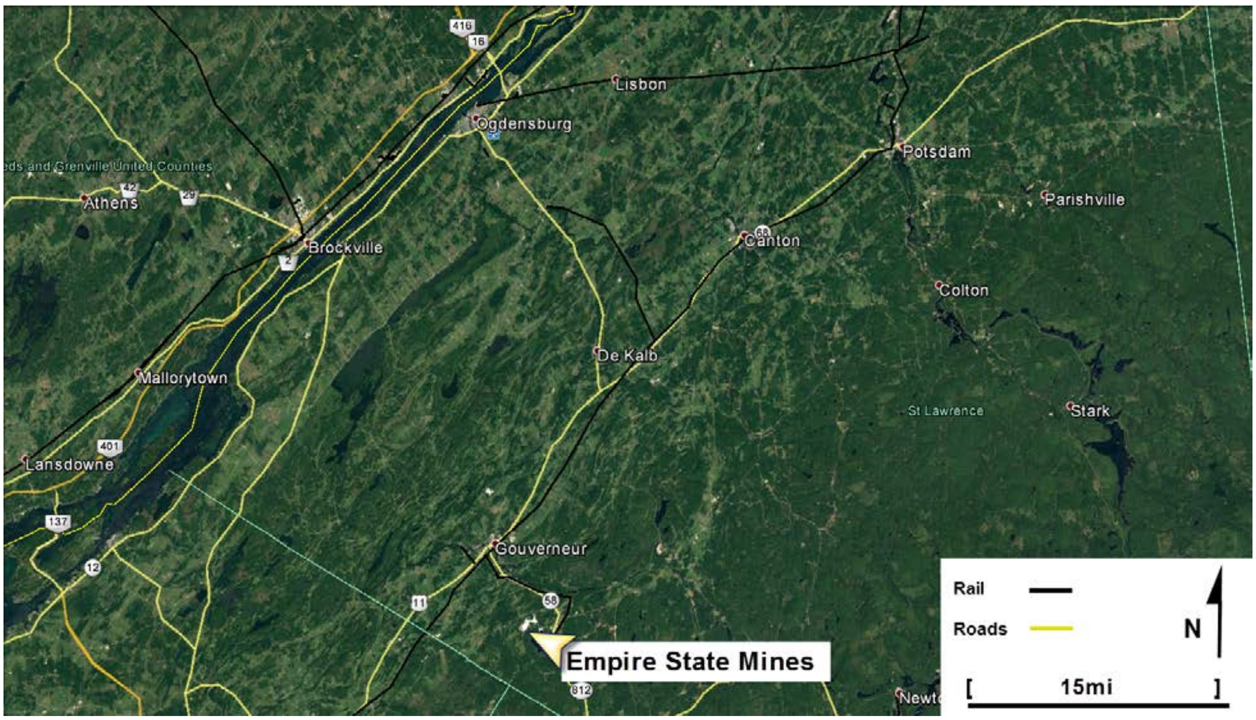
Source: SLZ (2017)

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The property is reached by traveling southeast from Gouverneur, NY for 7.9 mi along NY-812 S, through the town of Fowler, to the mine offices on Sylvia Lake Road. The site lies 38 mi south of Ogdensburg, NY via NY-812 S.

Figure 5.1: Site Accessibility



Source: JDS (2017)

5.2 Local Resources and Infrastructure

The nearest population center is Gouverneur with an estimated population of 7,000. The outlying rural areas have a population of approximately 35,000. All modern services, including hospital, hotel, and railway are present at Gouverneur. Syracuse, NY lies 100 mi to the southwest. Ottawa, Ontario, Canada lies 90 mi to the north.

5.3 Climate

The area has typical mid-continental climate with moderate summers and cold winters, moderated by the nearby Great Lakes. Average annual temperatures are 53° to 38°F. Summer highs may reach 85°F. Winter lows may reach -20°F. Annual average frost free days are 115. Annual average precipitation is approximately 40”, 70% occurs as snow. The mine and process facility will operate year-round. Weather is not expected to frequently or significantly affect operations at any time of the year.

5.4 Vegetation and Wildlife

The ESM project area is classified as hardiness zone 3b by the US Department of Agriculture (USDA). Tree species include hardwoods like sugar maple, black cherry, paper birch and American Beech. Common softwoods include white pine, red pine, scotch pine, and eastern hemlock. Ground cover consists primarily of saplings, various grasses and forbs.

Animal species include whitetail deer, eastern grey squirrels and many varieties of songbirds, fish and waterfowl.

The mine site is surrounded by heavily treed bedrock ridges with interspersed low-lying marsh areas. The area is covered by gravel and clay overburden.

5.5 Physiography

The ESM project is situated on the northwest flank of the Adirondack Mountains. The ESM mine site lies within heavily forested bedrock ridges and interspersed low-lying marsh areas. Elevation at the mine site is 710 ft above mean sea level (amsl). Relief throughout the area ranges from 384 to 1106 ft amsl.

Various classes of streams drain to the St. Lawrence River. The area contains numerous ponds and lakes. Soils vary from loamy sand soil to exposed bedrock.

Figure 5.2: Empire State Mines Mine Aerial View

Source: SLZ (2017)

Figure 5.3: Aerial Photo of Gouverneur, NY

Source : Landstat (2015)

5.6 Surface Facilities and Rights

The existing operation is located on lands owned or leased by SLZ. All utilities such as roads, rail, electricity, water, communications systems, tailing management facilities, waste rock disposal means, and the processing plant currently exist on-site and are in good condition.

A small management staff keeps up with site administration, maintenance, mine dewatering, and permits. Once operations are planned to commence, labour not available locally will be sourced from outside the region. The mine is located in a desirable area to live, so while a significant portion of the labour force may have to be brought into the area, the effort in doing so is expected to be reasonable and customary for a developed location in North America.

6 History

6.1 Management and Ownership

The Empire State Mines (ESM) operation is wholly owned by St. Lawrence Zinc Company, LLC (SLZ), a subsidiary of Titan. A history of ownership is listed in Table 6.1

Star Mountain Resources, Inc. purchased SLZ from Hudbay in November of 2015.

On December 30, 2016, Titan US purchased the shares of Balmat Holding Corporation, which in turn holds the shares of SLZ. Titan is a privately held company whose primary asset is ESM. Titan changed the name of the mine from Balmat to Empire State Mines in February 2017.

Table 6.1: History of Ownership

Date	Company	Activity
1915 – 1987	St. Joe Minerals Corporation & Predecessors. Purchased by Fluor Corporation in 1981.	Mined Edwards in 1915 and Balmat in 1930
1987 – 2001	Zinc Corporation of America (ZCA)	Purchased operation and mined through 2001
2003 – 2015	OntZinc (renamed Hudbay Minerals in December 2004)	Purchased ZCA and mined Balmat from 2005 to 2008
2015 – 2016	Star Mountain Resources Inc.	Purchased SLZ from Hudbay
2016 – Present	Titan Mining (US) Corporation	Purchased Balmat shares from Star Mountain and renamed Balmat mine to Empire State Mines (ESM)

Source: SLZ (2017)

6.2 Exploration History

In 1838, zinc was discovered in a prospect pit on the Balmat farm near the present Balmat No. 1 shaft location. Further zinc was discovered in the Balmat-Edwards-Pierrepont district from road excavations in 1908. Gossan was later recognized and subsequent core drilling defined the mineral resources of the Balmat No. 2 Mine in 1928. In 1945, surface drilling, down-plunge from surface showings, intersected the Balmat No. 3 Mine mineral resources. A systematic fence-drilling program across the Sylvia Lake Syncline (perpendicular to the plunge) discovered the mineral resources of Balmat No. 4 Mine in 1965. In 1979, the Pierrepont mine was discovered while drilling down-plunge from geochemical anomalies. Mine development and exploration drilling added significant reserves to the Hyatt mine in 1994, and to the Balmat No.4 Mine in 1996, with the expansion of the Mud Pond. The New Fold and Mahler resources were later discovered in the No. 4 Mine in 1997 and 2000.

The Balmat area has had an active mining history for the past 85 years. On average, during the period between 1908 (discovery of the Edwards mine) and 1979 (discovery of the Pierrepont mine), a mine was discovered every 17 to 18 years in the Balmat-Edwards-Pierrepont district. Road excavations exposed zinc mineralization that was developed into the Edwards (1908) and Hyatt (1917) mines.

6.3 Production History

Since 1915, six zinc mines have operated in the Balmat-Edwards district, collectively now known as Empire State Mines. Zinc was first produced from the Edwards mine in 1915 and from the Balmat No. 2 Mine in 1930. The other mines in the district are the Balmat No. 3, Balmat No. 4, Hyatt, and Pierrepont.

Mines were operated in the district by St. Joe Minerals Corporation's subsidiary companies including St. Joseph Lead Company and St. Joe Resources Company (all referred to as "St. Joe Minerals"), and its predecessors from 1915 to 1987. Zinc Corporation of America (ZCA) purchased the mines in 1987 and operated them until 2001, shutting down the Balmat operations when high grade feed from the Pierrepont mine was exhausted. OntZinc, renamed Hudbay Minerals Inc. in December 2004, purchased the idle Balmat assets in September 2003. The Balmat #4 Mine re-opened in 2006 and operated into 2008. The mine was placed on care and maintenance in August 2008.

From 2006 to 2008, Hudbay mined 855,000 t of mineralization grading 7% zinc from the Davis, Mud Pond, Mahler, Fowler, Upper Fowler and New Fold zones.

The Balmat #2, #3 and #4 Mines has produced 33.8 Mt of 8.6% Zn since operations began in 1930. The greater Balmat-Edwards-Pierrepont district has produced in excess of 43 Mt of 9.4% Zinc during the 76 years of operation by St. Joe Minerals and its predecessor companies. This is based on the formal reserves calculation prepared in 2001 by ZCA.

The existing Balmat mill was constructed in 1971 by St. Joe Minerals and has a nameplate capacity of 5,000 t/d. The mill has processed mineralized material from the Hyatt, Pierrepont and Balmat Mines. The Balmat No. 4 shaft is adjacent to the mill and accesses zinc mineralization from the 1300, 1700, 2100, 2500 and 3100 levels. All mine plan tons in this PEA will be hoisted from the 3100 level of the No. 4 shaft.

Table 6.2: Gross Historical Production

Mine	Year Discovered	Year Closed	Tons Mined (Mt)	Zinc Grade (%)
No. 2 Mine	1928	1998	17.8	8.7
No. 3 Mine	1945	1985	5.7	9.4
No. 4 Mine	1965	2008	10.2	7.9
Total			**33.8**	**8.6**

Source: SLZ (2017)

Table 6.3: Recent Annual Historical Production

Year	Ownership	Balmat No. 4 Mine		Pierrepont Mine		Concentrate Produced	
		kt	Zn%	kt	Zn%	kt	Zn%
1998	ZCA	579	6.7	166	12.8	102	55.5
1999	ZCA	627	6.5	106	13.5	93	55.4
2000	ZCA	581	6.1	134	12.1	88	55
2006	Hudbay	178	6.1	0	0	0	0
2007	Hudbay	367	7	0	0	38.6	57.2
2008	Hudbay	310	8	0	0	37.3	57.3
Total		**2,642**	**6.7**	**406**	**12.8**	**359**	**56.1**

Source: SLZ (2017)

6.4 Historic Mineral Reserves

A list of most recent mineral reserve estimates is set out in Table 6.4 for the previous owners of the project: St. Joe Minerals, Hudbay Mining & Smelting and, Star Mountain Resources. The historic mineral reserves are relevant as they detail the change in mineral reserves over time estimated by different persons and methods. The Company is not treating these historic estimates as a current Mineral Reserve and the Company is not basing its production decision on these historic estimates. The authors are unaware of complete methods, parameters or assumptions used to generate these historic estimates including cut off grades and dilution and cannot comment to their accuracy or reliability. A qualified person has not done sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves. The 1985 St. Joe Minerals estimate used different classification definition terms that are explained in the table notes (items 1, 2 & 3) and was published as an annual summary tabulation. HudBay Mining & Smelting's estimates were reported to NI 43-101 standards and categories and published with summary details in Annual Reports. The 2015 Star Mountain Mineral Reserve estimation was reported in a US SEC Industry Guide 7 Report with categories equivalent to NI 43-101, several of the assumptions and methods used are not considered current best practice with the resource wireframes not snapped to the drill holes and only single full length composites used for each intercept. The 1985 mineral reserves listed in the Fowler, Upper Fowler, Davis and Loomis areas are exclusive of the current Mineral Resource published in this Technical Report, it is unknown of any of these areas are included in the "1992 Low Grade Reserve" write-downs as detailed in Section 6.5. The Company has completed work to re-estimate Mineral Resources only in some areas covered by the previous historic mineral reserves by HudBay in 2005, 2006, and 2007 and by Star Mountain in 2015, as outlined in this Technical Report. Prior to establishing any Mineral Reserves, the Company plans to complete additional drilling activities and complete a Pre-Feasibility Study or Feasibility Study.

Table 6.4: Historic Mineral Reserves

Year	Proven		Probable		Proven and Probable	
	Mass (000's tons)	Zn Grade	Mass (000's tons)	Zn Grade	Mass (000's tons)	Zn Grade

1985[4]	1,159[1,3]	11.52	598[2,3]	9.81	1,758[3]	10.94%
2005[5]	763	10.90%	1,095	11.40%	1,859	11.20%
2006[6]	912	10.10%	1,163	11.40%	2,075	10.80%
2007[7]	1,000	9.50%	890	10.80%	1,891	10.20%
2015[8]	152	9.00%	394	9.20%	531	9.20%

Notes:

(1) Proven ore designates ore so well outlined by development and closely spaced diamond drilling that the risk of failure in continuity of the ore is reduced to a minimum.

(2) Probable ore refers to ore for which the risk of failure is greater than for proven ore, but for which there is sufficient justification in assuming continuity of the ore. Probable ore is substantiated by wider spaced diamond drilling and by little or no development. Probable ore includes ore of probable future value, but for the present rendered unavailable by reason of ground support, ground water or proximity to an operating shaft.

(3) Proven ore and probable ore are not equivalent to the CIM definitions of Proven Mineral Reserve and Probable Mineral Reserve, respectively. There is no similar concept to Inferred Ore under the CIM definitions.

(4) Sources: "St. Joe Resources Company Mining Division, Ore Grade Reserves as of October 31, 1985," prepared by Geology Department for St. Joe Resources Company. It is unknown if any portions of the 1985 ore grade reserves were included in "1992 Balmat Low Grade Reserves" and ultimately written down as detailed in section 6.5

(5) Source: " Balmat No. 4 Zinc Mine Re-Opening Feasibility Study. 2005" dated November 1, 2005, prepared by HudBay Minerals Inc.

(6) Source: "HudBay Minerals Annual Report 06" dated January 1, 2007, prepared by HudBay Technical Services for HudBay Mining and Smelting Co.

(7) Source: "HudBay Minerals Annual Report 07" dated January 1, 2008, prepared by HudBay Technical Services for HudBay Mining and Smelting Co.

(8) Source: "Industry Guide 7 Report: Mineral Reserves at the Balmat Mine, St. Lawrence County, New York" dated November 2, 2015 prepared by Practical Mining LLC for for Star Mountain Resources Inc.

6.5 Historic Mineral Ore Write-Downs

The Company plans to evaluate historic mineral reserves and remnants as targets for future exploration activities and will prioritize areas based on proximity to current Mineral Resources that may, if economic, extend the life of the Empire State Mine and/or add to Resources and take advantage of spare capacity in its 5,000 tpd nameplate capacity processing facility.. During the operations of the previous owners, proven ore, probable ore and inferred ore were periodically re-estimated, with part of this process re-assessing the continued feasibility, considering technical and economic conditions at the time. Proven ore and probable ore are not equivalent to the CIM definitions of Proven Mineral Reserve and Probable Mineral Reserve, respectively. There is no similar concept to inferred ore under the CIM definitions.

During these re-estimations, certain areas, or remnants, that were deemed technologically unlikely to be extracted due to the rock mass being supporting pillars and areas of lower grade that become marginally economic or uneconomic at the time were removed from the proven ore, probable ore and inferred ore estimations. These areas were "written down" and no longer considered in the economic operation. But the previous owners did calculate these areas with tons, grade and classification, designating them "Low Grade Reserves" and "Pillars" and filing internally as such, to allow for re-evaluation when technical or economic conditions change, listed in Table 6.5.

Table 6.5: Historic Mineral Ore Write Downs

Year	Area	Proven Ore(1),(4)			Probable Ore(2),(4)			Proven Ore and Probable Ore			Inferred Ore(3),(4)		
		Mass (000's tons)	Zn Grade	Contain-ed Zinc (tons)	Mass (000's tons)	Zn Grade	Contain-ed Zinc (tons)	Mass (000's tons)	Zn Grade	Contain-ed Zinc (tons)	Mass (000's tons)	Zn Grade	Contain-ed Zinc (tons)
1985	Balmat (No. 2 and No.4)(5)	824	7.11%	58,602	861	7.41%	63,772	1,685	7.26%	122,374	1,097	6.84%	75,021
1985	Balmat No. 2 Mine - Shaft Pillar(6)	-	-		223	7.63%	16,992	222.7	7.63%	16,992	-	-	
1992	Balmat Mine – Low-grade Reserves(7)	-	-		130	7.60%	9,895	130.162	7.60%	9,895	-	-	
2001	Mud Pond Pillars(8)	105	10.30%	10,815	-	-		105	10.30%	10,815	-	-	
1976	Balmat No. 3 Mine - Upper Gleason Pillars(9)	20	12.00%	2,400	-	-		20	12.00%	2,400	-	-	
1985	Pierre-pont(10)	7	6.00%	408	123	6.97%	8,575	129.9	6.92%	8,983	-	-	
1998	Hyatt Mine(11)	79	7.78%	6,166	102	4.75%	4,818	180.779	6.08%	10,984	316	6.37%	20,124
Total		1,035	7.57%	78,391	1,438	7.20%	104,052	2,473	7.40%	182,443	1,413	6.73%	95,145

Notes:

(1) Proven ore designates ore so well outlined by development and closely spaced diamond drilling that the risk of failure in continuity of the ore is reduced to a minimum.

(2) Probable ore refers to ore for which the risk of failure is greater than for proven ore, but for which there is sufficient justification in assuming continuity of the ore. Probable ore is substantiated by wider spaced diamond drilling and by little or no development. Probable ore includes ore of probable future value, but for the present rendered unavailable by reason of ground support, ground water or proximity to an operating shaft.

(3) Inferred ore designates ore for which quantitative estimates are based largely on assumed continuity or repetition justified by good geologic evidence. Inferred ore is indicated by few if any diamond drill holes and by little or no development. Inferred ore also refers to estimates of ore based on a total tonnage expectancy, or potential, for each deep level, as projected from past experience on upper levels, and is equivalent to the total tonnage expectancy for the level less the sum of total production to plus calculated reserves.

(4) Proven ore and probable ore are not equivalent to the CIM definitions of Proven Mineral Reserve and Probable Mineral Reserve, respectively. There is no similar concept to Inferred ore under the CIM definitions.

(5) Sources: "St. Joe Resources Company Mining Division, General Summary of Low Grade Reserves as of October 31, 1985," prepared by Geology Department, St. Joe Resources Company, and tabulations of "Low-grade Reserves for Balmat No. 2 Mine and No. 4 Mine," compiled by individual orebody, reserve classification and level, dated October 31, 1985, prepared by Geology Department, St. Joe Resources Company, and updated April 1990 by Geology Department, Zinc Corporation of America. The Balmat No.2 and No.4 Mines are dewatered, some rehabilitation may be required for access to individual areas.

(6) Sources: "St. Joe Resources Company Mining Division, General Summary of Low Grade Reserves as of October 31, 1985," prepared by Geology Department, St. Joe Resources Company, and tabulations of "Low-grade Reserves for Balmat No. 2 Mine and No. 4 Mine," compiled by individual orebody, reserve classification and level, dated October 31, 1985, prepared by Geology Department, St. Joe Resources Company, and updated April 1990 by Geology Department, Zinc Corporation of America. The Balmat No.2 Shaft is still in use as a secondary egress.

(7) Source: "General Summary of Formal and Low Grade Reserves – 1992" dated February 2, 1993, prepared by Geology Department, Zinc Corporation of America. The Balmat No.4 Mine is dewatered, some rehabilitation may be required for access to individual areas.

(8) Source: Derived from "2001 Mud Pond Ore Reserve" tabulation compiled by individual orebody, reserve classification and level, prepared by Geology Department, Zinc Corporation of America. The Balmat No.4 Mine is dewatered, some rehabilitation may be required for access to Upper Mud Pond Area.

(9) Source: Derived from "Pillars in Upper Gleason" tabulation and analysis dated December 31, 1976, prepared by Geology Department, St. Joe Resources Company. The Balmat No. 3 Mine surface area has been reclaimed and underground mine flooded.

(10) Source: "St. Joe Resources Company Mining Division, General Summary of Low Grade Reserves" as of October 31, 1985, prepared by Geology Department, St. Joe Resources Company. The Pierrepont Mine surface area has been reclaimed and underground mine flooded.

(11) Source: "Hyatt Mine – Total Salvage ore and low-grade ore reserves as of December 31, 1998," dated January 7, 1999, prepared by Geology Department, Zinc Corporation of America. The Hyatt Mine surface area has been reclaimed and underground mine flooded.

None of the mineralization in areas that were written down by the previous owners is included in the ESM's current mineral resource envelope and each of these estimates cover a separate area. A qualified person has not done sufficient work to classify these historical estimates as a current Mineral Reserve. The Company does not treat the historical estimates as a current Mineral Resource or Mineral Reserve. The Company believes that this historic proven ore, historic probable ore and historic inferred ore are relevant to its prospects to extract additional mineralized material at the Empire State Mine Project, however, the Company is not basing its production decision on the historical estimates. The Company is aware of some of the methods used to estimate the historic proven ore, historic probable ore and historic inferred ore based on standard polygonal estimation procedures, in plan or section and CAD-based area and volume calculation but not the particular details involved. The important assumptions and parameters including cut off grades and dilution used to calculate the historic proven ore, historic probable ore and historic inferred ore are not known to the Company. Generally, the work needed to upgrade the historic estimate to a Mineral Resource or Mineral Reserve includes channel sampling and/or diamond drilling, additional modelling of the mineralization and, underground inspections were applicable to confirm that the mineralization remains on site, some rehabilitation may be required for access to specific heading locations. The surface areas at the former Pierrepont, Hyatt and Balmat No. 3 Mines as listed in the second part of table 6.5, have been substantially reclaimed and the underground areas flooded, and as such will require re-permitting and de-watering to enable access to the underground areas.

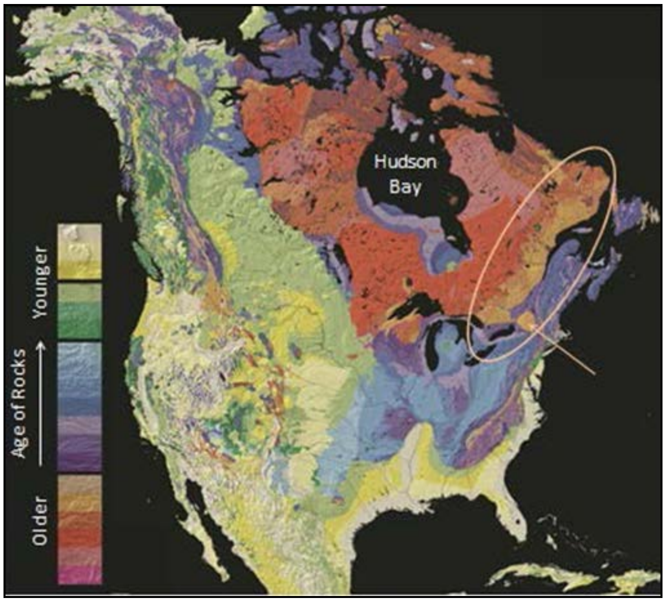
7 Geological Setting and Mineralization

7.1 Geological Setting

Empire State Mines (ESM) is located in a region with a very long and complex geological history. The host rocks were deposited during the mid-Proterozoic era between roughly 1300 to 1000 Ma (mega-annum, millions of years before present), near the edge of the North American craton. Due to their position near the margin of this tectonic domain, they were subject to tectonic forces that, over a billion years, assembled and broke up two supercontinents- Rodinia in the late Proterozoic, and Pangaea in the late Paleozoic to early Mesozoic. Zinc deposition is interpreted to have occurred contemporaneously with deposition of the rock units, which indicates that the originally tabular zinc bodies were intensely deformed and metamorphosed along with their host rocks through eons of varying tectonic forces.

The mine is located near the eastern edge of the Canadian Shield, a vast expanse of very old exposed bedrock which can be described as the core of the North American continent. The Canadian Shield was assembled in an ancient zone of prolonged tectonic convergence. During the Archean and Proterozoic eons, tectonic forces were focused towards the region that is now the Canadian Shield. As tectonic plates moved towards this zone they collided with each other, resulting in compressive forces that caused extensive uplift of continental crust high above sea level. The forces were active for millions of years, and material from advancing plates was gradually added to the crustal core. The added material is known as accreted terranes. The Canadian Shield was built as terranes agglomerated over time (Marshak, Stephen, Essentials of Geology, 2009). In Figure 7.1, the Canadian Shield can be seen as the red and orange band encircling Hudson Bay.

Figure 7.1: Regional Geology Setting



Source: SLZ (2017)

One of the final, major series of tectonic events that occurred before tectonic forces shifted away from the Canadian Shield is known collectively as the Grenville Orogeny. The Grenville Orogeny includes a series of exceptionally intense accretionary events which occurred during the Mesoproterozoic era, as assembly of the supercontinent Rodinia neared completion. The scale of the orogeny is analogous to the present day (Himalaya Tollo, Richard P.; Louise Corriveau; James McLelland; Mervin J. Bartholomew, 2004). The series of terranes that were accreted during the Grenville Orogeny are collectively known as the Grenville Province. The Adirondack Mountains, which contain the ESM mineralization, are part of the Grenville Province. In Figure 7.1, the Grenville Province, shown in light orange, is circled.

Following the Grenville events, tectonic forces shifted away from the Canadian Shield and rifting commenced. Mountain ranges underwent collapse (Tollo, Richard P.; Louise Corriveau; James McLelland; Mervin J. Bartholomew, 2004). Erosion outpaced uplift. Over billions of years of passive tectonism, the Canadian Shield was eroded to low relief. The area outboard from the Grenville Province, including the area that is now the Adirondacks, subsided below sea level and eventually accumulated a cover of Paleozoic sediment. Paleozoic sedimentary deposition began with the late Cambrian to early Ordovician Potsdam Sandstone, followed by a limestone-dolostone sequence (Derby, James; Fritz, Richard; Longacre, Susan; Morgan, William; Sternbach, Charles, 2013). Potsdam sandstone can be identified in the project area.

Magmatism accompanied both orogenesis and rifting, and as a result the Grenville Province contains many igneous intrusions of various ages, which have been metamorphosed at varying intensities. These are not thought to have been involved in mineral deposition at ESM.

Following the late Precambrian to early Cambrian era of passive tectonism and the late Cambrian to early Ordovician period of deposition, a new series of tectonic events began that would build the Appalachian Mountains. These events are called the Taconic, Acadian and Alleghenian orogenies. During the middle Ordovician Taconic and the mid to late Devonian Acadian orogenies, the area that would become the Adirondacks was buried, followed by uplift and exhumation during the late Pennsylvanian to Permian Alleghenian orogeny (Share, 2012). By the end of the Alleghenian orogeny, the Appalachians had reached heights comparable to the current Rocky Mountains (Hatcher, R. D. Jr., W. A. Thomas & G. W. Viele, eds. 1989). The Adirondacks had not yet been uplifted.

Uplift of the Adirondack dome is generally attributed to the passage of the North American plate over the Great Meteor Hotspot in the early Cretaceous. The theory lacks consensus because the Adirondack Dome lies somewhat south of the apparent track of the Great Meteor Hotspot, and because of a lack of direct evidence such as volcanic rock deposition attributable to hotspot volcanism. Taylor and Fitzgerald suggest the Adirondacks were formed through dissection of a plateau. In Figure 7.1, an arrow points to the Adirondack Mountains (Taylor, Joshua P. and Fitzgerald, Paul G., 2011).

7.2 Regional Geology

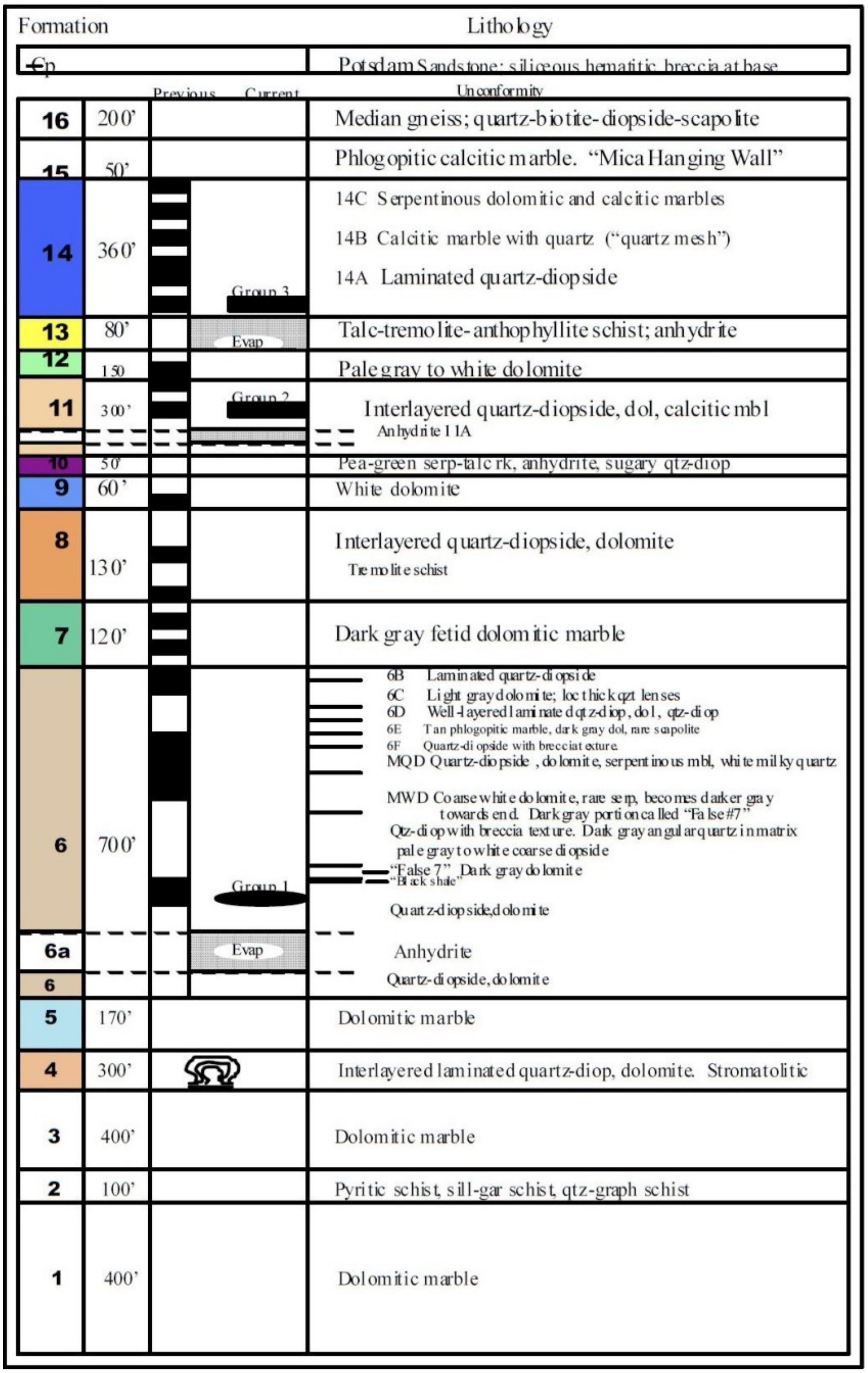
The Adirondacks are considered an outlier of the Grenville Province since they are nearly surrounded by Proterozoic sediments. The Adirondack dome may have been forced upwards through the Proterozoic sediments by the Great Meteor Hotspot. A narrow strip of Mesoproterozoic bedrock called the Frontenac Axis connects a section of the northwestern flank of the Adirondacks to the rest of the Grenville Province. The Adirondacks are lithologically and topographically divided into two main zones, the Highlands and Lowlands. The Lowlands comprise the relatively small northwestern portion of the Adirondacks, and the Highlands make up the main body of the Adirondack Dome. The Highlands and Lowlands are divided by the Carthage-Coulton shear zone (Mezger, K., van der Pluijm, B. A., Essene, E. J., Halliday, A.N., 1992). The Lowlands have been metamorphosed to amphibolite grade, the Highlands to higher granulite grade (McLelland, James M., Selleck, Bruce W., and Bickford, M.E., 2010,). ESM is located in the Adirondack Lowlands.

The rocks of the Adirondack Lowlands are part of the Grenville Supergroup. The Grenville Supergroup is a group of metamorphosed sedimentary terranes that compose a section of the Grenville Province known as the “Central Metasedimentary Belt” (Davidson, A., An Overview of Grenville Province Geology, Canadian Shield, in Lucas, S.B. and St-Onge, M.R., 1998,). The rocks of the Adirondack Lowlands were deposited in the Trans-Adirondack back arc basin prior to final accretion of the Grenville Province (Chiarenzelli, Jeff, Kratzmann, David, Selleck,Bruce, deLorraine, William, 2015). The Adirondack Lowlands have been divided into three stratigraphic formations: the Upper Marble Formation, the Popple Hill Gneiss, and the Lower Marble Formation. The zinc mineralization at ESM is contained in the Upper Marble Formation.

7.3 Local Geology

The Upper Marble Formation is a sequence of shallow water carbonates consisting of multiple series of dolomitized marbles and quartz diopsides with occasional schists and periodic occurrences of anhydrite. It is divided into 16 units. Geologists working in the Balmat-Edwards zinc district have recognized distinct marker horizons within the marble which allow them to identify favourable locations for zinc mineralization. The marker horizons include a pyritic schist, a dark grey dolomitic marble, and the periodic anhydrite beds. The anhydrites are of particular importance because zinc deposition appears to have followed anhydrite deposition. Units 6, 11 and 14 contain massive strataform sphalerite bodies occurring soon after anhydrite beds in the lithologic sequence. Units 6-10 locally host semi-massive crosscutting sphalerite bodies where structures intersect sphalerite deposits contained in unit 6, 11 or 14. Figure 7.2 shows the stratigraphic section for the ESM area.

Figure 7.2: Empire State Mines Stratigraphic Section

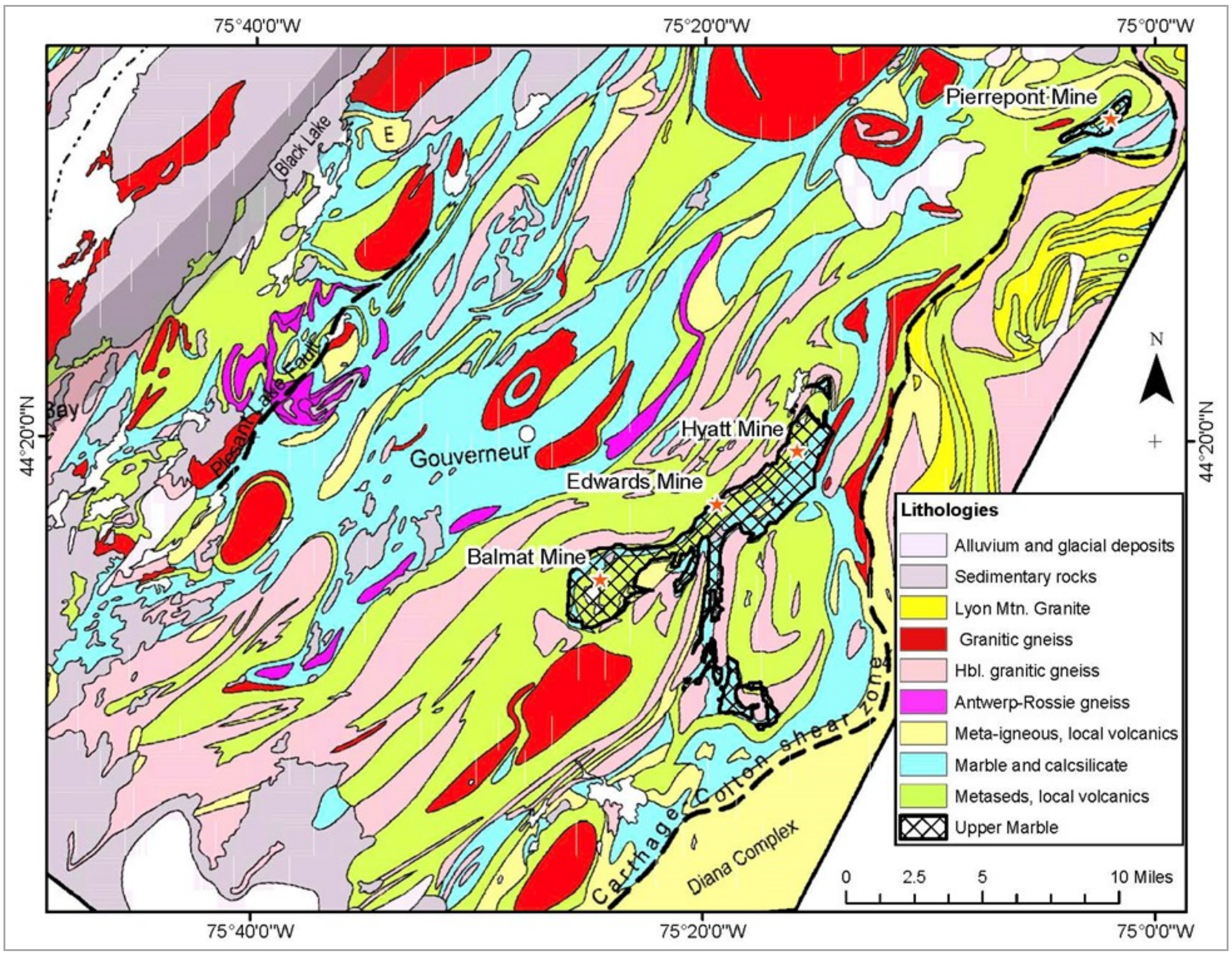


Source: SLZ (2017)

7.4 Property Geology

As a result of the intense tectonism in the ESM region's geologic history, the Upper Marble is extensively deformed. The predominant structure is the Sylvia Lake Syncline, a major southwest to northeast trending fold lying between the original Balmat mine and the Edwards mine. Aerial exposure of the Upper Marble Formation is limited, and the exposure generally trends along the axis of the syncline. Sphalerite (zinc sulphide) tends to occur within axial regions and limbs of local scale folds and faults associated with the Sylvia Lake Syncline. In Figure 7.3, the mapped surface expression of the Upper Marble Formation (hashed area) is shown superimposed on a geologic map of the Adirondack Lowlands. The locations of the zinc mines mark the axial trace of the Sylvia Lake Syncline.

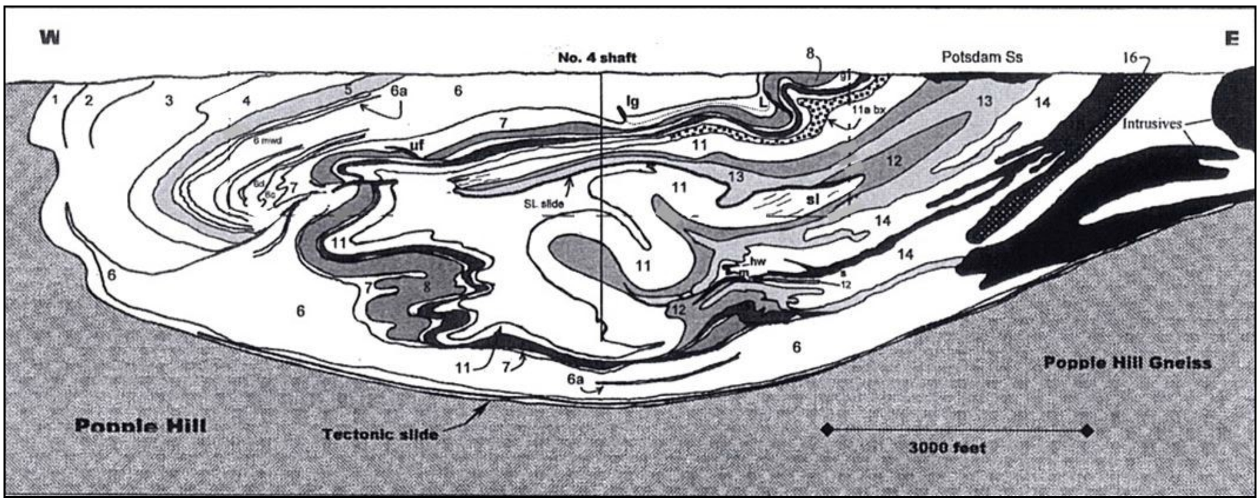
Figure 7.3: Local Geologic Setting



Source: SLZ (2017)

The zinc deposits at ESM are thought to have been syn-depositional, meaning they were deposited in sequence with the marbles that host them. Their original geometries would have been tabular as a result of being deposited on relatively flat areas of a sedimentary basin. Their current morphologies and positions are a response to ductile-brittle kinematic stresses. Extreme contrasts in ductility exist in the Upper Marble Formation, ranging from very ductile anhydrite and sulphide (sphalerite) beds to moderately ductile dolomitic marble to moderately brittle calcitic and serpentinous dolomitic marble to brittle silicious interlayered quartzite and diopside. Anhydrite and sulphide beds are relatively thin, and sulphide beds are spatially restricted, but their tendency to occur together consolidates ductile zones. When exposed to stress, the brittle rocks fractured, and the structures evolved into thrust faults in the ductile rocks. The thrust faults served to propagate folds. The tendency of folds to form in the most ductile regions caused the sphalerite to be concentrated in the noses of folds. The mine geologists have also suggested that sphalerite may have been remobilized towards the noses of folds during multiple episodes of metamorphism. Figure 7.4 is a cross section through the ESM area which illustrates the extent of deformation of the Upper Marble Formation.

Figure 7.4: Section through the No. 4 Shaft



Source: SLZ (2017)

7.5 Mineralization

Massive and semi-massive sphalerite-bearing deposits occur in siliceous dolomitic and evaporite-bearing marbles of the Upper Marble Formation of the Balmat-Edwards marble belt. These zinc-sulphide deposits lie in the core of the Sylvia Lake Syncline, a major poly-deformed fold lying between Balmat and Edwards. Zinc mineralization tends to follow evaporate deposition in the stratigraphic sequence. The region has experienced multiple metamorphic and intrusive events and large-scale ductile structures are common.

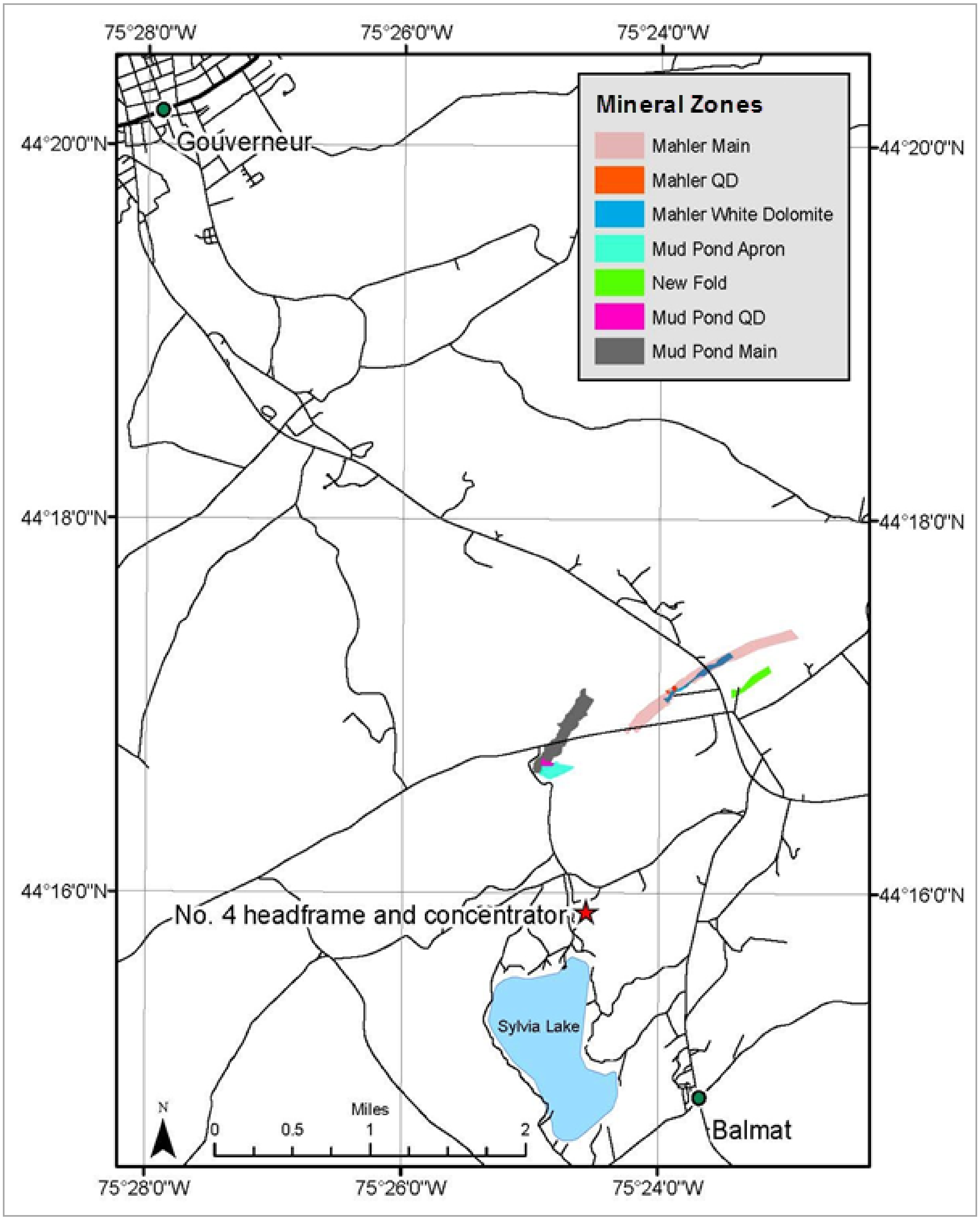
The ESM property contains 14 known zones of zinc mineralization. The deposits tend to occur in clusters. Three clusters have been defined consisting of three to five deposits each. Geometry of mineralization varies, ranging from tabular to podiform, shallow to steep. Areas defined to date contain tonnages ranging from roughly 0.5 Mt to over 10 Mt. Typical thickness ranges from two feet to 12 ft thick.

Mineralization tends to be very continuous along strike, ranging from 50 to 800 ft. Plunge-lengths may exceed 6,000 ft. Figure 7.5 shows the locations of zinc mineralized bodies currently being considered for future production.

Massive sphalerite-bearing zones are stratiform, and semi-massive zones are crosscutting and stratabound. ESM geologists conceptualize a parent-daughter relationship, where the stratiform mineralization is the parent and the crosscutting zone is the daughter. The parent-daughter model suggests that daughters are formed from sphalerite remobilized from parents during metamorphism. The sphalerite migrates along fault surfaces up and down dip from the parents, potentially as far as the Unit 10 anhydrite. It is thought that ductile flow of Unit 10 anhydrite closes fault surfaces and halts migration of remobilized sphalerite. Daughter zones share similar trace element geochemical signatures with their parent zones. They often contain significant quantities of occluded wall rock material. The geologists have experienced exploration success using the parent-daughter model, defining four new zones in the 1990's.

The mineralization at ESM has been classified as sedimentary exhalative (Sedex) in origin. The composition of the mineralization is unique, composed of primarily massive sphalerite and only minor galena and pyrite. The zinc-lead ratio is approximately 35:1. ESM has slightly higher-than-average grade for a sediment-hosted lead-zinc deposit. Typical grades of sediment-hosted lead-zinc deposits may average 7.9% Pb and Zn combined. The average grade was 8.6% Zn, while the average for the greater Balmat-Edwards zinc district is even higher at 9.4% Zn. Some ESM geologists have theorized that intense metamorphism may have concentrated the sphalerite, perhaps fractionating zinc sulphide (sphalerite) from lead and silver sulphide (galena) and remobilizing them to different locations leading to the high zinc grades observed at ESM.

Figure 7.5: Location of Zinc Mineralized Zones Review



Source: SLZ (2017)

8 Deposit Types

ESM deposits are broadly classified as sedimentary exhalative (Sedex) in origin, forming initially in a marine sequence of carbonates and evaporates. They were deeply buried, metamorphosed to amphibolite grade and strongly deformed during the late Precambrian Grenville Orogen.

8.1 Sedex Type Deposits

The term Sedex is derived from the words *sed*imentary and *ex*halative to denote sedimentary exhalative processes. Multiple theories have been suggested for the process of formation of Sedex deposits. In a 2009 USGS open-file report, Emsbo set forth a set of criteria for the assessment of sedimentary exhalative deposits based on available work. Characteristics of Sedex deposits were summarized based on empirical, physiochemical, geologic, and mass balance data. In brief summary, Emsbo's synthesis of Sedex deposit data indicates that the deposits are formed by the following processes:

Sedex deposits are formed in saltwater sedimentary basins within extensional tectonic domains. Large volumes of brine must migrate through the basin to generate Sedex deposits. The brines are generated by extensive and rapid seawater evaporation on large evaporative carbonate platforms. The brine is denser than sea water, so it sinks. It may infiltrate porous terrigenous basin fill sedimentary layers. As it migrates through the terrigenous sediments towards the lowest parts of the basin it leaches metals. Temperature increases as basin depth increases, so the brines heat up. When the brine encounters extensional fault surfaces it may migrate up the faults to the basin floor. Once exhaled into the basin, brines interact with the distal basin facies rocks, which are amenable to H_2S generation, which precipitates the metals as zinc and lead sulphide.

These processes as they relate to ESM are discussed below.

8.2 Sedimentary Basin: Carbonate Platform and Brine Generation

Sedex deposits are formed from brines generated by extensive and rapid seawater evaporation. Large evaporative carbonate platform areas are needed to produce the volumes of brine required to form Sedex deposits. Evaporation is rapid in low latitudes and brines are concentrated best in confined basins with restricted flow to the open ocean (Emsbo, 2009). These evaporative conditions are well recorded in the sedimentary record at ESM. The periodic anhydrite beds at ESM, as well as the dolomitization of the Upper Marble are indicative of evaporative conditions. A paleolatitude reconstruction by Cocks and Torsvik, places the area at a latitude conducive to rapid evaporation during the time of deposition (Cocks, L. Robin M. and Torsvik, Trond H., 2005). The rocks were deposited in the Trans-Adirondack back arc basin, an extensional environment with restricted flow to the open ocean. The carbonate platform represents the sedimentary basin's proximal facies (Chiarenzelli, Jeff, Kratzmann, David, Selleck, Bruce, deLorraine, William, 2015).

8.3 Sedimentary Basin: Rift-Fill Clastics and Supply of Metals

As brines are generated on the evaporative carbonate platform, they begin to sink due to their increased density. Sedimentary basins that host Sedex deposits characteristically have a thick layer of coarse clastic syn-rift oxidized terrigenous sediments underlying the evaporites in the sedimentary sequence. When the dense brines encounter this layer, the coarse permeable terrigenous sediments provide the fluid pathway for the dense brines to migrate laterally towards the lowest regions of the basin. The oxidized terrigenous sediments also provide the metal source for brines that form Sedex deposits. As the brines migrate, metals are scavenged and transported in the brine as chloride complexes. Oxidized syn-rift sediments buffer mineralized material fluids to compositions amenable to metal scavenging because they are low in organic carbon and high in reactive iron (Emsbo, 2009.)

Mass balance studies indicate that large volumes (thousands of km^3) of clastic sediments are required to generate enough metals to form a Sedex deposit. Fluid inclusion studies indicate that Sedex deposits are formed from brines with temperatures between 100-200°C. Metals are most soluble in this temperature range. Brines increase in temperature as they migrate because basin temperature increases with depth. Sedimentary fill in the basin must reach at least 3 km depth to generate the required temperatures (Ibid) At ESM, the clastic sequence may be represented in the Popple Hill Gneiss, which underlies the Upper Marble Formation. The Lower Marble Formation, which underlies the Popple Hill Gneiss, also includes some clastic members. The original extent and thickness of the clastics is difficult to determine because the Grenville Supergroup is allocthonous; the rocks have been thrust out of depositional position and extensively deformed.

8.4 Tectonic and Sedimentary Structure

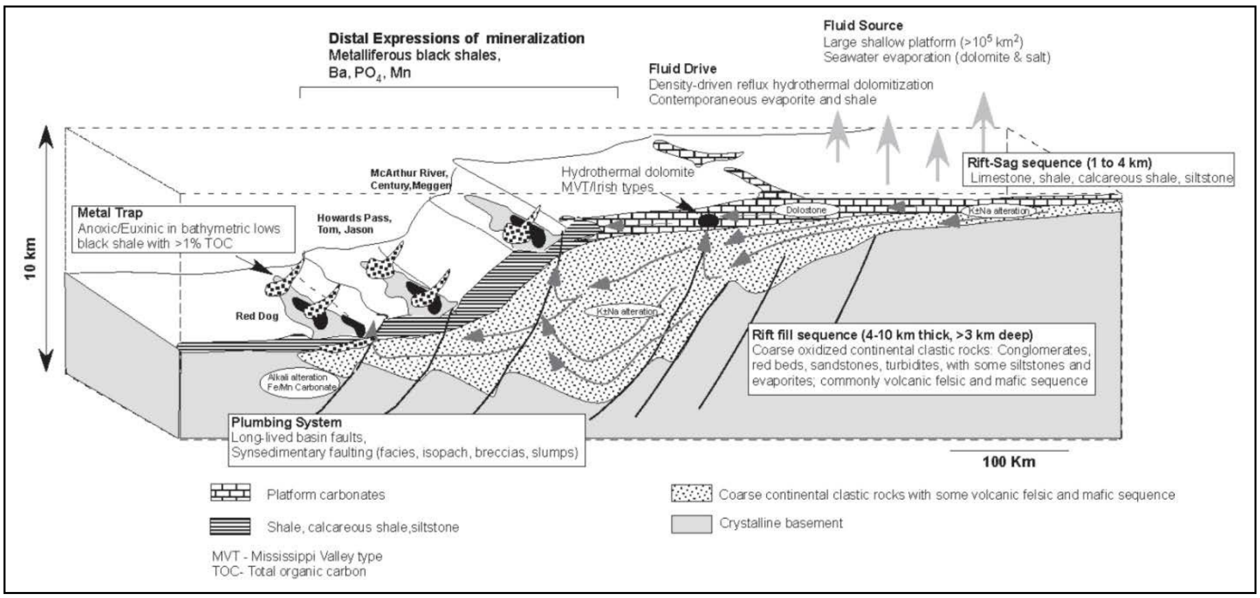
Warm, metal-laden migrating brines may eventually encounter extensional fault surfaces and migrate up the faults to the basin floor. Workers describing sedimentary basins have divided the basins into three orders of scale. First-order sedimentary basins which host Sedex deposits are greater than 100 km in length. Within the basin, second-order basins occur on the scale of tens of kilometres. Second-order basins are controlled by extensional faults forming half grabens in the basin. The Sedex model suggests that brines migrate up these faults. Some indicators of second-order basin bounding faults include syn-sedimentary faulting (evidenced as abrupt platform- slope facies transition) and intraformational breccias. Faults that were fluid conduits may be identified by Fe and Mn alteration and/or silicification, and sometimes tourmalinization. Third- order basins, on the scale of a few kilometres, represent bathymetric lows. Sedex deposits typically occur in third-order basinal areas within a few to tens of kilometres of second-order faults. Some indicators of bathymetric lows, where metals are likely to be deposited, include increasing debris flow thickness and increasing organic matter and pyrite concentrations in reduced sediments representing distal basin facies. At ESM, intense metamorphism has obliterated the more subtle sedimentary features that characterize Sedex deposits, and post- depositional deformation has overprinted tectonic features.

8.5 Deposition of Sulphides

Dense brines exhaled onto the basin floor tend to pool in bathymetric lows. These lows occur in deeper distal basin facies, which tend to be anoxic. The distal facies is typically represented by fine-grained clastic sedimentary rocks like shale. Sedex-hosting shales are unusually high in organic matter. The reducing conditions of third-order basins preserve organic matter. Hydrogen sulphide (H_2S) is generated in this depositional environment by bacterial sulphate reduction. Bacteria living in the highly carbonaceous distal sediments or thermal vents oxidize the organic compounds in the shale while reducing sulphate (SO_4^{2-}) from sea water to generate H_2S. The H_2S reacts with the pooled brines and precipitates the contained metals as zinc sulphide (sphalerite, (Zn,Fe)S))and lead sulphide (galena, (PbS)). Another possible mode of generation of H_2S is by thermogenic reduction of organic matter. The ESM deposits occur in proximal facies rocks as opposed to third-order basin distal facies rocks, which is at variance with the Sedex model. The Upper Marble does contain a pyritic schist unit underlying the marble units that contain zinc deposits.

Sedex deposit formation may be limited to Proterozoic and Phanerozoic time since marine sulphate (SO_4^{2-}) likely did not exist prior to the accumulation of oxygen in the atmosphere. ESM was deposited within this timeframe. Sedex deposits may correspond with regional and global anoxic events, which would have helped preserve higher concentrations of organic carbon during transport to anoxic distal basin facies.

Figure 8.1: Illustration of the Process of Formation of Sedex Deposits



Source: Emsbo (2009)

9 Exploration

Regional zinc exploration in the Balmat-Edwards marble belt, as well as the northwest Adirondacks was carried out almost exclusively by St. Joe Minerals since the 1960's. Despite the fact that no systematic regional exploration work was carried out since 1986, five new mineralized bodies were discovered in the district within the last 25 years (three in the Balmat mine and two in the Hyatt mine).

Resource potential of the Balmat-Edwards district is divided into three categories: Balmat mine, Balmat-Edwards segment and district wide. In the last 19 years alone, including 12 years of curtailed production, three new mineralized deposits (New Fold, Mahler and NE Fowler) were discovered in the Balmat mine.

Past exploration successes indicate that it is possible that several zones remain to be discovered in the Balmat mine, the Balmat-Edwards segment and throughout the district. The implementation of the new exploration model will greatly increase the likelihood of discovery of new mineral resources in the district.

More recent exploration activity included a 21,000 ft diamond drilling program in 2005 by Hudbay along with 435 ft of exploration drifting. This program was aimed at upgrading approximately 400,000 tons of Inferred resource to an indicated classification (Hudbay, 2005).

The zones have been primarily developed during 'in-mineralization' ramping which provides poor access and drill angle for infill and exploration drilling. A lack of exploration budget compounded this issue and resulted in a wide spaced delineation of the resource, misinterpretations of localized geometry and high mine dilution rates (Hudbay, 2010). Future mine development techniques are currently planned to include ramp development along the footwall of the resource, which will provide opportunity for localized infill drilling programs to correct past issues.

In 2008, Geotech Ltd of Aurora, Ontario flew a helicopter borne VTEM (versatile time domain electromagnetic) geophysical survey over the Adirondack Lowlands of northern New York on behalf of Hudbay Minerals. The survey area covered a nominally rectangular area of 58 by 30 mi, including the greater Balmat mining district.

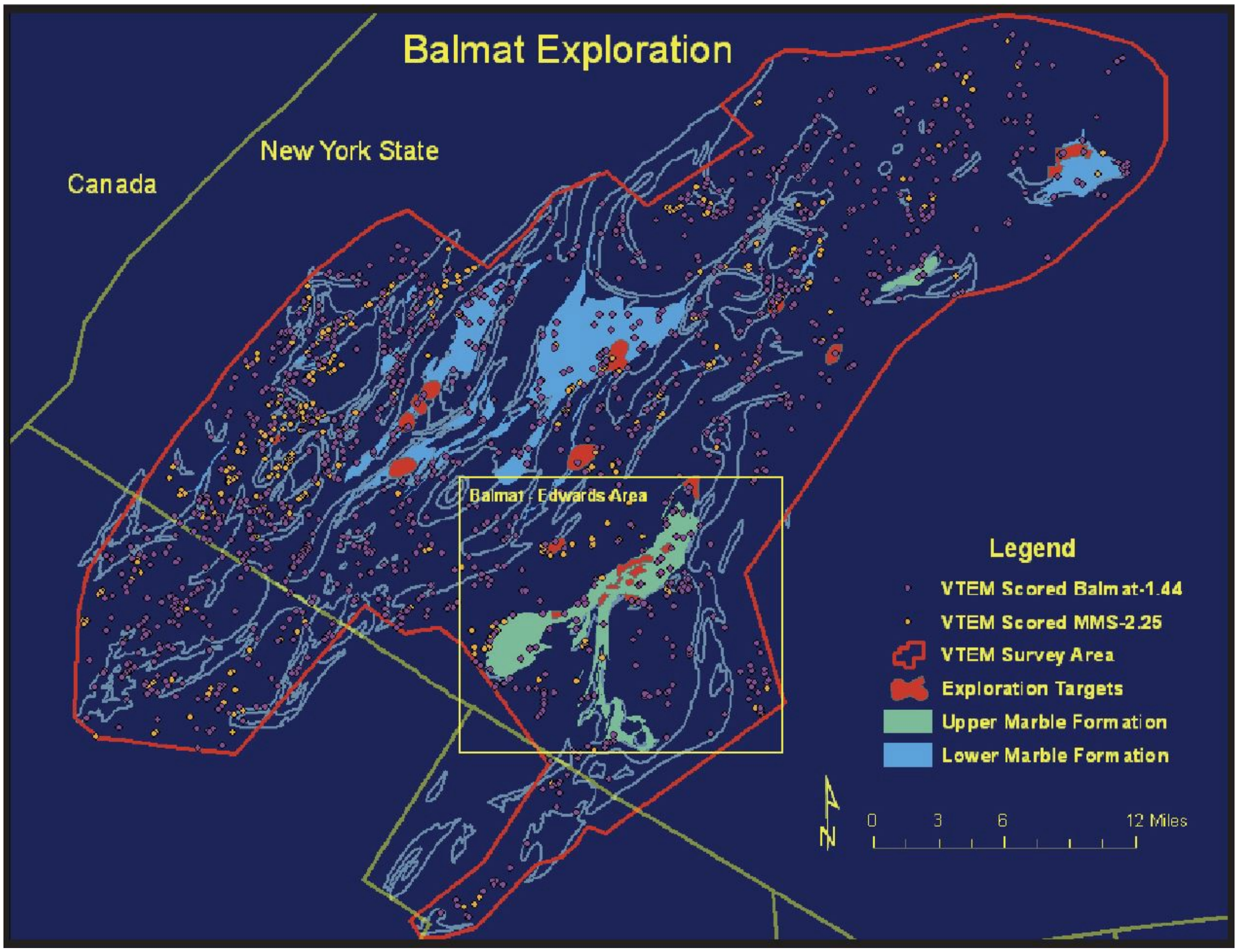
Flight lines were flown on 650-foot line spacing. The geophysical database was forwarded to the geological department at ESM for interpretation and anomaly ranking based on correlation of observed physical parameters and deposit characteristics. The interpretative team determined that linear anomalies parallel regional structural fabrics and trends, known pyrite-rich stratigraphic units were readily detected and that anomalies in massive carbonate sequences are, at best, weakly responsive.

The interpretative team also defined the basic ranking criteria to be based on anomalies of "Balmat deposit type mineralization body" sized lengths over two or three parallel flight lines. The anomalies themselves should reflect known geological characteristics, meaning those in areas of carbonate and calc-silicate host rocks should not be as responsive as those in pyrite bearing or graphitic sequences. A series of high quality targets were delineated within and around the district, and additional targets are being developed in conjunction with historical data synthesis.

Two areas are present within the Balmat district but outside of the existing mine footprint and eight areas lie within the existing mine footprint. Figure 9.1 shows the area covered by the geophysical survey areas with results processed to represent two target types identifed in the district; Balmat style and metamorphic massive sulfide (MMS) style deposits. The former is characterized by dominantly sphalerite with minor accessory sulfides (pyrite, pyrrhotite, galena) while the latter contains a much a higher ratio of these sulfides relative to sphalerite.

Star Mountain did not conduct any exploration work on the property during ownership. Titan Mining Corp commenced a surface exploration drilling program at ESM in February 2017 that is in progress as of the effective date of this report (Drilling contractor field activities temporarily paused June 30th).

Figure 9.1: Geophysical Survey Area



Source: SLZ (2017)

10 Drilling

Drilling at ESM has been exclusively core drilling. The mine owns a Diamec 262 underground drill using AW-34 size core. Three contract Longyear underground drills that use BQ size core were utilized during the period after 2005. The drillhole database contains 4,317 drill holes completed at various times in the project's history within the Balmat and Edwards areas. Of these holes, a total of 1,605 were drilled from surface and the remaining 2,712 were underground. Most of the holes are peripheral to the current project area. The mineral inventory estimate was calculated using assay values from 633 holes.

According to ESM geologists, core was handled in the following manner by the mine geology department during the most recent phase of production. Core was removed from the drill string by the driller and placed in a wooden core box. Wooden blocks were used to mark the ends of individual core runs. The geologist then logged the core and selected and marked the intervals to be prepared for assay samples.

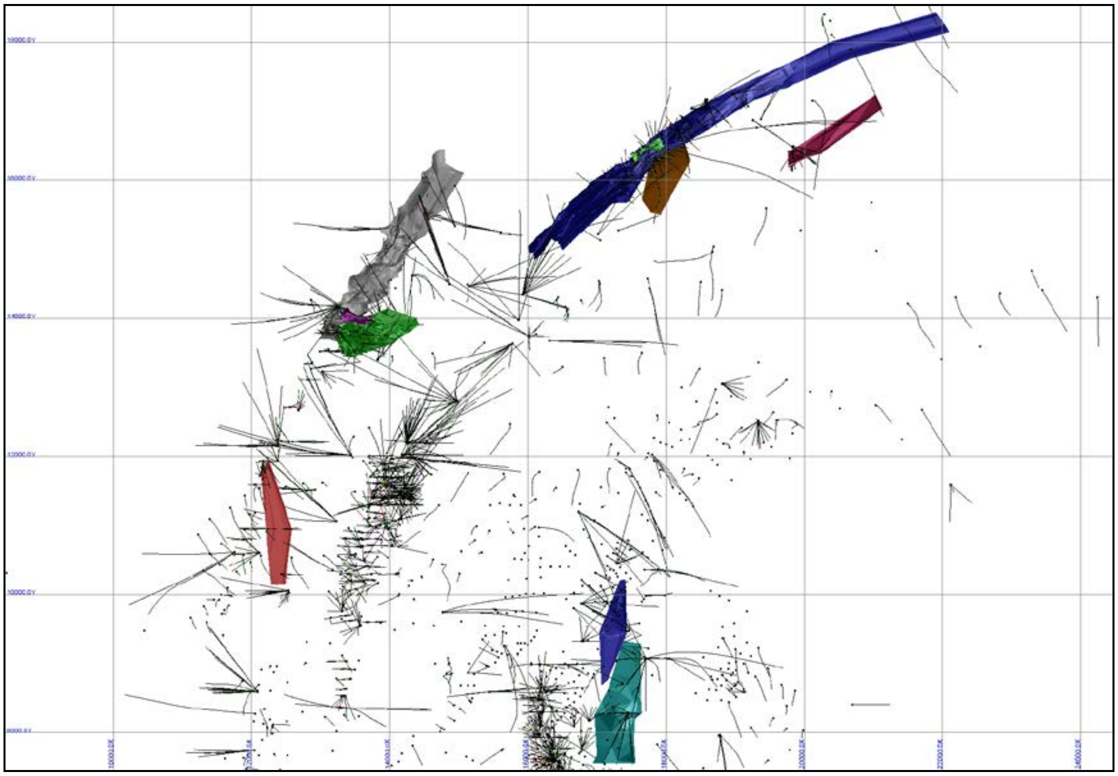
The core was then transported to the surface where the marked assay samples were split. One half split was returned to the core box, the other half split was sent to the assay laboratory.

The geology logs of the drill holes and the assay results are archived as hard copy and entered into a digital database.

Drilling conditions in the Upper Marble Formation are generally very good, and core recovery is typically excellent. Zinc mineralization is visible, and sample intervals are chosen by trained geological staff. Samples are analyzed by a reputable independent assay laboratory.

The authors are not aware of any issues that would negatively impact the accuracy and reliability of drill sample results at ESM although the high variability in sample lengths and not sampled (NS) within the mineralized zones needs to be reviewed for future work.

Figure 10.1: Map Showing the Distribution of Drilling



Source: Tuun (2017)

When the mine was shut down in 2008, due to low zinc prices, significant mineralization had been defined by Hudbay. A current mine plan has been prepared for the next phase of mining based on existing drill data. Delineation and exploration drilling could resume from underground drill platforms after the mine resumes production.

10.1 Drilling Summary

A total of 4,317 diamond drill holes have been completed historically, totalling 2,561,297 ft, as shown in Table 10.1

Table 10.1: Project Drilling by Year

Year	No. Holes	Footage Drilled
Pre-2000	3,811	2,366,540
2000	34	23,684
2001	12	3,539
2004	5	3,143
2005	98	47,312
2006	126	43,907
2007	82	32,165
2008	143	37,438
2009	6	3,567
TOTAL	**4,317**	**2,561,297**

Source: SLZ (2017)

10.2 2017 ESM Drilling Program – In Progress

The Empires State Mines' 2017 drilling exploration program is not yet complete with surface and underground drilling temporarily paused in June and July respectively. Initial assay results have been received for most drill holes, these results have been excluded from the Mineral Resource estimation in this report pending further verification. Final QA-QC checks will be completed at the conclusion of the program when 5% of the samples will be submitted to a second laboratory.

Diamond core drilling contractor Longyear mobilized on site and surface drilling commenced on February 12, 2017. Surface drilling was conducted at one regional exploration target (Sully: one hole) and two mine site exploration targets (Mud Pond Apron Extension: two holes; and Mud Pond Upper Extension: six holes). Nine holes totaling 16,071 ft of surface drilling were completed as of June 30, 2017. Surface drilling was paused at the end of June 2017 and approximately 50,000 ft remain to be drilled from the original contract. Additional drill programs have been designed for the Sully exploration target and the Gap Zone, both of which are situated between the Empire State Mine and Hyatt Mine along the general trend of known zinc mineralization.

Drilling at the Sully exploration target did not encounter significant mineralization, but did establish continuity of the controlling shear zone with minor zinc mineralization. Drilling in the Mud Pond Apron Extension confirmed the continuation of zinc mineralization between the end of the Mineral Resource shell and the down-plunge historic drill hole DD1097-F (11' at 13.4% Zn). Drilling in the Mud Pond Upper Extension zone also confirmed the continuation of zinc mineralization between historic drill holes.

Other ongoing exploration activities include a systematic review of historic exploration which is focused on digitization and interpretation of previous work. The goal is to identify regional targets that warrant follow up and generate new targets by integrating the various data types (geology, drilling, geochemistry and geophysics). The design for an airborne gravity gradiometry survey has been proposed with the aim of directly detecting large and high grade zinc deposits, and is anticipated to commence in November 2017. Lease renewal and payment activities are ongoing and new lease agreements for expired leases and options are expected to be negotiated and entered into with the relevant owners in 2017.

Major Drilling International Inc. was awarded a 40,000 ft underground drilling program contract, principally for upgrading of Inferred Mineral Resources to Indicated Mineral Resources with an allowance for delineation of targets developed from the surface drilling.

Underground drilling totaling 9,099 ft in 16 drill holes was completed by Major Drilling International Inc. between May 24, 2017 and July 9, 2017. Two underground drill locations were used to target Mineral Resource infill drilling to upgrade from Inferred to Indicated Mineral Resources at the Mud Pond Apron Extension and Mahler zones. Lateral extensions at the Mud Pond Apron Extension were also tested. Underground drilling is planned to resume in October 2017 with 30,901 ft remaining on the 40,000 ft contract with Major Drilling International Inc. The remaining drilling from existing underground drill locations will target further upgrading of the Mahler Mineral Resource from Inferred to Indicated Mineral Resource, the Mahler zone up dip and the Mud Pond Apron Extension zone down dip.

Two underground drill holes at Mud Pond Apron Extension targeted infilling and further defining the current Mineral Resource – DDH 2208-F & 2209-F, with 2208-F not intersecting the mineralized target and 2209-F intersecting similar to expected. Eight drill holes, 2206-F to 2207-F & 2210-F to 2216-F, targeted lateral extension of the Mud Pond Apron and all intersected the target horizon, but with narrow widths and moderate to low zinc grades. Four drill holes, DDH 2217-F to 2220-F, targeted infilling and upgrading of the Inferred Mineral Resource at the Mahler zone. All holes intersected the target horizon with the results received for holes 2217-F and 2218-F confirming the higher grades present in the Mahler zone. Drill hole 2218-F intercepted the nose of a fold so the true width of this intersect is difficult to calculate.

Table 10.1: Surface Drilling Program Significant Intersections

Hole No.	Target	From (ft)	To (ft)	Interval (ft)	True width (ft)	Zn assay
2429	Sully	no significant results				
2430	MP Apron Ext	2,759.40	2,766	6.6	6.3	8.96
2431	MP Apron Ext	2,794.20	2,797	2.4	2.3	11.65
2432	MP Upper Ext	963.50	966	2.9	2.9	10.15
2432	*plus*	1,226.20	1,230	3.6	3.6	5.83
2433	MP Upper Ext	abandoned				
2434	MP Upper Ext	1,276.60	1,281	4.2	4.1	3.88
2434	*plus*	1,287.80	1,291	3.2	3.1	9.75
2434	*plus*	1,371.60	1,374	2.0	2.0	7.13
2435	MP Upper Ext	1,052.90	1,054	1.0	0.9	12.5
2435	MP Upper Ext	1,092.10	1,094	2.1	1.9	18.66
2435	*Inc.*	1,092.10	1,093	0.7	0.6	42.3
2435	MP Upper Ext	1,354.30	1,355	0.7	0.7	12.25
2436	MP Upper Ext	1,371.00	1,378	7.4	7.0	8.14
2436	MP Upper Ext	1,390.00	1,392	2.0	1.9	10.9

2437	MP Upper Ext	883.20	883	0.2	0.2	9.44
2437	*plus*	929.70	931	1.2	1.2	7.95
2437	*plus*	1,182.10	1,186	4.0	4.0	9.32
2437	*Inc.*	1,183.95	1,185	1.1	1.1	21.6
2437	*plus*	1,246.00	1,246	0.3	0.2	19.25

Table 10.2: Underground Drilling Program Significant Intersections

Hole No.	Target	From (ft)	To (ft)	Interval (ft)	True width (ft)	Zn assay
2205-F	MP Apron Ext	abandoned				
2206-F	MP Apron Ext	446.40	449	2.1	1.3	1.84
2207-F	MP Apron Ext	no significant results				
2208-F	MP Apron	363.20	367	3.4	na	1.64
2209-F	MP Apron	210.20	220	10.2	9.8	14.46
2209-F	MP Apron Ext	236.00	239	2.7	2.6	16.7
2210-F	MP Apron Ext	301.00	304	2.9	2.1	12.4
2211-F	MP Apron Ext	441.50	443	1.7	na	4.82
2212-F	MP Apron Ext	422.00	423	0.6	0.2	4.51
2212-F	MP Apron Ext	424.90	427	1.8	0.7	5.22
2213-F	MP Apron Ext	337.40	340	2.4	1.5	6.5
2214-F	MP Apron Ext	240.50	242	1.4	1.2	7.77
2214-F	*plus*	257.80	261	3.5	3.0	11.44
2215-F	MP Apron Ext	411.20	412	0.8	0.5	3.25
2216-F	MP Apron Ext	447.80	448	0.5	0.3	3.25
2217-F	Upper Mahler	529.20	532	2.9	2.6	11.3
2217-F	*plus*	539.00	544	4.7	4.2	8.68
2217-F	*plus*	550.10	557	7.3	6.5	19.66
2217-F	*plus*	578.10	589	11.3	10.1	10.78
2218-F	Upper Mahler	599.55	635	35.7	17.9?	22.51
2218-F	*plus*	692.00	735	42.5	21.3?	19.18
2218-F	*Inc.*	724.15	735	10.4	9.3	37.21
2219-F	Upper Mahler	Assay Results Outstanding				
2220-F	Upper Mahler	Assay Results Outstanding				

11 Sample Preparation, Analyses and Security

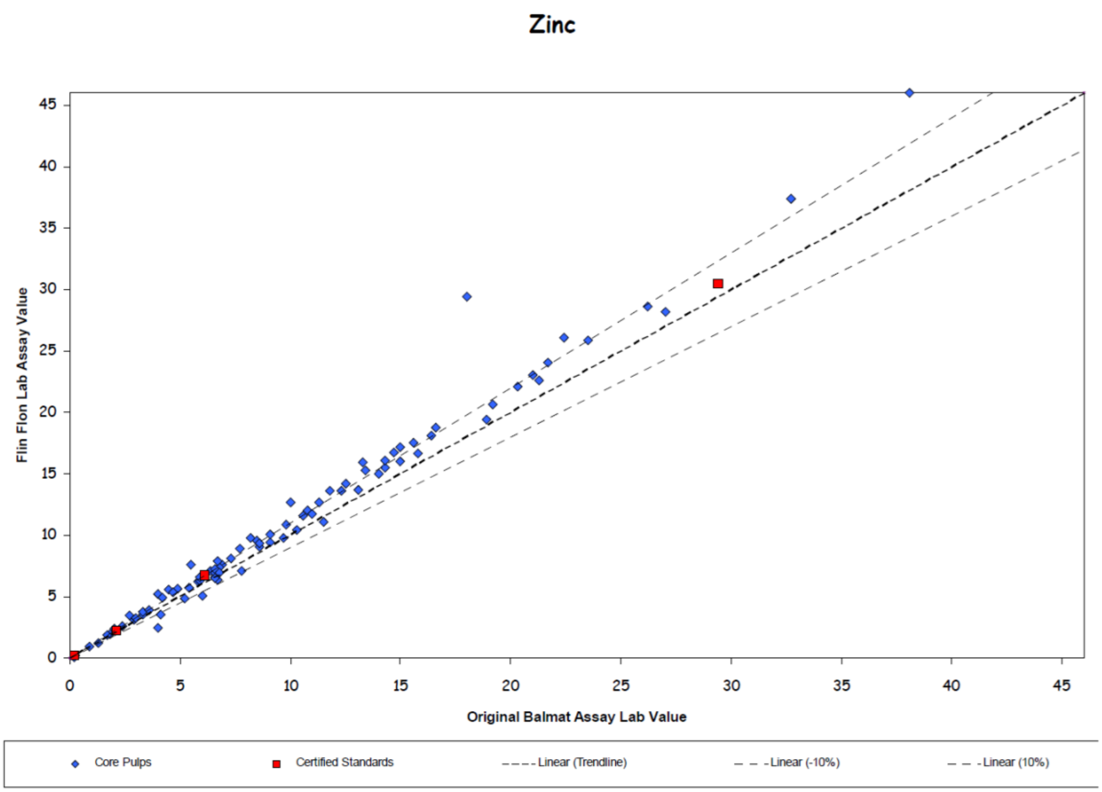
11.1 Historical Sampling

Prior to the 2003 acquisition of the ESM by Hudbay Minerals, all assaying was performed at the ESM assay laboratory. Fine pulps from the core drilled between the years 1995 and 2000 were stored at the ESM #2 core facility. Pulps were marked with drill hole identification and assay interval.

Assays from these years were not supported by a defined quality assurance/quality control protocol. Hudbay Minerals selected 86 fine pulps from this population, representing six ESM resource areas to test for analytical integrity for the 1995 to 2000 drilling. The pulps were packaged inside 5-gallon buckets along with four certified reference standard samples and shipped to Hudbay's Flin Flon, Manitoba assay laboratory for check analyses. The Flin Flon laboratory visually inspected each pulp to assess oxidation and preparation effectiveness with particular attention paid to sample size grading. Zinc assays were completed for each sample.

The Flin Flon laboratory reported consistently higher results than those obtained by the ESM lab. The certified reference standards were all within acceptable limits.

Figure 11.1: Hudbay Flin Flon Lab Check Assays of ESM 1995-2000 Pulps



Source: SLZ (2017)

11.2 Sampling Post-2005

All drill hole core samples from the 2005 to 2010 diamond drilling were sent to the ALS Chemex lab in Mississauga, Ontario). The QAQC program initiated by Hudson Bay Mining and Smelting Co., Limited (HBMS) was followed using the protocol:

Blank samples, consisting of material barren of any visible sulphides were inserted into the sample stream before being sent for assay. Every 50th sample the core loggers send to the assay lab is a blank sample from the above material. The sample is consistently placed every 50th, regardless of the type of material sampled previous to the blank one.

The blank samples are considered barren having undetectable limits for base metals. If assay results on the blanks are above three times the detection limit, the assumption is that there has been contamination at the sample preparations stage (primary crusher) due to improper cleaning of equipment between samples. These procedures were not strictly followed with a limited number of blank samples submitted to the laboratory from different sources and assumed by the geologists to be free of zinc mineralization.

This has proven to not be the case, all results are greater than three times the detection limit of 0.01% Zn. Further work with blank samples from common source than can be proven to be free of zinc mineralization is recommended in the future.

Table 11.1: Blanks (Barren of Zinc) Submitted to Chemex for Assay

Sample ID	Zn %	Zn ppm	Certificate	Date	Description
BAL-AN-1	0.06	462	TO05031733	2005-05-04	ANHYDRITE ROCK CHUNK
BA-DOL-2	0.34	NS	TO05045120	2005-06-20	LT GY DOL, UNIT #4 SURF BY ENTRANCE
BAL-QD-3	0.98	9470	SD05099834	2005-11-30	QTZ-DIOP BLANK FROM UNIT 4,NFCONTR DRIFT
BAL-DOL-4	0.77	6450	SD06000742	2006-01-14	
BAL-QD-4	0.08	NS	SD06050196	2006-06-22	ROCK CHUNK (BLANK)
BAL-QD-5	0.45	NS	SD06118074	2007-01-05	
BAL-DOL-6	0.28	NS	SD07030823	2007-04-09	
BAL-DOL-7	NS	662	SD08048097	2008-05-14	

Source: SLZ (2017)

Insertion of Certified Standards Internationally certified samples of known grades were prepared and purchased from Ore Research and Exploration Pty Ltd. (an Australian company) by HBMS in 2004. HBMS supplied five different grades of material (grab samples) from the mines in the Flin Flon camp that represented at least 90% of the grades encountered at the mines. Ore Research crushed the samples then calculated the expected grades based on the average of assay results from eight independent lab analyses. Standards are the most important QAQC samples because their expected assay value is known (therefore all subsequent assay results should be very close to this average of eight results value).

Table 11.2: Flin Flon Mine QAQC Certified Standards Supplied by HBMS

	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Fe (%)	As (%)
STANDARD A-4	0.225	4.1	0.423	0.219	0.03	9.24	0.02
STANDARD B-4	0.838	11.9	1.02	2.12	0.09	15.06	0.03
STANDARD C-4	3.16	19.2	4.5	6.11	0.1	22.2	0.05
STANDARD E-4	0.746	12.7	1.17	29.4	0.56	20.6	0.1

Source: SLZ (2017)

All standards come finely crushed in foil packages clearly labeled with the standard type (A-4, B-4, C-4, or E-4).

Although these certified standards were prepared for HBMS's specific requirements, the standards were inserted into the mainstream of samples at Balmat as a QAQC check on the Chemex lab's assay results.

In 2008, two new standards were prepared by Ore Research & Exploration Pty Ltd specifically using sulphide reference material form the Balmat mine: Standards (G-5 & H-5). The standards were certified with round robin assaying at 15 laboratories.

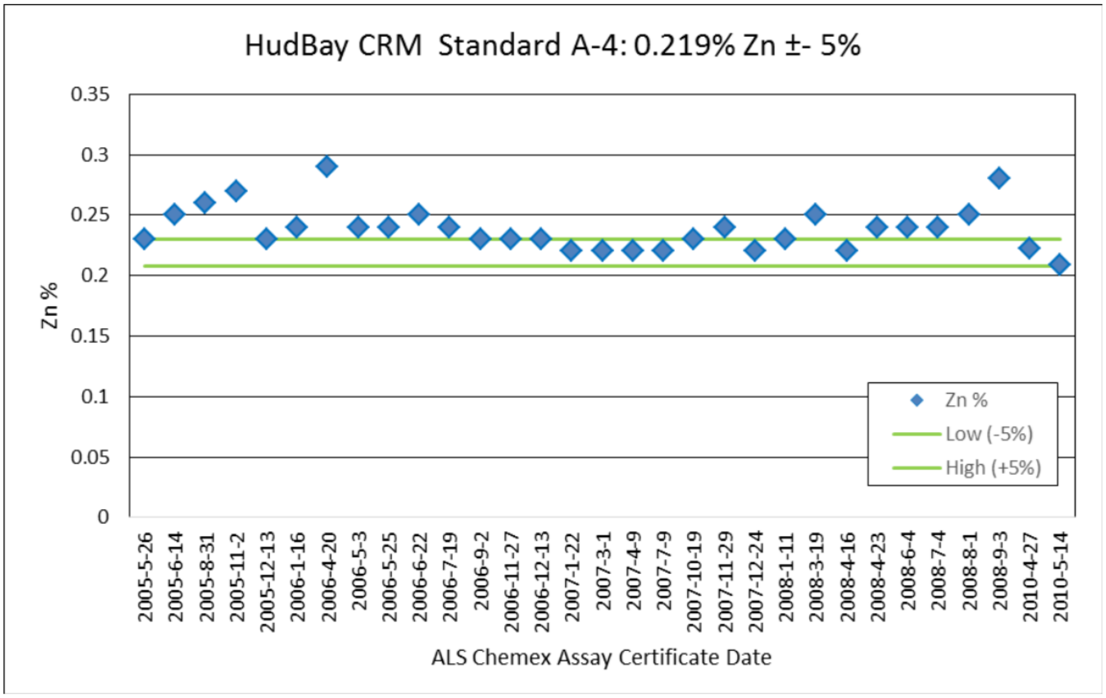
Table 11.3: ESM QAQC Certified Standards Supplied by Ore Research & Exploration Pty Ltd June 2008

	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Fe (%)	As (%)
STANDARD G-5	0.097	3.50	0.060	9.97	0.076	1.49	0.009
STANDARD H-5	0.038	3.81	0.043	22.9	0.075	1.59	0.004

Source: SLZ (2017)

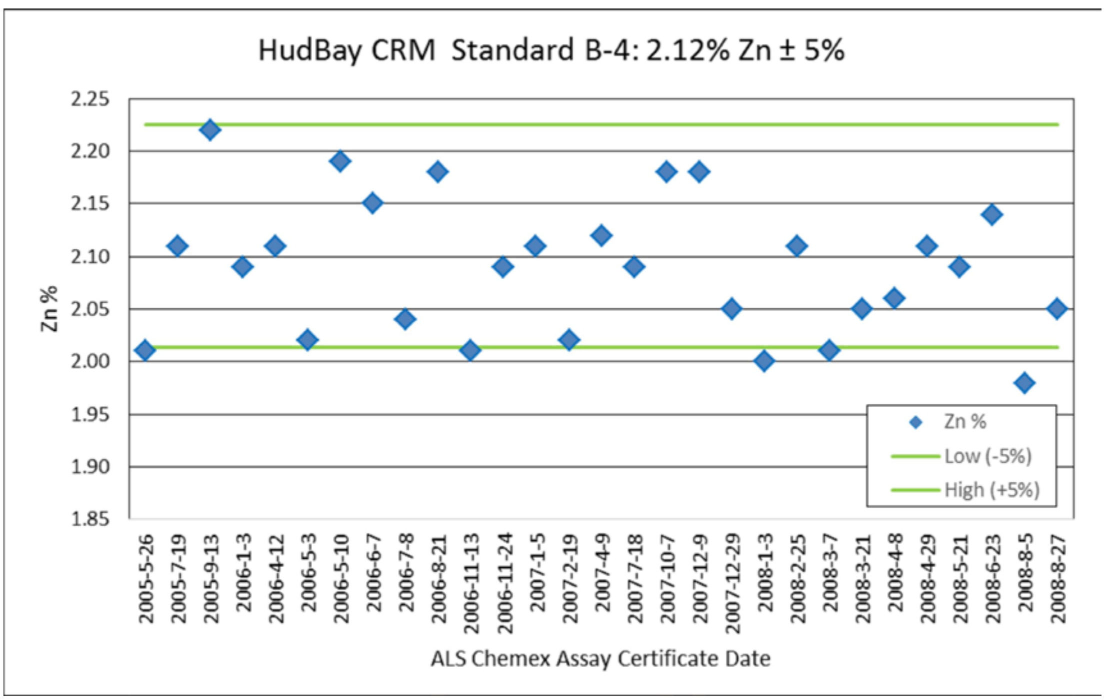
The core loggers insert one certified standard per 20th sample.

Figure 11.2: Hudbay CRM Standard A-4



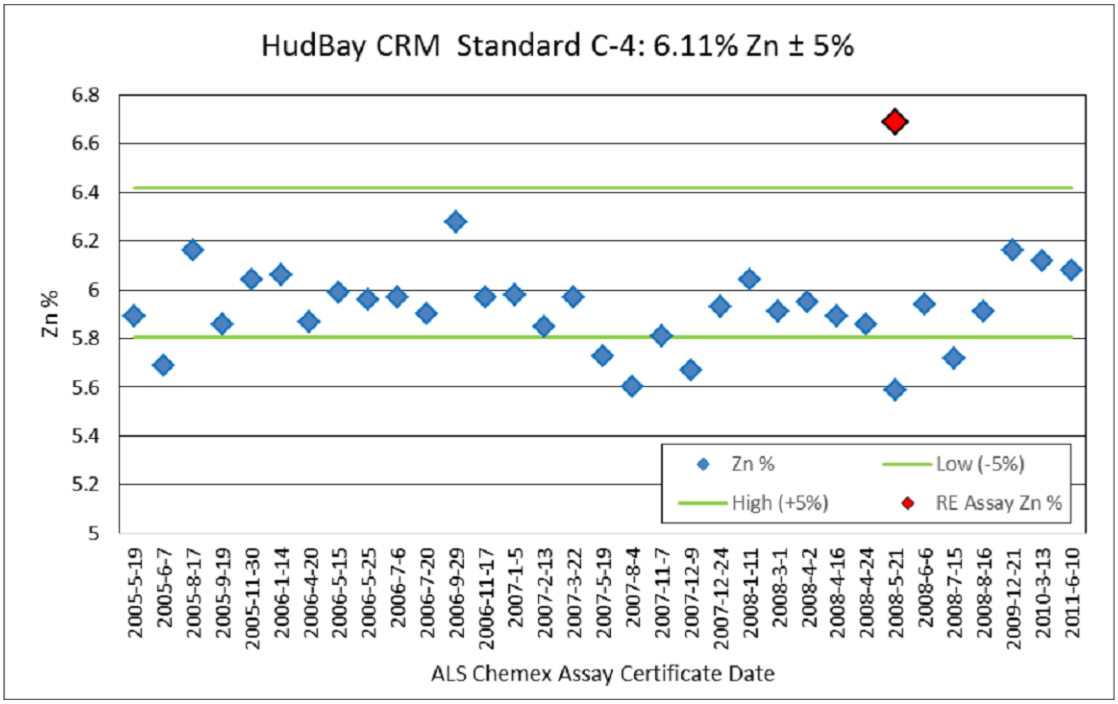
Source: SLZ (2017)

Figure 11.3: Hudbay CRM Standard B-4



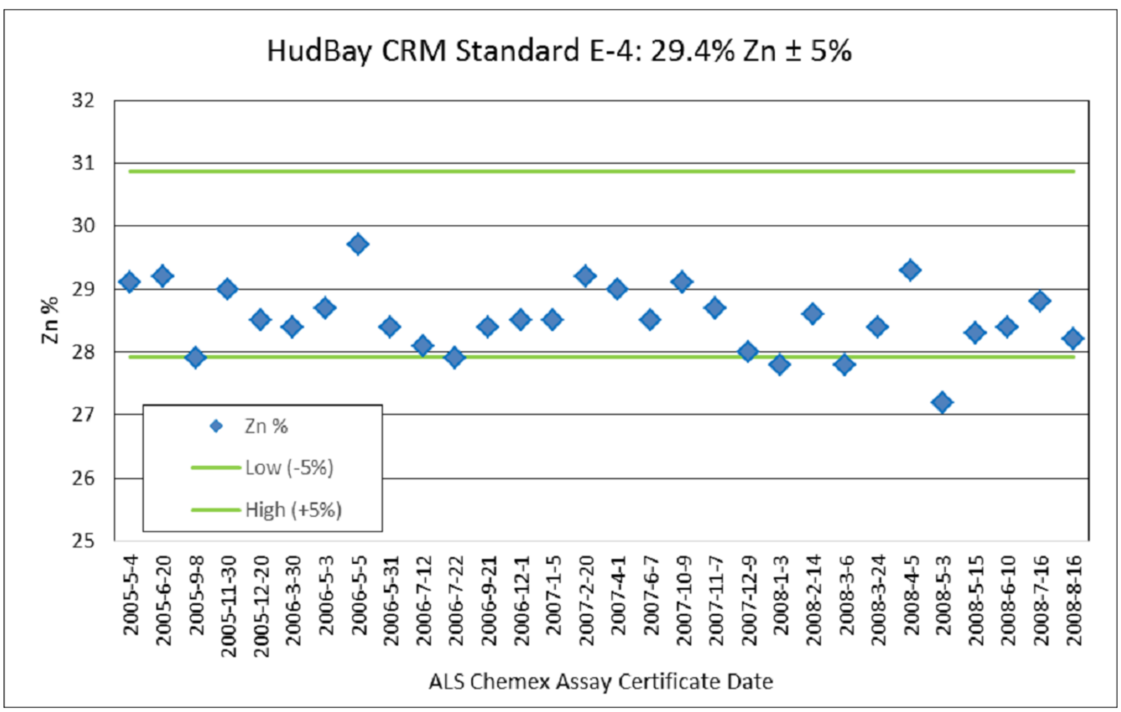
Source: SLZ (2017)

Figure 11.4: Hudbay CRM Standard C-4



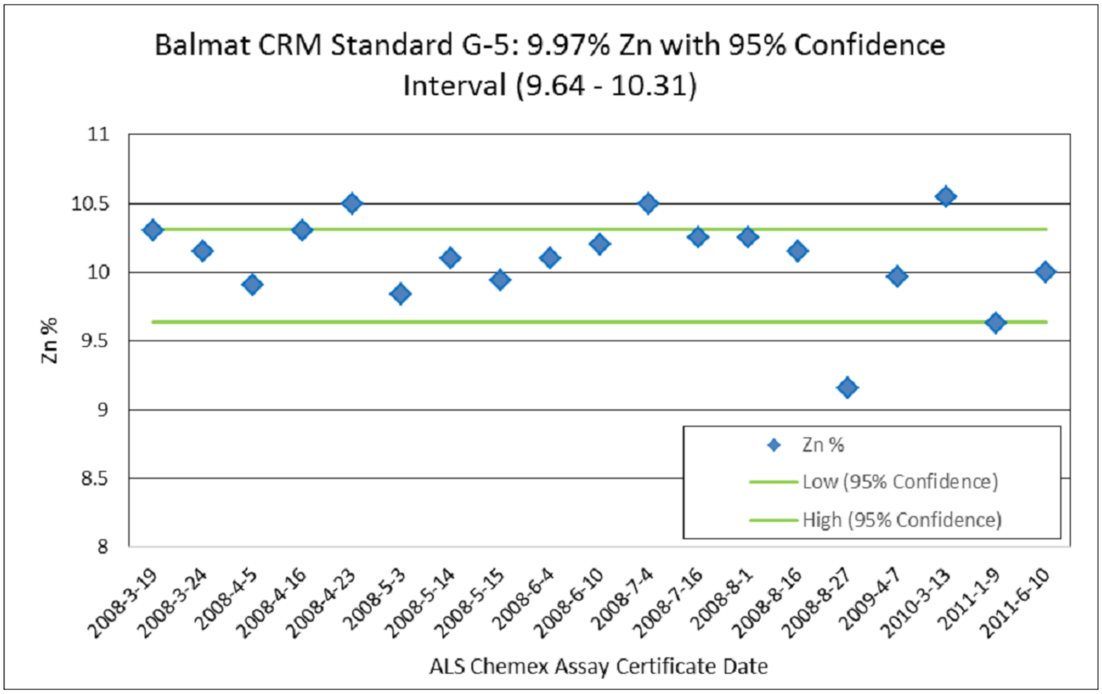
Source: SLZ (2017)

Figure 11.5: Hudbay CRM Standard E-4



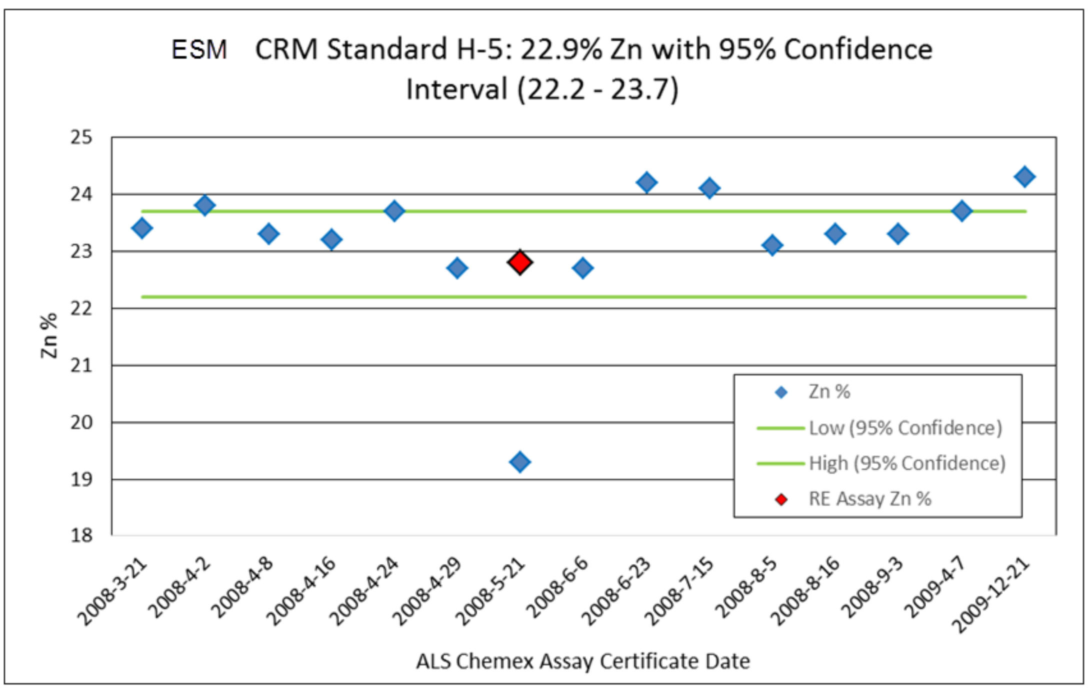
Source: SLZ (2017)

Figure 11.6: Balmat Standard G-5



Source: SLZ (2017)

Figure 11.7: ESM Standard H-5



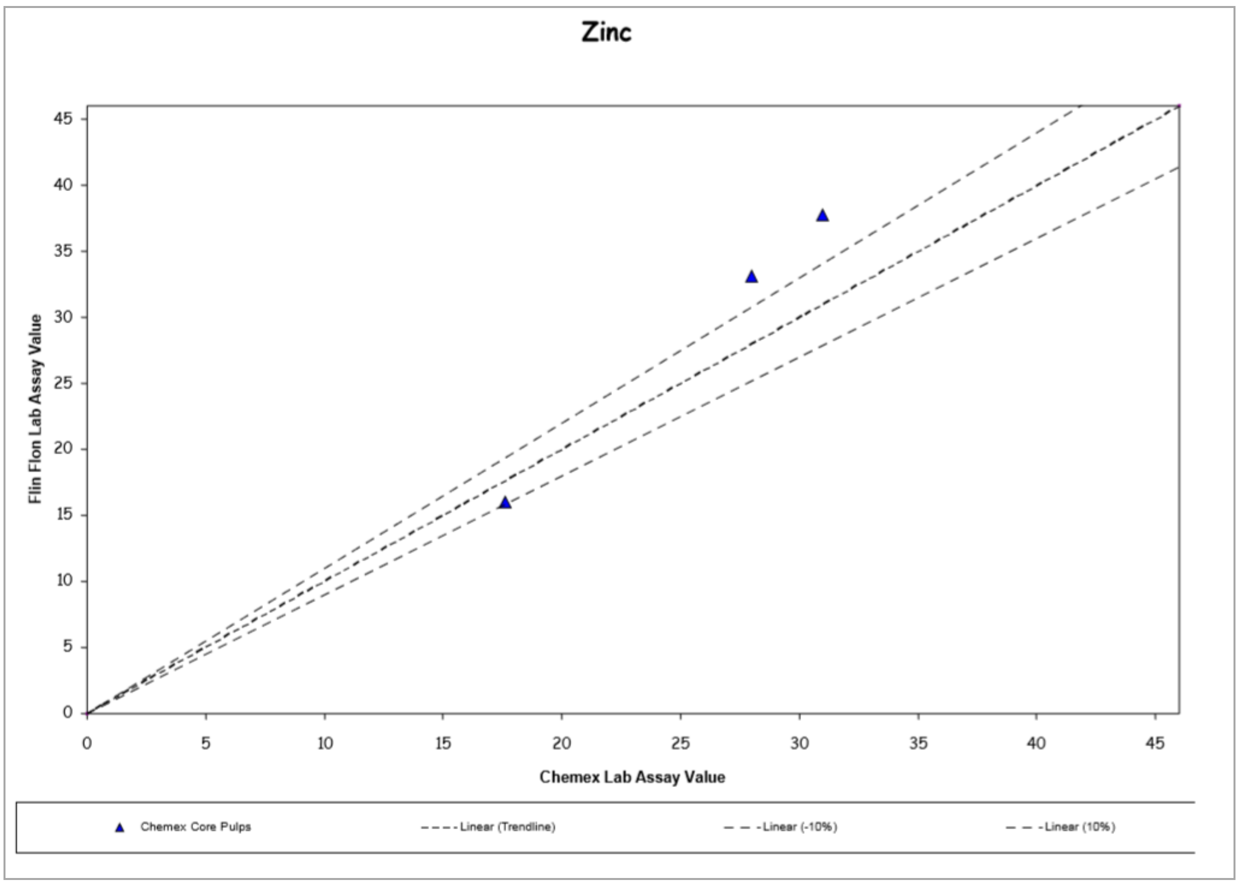
Source: SLZ (2017)

Thus there is good correlation between the assay results from the ALS Chemex lab and the certified assay value. The exception is 25-May-2008 Assay Certificate SD08050843 where the C-4 & H-5 standards are outside the recommended limits, this assay batch was re-run, including the standards with the re-run results for H-5 acceptable and C-4 now above the recommended limit.

Written procedures from 2005 indicate: Duplicates Every 20th sample is pulverized and split at Chemex with the split portion returned to Balmat, and the samples are then forwarded to the Flin Flon assay lab. The Flin Flon lab assays this split portion and the assay results are compared back to the original assay results from Chemex.

The results of the duplicates as of June 22, 2005 are shown in Figure 11.8.

Figure 11.8: Comparison of ALS Chemex Assay Lab Results and Flin Flon Assay Lab Results for Three Fine Pulp Samples (June 22, 2005)



Source: SLZ (2017)

The duplicates are considered an external independent check on the ALS Chemex assay lab results.

Additional QAQC checks showed (July 2005), as seen in the above graph, there is a >10% discrepancy between the assay results from the Flin Flon lab and Chemex on samples >25% Zn. However, there were not enough samples to draw any definitive conclusions.

A further 23 samples including six samples >30% were sent to Flin Flon in July 2005. The results from these check assays have not been located or if any check assays for the years 2006 to 2010 were sent to Flin Flon. The effectiveness of this check assay program cannot be evaluated on the limited results from June 22, 2005 and as such must rely on the CRM's submitted to the original lab for years 2005-2010.

11.3 Density Data

Historically, during operations, the mine had assumed a mineralized material bulk density of 0.100 t/ft^3 or a specific gravity of ~3.20. In 2005, a series of tests began to substantiate that assumption. The analytical method used was the 'Archimedes Method' or weight-in-air/weight-in-water.

A collection of 128 samples yielded a regression curve which was then used to estimate SG based on the zinc assay. A possible flaw in that calculation was that the skewed sampling meant that the extreme zinc% outliers may have biased the calculated density and thus the estimated tonnage.

Site personnel continued taking samples for SG and modified the regression curve (with a total of 157 samples) to incorporate gangue minerals (5%-calcite; 40%-diopside; 40%-dolomite; and 15%-quartz).

The database now totals 308 samples, of which 19 are waste or the zone code was not entered (mean SG 3.01). Table 11.4 summarizes the samples by their zone. The mine staff used the SG conversion to Imperial units T/ft = (SG x 62.4/2000).

Table 11.4: Specific Gravity Tests

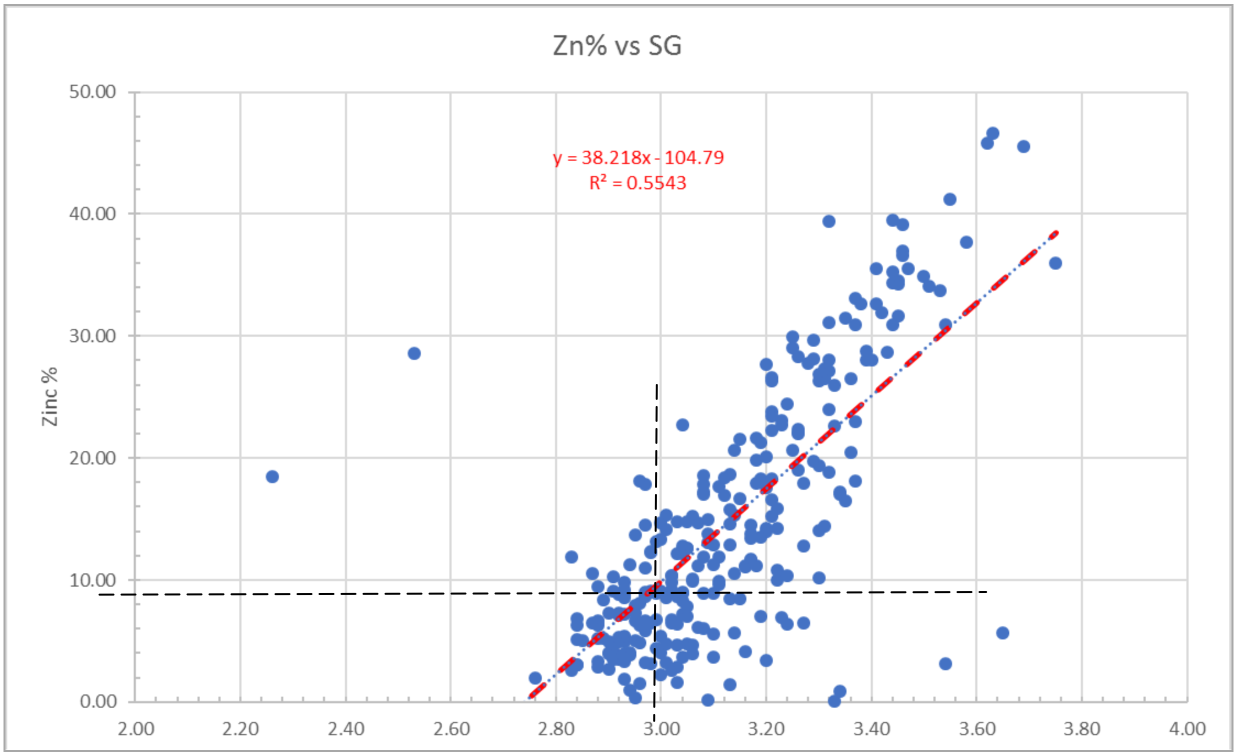
Zone Name	Zone	# Of SG Tests	Mean SG	Density (t/ft^3)
Davis	10	0	NC	NC
Cal Marble	20	0	NC	NC
Cal Upper	21	0	NC	NC
Sylvia Lake	30	0	NC	NC
Mud Pond Main	40	11	3.159	0.0986
Mud Pond Apron	41	84	3.144	0.0981
Mud Pond Apron Qtz-Diop	43	11	3.307	0.1032
Mahler Main	50	98	3.073	0.0959
Mahler White Dolomite	51	27	3.065	0.0956
Mahler Quartz Diopside	52	34	3.061	0.0955
NE Fowler	60	23	3.137	0.0979
New Fold	70	1	3.26	0.1017
TOTAL		**289**	**3.123**	**0.0975**

Source: SLZ (2017)

Eight core samples and one muck pile sample were taken by the as a cross-check of grade and SG. Results were within the above expected ranges.

A new regression curve for all the current data is shown in Figure 11.9.

Figure 11.9: SG vs Zn% Scatter Plot and Regression Curve



Source: Tuun (2017)

If one assumes that the above curve is a reasonable fit and representative, then the historic mined grade of 8.6% Zn (from 33.8 Mt of material mined historically) might have averaged SG of ~2.95 (intersection of the dashed lines).

The QP believes that the current level of SG testing is adequate for this Resource Estimate, but would recommend that testing of all the zones be continued.

11.4 Quality Assurance and Quality Control Programs

Hudbay's practice at ESM was fully compliant with quality assurance and quality control protocols of the time, and used the preparation and analytical services of certificated commercial laboratories. SLZ staff has continued to follow the protocols which include the insertion of blanks and standards as follows:

Blank samples are inserted into the assay sample stream at intervals of 50 samples. One of four commercially available certified reference standards is inserted at intervals of 20 samples.

Finally, the analytical laboratory prepares a duplicate pulp for each 20th sample and returns it to the Balmat geology department.

The certified reference standards were obtained from Ore Research and Exploration Pty Ltd. ALS Chemex was the commercial laboratory used for the 2005 drilling campaign and the 2006 to August 2008 operations period. Exploration done during the 'care and maintenance' years has continued to follow this protocol with samples being sent to ALS Chemex for assaying.

11.5 Adequacy Statement

It is the authors' opinion that these protocols and practices are adequate to ensure the integrity of the assay database.

12 Data Verification

12.1 Verifications by the Authors of this Technical Report

The authors of this report have reviewed the drill hole data set provided which consisted of 4,317 holes from which a subset of 633 were used for the current Mineral Resource estimate. The authors reviewed assay data for all available holes, representing about 95% of the data. Assay values from the database were verified by correlation with original assay certificates and by review of QA/QC procedures and results.

SLZ personnel provided the authors with the ESM digital database and some of the corresponding raw data files (source data) for the validation. The authors reviewed all relevant data and recommended corrections and additions prior to preparing the Resource Estimate. The data subset used for the verification process was selected in an attempt to represent the data spatially and temporally.

Values were compared for direct correlation, record-by-record, between the original source data and the database. The intent of the data validation was to demonstrate a positive correlation between source data and the database covering the data, which establishes reasonable confidence in the data for use in the Mineral Resource estimate.

Data categories reviewed and any limitations include:

Collar locations: raw collar survey reports were sometimes not available on the written logs, however, the site surveyor was able to provide survey verification from his files. Collar survey data was manually recorded on geology logs for most of the holes, and that data was compared to the collar file in the database. The data recorded on the geology logs appears to be approximate location, not surveyed location, as most are recorded as whole numbers. Wherever noted, collar entries were corrected.

Downhole surveys: raw downhole survey reports were unavailable. Survey data was manually recorded on geology logs under the header “Tro-Pari survey.” The Tro-Pari records were compared to the survey file in the database. These tended to match, but the authors observed occasional instances of rounding the depth record to the nearest 5 feet or dropping a decimal from the dip or azimuth record. Corrections were made as required.

Lithology: scanned paper geological logs were provided, however the database used for the resource estimate did not include a geology field, so a review was not performed.

Sample intervals: sample intervals were written on sample bags and recorded by the assay laboratory as part of the sample ID. The intervals on the assay certificates were compared to intervals in the assay field of the database. Three mismatches were identified. These were compared to the geology logs, and it was determined that the assay laboratory made a recording error and the database value was correct.

Assays: original ALS Chemex assay result certificates in digital format for later years 2005-2009 were compared with the database. Mismatches were noted. It appears that the database was not maintained and checked digitally prior to or following mine closure, an error rate of 1.7% was identified, whereby 45 errors were found within a dataset of 2,683 assays. All errors noted were corrected prior to resource modelling. Of note were that the holes 1996-F to 2001-F had 'visual' grade estimates only as the original samples were lost during shipment to the lab. Those holes were adjusted to show as not sampled (NS) and not used for estimation purposes. During the site visit the Author examined several drill core intersections for mineralized intervals and verified that sphalerite (zinc sulfide) mineralization was present in the drill core. The Author collected eight verification samples from previously spilt drill core for personal submittal to ALS Minerals Vancouver BC laboratory on March 1st, 2017 . The samples collected were approximately 10cm long to confirm the clearly visible sphalerite mineralization, interbedded waste and assess specific gravity (Table 12.1). The samples Zinc grade results confirmed the visible sphalerite mineralization noted by the author and the specific gravity results are within the ranges anticipated by the zinc grades. The Author also visited the underground workings with sphalerite mineralization observed in the Mahler 3830 drift and Mud Pond QD 2730 stope areas.

Table 12.1: Validation Assays

Sample#	Hole ID	Depth (ft)	Zone	Zn%	SG
933913	2038-F	160	Mud Pond Main	1.82	2.91
933914	2038-F	139	Mud Pond Main	20.60	3.11
933915	1619-F	550	Mahler Main	>30	3.68
933916	2200-F	1346	Waste	0.097	2.85
933917	1847-F	268.5	Mud pond Apron	29.60	3.26
933918	1674-F	356.9	Inter-bedded Waste	0.058	3.10
933919	1578-F	425.9	Mahler Main	3.00	2.84
933920	1573-F	1973	NE Fowler	17.45	3.32

12.2 Adequacy Statement

It is the Author's opinion that the drill core procedures with collection of information for inclusion in the drill hole database provided, is sufficiently accurate and precise for the Mineral Resource Estimation purpose in this Technical Report.

13 Mineral Processing and Metallurgical Testing

13.1 Test Work Summary

A test program was undertaken by Hudbay in 2005 to confirm the processing requirements of selected mineralized material zones from the Empire State Mines. These mineralized material zones were selected based on projected tonnage, mineralized material type, and sample availability.

Flotation tests were completed by Hudbay personnel in the EMS laboratory, under the guidance of Fred Vargas, the metallurgical consultant who developed the pHLOTEC flotation process in use at the ESM mine since 1984. As well, a representative for SGS Lakefield Research, performed site reviews to ensure that the program was at FS level requirements. SGS Lakefield Research assisted with development of the scope of work, review and analysis of batch test data, supervision of the locked cycle tests and interpretation of results.

The metallurgical testing and operational results from 2006 to 2008 support a zinc recovery of 96% and a zinc concentrate grade of 56% for the re-start of operations. The mineralized zones to be mined are a continuation of the mineralization mined from 2005 to 2008.

13.2 Mineralized Material Sampling and Representation of Deposits

There are three mineralization types at the ESM. At the time that metallurgical test work began, the production tonnage and mix in the concentrator of the three types was not available. Accordingly, the test work program was designed to evaluate each mineralization type individually, with the results mathematically combined as appropriate.

Type 1 mineralization make up the bulk of the tonnage (70.2%) for the life of mine. Type 1 mineralization is characterized by 600 to 1,200 ppm mercury content and 1.6 to 2.3% iron. Mud Pond and Mahler represent the highest tonnage of Type 1 mineralization and were selected for test work.

Type 2 mineralization is the second largest group in terms of tonnage (23.1%) for the life of mine and is characterized by 200 to 300 ppm mercury content and 2.9 to 4.9% iron. Sylvia Lake was the only Type 2 mineralized materials available in quantity and was selected for test work.

Type 3 mineralization represents only 6.7% of total mineralized material, all from the Cal Marble mineralized material body. Type 3 material is characterized by less than 50 ppm mercury and high, relative to the other ESM mineralized material types, iron (4.8 to 5.9%). The available sample of Cal Marble material was only 18 kg of drill core. As a result the test regime for Type 3 material was less in comparison to the Type 1 and 2 material.

The test work was split into two phases, phase one concentrated on the Type 1 material that comprise the majority of the tonnage for the current resource.

13.3 Test Methodology

Flotation test conditions (fineness of grind, reagent regime, and flowsheet) were based on the established operating practices of the ESM concentrator, optimized as necessary for the particular requirements of the mineralized material zones being tested.

The existing flotation circuit consists of a lead flotation circuit followed by zinc flotation. Lead grades for the mineralization to be mined are only 0.02%, and as such, lead flotation was not included in the test work. The zinc flotation circuit consists of rougher flotation followed by scavenger flotation. The scavenger concentrate returns to the head of the rougher circuit. Rougher concentrate undergoes two stages of cleaner flotation. Cleaner tailings are returned to the previous stage of flotation in the traditional manner.

Kinetic test work indicated that the scavenger concentrate could be combined with the rougher concentrate and sent to the cleaner circuit, in an open circuit manner, with no detrimental impact on grade or recovery. This open circuit roughing approach was used in the locked cycle flotation work.

Tests conducted on Type 1 material concentrated on two variables; mine dilution and grind size. Dilution was selected as a test variable as it was seen as a potential risk given the nature of the deposit and the mining method. High dilution typically results in reduced recovery performance of milling circuits. Mining dilution cases were selected to provide for the projected standard dilution, high dilution, and low dilution. The target cases can be seen in Table 13.1.

Table 13.1: Dilution Cases for Test Work

Deposit	Description	Mud Pond	Mahler
Low Grade (High Dilution)	Mineral (%)	56	40
	Waste (%)	44	60
	Dilution (%)	80	150
	Target Grade (% Zn)	6.6	7.4
Forecast Grade (Standard)	Mineral (%)	77	57
	Waste (%)	23	43
	Dilution (%)	30	75
	Target Grade (% Zn)	9.2	105
High Grade (Low Dilution)	Mineral (%)	91	80
	Waste (%)	9	20
	Dilution (%)	10	25
	Target Grade (% Zn)	10.8	14.7

Source: SLZ (2017)

Fineness of grind was selected as the test variable to ensure that historical concentrator grind was applicable to the new mineralized zones. Tests at different fineness of grind were conducted on the standard dilution case only. Target grinds were selected as standard, coarse, and fine. Standard grind was selected at the historical plant value of 85% passing 210 µm. Coarse grind was selected at 75% passing 210 µm. Fine grind was selected at 95% passing 210 µm.

Flotation material charges were blended from samples of mineralization and waste rock at the mass ratio predicted by the geology department. These charges were subsequently assayed for zinc content. The sample composition was adjusted as required with waste rock or mineralized material to obtain target zinc grades.

Batch flotation tests were conducted to provide kinetic information on each mineralized material zone at the specified dilution and grind. Rougher flotation kinetics, first stage cleaner kinetics, and second stage cleaner kinetics were performed. This kinetic information was used to determine the flotation conditions for locked cycle test work

13.4 Assays

The laboratory atomic adsorption (AA) analyzer at ESM was used to determine the zinc assays of samples from the test work. Duplicate samples were then shipped to the Hudbay Flin Flon Assay Laboratory. Other elements were determined by induced-coupled plasma (ICP) at the Hudbay Flin Flon Assay Laboratory. Zinc in zinc concentrate for the locked cycle work was determined by wet chemical analysis at the Hudbay Flin Flon Assay Lab.

13.5 Mineralogy

Un-pulverized portions of the samples from locked cycle tests were retained for mineralogical analyses as required. These samples include the final zinc concentrate, final tails, the last cycle first cleaner tails, and the final cycle second cleaner tails from each locked cycle test.

13.6 Bond Work Index

Blended samples of Mud Pond material, Mahler material, Mahler Quartz Diopside waste, Mahler white dolomite waste, and Mahler contact waste were sent to Lakefield Research for Bond Work Index tests.

The Ball Mill Work Index (BWI) measured on Mahler mineralization was 8.3 kWh/t.

A target grind size of 85% passing 210 µm was selected during the batch flotation test work. Thirteen minutes and thirty seconds of grinding time in the laboratory test was required to achieve this target for both Mud Pond and Mahler mineralized material. This would indicate that Mud Pond has a similar Bond Work Index to Mahler.

13.7 Batch Flotation Conditions

A series of batch kinetic flotation tests were conducted on Mud Pond and Mahler material. These tests were conducted at varying grind and dilution cases to determine their impact on zinc grade and recovery.

The reagents used in these flotation tests were consistent with those used in the pHLOTEC process. The pHLOTEC process has been used at the ESM since 1984. This process uses sodium cyanide (NaCN) and sodium sulphide (Na_2S) to condition the feed prior to flotation at a natural pH. The pHLOTEC process does not require pH modifiers such as lime or soda ash. The pulp potential (Eh) and pH were periodically monitored during flotation. The pH ranged from eight to just over nine.

Eh values prior to copper sulphate ($CuSO_4$) addition ranged from –165 mV to –101 mV. Eh values after $CuSO_4$ addition ranged from –96 mV to – 51 mV.

Reagent additions to flotation were as follows:

- NaCN and Na_2S were added to the grinding stage at 48.6 g/t and 97.2 g/t respectively;

- $CuSO_4$ was used as an activator. It was added at the start of rougher flotation and scavenger flotation. Addition at the rougher stage ranged from 170 to 291 g/t, and was adjusted based on predicted head grade. Addition at the scavenger stage was 24.3 g/t;
- Potassium amyl xanthate (PAX) and Aero-promoter 3477 were the collectors used. PAX addition ranged from 8.7 to 9.7 g/t; while 3477 addition ranged from 3.8 to 4.8 g/t; and
- The frother used was methyl isobutyl carbinol (MIBC). Frother addition to the rougher scavenger stage ranged from 7.4 to 9.7 g/t. Frother addition to the first cleaner stage ranged from 3.6 to 7.4 g/t. Frother addition to the second cleaner stage ranged from 5.4 to 7.4 g/t. Total frother addition ranged from 18.5 to 24 g/t.

Batch kinetic flotation times were selected to ensure that fully developed kinetic curves could be generated. Rougher and scavenger flotation times were selected at seven minutes initially. Initial tests indicated that six minutes was sufficient to develop the curves, and the subsequent cleaner flotation tests used six minutes of rougher flotation time. Both first and second cleaner flotation times were three minutes.

Flotation details may be found in Appendix 6 of the *Balmat No.4 Zinc Mine Re-opening Feasibility Study* dated October 2005 produced by Hudbay Minerals (Hudbay, 2005) for the following information:

- Records of the individual flotation conditions for Mud Pond and Mahler mineralized material;
- Record of all available assays for the Mud Pond and Mahler kinetic flotation test work;
- Tables of the individual test weights, grades and recoveries for Mud Pond and Mahler mineralized material;
- Graphs of the individual zinc kinetic test weights, grades and recoveries for Mud Pond and Mahler mineralized material;
- Individual grade recovery curves;
- Flotation conditions for the locked cycle work;
- Individual test analysis of mass and zinc unit stability for the locked cycle work; and
- Summary tables for Mud Pond and Mahler grade and recovery for zinc, iron, calcium, magnesium, and mercury.

13.8 Mud Pond Flotation Kinetics

Average results of Mud Pond batch tests are shown in Table 13.2 to 13.4.

Table 13.2: Mud Pond Rougher Flotation Kinetics

Mud Pond	Rougher Flotation Analysis		
Dilution Case	Grind Case	Grade (% Zn)	Recovery (% Zn)
Standard	Standard	28.0	98.7
Standard	Fine	23.1	99.3
Standard	Coarse	25.2	99.2
High	Standard	21.0	99.0
Low	Standard	31.5	99.6

Source: SLZ (2017)

Mud Pond rougher flotation results at the standard dilution and standard grind case resulted in an average 28.0% zinc concentrate grade and a recovery of 98.7%. All other feed conditions resulted in slightly higher recoveries. These higher recoveries are within acceptable experimental error and are considered equivalent. Fine grinding, coarse grinding and high dilution cases resulted in lower concentrate grades. The low dilution case resulted in a higher grade than the base case.

Table 13.3: Mud Pond First Cleaner Flotation Kinetics

Mud Pond	1st Cleaner Flotation Analysis		
Dilution Case	Grind Case	Grade (% Zn)	Recovery (% Zn)
Standard	Standard	47.9	94.1
Standard	Fine	43.4	97.1
Standard	Coarse	47.1	97.6
High	Standard	39.5	96.0
Low	Standard	48.5	97.9

Source: SLZ (2017)

Mud Pond first cleaner flotation kinetics at the standard dilution and standard grind case averaged 47.9% zinc concentrate grade and a recovery of 94.1%. All other feed conditions resulted in higher recoveries. Fine grinding, coarse grinding and high dilution cases resulted in lower concentrate grades. The low dilution case resulted in a higher recovery and grade than the base case.

Table 13.4: Mud Pond Second Cleaner Flotation Kinetics

Mud Pond	Second Cleaner Flotation Analysis		
Dilution Case	Grind Case	Grade (% Zn)	Recovery (% Zn)
Standard	Standard	54.4	94.6
Standard	Fine	51.9	94.8
Standard	Coarse	52.5	96.7
High	Standard	47.6	96.1
Low	Standard	55.4	96.5

Source: SLZ (2017)

Mud Pond second cleaner flotation kinetics at the standard dilution and standard grind case resulted in a 54.4% zinc concentrate grade and a recovery of 94.6%. All other feed conditions resulted in higher recoveries. The low dilution case resulted in a higher recovery and grade than the base case. All other feed conditions resulted in a lower grade.

Overall, the results from the Mud Pond test work indicate that higher grades and equivalent or higher recoveries can be achieved with low dilution (i.e. higher feed grades). All other cases resulted in higher or equivalent recoveries at lower concentrate grades.

13.9 Mahler Flotation Kinetics

Average results of Mahler batch tests are shown in Tables 13.5 to 13.9.

Table 13.5: Mahler Rougher Flotation Kinetics

Mahler Dilution Case	Rougher Flotation Analysis		
	Grind Case	Grade (% Zn)	Recovery (% Zn)
Standard	Standard	31.3	99.2
Standard	Fine	31.4	98.7
Standard	Coarse	38.7	98.0
High	Standard	27.2	98.4
Low	Standard	38.6	97.1

Source: SLZ (2017)

Mahler rougher flotation results at the standard dilution and standard grind case resulted in a 31.3% zinc concentrate grade and a recovery of 99.2%. All other feed conditions resulted in slightly lower recoveries. These lower recoveries are within reasonable experimental error and can be considered equivalent. The exception to this is the low dilution feed case, which had a measurably lower recovery. The low dilution case had an unexpectedly high feed grade of approximately 20% zinc. The flotation times and reagent addition for the low dilution tests were too low to recover the high contained zinc values. Fine grinding resulted in an equivalent concentrate grade. Coarse grinding and low dilution cases resulted in higher concentrate grades. The high dilution case resulted in a lower grade concentrate than the base case.

Table 13.6: Mahler 1st Cleaner Flotation Kinetics

Mahler Dilution Case	First Cleaner Flotation Analysis		
	Grind Case	Grade (% Zn)	Recovery (%)
Standard	Standard	44.0	97.4
Standard	Fine	47.6	91.8
Standard	Coarse	48.9	96.4
High	Standard	46.3	94.8
Low	Standard	49.5	97.3

Source: SLZ (2017)

Mahler first cleaner flotation kinetics at the standard dilution and standard grind case resulted in a 44.0 % zinc concentrate grade and a recovery of 97.4%. All other feed conditions resulted in lower recoveries. The exceptions to this are the coarse grind and low dilution case, which resulted in equivalent recoveries. All other feed cases resulted in higher grades than the base case.

Table 13.7: Mahler 2nd Cleaner Flotation Kinetics

Mahler Dilution Case	Second Cleaner Flotation Analysis		
	Grind Case	Grade (% Zn)	Recovery (%)
Standard	Standard	59.3	92.3
Standard	Fine	55.1	85.3
Standard	Coarse	54.5	95.5
High	Standard	60.2	93.8
Low	Standard	52.3	95.8

Source: SLZ (2017)

Mahler second cleaner flotation kinetics at the standard dilution and standard grind case resulted in a 59.3% zinc concentrate grade and a recovery of 92.3%. The coarse grind, high dilution and low dilution cases resulted in higher recoveries. The fine grind case had a significantly lower recovery than all other cases. The high dilution case had a roughly equivalent grade to the base case. All other cases had a lower grade than the base case. Performance in terms of grade and recovery for the low dilution case was low due to the extremely high feed grade of approximately 20% zinc as previously discussed.

Overall flotation results for Mahler mineralization indicated higher grade concentrates at equivalent or slightly lower recoveries than were produced with Mud Pond material. Unlike Mud Pond, clear relationships between dilution, grind, and grade/recovery results could not be identified.

13.10 Locked Cycle Flotation Test Work

Locked cycle tests were performed on Mahler and Mud Pond mineralization. Locked cycle flotation tests are semi-continuous and provide a better estimate of full scale results rather than batch tests only. The locked cycle test flotation stages included a single rougher stage and two stages of cleaning. The rougher flotation test stage was conducted to produce a rougher concentrate without a scavenger concentrate. In all cases, cleaner tailings material was recycled to the previous stage of flotation in the subsequent cycle.

The flotation conditions for the locked cycle tests were the same for both material types. The exception is that the frother additions to the later cycles of the Mahler material test were reduced in order to maintain a proper froth texture. The time for the rougher flotation was three minutes. The time for the first cleaner flotation was 2 minutes. The time for the second cleaner flotation was two minutes.

Reagent addition was consistent with the conditions used in batch flotation.

Table 13.8: Locked Cycle Test Reagents

Reagent	Unit	Value
NaCN	g/t	48.6
Na_2S	g/t	97.2
$CuSO_4$	g/t	316
PAX	g/t	8.7 – 9.7
3477	g/t	3.9
MIBC	g/t	17.5

Source: SLZ (2017)

Locked cycle tests were conducted with six cycles for all tests. Duplicate locked cycle flotation tests yielded consistent metallurgical predictions, as shown in Table 13.9.

Table 13.9: Locked Cycle Test Results

Sample	Element	Units	Mud Pond			Mahler			Wt. Avg.
			LC-1	LC-2	Average	LC-1	LC-2	Average	Total*
Head Assay	**Zn**	**%**	**9.7**	**9.6**	**9.65**	**10.7**	**10.7**	**10.7**	**10**
	Fe	%	0.8	0.8	0.8	0.9	1	0.95	0.85
	Pb	%	0.051	0.495	0.273	0.005	0.005	0.005	0.184
	Mg	%	6.9	6.8	6.85	8.7	9	8.85	7.52
	Ca	%	16.4	15.4	15.9	15.3	13.7	14.5	15.43
	Hg	ppm	-	178	178	137	142	140	165
Conc. Assay	**Zn**	**%**	**59.9**	**61**	**60.45**	**58.7**	**60.8**	**59.75**	**60.22**
	Fe	%	3.2	2.9	3.05	3.5	3	3.25	3.12
	Pb	%	0.235	0.17	0.203	0.01	0.009	0.01	0.138
	Mg	%	0.48	0.33	0.41	0.74	0.53	0.64	0.48
	Ca	%	0.43	0.26	0.35	0.67	0.25	0.46	0.38
	Hg	ppm	-	1150	1150	751	762	757	1019
Tail Assay	**Zn**	**%**	**0.31**	**0.48**	**0.4**	**0.27**	**0.25**	**0.26**	**0.35**
	Fe	%	0.35	0.41	0.38	0.3	0.6	0.45	0.4
	Pb	%	0.017	0.028	0.022	0.044	0.042	0.043	0.029
	Mg	%	8.06	7.89	7.98	10.4	10.74	10.57	8.84
	Ca	%	19.33	18.07	18.7	18.48	16.46	17.47	18.29

Table 13.9: Locked Cycle Test Results (continued)

Sample	Element	Units	Mud Pond			Mahler			Wt. Avg.
			LC-1	LC-2	Average	LC-1	LC-2	Average	Total*
	Hg	ppm	-	6.73	6.73	3.85	13.3	8.58	7.35
Conc.	**Zn**	**%**	**97.3**	**95.8**	**96.55**	**97.9**	**98**	**97.95**	**97.02**
Rec.	Fe	%	62.7	56.2	59.45	71.8	51.1	61.45	60.12
	Pb	%	72.7	51.6	62.15	33.1	30.7	31.9	52.067
	Mg	%	1.09	0.73	0.91	1.52	1.02	1.27	1.03
	Ca	%	0.41	0.25	0.33	0.78	0.32	0.55	0.4
	Hg	ppm	-	96.86	96.86	97.7	92.41	95.06	96.26

*Weighted average total, calculated as 2/3 Mud Pond and 1/3 Mahler by weight

Source: SLZ (2017)

These results represent the best performance achieved with the flotation conditions used with Type 1 mineralization. The combined average in the table represents the expected flotation results when processing Mud Pond material as 2/3 of the plant feed and Mahler as 1/3 of the plant feed at a 60.2% zinc concentrate with a 97% recovery.

Locked cycle test work was also conducted at SGS Lakefield Research laboratory on a 40 kg sample of blended Mud Pond and Mahler material. The primary purpose of this test was to produce a sample of zinc concentrate for autoclave testing. The test was designed such that it could provide independent verification of the test results from the locked cycle work conducted at Empire State mine. Four 10 kg cycles were run. The locked cycle test work at SGS Lakefield included a true scavenger flotation stage. These tests resulted in a zinc grade of 60.8% and a recovery of 97.1%. The 8.0% zinc head grade for this test was lower than the locked cycle tests conducted at Empire State mine. The tailings from this test graded 0.27% zinc, also lower than the ESM results. These results were consistent with the locked cycle work conducted at Empire State mine.

For comparison, the typical zinc recovery in previous years of plant operation was 94.5% to a concentrate grading 55% zinc (Hudbay, 2005). This plant performance was achieved while running the original resources which are somewhat different from those of today. The original resources had lead values that justified production of a lead concentrate, and as a result, zinc losses to lead concentrates were incurred. As well, the original Pierrepont resource had significant talc values that adversely impacted grade and recovery in the zinc circuit.

13.11 Metallurgical Forecast

The following factors were considered in the preparation of the metallurgical forecast:

- Locked cycle test results at ESM and Lakefield;
- Historical ESM concentrator performance;
- The proportions of Type 1, Type 2 and Type 3 mineralization in concentrator feed;
- The relative pyrite contents of the three material types; and
- The relative iron contents in sphalerite for the three mineralization types.

The locked cycle grade of 60% was reduced to account for the impact of iron content in sphalerite increasing with Type 2 and 3 mineralized material, increased predicted head grades of iron, and expected plant inefficiencies. This resulted in a predicted grade of 56% zinc.

The locked cycle recovery of 97% was reduced to 96% to account for expected plant inefficiency compared to the test condition. The geological estimate of future lead head grades is low, and therefore the lead circuit will not be run. As such no recovery penalty was applied for losses of zinc in lead flotation.

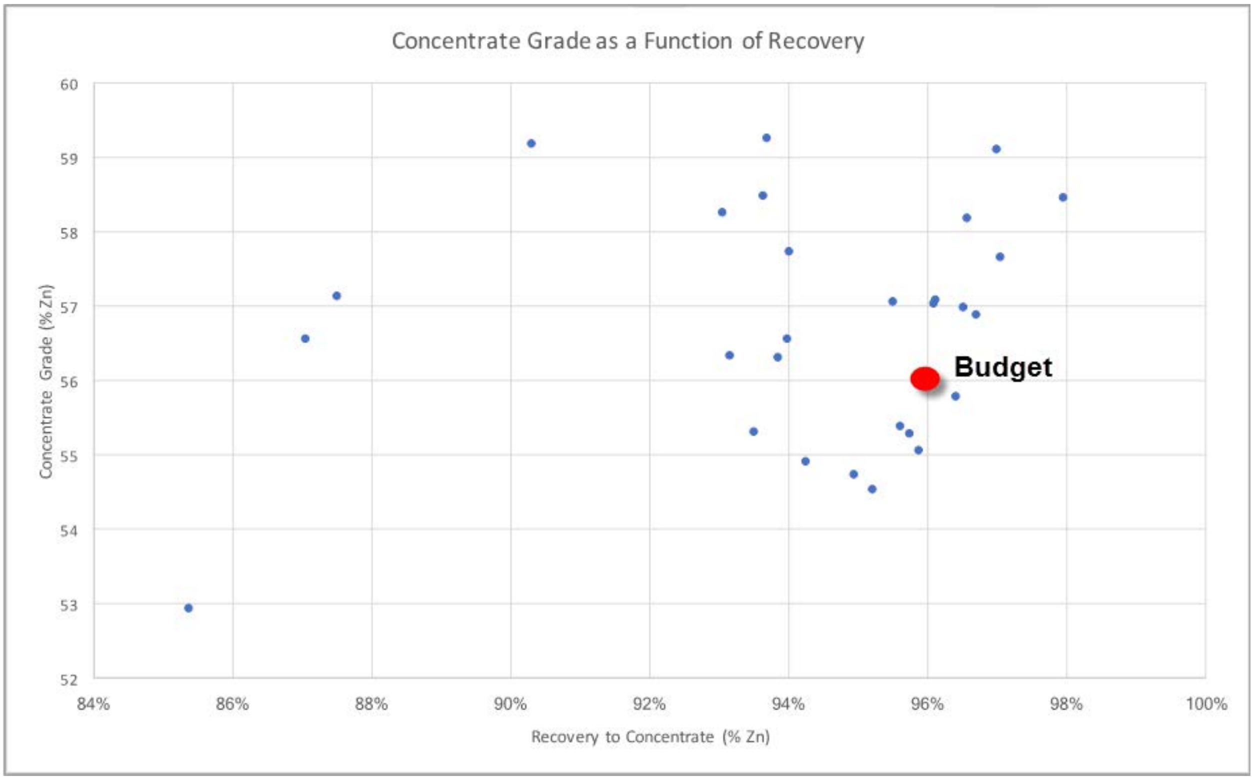
Secondary elements such as iron, lead, magnesium, calcium, and mercury were balanced over the same cycles determined to balance the zinc. Mercury values in concentrate were over 1,000 ppm. The mercury head grades and high recovery to zinc concentrate contribute to these values.

13.12 Metallurgical Assumptions

- The pHLOTEC process can be used to process mineralized plant feed at ESM;
- The sphalerite at ESM Type 1 mineralization exhibits fast kinetics at a coarse grind;
- Locked cycle tests on Type 1 mineralization produced an average zinc recovery of 97% to concentrate grading 60% zinc;
- A zinc concentrate grade of 56%, and zinc recovery of 96% are considered to be readily achievable results in the plant; and
- Mine head grade estimates have increased since the completion of the test program. This is assumed to have a favourable impact on metallurgical performance, but has not been taken into consideration in the grade and recovery forecast at this point.

Testing completed for the Hudbay 2005 Feasibility Study, identified zinc recovery of 96% with a concentrate grade of 56% Zn. These figures were used for budget purposes during the years Hudbay operated the ESM from 2006 to 2008. The following figure shows zinc concentrate grade as a function of zinc recovery, using month end data for the same period.

Figure 13.1: Monthly Concentrate Grades and Recoveries; 2006 to 2008



Source: JDS (2017)

The operating results are somewhat scattered but demonstrate that the targets of 96% Zn recovery with a concentrate grade of 56% Zn is achievable. The period from February 2007 to October 2007 met these targets, averaging 96.7% recovery at an average concentrate grade of 57.4% Zn. The average head grade during this period was 7.08% Zn.

The Tetratech Fatal Flaw review of October 2014 (Tetra Tech, 2014) also supported the same recovery and concentrate grades as the basis for mill operation. Table 13.10 contains the recommended zinc recoveries and grades for operations re-start.

Table 13.10: Recovery and Concentrate Grade Estimates

Parameter	Unit	Concentrate
Zn Recovery	%	96
Zn Concentrate Grade	%	56

Source: TR (2017)

14 Mineral Resource Estimate

Empire State Mines (ESM) consists of several historic past-producers in the Fowler, NY area. While resources and reserves have recently been reported (IG 7) to US regulators, this Resource Estimate was prepared with both revised, and supplemental data collected since the 2005 Feasibility Study and therefore supersedes all previous reports.

14.1 Introduction

This Mineral Resource Statement for the Empire State Mines Zinc Project has been prepared under the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) guidelines. Historically, the mining operations (aka Balmat mine: from 1930) produced some 33.8 Mt of 8.6% Zn. The mine was put into 'care and maintenance' in 2008 when low metal prices and high operating costs negatively impacted mine economics.

Previous mine owner Star Mountain Resources published a US Industry Guide 7 Mineral Reserves Report (IG7) in November 2015. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. This Mineral Resource Estimate re-examined the existing data for the purposes of producing an updated Resource Estimate following NI 43-101 guidelines that form the basis of this Preliminary Economic Assessment (PEA).

This Mineral Resource Estimation was completed by Allan Reeves, P. Geo., of Tuun Consulting Inc., an independent Qualified Person as defined under NI 43-101 requirements.

The effective date of the resource statement is April 6, 2017 and follows the guidelines of the generally accepted CIM 'Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines (as adopted on November 23, 2003). Also considered was the 'Guidance on Commodity Pricing used in Resource Estimation and Reporting' adopted by the CIM Council on November 28, 2015. The guidance provides additional clarity on the CIM definition of "reasonable prospects of eventual economic extraction".

For resource estimation, Dassault Systemes Geovia GEMSTM Version 6.7.1.1 (GEMS) software was utilized to validate the provided wireframe solids; basic statistics and geostatistics; Variography; block modelling; and reporting of the Mineral Resource.

ESM provided Tuun with eleven revised geologic solids for evaluation. The solids represent both remnant historic resources and potential future resources.

14.2 Resource Database

The original diamond drill hole resource database was supplied as four 'comma separated value' spreadsheets. The spreadsheets contained the drill hole collar coordinates and hole length; down hole surveys; assays; and geology. The 46 channel samples that had been used for the 2005 estimation were not included in this work as backup data on location, grade, and sampling methodology was inadequate.

The drill hole database was imported into GEMS and error reporting checks done to locate input errors. Out of the 4,317 holes in the database, three were found to have an error in the down hole survey length; ten had minor typos in the down hole survey azimuth/dip; twenty-four had minor edits to collar locations; and fourteen holes had errors in the assay interval sequences (overlaps/hole lengths). These errors were reported back to the site geologists for follow-up and correction. SLZ then provided Tuun with a revised GEMS database complete with as-built workings, resource solids and corrected and supplemental drill hole information.

Holes that intersected the resources were flagged so that data analysis was restricted to a smaller subset of the overall database. A total of 633 holes and 2970 assays, were identified (Table 14.1) and utilized in the Resource Estimate.

Table 14.1: Drill holes used in the Resource, by Year

Year	No. of Holes	Footage Drilled
Pre-2000	142	126,407.0
2000	33	23,384.0
2001	12	3,539.0
2004	5	3,143.0
2005	98	47,312.0
2006	120	42,446.3
2007	77	31,028.5
2008	140	36,931.6
2009	6	3,567.0
TOTAL	**633**	**317,758.4**

Source: Tuun (2017)

All other holes were either distal exploration holes or holes defining the historic underground workings not relevant to this study.

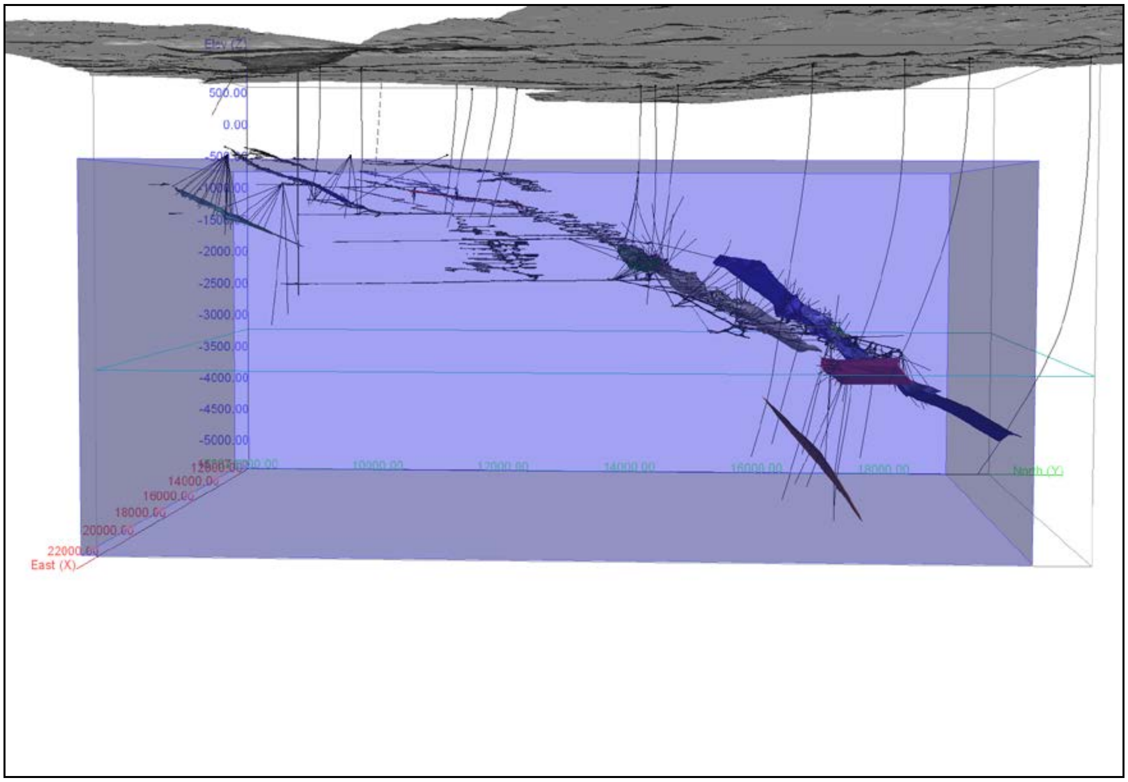
Wireframes consisted of two types of information: surveyed underground workings (as-built) and resource shapes as determined by site geologists. Plans, sections, and backup documentation identifying 11 main resource zones in the deposit were also provided.

It is the QP's opinion that the quality of the drill hole data and wireframes provided are adequate for the estimation of resource tons and grade for this PEA.

14.3 Surfaces and Solids

The surface, workings, drill holes, resource solids and block model bounding box are shown in Figure 14.1. The depth of resources varies from the -675 ft level to -3,900 ft.

Figure 14.1: Surfaces, Solids and Drill Holes



Source: Tuun (2017)

14.3.1 Topography

The main mine workings and resource solids are well below the topographic surface as can be seen in Figure 14.1.

14.3.2 Mine Workings (As-built voids)

The original underground workings totalled in excess of 250 small wireframes, many of which were improperly closed, thus preventing volumetric calculations. The wireframes were sent to MaptekTM for repairs and validation. The as-built solids were then combined into one solid which was subsequently validated in both Vulcan™ and GEMS™ software packages.

The complexity of the as-built wireframe meant that clipping the mined-out areas from the resource wireframes could create new solids errors. To avoid that issue, the void volumes were subtracted from the resources during the block modelling phase.

14.3.3 Resource Wireframes (Solids)

ESM provided 11 key mineral domains which are constrained by the well-documented geologic horizons described in Sections 7.4 to 7.5 of this report. While 12 domains were identified, Cal Upper domain was discarded as it only had 3 drill holes. The mineralized zones have been identified as in Table 14.2.

Table 14.2: Mineral Zone Domains

Mineral Zone	Zone Code
Davis	10
Cal Marble	20
Sylvia Lake	30
Mud Pond Main	40
Mud Pond Apron	41
Mud Pond Quartz Diopside	43
Mahler Main	50
Mahler White Dolomite	51
Mahler Quartz Diopside	52
NE Fowler	60
New Fold	70

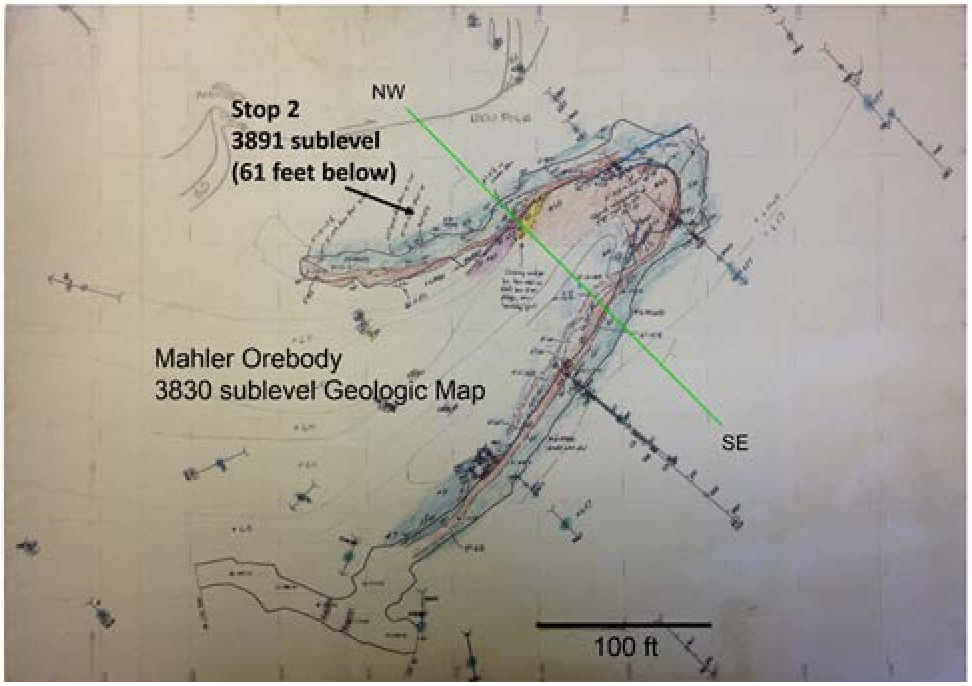
Source: Tuun (2017)

Decades of face-mapping was used to develop the wireframes in 2009. Level plans such as the one shown in Figure 14.2 give credence to the detail of the work completed adjacent to the mine workings.

Away from the workings however, the wireframes necessarily relied upon both surface and underground drill holes. Deviation in the azimuth or dip of the drill holes often increased due to the structural complexity of the host rocks causing hole deflection.

The 2009 wireframes also had been constructed along vertical cross-sections. That methodology was updated in February to March 2017 by re-interpretation and adjustment of polylines to ‘snap’ to drill hole intercepts. The revised 2017 mineral zone wireframes were used for this resource estimation.

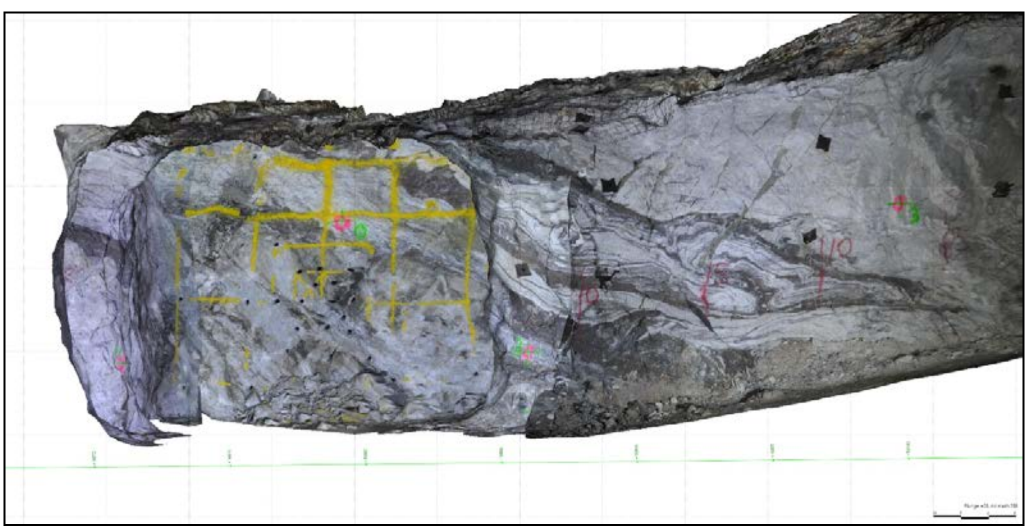
Figure 14.2: Mahler Level Mapping



Source: John Johnson U/G Tour Presentation (ESM)

During the site visit, it was observed that the lithologic units are host to poddy and semi-discontinuous mineralization that pinches and swells within the stratigraphic horizon. Note that the dark sphalerite-rich bands can contain variable light-coloured dilution of primarily white dolomite, or a greenish quartz diopside (Figure 14.3).

Figure 14.3: Mahler 3891 Sub-level



Source: John Johnson U/G Tour Presentation (ESM)

The drill holes were 'passed' through the mineral domain solids to determine which historical holes were relevant. Holes that hit a solid had a flag [rock code] added to the header to facilitate identification and subsequent analyses. Due to the orientation of the drill holes it was possible to intersect more than one zone (Table 14.3).

Table 14.3: Summary of Resource Wireframes

Zone Name	BM Code	Colour	Volume (ft^3)	No. Holes	No. Assays
Davis	10	Dk. Red	1,396,757	16	43
Cal Marble	20	Aqua	5,095,184	25	31
Sylvia Lake	30	Blue	3,369,493	31	49
Mud Pond Main	40	Tan	15,658,107	193	161
Mud Pond Apron	41	Green	4,201,802	98	161
Mud Pond QD	43	Violet	1,991,859	40	157
Mahler Main	50	Dk. Blue	19,391,327	205	601
Mahler WD	51	Lt Blue	3,978,665	64	167
Mahler QD	52	Bright green	661,469	34	90
NE Fowler	60	Lt Orange	6,313,186	7	21
New Fold	70	Red	12,270,894	23	112

Source: Tuun (2017)

14.4 Assay Data Evaluation

Various statistical tools were used to examine the characteristics of the dataset. Preliminary statistics were conducted in Excel. Note that the Excel statistics provided a guide only as they were based on the preliminary geologic 'codes' attached to the assays by the mine geologists. The preliminary statistics were useful in identifying sampling bias, outliers and unusual sample lengths.

GEMS software contains a comprehensive set of statistical tools to examine the characteristics of a dataset. In addition to basic or 'descriptive' statistics; histograms and probability plots were used to further analyze the data.

14.4.1 Basic Assay Sample Length Statistics

Assay lengths were inconsistent due primarily to barren interbeds within the main lithologic zones. Discussion with the site geologist identified that the un-sampled intervals were considered barren (white dolomite, qtz.-diopside etc.) and therefore both implicit and explicit missing intervals are to be calculated with a zero-grade zinc% assay. (Pers. Comm. J. Johnston, ESM)

Table 14.4 summarizes the basic assay length statistics as determined in Excel. Of note are high values for variance, skewness and kurtosis.

Skewness is a measure of symmetry, or more precisely, the lack of symmetry. A distribution, or data set, is symmetric if it looks the same to the left and right of the center point.

Kurtosis is a measure of whether the data are heavy-tailed or light-tailed relative to a normal distribution. That is, data sets with high kurtosis tend to have heavy tails, or outliers. Data sets with low kurtosis tend to have light tails, or lack of outliers. A uniform distribution would be the extreme case.

Table 14.4: Excel Statistics for Assay Length

ZONE=>	Units	Davis	CalMar	Sylvia	MP	MP-A	MPQD	Mahler	MAWD	MAQD	NEF	NF
Number	#	19	30	53	360	436	191	770	367	320	51	166
Min	ft	1	1	1	1	0.25	0.3	0.1	0.2	0.2	0.5	0.3
Max	ft	181.7	10	39	38	29.5	25	53.5	32.4	36	8	19
Mean	ft	6.7	4.6	8.9	7	4.8	4.1	5	4.7	3.6	2.5	3.6
Median	ft	6	4.3	6	4.5	3.5	2.8	3.1	3.2	2.4	1.5	2.5
St. Dev.	Ft	4.82	2.59	8.12	6.66	4.36	4.04	5.66	4.95	4.06	2.05	3.42
Variance	ft	23.2	6.7	65.9	44.4	19	16.3	32.1	24.6	16.5	4.2	11.7
Skewness	ft	1.1	0.3	2	1.9	1.8	2.2	1.9	2.1	3.3	1.3	1.8
Kurtosis	ft	0.5	-1	4.5	4.3	4.3	6.4	3.7	4.6	16.2	0.6	3.7

Source: Tuun (2017)

The high variability in the sample lengths is unusual in this QP’s experience. For base metals such as this deposit, a consistent sample length of ~5 ft with shorter runs at contacts is more appropriate, and SLZ is currently implementing this practice.

14.4.2 Basic Zinc Assay Statistics by Zone

Examining the zinc assays in Excel shows (Table 14.5) the same high variability, skewed data and presence of outliers noted with the sample lengths. The sample selection and lithologic coding is handled differently within GEMS software.

Drill holes were ‘passed’ through the mineral domain solids to determine which historical holes were relevant. Holes that hit a solid had a flag [the mineral zone letters: E.g. ‘MA’ or MP-QD’] added to the header table to facilitate identification and subsequent analyses. Due to the orientation of the drill holes it was possible to intersect more than one zone (Table 14.5). Assay intervals that passed through the resource domain solids were coded with respective domain rock codes.

Table 14.5: Excel Statistics of Zinc Assay Data

ZONE=>	Units	Davis	CalMar	Sylvia	MP	MP-A	MPQD	Mahler	MAWD	MAQD	NEF	NF
Number	#	19	30	53	360	436	191	770	367	320	51	166
Min	ft	1	4.2	0.2	1.5	0.7	2	0.3	0.1	0.1	0.1	0.1
Max	ft	28.4	25	26.1	42.2	57.8	46.7	54.5	54.6	52.8	38.8	54.5
Mean	ft	8.7	12.5	9.6	12.9	12.8	13	17.7	18.8	13.5	6.4	16.8
Median	ft	6.9	11.4	7.6	11.5	9.7	9.8	13.9	14.6	10.4	3.2	12.8
St. Dev.	Ft	7.8	5.4	7.5	8.2	9.8	9.4	12.8	14	9.5	7.8	12.6
Variance	ft	61.4	28.6	56.4	66.4	95.8	88.7	163	195.9	9.1	60.5	157.9
Skewness	ft	1.5	0.7	0.6	1.1	1.5	1.5	0.6	0.7	1.5	2.9	0.9
Kurtosis	ft	2.2	-0.1	-0.7	1	1.9	1.9	-0.9	-0.1	0.9	10	-0.1

Source Tuun (2017)

In general, the Excel statistics suggest potential problems resulting from inconsistent sample lengths along with a possible selection bias towards which assays should be included (or excluded) from the estimation.

When analyzing the data in GEMS, the assays were tagged with the zone codes generated by a cross-table transfer of information from the zone solids intersections. See Table 14.6 for the statistics of zinc samples that fall within the mineralized wireframes.

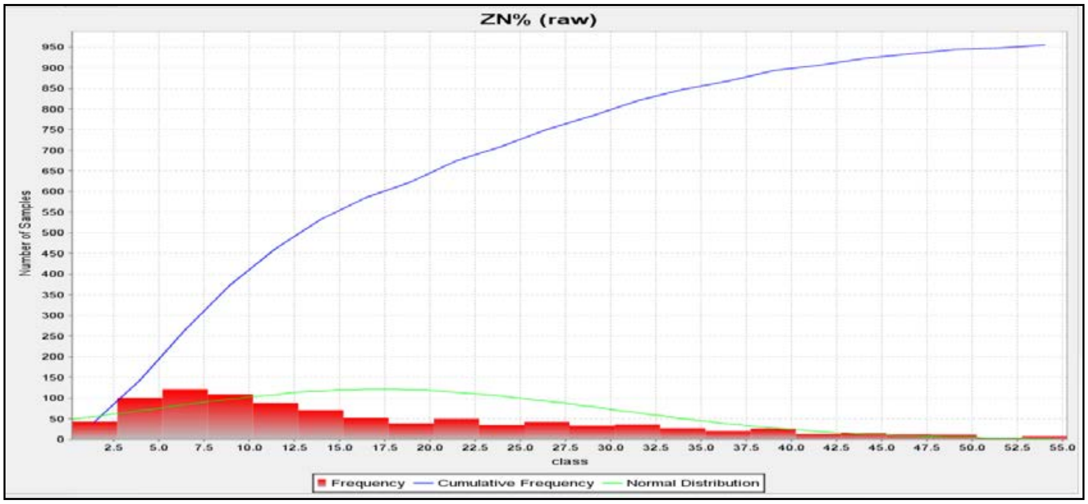
Table 14.6: GEMS Statistics of Zinc Assay Data

ZONE=>	Units	Davis	CalMar	Sylvia	MP	MP-A	MPQD	Mahler	MAWD	MAQD	NEF	NF
Number	#	36	30	49	378	216	139	591	165	90	20	110
Min	ft	1	4.2	0.2	1.5	1.4	3	0.4	1	3.1	1	2
Max	ft	33.5	25	26.1	39.6	52.7	46.7	54.5	54.6	44.1	38.1	54.5
Mean	ft	12.4	12.8	9.7	12.5	14.3	13.9	18.9	24.6	17.3	9	19.8
Median	ft	10.4	11.6	7.6	11	11.6	10.4	15.3	24	14.9	8.2	15.6
St. Dev.	Ft	8.3	5.5	7.6	7.6	10.1	10	13	14.3	10.3	8	12.9
Variance	ft	68.1	30.6	57.4	57	103.2	99.3	170.1	203.1	106.4	63.4	166.2
Skewness	ft	0.8	0.5	0.6	1	1.3	1.3	0.7	0.2	0.6	2.3	0.6
Kurtosis	ft	2.7	2.3	2.2	3.6	4.6	4.1	2.6	2.1	2.4	9.2	2.4

Source: Tuun (2017)

Basic statistics and histograms were created in GEMS. Figure 14.4 shows the combined assay data results for all zinc assay intercepts within the twelve zones. The breadth of the distribution curve shows the range of variability in sample values. There is quite a long tail which was indicated by the kurtosis values, along with what appear to be multiple populations. The small multiple populations may be real (zonal) but also could be a relic of the sampling methodology.

Figure 14.4: GEMS Histogram of All Zinc Assays in Mineralized Zones



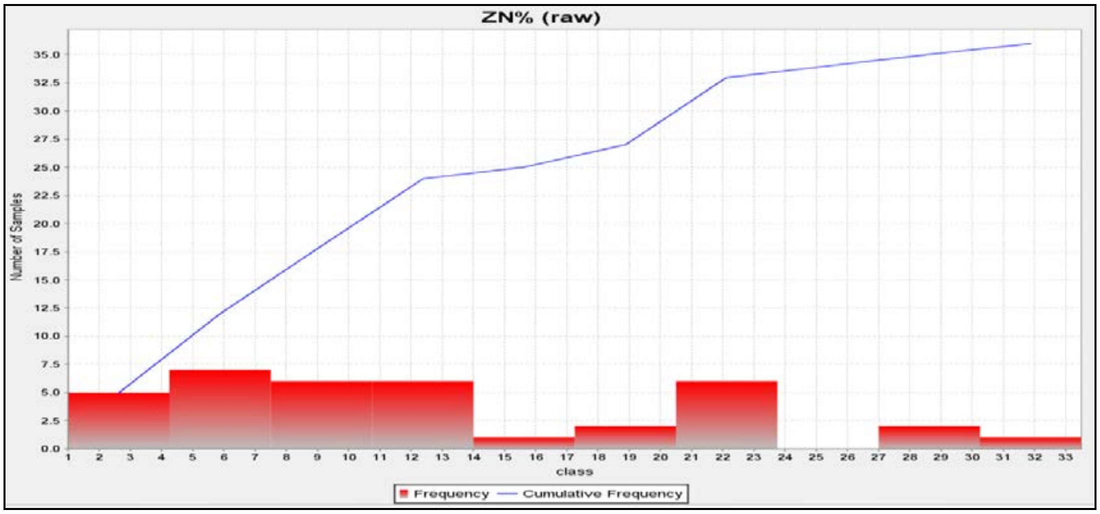
Source:Tuun (2017)

The following subsections are a closer examination of the zinc assays by each mineral zone.

14.4.2.1 Davis (Zone 10)

The Davis zone represents potentially recoverable pillars and minor unmined stope periphery material. Unfortunately, the paucity of raw data (36 samples) limits confidence in an estimation. The erratic sampling and a second population is evident in the histogram for percent zinc (Zn%).

Figure 14.5: Davis Zinc Assays

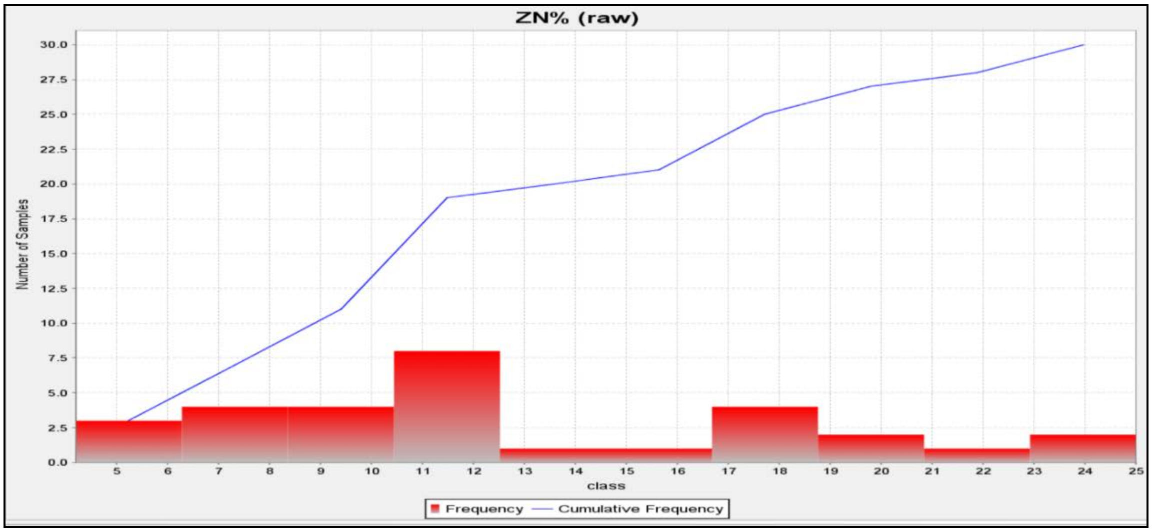


Source: Tuun (2017)

14.4.2.2 Cal Marble Main (Zone 20)

The Cal Marble sampling shows outliers at >23% zinc along with a possible secondary population at 18% zinc and a third at >23% zinc. Given that there are only 30 samples in this dataset, it is possible that the other populations may be an artifact of the selective sampling methodology.

Figure 14.6: Cal Marble Zinc Assays

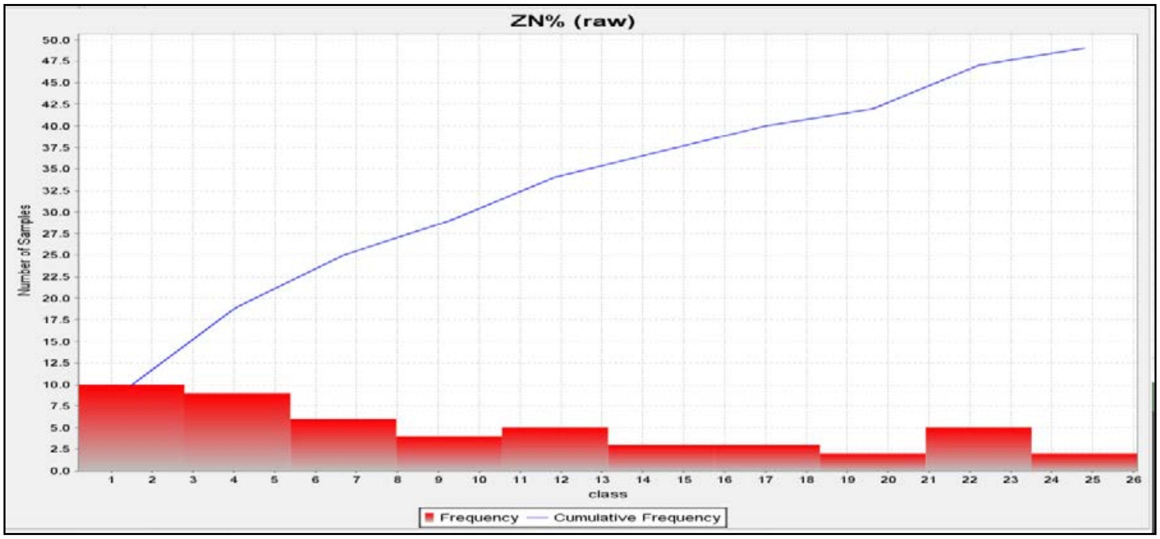


Source: Tuun (2017)

14.4.2.3 Sylvia Lake (Zone 30)

Sylvia Lake has 49 samples which show high variability and a lower overall zinc grade than other zones. There is a clear outlier population at >21% zinc.

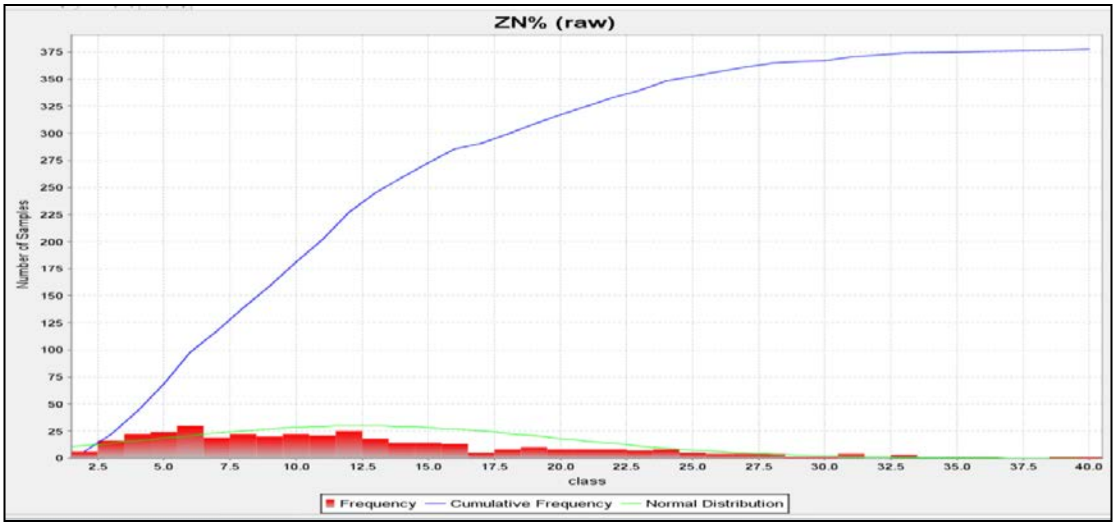
Figure 14.7: Sylvia Lake Zinc Assays



14.4.2.4 Mud Pond Main (Zone 40)

The bulk of the Mud Pond assays are less than 17% zinc, but the range of values extends to >38% zinc. This was not unexpected due to the high variance, skewness and kurtosis values seen in the Excel analysis.

Figure 14.8: Mud Pond Main Zinc Assays

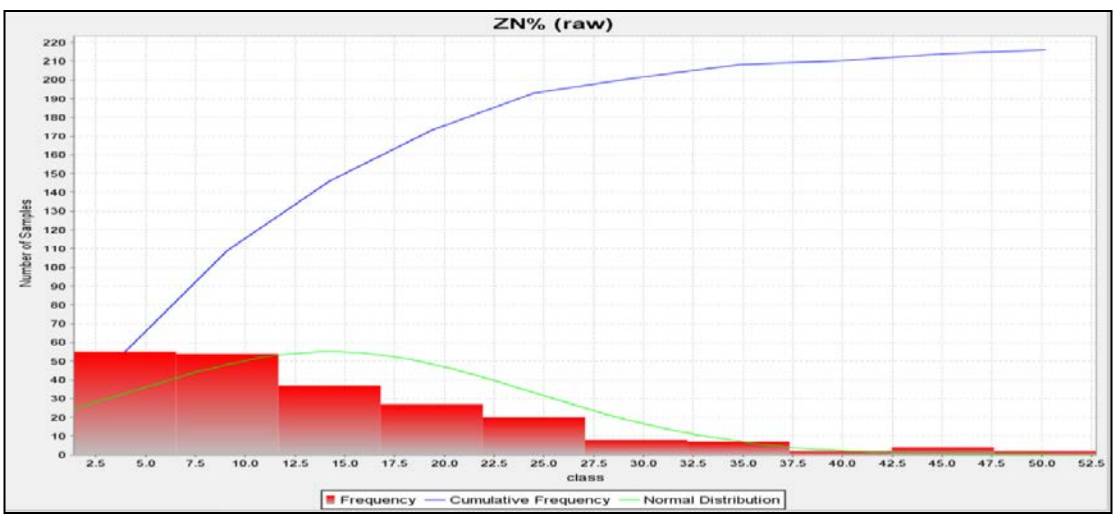


Source: Tuun (2017)

14.4.2.5 Mud Pond Apron (Zone 41)

The Mud Pond Apron has a reasonable distribution (only a few outliers to consider) based upon a moderate number of raw assays (216).

Figure 14.9: Mud Pond Apron Zinc Assays

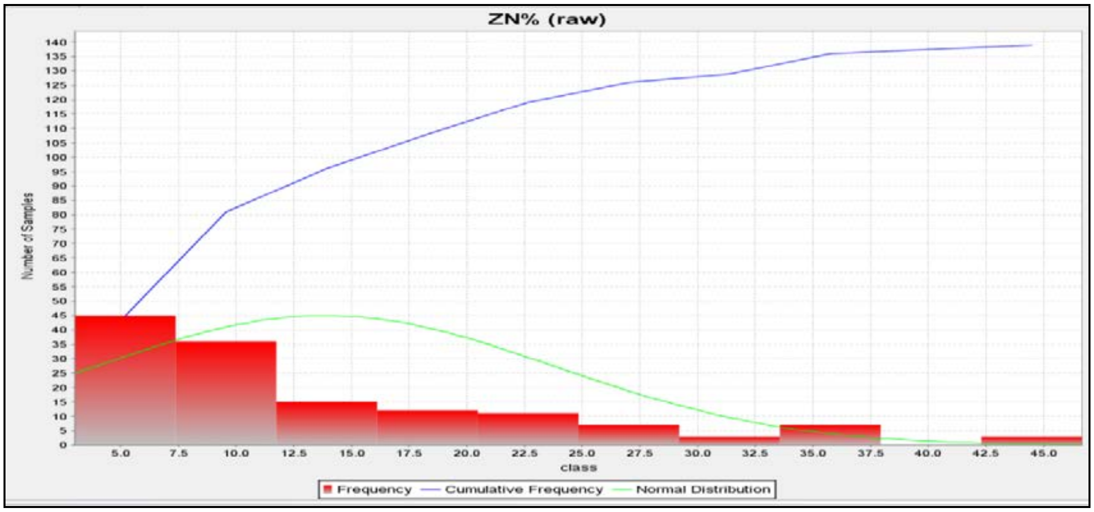


Source: Tuun (2017)

14.4.2.6 Mud Pond Apron Quartz Diopside (Zone 43)

The Mud Pond Quartz Diopside zone has a mean grade and distribution that is like Mud Pond Apron but has extreme outliers and fewer assays (139).

Figure 14.10: Mud Pond Quartz Diopside Zinc Assays

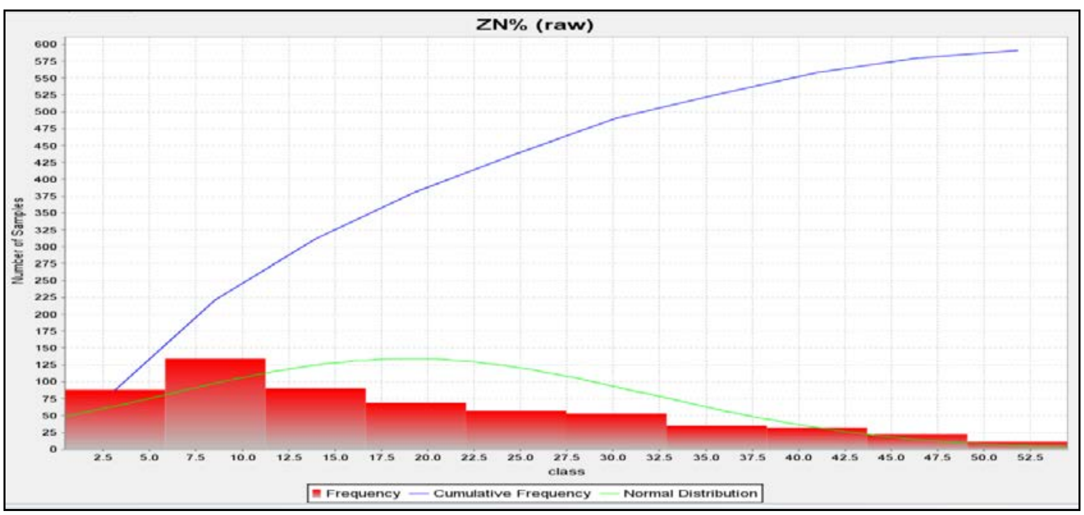


Source: Tuun (2017)

14.4.2.7 Mahler Main (Zone 50)

The Mahler Main zone is one of the largest and best-drilled zones on the property. As can be seen on Figure 14-9, the zinc assays (591 samples) have a more normal distribution than seen with some of the other zones. The variability and long tail were indicated by the basic statistics.

Figure 14.11: Mahler Main Zinc Assays

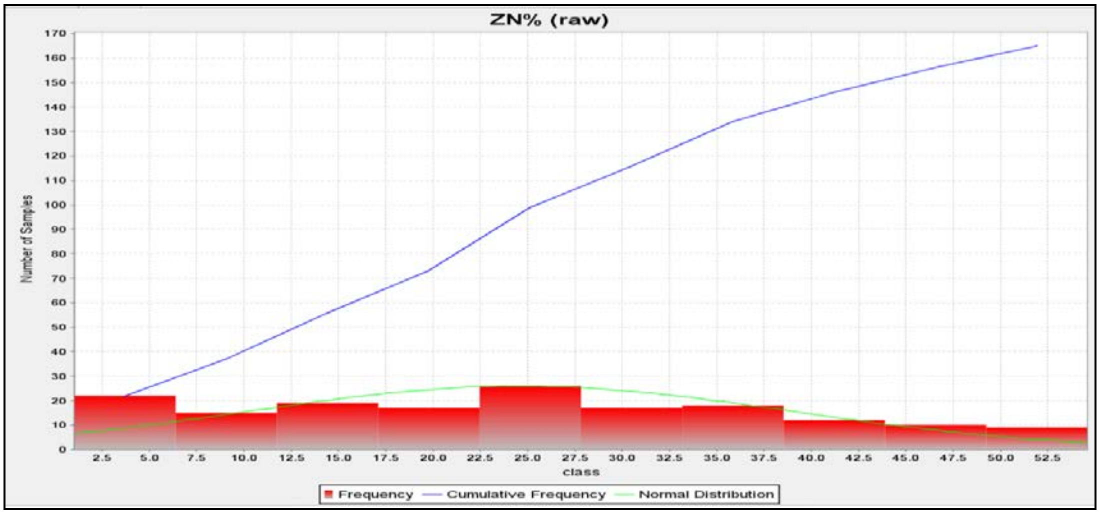


Source: Tuun (2017)

14.4.2.8 Mahler White Dolomite (Zone 51)

The Mahler White Dolomite has a broad range of values as evidenced by the calculated variance of >200.

Figure 14.12: Mahler White Dolomite Zinc Assays

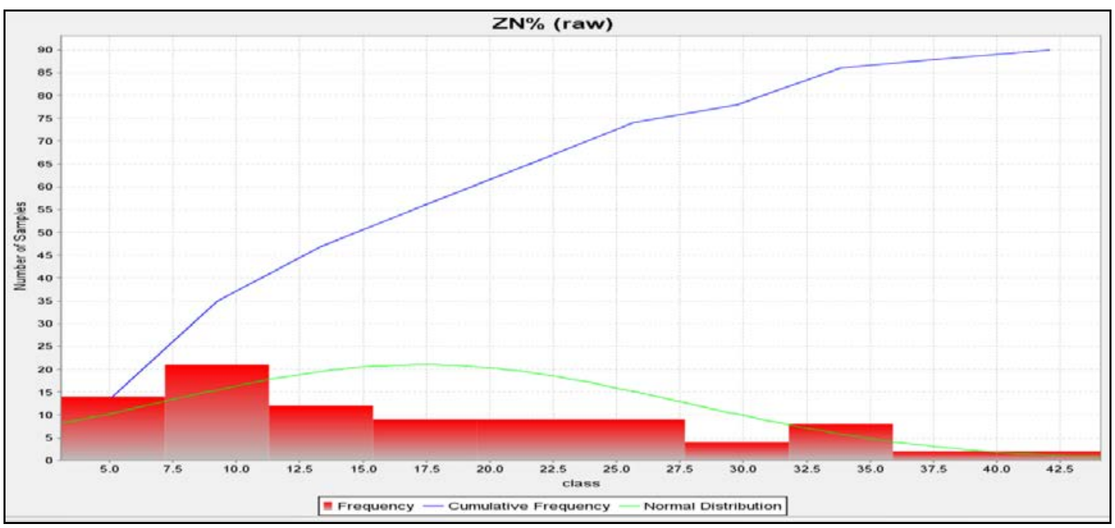


Source: Tuun (2017)

14.4.2.9 Mahler Quartz Diopside (Zone 52)

The Mahler quartz diopside shows the same tendency to erratic high outliers seen with most of the other zones. The secondary population may be real and could be examined for possible future modelling.

Figure 14.13: Mahler Quartz Diopside Zinc Assays

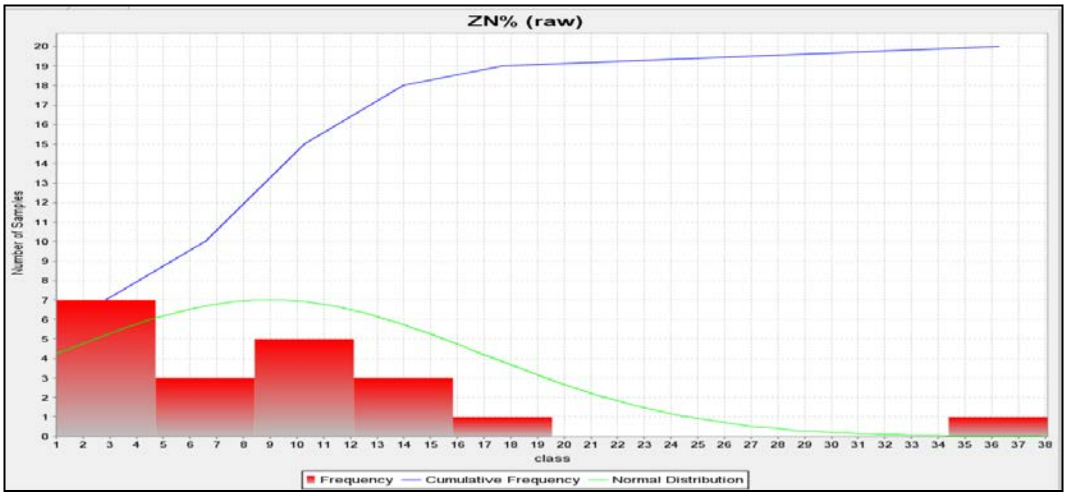


Source: Tuun (2017)

14.4.2.10 NE Fowler (Zone 60)

The remnant resource at the historic NE Fowler zone is poorly delineated with only 20 samples to define the zinc grade. Also in evidence is one extreme outlier which was highlighted by the kurtosis value of >9, and bi-modality.

Figure 14.14: NE Fowler Zinc Assays

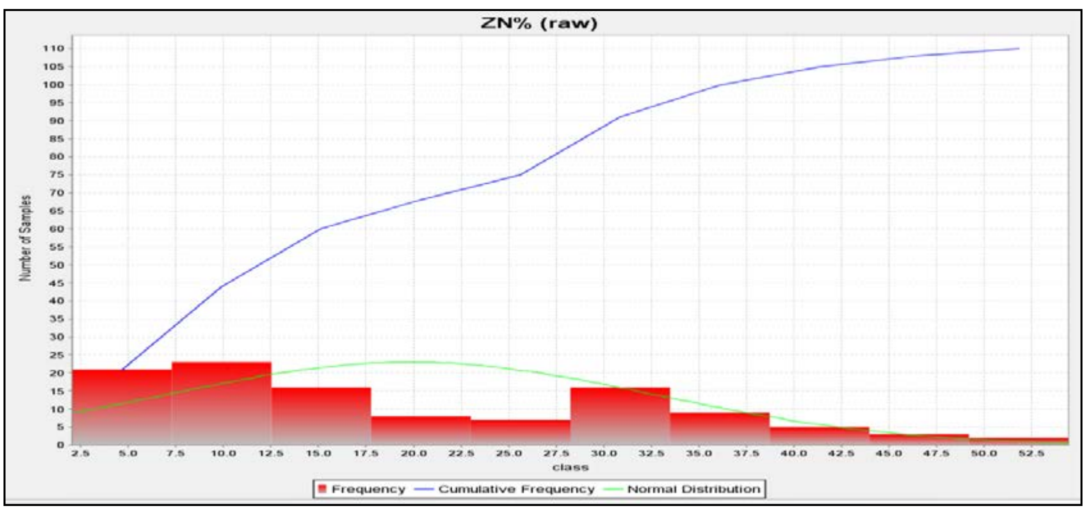


Source: Tuun (2017)

14.4.2.11 New Fold (Zone 70)

The New Fold zone shows a more distinctive bi-modal histogram that may be an artifact of the selective assaying, or a second population not properly identified. The high variance value reflects the very broad range of values.

Figure 14.15: New Fold Basic Assay Statistics



Source: Tuun (2017)

14.4.2.12 Summary of Raw Assay Statistical Analysis

The highly variable sample lengths and extreme outlier Zn% grades strongly suggest that compositing and capping is required for the zones within this deposit. An inspection of the dataset reveals that there are very short intervals with high zinc assays (e.g. hole 1701-F is 15%Zn over 0.1ft) while others have high grades spread over very long intervals (e.g. 1734-F is 27.7%Zn over 40.5ft.)

In some zones, a second high grade population has been noted, but the QP is uncertain if this is an artifact of the sampling methodology or if there are true population(s) that could be modelled with more assay information. The local geology describes “parent-daughter” mineralization and this histogram feature may be a graphic representation of that mineralization event.

14.5 Compositing

GEMS composite statistics are calculated by the using the ‘hole vs wireframe intercept’ to control which assays are to be composited. All implicit and explicit missing sample intervals within the wireframes are considered as zero grade. As previously noted, historic sampling by the geologists and assaying by the on-site laboratory confirmed the intervals as barren internal dilution.

14.5.1.1 Composite Statistical Analysis

The block model size chosen by SLZ is 15 ft x 15 ft x 15 ft which is based on the existing equipment fleet and historical mining widths. The QP, reviewed both 7.5’ and 5’ composite lengths and decided 5 ft. composite lengths should be used.It is believed that the shorter interval does better represent the internal dilution resulting from the implicit and explicit missing intervals being set to zero grade. In addition, the extra composites help improve resolution during the estimation phase. The minimum acceptable length for a composite was set at 50% or 2.5 ft. This meant that shorter intervals would not be created, thus minimizing a volume-variance problem.

The GEMS methodology chosen for compositing was done by cross-table intercepts at a 5 ft composite interval but by equalizing the interval length based on the intercept from the wireframe. A simple example of this method is that a 12 foot solids intercept would have created three 4 ft intervals instead of the simpler method which would have created two 5 ft lengths and ignored the final 2 ft interval.

The raw zinc% assays were composited without capping as a check on the impact to the outlier values. If the ‘second population’ (massive sulphide vs disseminated stringers?) is moderated, then the possibility exists that it was a relic of the assay sampling process and no further restrictions need to be applied to the estimation process.

If the uncapped composite statistics still show skewness and high kurtosis values, then a cap will be applied on the composited assays. Further work would be needed to ascertain whether internal high grade zones could be modelled.

14.5.1.2 Summary Observations of All Zinc Composites

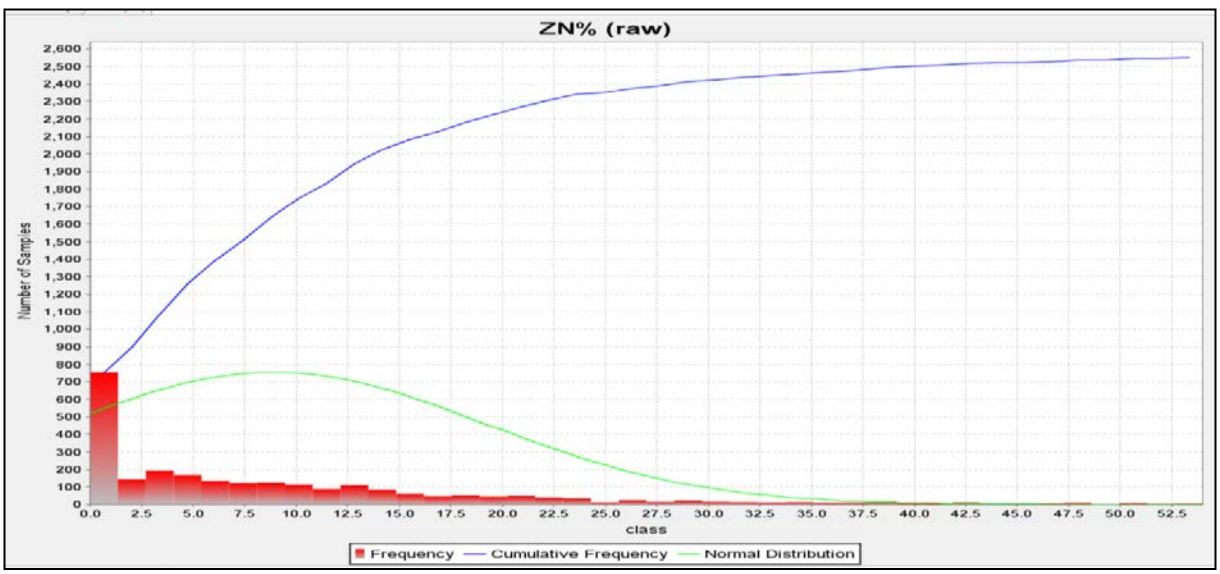
As a high-level look at the tenor of the mineralization, all the zone composites were combined to create an overview histogram and distribution/cumulative frequency curves. For simplicity, the Table 14.7 and Figure 14.16 summarize all composited zinc values.

Table 14.7: Statistics of All Zinc Composites within Mineral Zones

Variable	Zinc% (Comp)
Number	3,438
Minimum	0
Maximum	57.82
Mean	9.59
Median	6.44
Variance	116.22
Std. Deviation	10.78
Coeff. Of Variation	1.12
Skewness	1.56
Kurtosis	5.39

Source: Tuun (2017)

Figure 14.16: Distribution of all Zn% Composites



Source: Tuun (2017)

The long tail and high variance is very similar to what was seen on a zone by zone basis in the raw assay data.

14.5.2 Control of High Grade Zinc% Outliers

The QP considered four ways to treat the "outliers" or very high grade samples identified during the statistical analysis:

- Apply a cap to the raw assay grade;
- Composite the assays and apply a cap;
- Composite the assays and do not cap;
- Composite the assays, use a cap and limit the influence of outliers.

Given the unusual sample intervals, variable sample lengths and a few extreme outlier Zn% values encountered, the QP selected option #2: composite the assay values and then during block modelling apply an outlier cap at the 95th percentile zinc value.

The GEMS calculated statistics for each of the zones is summarized in Table 14.8 along with the selected cap limit for block estimation.

Table 14.8: Statistics of Zinc Composites by Mineral Zone

ZONE=>	Units	Davis	CalMar	Sylvia	MP	MPA	MPQD	Mahler	MAWD	MAQD	NEF	NF
Number	#	35	34	112	1113	459	221	911	240	133	12	168
Min	%Zn	0	0	0	0	0	0	0	0	0	0	0
Max	%Zn	14.2	23.4	26	44.56	57.82	39.38	54	54.2	42.5	30.36	54.45
Mean	%Zn	2.92	10.66	3.8	6.26	8.02	8.4	12.36	20.63	10.75	9.56	10.94
Variance	%Zn	13.7	34.52	45.89	57.76	80.07	76.59	146.33	228.91	103.88	75.55	115.83
St. Dev.	%Zn	3.7	5.88	6.78	7.6	8.95	8.75	12.1	15.13	10.19	8.69	10.76
Coeff. Of Var.	%Zn	1.26	0.55	1.12	1.21	1.12	1.04	0.98	0.73	0.95	0.91	0.98
Skewness	%Zn	1.4	0.48	1.18	1.28	1.73	1.62	1.22	0.56	1.33	1.38	1.35
Kurtosis	%Zn	4.34	2.71	3.55	4.16	7.07	5.48	3.82	2.26	4.63	3.47	4.88
95th Pctile	%Zn	10.63	22.1	22.2	21.7	26.34	30.1	32.3	49.4	31.52	27.04	33.3
97.5th Pctile	%Zn	13.4	22.75	22.3	24.3	31.92	36	39.02	51.78	42.5	30.36	39.49

Source: Tuun (2017)

While the high grades have been limited, it is worthy of mention that compositing with zero Zn% grades in the un-sampled zones also has a negative impact. One can assume that internal dilution would be elevated.

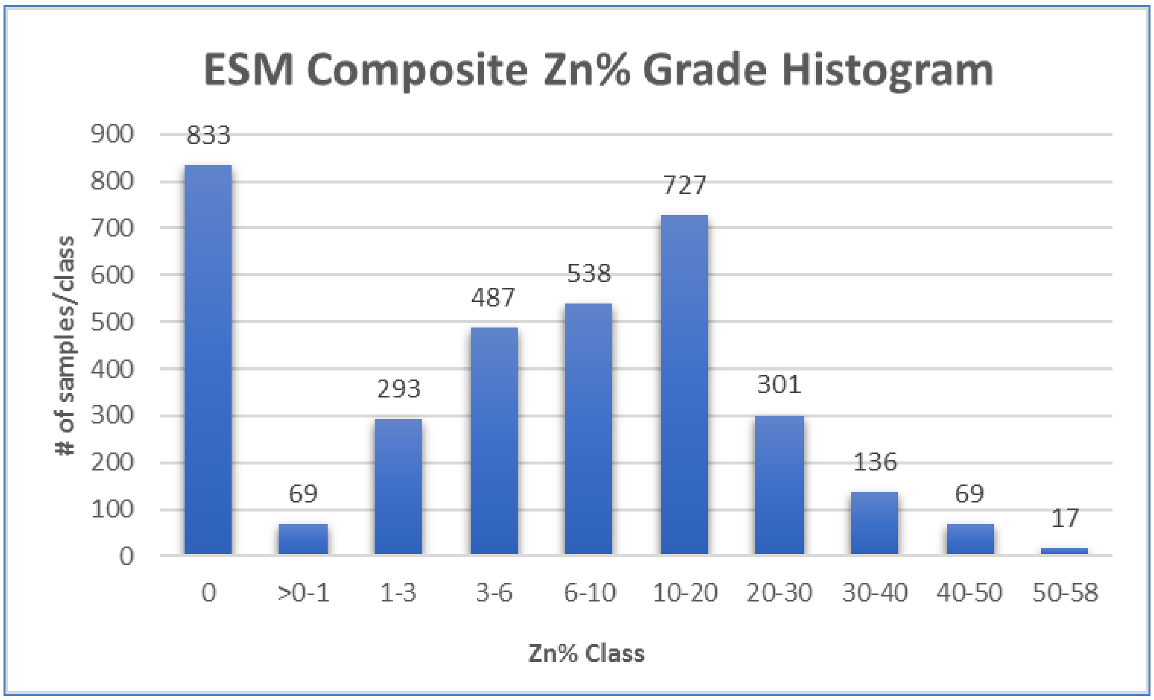
The impact of the assumed zero-grade values can be inferred from Table 14.9 and Figure 14.17.

Table 14.9: Zinc Composites by Class

Zinc % Class	Number of Samples	N%
0	833	24
0-1	69	2
3-6	487	14
6-10	538	16
10-20	727	21
20-30	301	9
30-40	136	4
40-50	69	2
50-58	17	0

Source: Tuun (2017)

Figure 14.17: Histogram of Zn% Composites



Source: Tuun (2017)

It is likely that some proportion of the 'zero' values within the >0 to 3% class, with the outcome that more material might lie above cut-off than expected.

The variability of the mineralized horizon thickness as seen during the underground tour makes it prudent to not over-estimate the extreme highs but it was also recognized that the un-sampled intervals may carry some grade.

Conversion of assumed zero grade by sampling those intervals is highly recommended.

While it is recognized that the methodology of assuming zero grade for implicit and explicit missing intervals is very conservative, it is anticipated to produce diluted zinc grades that may be similar to historically mined grades and will be reflective of remaining zinc resources.

14.6 Specific Gravity

Historically the mine had assumed an mineralized material density of 0.100 t/ft^3 or ~3.20. In 2005, a series of tests began to substantiate that belief. The analytical method used was the 'Archimedes Method' or weight-in-air/weight-in-water.

A collection of 128 samples yielded a regression curve which was then used to estimate SG based on the zinc assay. A possible flaw in that calculation was that the skewed sampling meant that the extreme zinc% outliers may have biased the calculated density and thus over-estimated tonnage.

Site personnel continued taking samples for SG and modified the regression curve (with a total of 157 samples) to incorporate gangue minerals (5%-calcite; 40%-diopside; 40%-dolomite; and 15%-quartz). The QP does not believe that the modification was warranted.

The database now totals 308 samples, of which 19 are waste or the zone code was not entered (mean SG 3.01). Table 14.10 summarizes the samples by their zone which was determined by the site geologists.

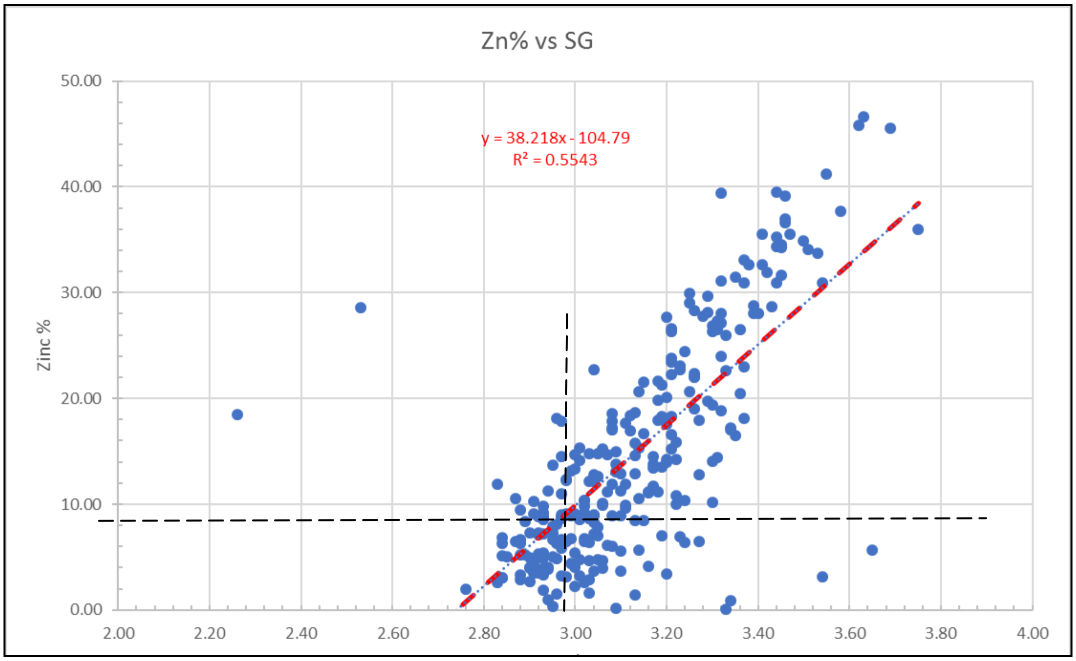
An updated regression curve for the current data is shown in Figure 14.18.

Table 14.10: Specific Gravity Tests

Zone Name	Zone	# Of SG Tests	Mean SG	Density (t/ft^3)
Davis	10	0	NC	NC
Cal Marble	20	0	NC	NC
Cal Upper	21	0	NC	NC
Sylvia Lake	30	0	NC	NC
Mud Pond Main	40	11	3.159	0.0986
Mud Pond Apron	41	84	3.144	0.0981
Mud Pond Apron Quartz Diopside	43	11	3.307	0.1032
Mahler Main	50	98	3.073	0.0959
Mahler White Dolomite	51	27	3.065	0.0956
Mahler Quartz Diopside	52	34	3.061	0.0955
NE Fowler	60	23	3.137	0.0979
New Fold	70	1	3.26	0.1017
TOTAL		**289**	**3.123**	**0.0975**

Source: Tuun (2017)

Figure 14.18: SG vs Zn% Scatter Plot and Regression Curve



Source: Tuun (2017)

If the above curve is a reasonable fit and representative of the historic mining, then the mined grade (33.8 Mt of 8.6% Zn since 1930) might have had an average SG of ~2.95 (intersection of the dashed lines).

Also of note on Table 14.11, the average SG is essentially the same for Mahler Main, Mahler WD and Mahler QD which tends to support the QP's opinion that modification of the SG curve for gangue minerals composition is not required.

Because the zinc assay statistical analysis exposed significant outliers and a potentially biased sampling technique, the QP did not use the regression curve to calculate density from zinc assays. Instead, the densities summarized in Table 14.11 were assigned to each of the wireframes. Where there was no data, the mean SG of 3.123 (0.0975 t/ft^3) was used.

Table 14.11: Density used for Resource Wireframes

Zone Name	Zone	Mean SG	Mean Density (t/ft^3)
Davis	10	3.123	0.0975
Cal Marble	20	3.123	0.0975
Cal Upper	21	3.123	0.0975
Sylvia Lake	30	3.123	0.0975
Mud Pond Main	40	3.159	0.0986
Mud Pond Apron	41	3.144	0.0981
Mud Pond Apron Qtz-Diop	43	3.307	0.1032
Mahler Main	50	3.073	0.0959
Mahler White Dolomite	51	3.065	0.0956
Mahler Quartz Diopside	52	3.061	0.0955
NE Fowler	60	3.137	0.0979
New Fold	70	3.123	0.0975
Waste (background)	900	2.8	0.0874

Source: Tuun (2017)

The QP believes that the current level of SG testing is adequate for this Resource Estimate, but would recommend that testing of all zones be continued.

14.7 Geostatistical Analysis and Variography

Mineral deposits often have spatial variability that tends to be strongest in one direction. This is termed anisotropy and samples in this direction have lower variability than samples in other directions. A semi-variogram is a graph used to show this variability.

The horizontal axis of the semi-variogram shows the distance between pairs of samples being compared while the vertical axis shows the variability (half of the variance) of the samples at specific distances (lag intervals).

The semi-variogram model consists of four key parts: the nugget, sill, range and model type. The nugget (C0) describes the variability at very short distances and could be a result of emplacement processes; differences in the sampling and assaying techniques; or perhaps contamination.

The sill is the point at which the curve approaches a constant value, and the distance that point is reached is called the range. The type of models that can be used to fit the data are commonly the Spherical, Exponential and Gaussian models.

Spatial continuity of all eleven mineralized zones suitable for the resource estimate was attempted with normalized variograms using Geovia GEMS™ Version 6.7.1 software. The anisotropy was assessed using Azimuth, Dip, and Azimuth (ADA) rotation. For the majority of the wireframes there was minimal composite data which resulted in not being able to generate robust semi-variograms.

To maintain estimation and reporting consistency, Tuun opted to use the Inverse Distance Squared (IDS or ID2) method for grade modelling of all mineral zones.

14.8 Block Model Definition

Mining operations used a block model size of 15 ft x 15 ft x 15 ft, so Tuun maintained that size for the estimation. The block model origin coordinates, block size and rotation are summarized in Table 14.12.

Table 14.12: Block Model Origin and Rotation

Origin	Block Size (ft)	# of Blocks
12,750 E	15	630
7,425 N	15	745
-925 El (max)	15	200
Rotation	-25	

Source: Tuun (2017)

Given the true thickness of the mineral zones observed, future block models at a 5 ft level thickness might be considered, particularly if full interval assaying is conducted at tighter sample lengths throughout the zones.

14.9 Grade Estimation

Block model grades were estimated in four passes using the IDS method. Models for the Nearest Neighbour (NN) and the Mean Value of Composites Used (MVCU) were also created. The NN and MVCU block models were used for comparative and validation purposes.

The classification methodology used was that blocks meeting the criteria for:

- Pass 1 needs to use three holes to be flagged as Measured;
- Pass 2 – Indicated;
- Pass 3 – Inferred; and
- Pass 4 – Target For Future Exploration.

The three classification passes used the minimum and maximum samples and searches as summarized in Table 14-11. Search ellipses were based on preliminary geostatistics and discussion with SLZ geologists.

For grade estimation, the search ellipses were rotated to align with each domain. The variograms were fitted using the GEMS “Azimuth-Dip-Azimuth” rotation method. The methodology to set up this rotation is outlined as follows:

- The first axis rotation (“AZ”) represents the true azimuth of the anisotropy X axis (Principal Azimuth – true strike);
- The second rotation (“DIP”) represents the dip angle of the anisotropy X axis (Principal Dip – negative downwards); and
- The third rotation (“AZ”) represents the azimuth of the anisotropy Y axis (Intermediate Azimuth).

Pass #4 blocks would have the block model 'Class' value of four (4) and would be used only as a guide to determining any "Targets For Future Exploration" (TFFE). The pass search was variable and impacted only four of the zones: Sylvia Lake, Mahler Main, NE Fowler, and New Fold.

Table 14.13: Search Ellipse Parameters

	Search Pass #	Orientation			Distances			Composites		Maximum
		AZ	DIP	AZ	X (ft)	Y (ft)	Z (ft)	Min	Max	Comps/Hole
Davis	P1	0	-10	0	100	50	100	7	15	3
	P2	0	-10	0	150	100	150	5	15	2
	P3	0	-10	0	300	150	300	1	15	2
Cal Marble	P1	0	-20	0	100	50	100	7	15	3
	P2	0	-20	0	150	100	150	5	15	2
	P3	0	-20	0	350	250	350	1	15	2
Sylvia Lake	P1	0	-20	0	100	50	100	7	15	3
	P2	0	-20	0	150	100	150	5	15	2
	P3	0	-20	0	300	150	300	1	15	2
Mud Pond	P1	37	-15	0	100	75	100	7	15	3
	P2	37	-15	0	150	100	150	5	15	2
	P3	37	-15	0	450	150	450	1	15	2
Mud Pond Apron	P1	60	-5	250	75	30	75	7	15	3
	P2	60	-5	250	150	50	150	5	15	2
	P3	60	-5	250	300	100	300	1	15	2
Mud Pond QD	P1	60	-5	250	75	30	75	7	15	3
	P2	60	-5	250	150	50	150	5	15	2
	P3	60	-5	250	300	100	300	1	15	2
Mahler Main	P1	50	-15	230	100	75	100	7	15	3
	P2	50	-15	230	250	150	250	5	15	2
	P3	50	-15	230	400	250	400	1	15	2
Mahler White Dolo.	P1	50	-15	230	100	50	100	7	15	3
	P2	50	-15	230	150	75	150	5	15	2
	P3	50	-15	230	300	150	300	1	15	2
Mahler Quartz Diop.	P1	50	-15	230	75	50	75	7	15	3
	P2	50	-15	230	150	75	150	5	15	2
	P3	50	-15	230	300	100	300	1	15	2
NE Fowler	P1	25	-50	0	75	50	75	7	15	3
	P2	25	-50	0	150	100	150	5	15	2
	P3	25	-50	0	300	200	300	1	15	2
New Fold	P1	50	-5	230	75	50	75	7	15	3
	P2	50	-5	230	150	75	150	5	15	2
	P3	50	-5	230	300	125	300	1	15	2

Source: Tuun (2017)

14.10 Model Validation and Sensitivity

The grade models were visually validated by comparing the blocks estimated by the various techniques with actual drill hole composite data on both section and in plan view. Table 14.19 shows the colour legend.

Figure 14.19: Legend for Zinc% Values

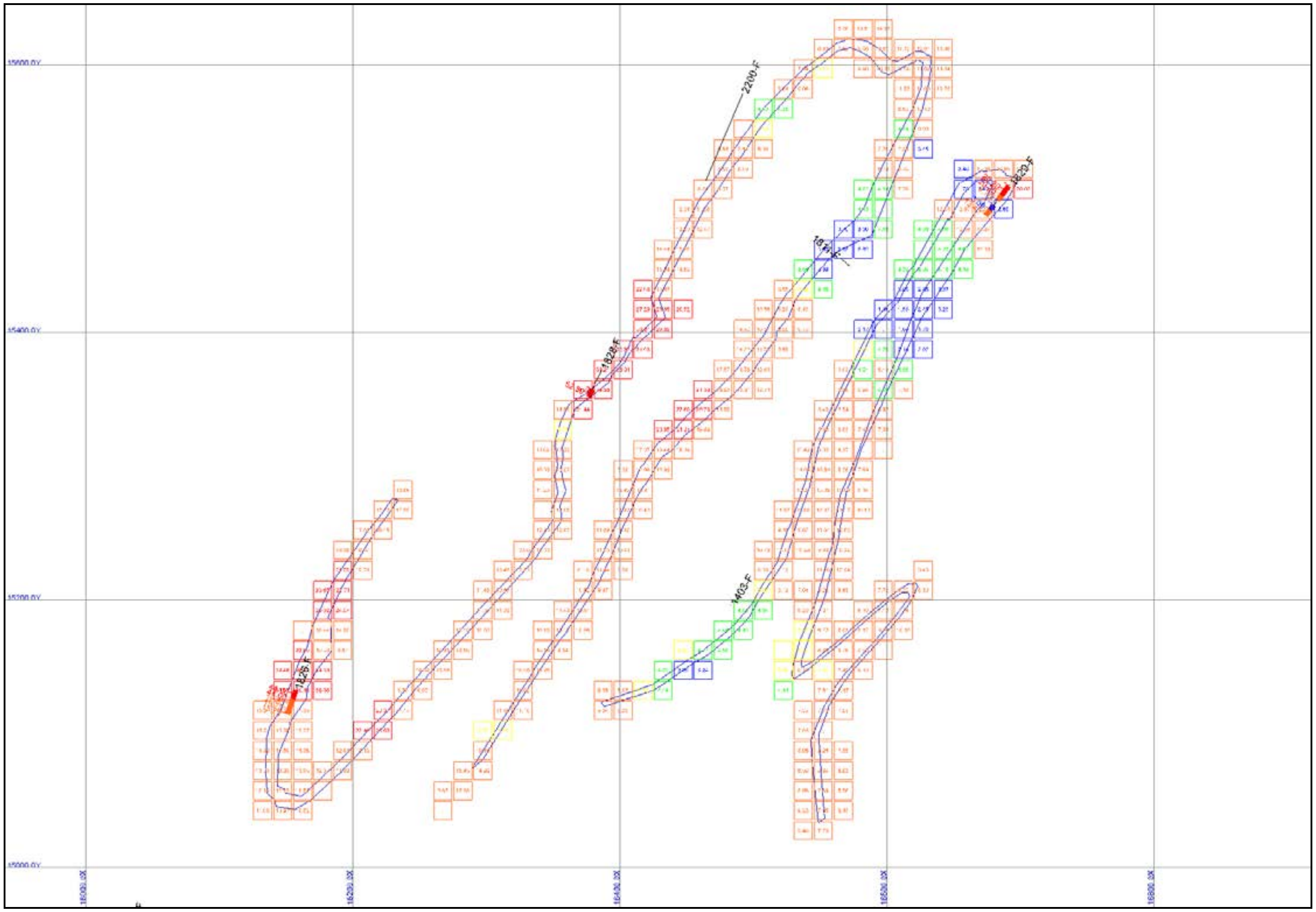
>= Lower Bound	< Upper Bound	Colour
0.00500	4.00000	RGB 0 0 255
4.00000	5.00000	RGB 0 255 0
5.00000	6.00000	RGB 255 255
6.00000	20.00000	RGB 255 104
20.00000	100.00000	RGB 255 0 0

Source: Tuun (2017)

The following Figures 14.20 to 14.22 show level plans which represent the blocks and drill hole composites for the largest mineral zones: Mahler Main and Mud Pond Apron. Also included is a plan showing Mahler Main and the adjacent Mahler WD and Mahler QD mineral zones.

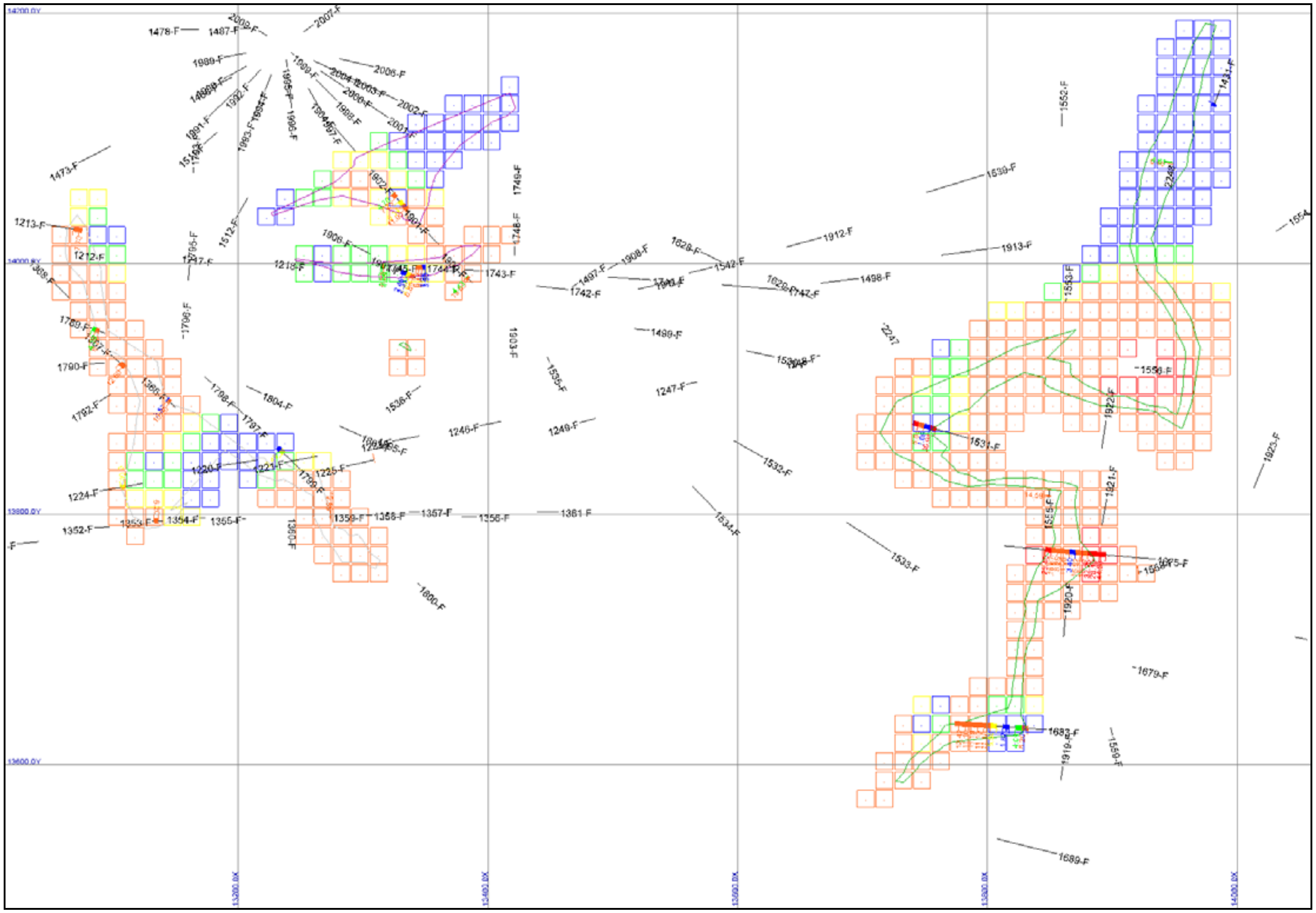
All level plans have a window width of +/- 7.5 ft to match the block model.

Figure 14.20: Mahler Main Zinc Grades (Level -2070)



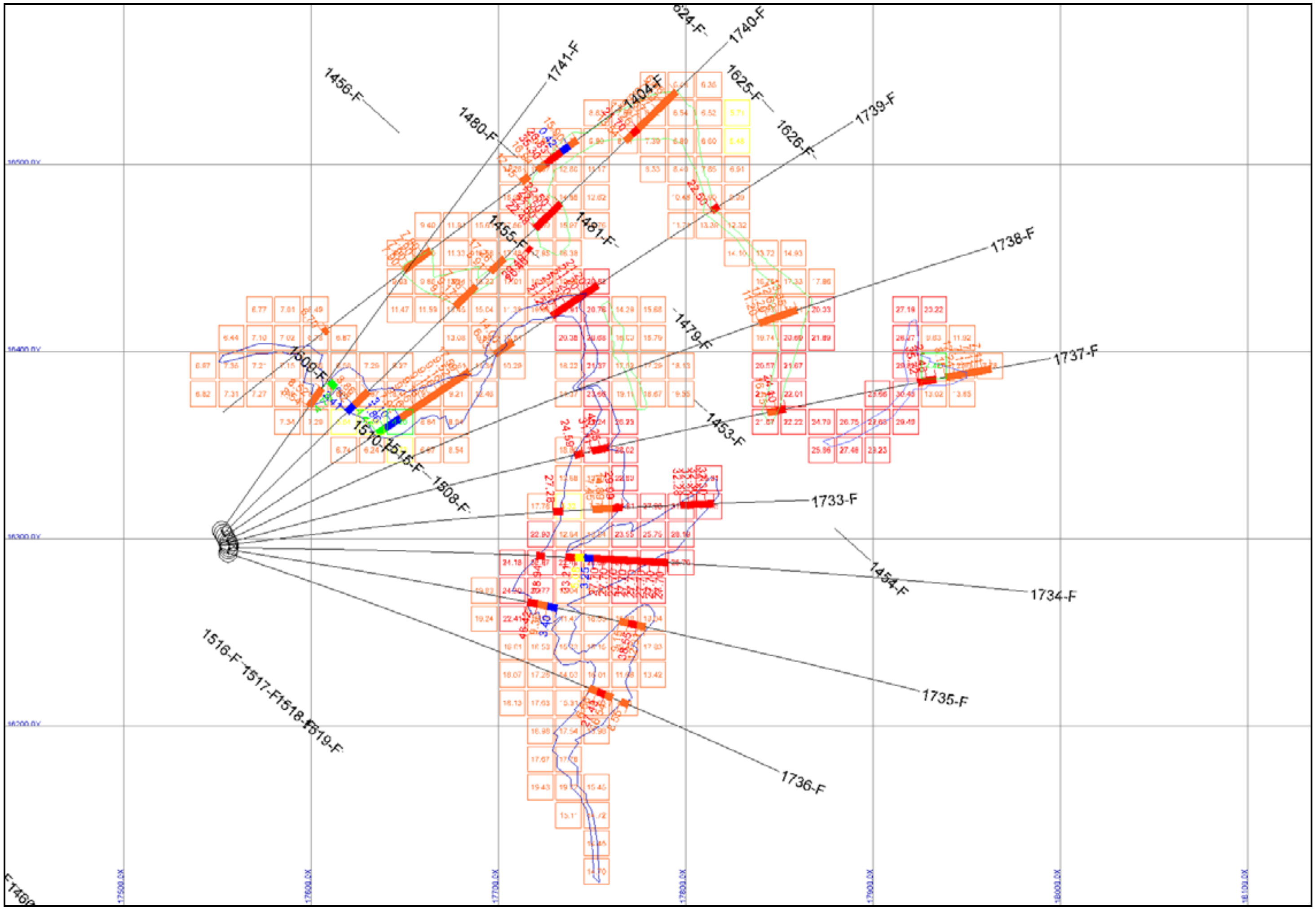
Source: Tuun (2017)

Figure 14.21: Mud Pond Main Block Grades (Level -2025)



Source: Tuun (2017)

Figure 14.22: Mahler Main, Mahler QD and Mahler WD Block Grades (Level -2835EL)



Source: Tuun (2017)

The Nearest Neighbour (NN) model and Mean Value of Composites Used (MVCU) models were generated at a 0% zinc cut-off for comparison to the IDS model. Table 14.13 shows the estimates.

Table 14.14: Comparison of Estimation Methods

Method	Tons	Zinc %
NN	6,185,700	10.95
IDS	6,185,700	10.51
MVCU	6,185,700	10.55

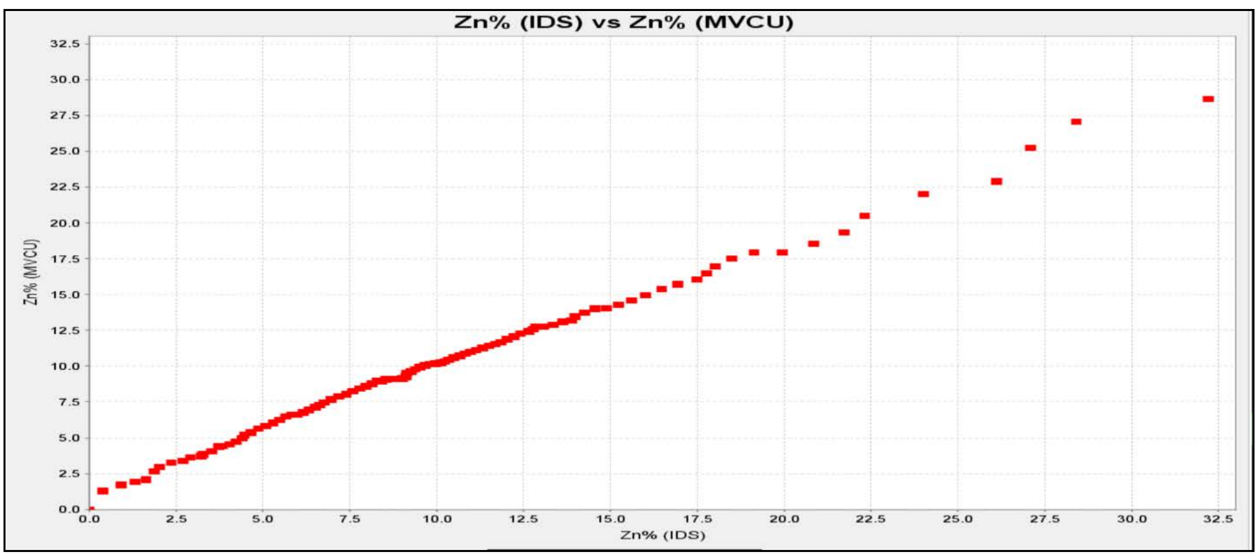
Source: Tuun (2017)

The Nearest Neighbour model represents a slightly biased estimate due to local outliers. The MCVU does not take into account any form of weighting but is reasonable when a large number of samples are available within the block. Tuun believes that overall, the IDS method was appropriate for the PEA resource estimation.

Tuun also created a Q-Q Plot of the IDS model estimates versus the "well-informed" block composite grades (MVCU) as a cross-check. The well-informed blocks are the arithmetic mean of all the composites used to estimate the block grade.

In the deposit (Figure 14-20) the block estimate comparison of the composites is very similar (0.88 correlation). Overall the QQ-Plot shows that the estimate supports the visual inspection of the blocks presented in the previous section.

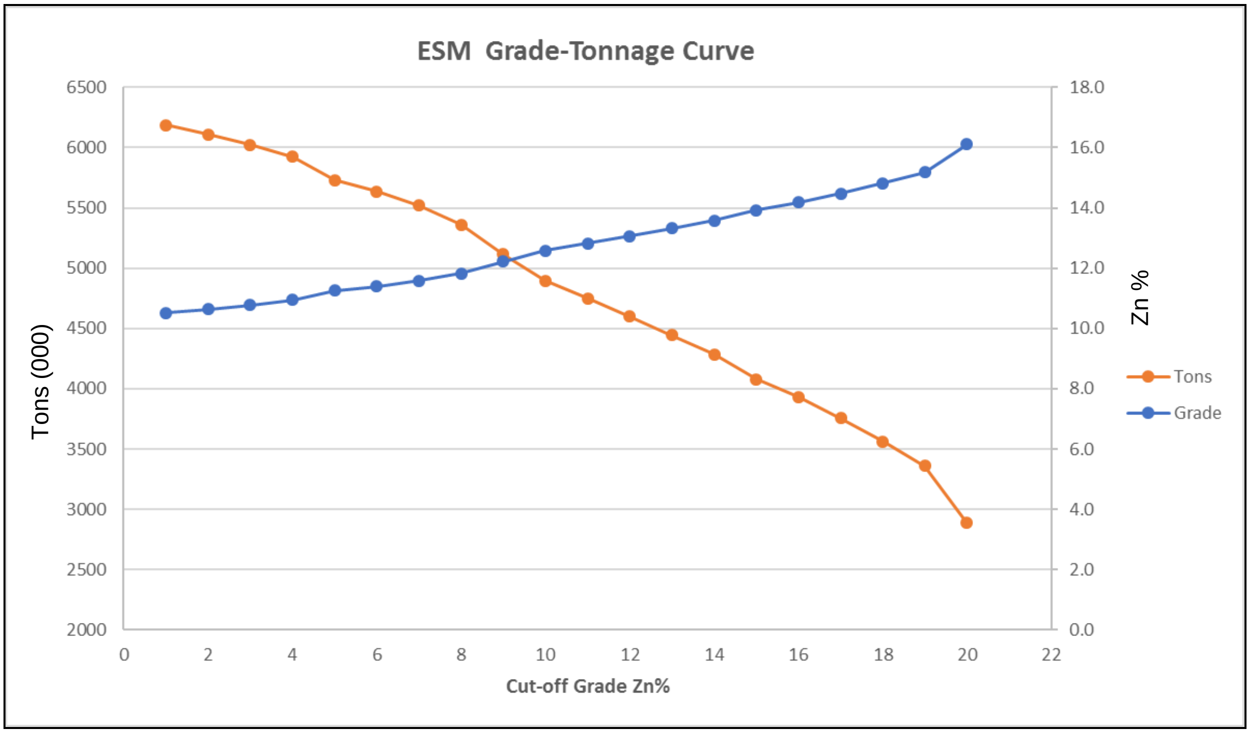
Figure 14.23: QQ-Plot of IDS estimates versus Mean Composite Grade



Source: Tuun (2017)

As a final check, a grade-tonnage curve was generated to assess the estimates (Figure 14.24).

Figure 14.24: Grade-Tonnage Curve



Source: Tuun (2017)

All indications are that the Inverse Distance Squared (IDS) estimation methodology is a good fit, particularly within the historically mined grade range of 6 to 12% zinc.

14.11 Mineral Resource Classification

The ESM zinc deposit block model quantities and grade estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves. The grade estimation was done by Mr. Allan Reeves P.Geo. of Tuun Consulting Inc. (Tuun).

This Mineral Resource classification considered the geological continuity of the mineralized zones and the quality and quantity of exploration data supporting the estimates. The effective date of the Mineral Resource statement is April 6, 2017.

The estimate follows the guidelines of the generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines” (as adopted on November 23, 2003).

On November 28, 2015 CIM Council adopted a submittal by the Commodity Price Sub-Committee of the CIM Best Practices Committee – “Guidance on Commodity Pricing used in Resource Estimation and Reporting”.

Tuun Consulting is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource estimation.

The mineralization generally exhibits good geological continuity and has been investigated at an adequate spacing with reliable and accurately located sampling information. Tuun considers that blocks estimated during the first estimation pass by at least three drill holes, can be classified in the Measured category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Blocks that were estimated during the second pass were classified as an Indicated category and those in the third pass as Inferred. Tuun believes that the level of confidence is sufficient to allow appropriate application of technical and economic parameters for this PEA.

With respect to the CIM definition of “reasonable prospects of eventual economic extraction”, Tuun considered that the resource had been mined historically but in “care and maintenance” since 2008. Historic grades were examined with respect to resource estimation to provide additional validation and confidence in the modelling technique utilized.

14.12 Mineral Resource Statement

The Mineral Resource statement has been prepared under the CIM Definition Standards for Mineral Resources and Mineral Reserves (adopted by CIM Council on May 10, 2014) which defines:

“Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase ‘reasonable prospects for eventual economic extraction’ implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word ‘eventual’ in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage ‘eventual economic extraction’ as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

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

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

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Prefeasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

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

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

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

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

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

This Mineral Resource is based on drill data, mining contacts, and the guidance of the on-site personnel that created the resource wireframes. The information was reviewed and all work believed to have been executed in a professional manner based on the standards of care at the time.

In Tuun's opinion, the existing sample data is considered to be adequate for estimating the Mineral Resource for the purposes of this PEA. All mineral zones combined are summarized in Table 14.15.

Table 14.15: Empire State Mines – Mineral Resource Estimate

Zn % Cut-off	Tons	Zn%
Measured		
>6.0%	850,100	13.19
Indicated		
>6.0%	1,307,900	13.35
Measured + Indicated		
>6.0%	2,158,000	13.29
Inferred		
	2,276,600	13.37

Notes:
1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
2. The UG mining economics used operating costs of US$70.00/ton and a commodity price of US$1.00/pound at 96% recovery.
3. Mineral resources are reported 'in-situ' using a cut-off grade of 6% Zn to determine 'reasonable prospects for eventual economic extraction'.
4. Tonnages are reported to the nearest 100 tons, and grades are rounded to the nearest two decimal places.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tons, and grade.

Source: Tuun (2017)

Table 14.16: Zn% Measured and Indicated Resources by Zones

	Davis		Cal Marble		Sylvia Lake		Mud Pond		MP-Apron		MP- QD	
	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%
Measured												
>10 %	-	0	-	0	22,300	13.76	108,000	13.23	28,700	14.00	16,600	13.73
>9%	-	0	-	0	25,900	13.16	134,800	12.49	33,400	13.02	24,600	12.31
>8%	-	0	-	0	31,700	12.29	167,200	11.72	35,900	13.02	35,100	11.15
>7%	-	0	-	0	37,600	11.54	195,800	11.1	38,800	12.62	49,300	10.11
>6%	400	6.24	-	0	44,500	10.77	231,400	10.38	43,400	11.98	61,900	9.37
>5%	600	5.26	-	0	52,100	10.00	275,400	9.61	47,600	11.41	72,300	8.81
>4%	800	4.31	-	0	63,200	9.04	311,600	9.01	52,900	10.73	81,800	8.31
>3%	2,700	3.39	-	0	74,100	8.21	353,800	8.36	57,600	10.14	87,700	7.99
Indicated												
>10 %	100	11.64	6,900	12.26	23,500	13.47	93,200	13.54	53,200	13.23	1,400	12.66
>9%	200	10.90	12,300	11.50	27,700	12.87	109,000	12.96	66,000	12.51	2,200	11.51
>8%	200	10.32	19,900	10.74	31,900	12.3	120,600	12.53	83,400	11.68	4,800	9.89
>7%	300	9.68	30,100	9.96	42,400	11.09	134,100	12.02	97,100	11.09	7,300	9.06
>6%	600	8.53	35,600	9.58	47,300	10.62	148,700	11.48	115,800	10.34	9,400	8.43
>5%	1,600	7.03	42,800	9.08	54,300	9.96	164,300	10.91	136,400	9.62	11,300	7.97
>4%	2,500	5.90	45,400	8.88	60,900	9.36	180,000	10.35	157,900	8.92	12,600	7.63
>3%	2,500	5.15	45,900	8.83	65,100	8.99	191,300	9.95	179,700	8.27	13,900	7.21

Notes:
1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
2. The UG mining economics used operating costs of US$70.00/ton and a commodity price of US$1.00/pound at 96% recovery.
3. Mineral resources are reported 'in-situ' using cut-off grade of 6% Zn to determine 'reasonable prospects for eventual economic extraction'.
4. Tonnages are reported to the nearest 100 tons, and grades are rounded to the nearest two decimal places.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tons and grade.

Source: Tuun (2017)

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Table 14.17: Zn% Measured and Indicated Resources by Zones – continued

	Ma-Main		Ma-WD		Ma-QD		NE Fowler		New Fold	
	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%
Measured										
>10%	236,200	17.37	77,000	19.43	5,700	17.12	0	0	48,500	14.56
>9%	257,300	16.73	79,400	19.12	5,900	16.81	0	0	56,200	13.86
>8%	279,400	16.08	80,100	19.03	6,300	16.25	0	0	60,400	13.48
>7%	296,700	15.58	81,900	18.79	6,400	16.19	0	0	63,700	13.18
>6%	311,800	15.14	82,100	18.75	6,600	15.85	0	0	68,000	12.75
>5%	321,900	14.84	82,300	18.72	6,900	15.41	0	0	73,700	12.20
>4%	329,300	14.61	82,700	18.65	7,100	15.03	0	0	75,500	12.02
>3%	332,600	14.50	82,700	18.65	7,300	14.85	0	0	75,800	11.98
Indicated										
>10%	436,800	17.62	68,200	19.65	15,900	13.88	0	0	141,400	14.36
>9%	473,600	16.99	73,000	18.98	19,800	13.02	0	0	178,700	13.34
>8%	512,400	16.35	76,900	18.45	23,500	12.30	0	0	206,400	12.69
>7%	552,600	15.70	79,200	18.13	26,500	11.77	0	0	230,900	12.14
>6%	590,900	15.11	80,300	17.97	29,700	11.21	0	0	249,600	11.72
>5%	625,900	14.57	81,400	17.80	31,500	10.89	0	0	267,200	11.32
>4%	670,600	13.90	82,300	17.67	32,500	10.69	0	0	279,700	11.02
>3%	710,600	13.32	82,400	17.63	32,800	10.62	0	0	288,100	10.8

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
2. The UG mining economics used operating costs of US$70.00/ton and a commodity price of US$1.00/pound at 96% recovery.
3. Mineral resources are reported ‘in-situ’ using an incremental cut-off grade of 3% Zn to determine ‘reasonable prospects for eventual economic extraction’.
4. Tonnages are reported to the nearest 100 tons, and grades are rounded to the nearest two decimal places.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tons and grade.

Source: Tuun (2017)

Table 14.18: Zn% Inferred Resources by Zones

	Davis		Cal Marble		Sylvia Lake		MP-Main		MP-Apron		MP-QD	
	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%
>10%	0	0	252,100	13.57	26,500	14.6	216,700	21.72	6,100	13.65	0	0
>9%	0	0	324,800	12.66	33,800	13.5	257,700	12.21	8,100	12.6	0	0
>8%	0	0	398,400	11.9	36,700	13.11	297,100	11.72	12,500	11.11	0	0
>7%	0	0	428,400	11.6	37,200	13.03	332,900	11.27	18,000	10.01	0	0
>6%	200	8.1	440,200	11.46	38,200	12.86	369,300	10.8	23,600	9.18	0	0
>5%	300	5.37	446,200	11.38	39,400	12.64	407,800	10.3	27,900	8.58	0	0
>4%	1500	4.34	447,500	11.36	40,900	12.35	449,200	9.76	34,100	7.85	0	0
>3%	9100	3.56	448,500	11.34	42,200	12.08	473,900	9.45	37,500	7.47	0	0

	Mahler Main		Ma-WD		Ma-QD		NE Fowler		New Fold	
	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%	Tons	Zn%
>10%	212,200	15.29	175,600	21.34	2,000	11.54	243,600	17.64	364,400	16.62
>9%	251,700	14.35	176,000	21.27	5,500	10.23	274,600	16.73	439,800	15.4
>8%	280,500	13.75	177,600	21.2	6,700	9.96	278,800	16.61	482,100	14.79
>7%	306,700	13.22	178,600	21.12	6,800	9.92	280,800	16.55	511,200	14.38
>6%	329,100	12.76	180,700	20.95	6,800	9.92	348,500	14.61	539,400	13.97
>5%	344,700	12.44	182100	20.83	6,800	9.92	367,600	14.14	570,800	13.5
>4%	356,600	12.17	184,200	20.64	6,800	9.92	458,200	12.22	601,100	23.04
>3%	416,800	10.89	185,000	20.57	6,800	9.92	480,100	11.82	724,400	11.46

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
2. The UG mining economics used operating costs of US$70.00/ton and a commodity price of US$1.00/pound at 96% recovery.
3. Mineral resources are reported 'in-situ' using a cut-off grade of 6% Zn to determine 'reasonable prospects for eventual economic extraction'.
4. Tonnages are reported to the nearest 100 tons, and grades are rounded to the nearest two decimal places.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tons and grade.

Source: Tuun (2017)

15 Mineral Reserve Estimates

15.1 Mineral Reserve Non-Compliance

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 tons and/or mill feed).

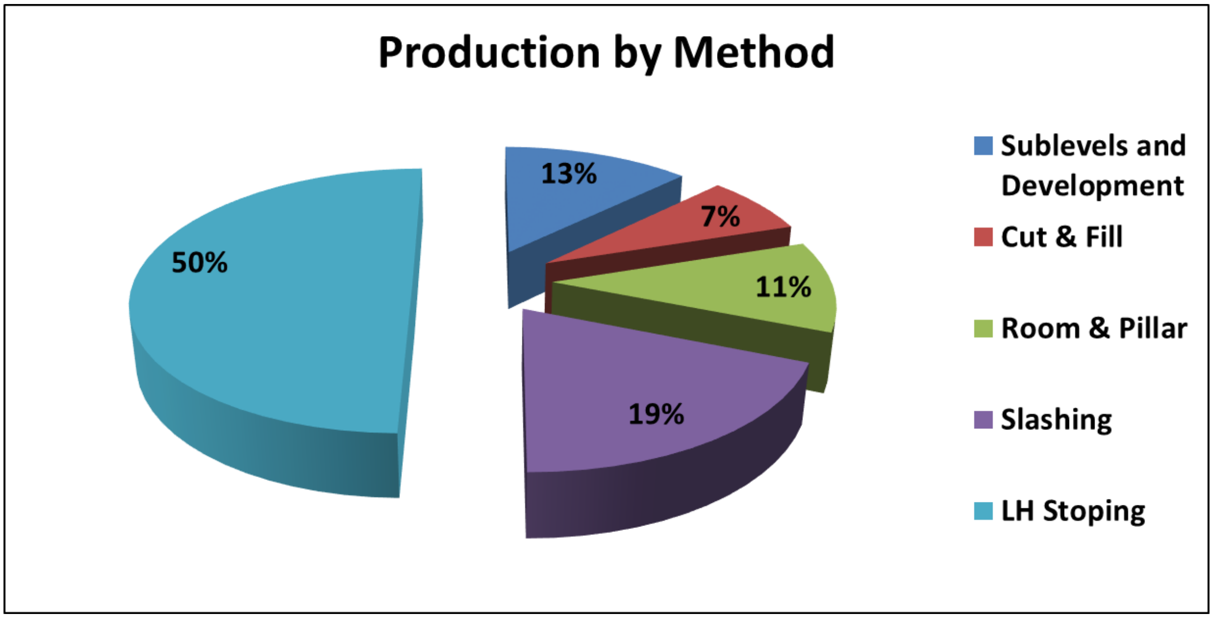
Mine plan tons were derived from the resource model described in the previous section. Measured, indicated and Inferred resources were used to establish mine plan tons.

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

16 Mining Methods

The mine plan tons at the ESM deposit will be extracted using a combination of longhole stoping (LH), cut and fill (C&F), slashing (SLS), room & pillar (PLR), and development (SLO) underground mining methods with rock backfill. The proposed mine plan is expected to reach an initial target production rate of 800 tons per day (t/d) and ramp up to 1,800 t/d. The overall mine life will be eight years. Figure 16.1 below outlines a summary of mine method use at ESM.

Figure 16.1: Mine Production by Method

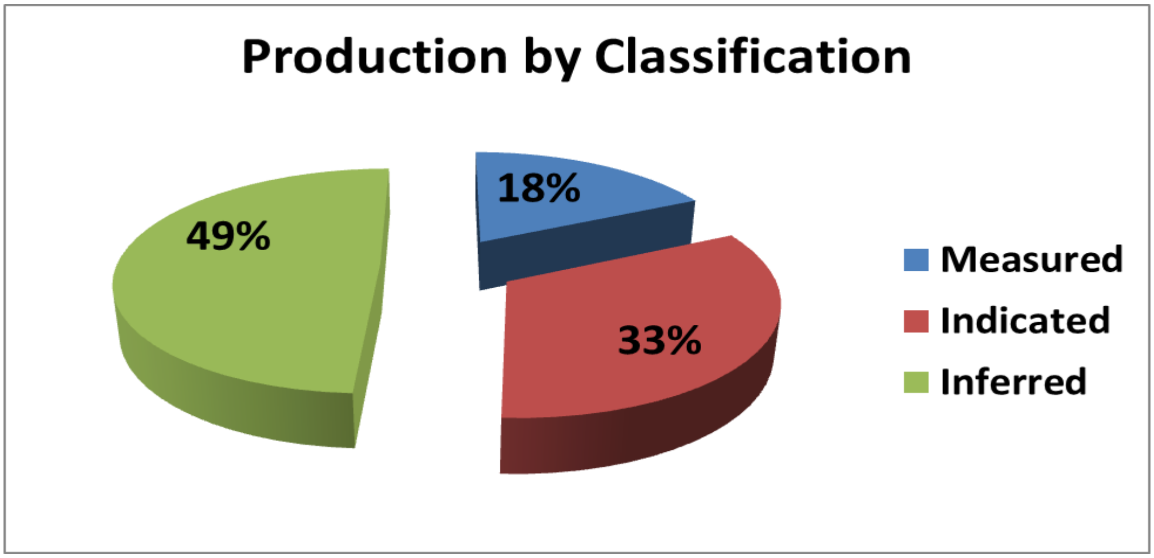


Source: JDS (2017)

The ESM deposit will be accessed from surface via the number 4 shaft, and all mineralized material and some waste rock will be hoisted out of the mine via that shaft. In addition to the existing development and raises, new lateral development and ramping will be required to access mineralized zones. To supplement the ventilation provided by the raises, as the ramps are being driven, shorter internal ventilation drop raises will ensure air delivery to the active development face.

Measured, Indicated and Inferred Mineral Resources were included in the mine design and schedule optimization process. The PEA LOM plan tons per mineralization classification is shown in Figure 16.2 below.

Figure 16.2: Production by Classification



Source: JDS (2017)

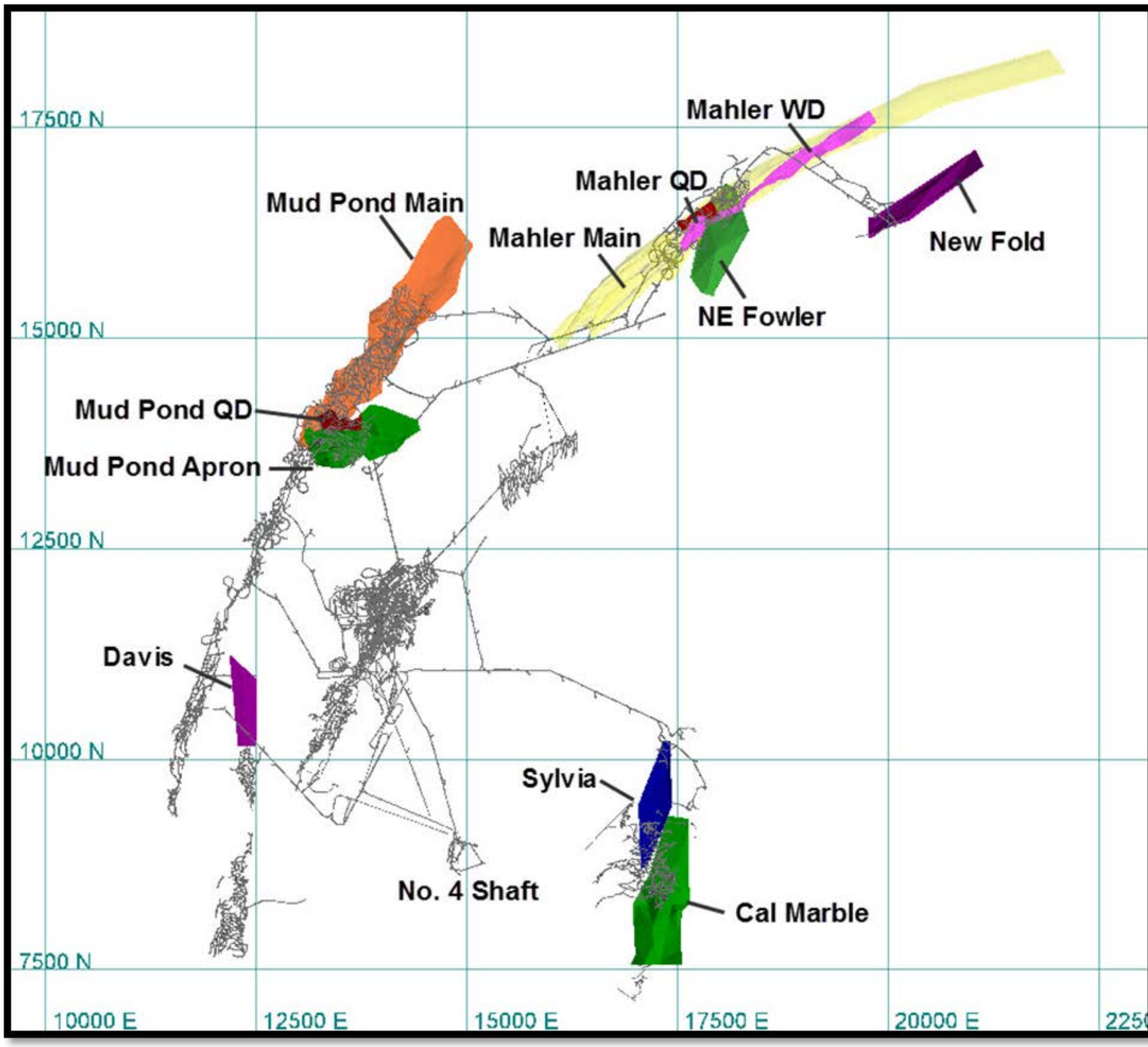
16.1 Deposit Characteristics

ESM hosts six zinc-rich mineralized zones, known as Mahler, New Fold, Mud Pond, North East (NE) Fowler, Sylvia Lake and Cal Marble. These zones are formed within structural folds of the host rock, with the thickest zones near the apex of the fold. Deposits are distributed throughout the property within a 6,000 ft radius and between 1,400 ft and 5,800 ft below surface.

Mineralized zones generally strike NE-SW from 450 to 6,000 ft with a width of 100 to 500 ft and dip 20 to 60º. On mining scales, extreme local variations in the dip and orientation are not uncommon.

All zones, except NE Fowler, are connected to existing infrastructure underground and many have not been fully delineated and remain open for further exploration and resource expansion. The currently understood interpretation of the mineralized zones is depicted in Figure 16.3.

Figure 16.3: Empire State Mine Resource Interpretation



Source: JDS (2017)

16.2 Mineral Resources Within the PEA Mine Plan – Estimation Process

To determine the mine plan tons at ESM, the following process was utilized:

- Analyze geologic resource model for geometric properties, such as mineralized zone width, depth, length, and continuity;
- Select the mining methods best suited for the deposit based on geometry, economics, and geotechnical parameters;

- Determine an economic cut-off grade based on expected operating cost, mining recovery, mining dilution, and commodity price assumptions;
- Identify the blocks in the model that are above cut-off, and design production stope shapes around these blocks;
- Query the production stope shapes for in situ tonnage and grade data, apply mine dilution, and check the diluted stope grades against the cut-off grade, removing all stopes that fall below cut-off; and
- Develop a mine plan around the economically viable production stopes and run economic models on various production scenarios.

16.3 Resource Model Sub-Blocking

JDS used the resource block model discussed in Section 14 of this report for mine planning purposes. The block model was sub-blocked down to 2.5 ft x 2.5 ft x 2.5 ft to gain resolution of mineralized material blocks near the waste/mineralized material contact and to better estimate planned mine dilution.

Sub-blocking an existing block model effectively reduces only the blocks that are in contact with a resource boundary and removes those blocks, which extend into a waste zone. As such, there is generally a minor loss of tonnage during sub-blocking exercises. Table 16.1 below summarizes the change in block model resource at a 6.0% Zn cut-off before and after the sub-blocking exercise.

Table 16.1: Mineral Resource Before and After Sub-blocking

Model Comparison			Percent Block Model		Sub-blocked Model		Difference
GRADEGROUP	ROCKGROUP	CLASS Grade	Tonnage T x 1000	Zn% Grade	Tonnage T x 1000	Zn% Grade	
MEASURED	**10**	1	0.4	6.3	0.4	6.3	103%
	20	1	0.0	0.0	0.0	0.0	0%
	30	1	44.9	10.8	44.9	10.8	100%
	40	1	246.8	10.4	250.8	10.4	98%
	41	1	60.0	12.0	61.2	12.1	98%
	43	1	83.5	10.7	85.0	10.7	98%
	50	1	356.2	15.0	360.8	15.0	99%
	51	1	89.6	19.2	90.2	19.2	99%
	52	1	7.7	15.7	7.8	15.8	99%
	60	1	0.0	0.0	0.0	0.0	0%
	70	1	69.4	12.7	69.4	12.7	100%
	Total	**1**	**958.7**	**13.3**	**970.5**	**13.3**	**99%**
INDICATED	**10**	2	1.3	8.2	1.8	8.5	72%
	20	2	43.0	9.1	42.8	9.1	101%
	30	2	48.0	10.6	48.4	10.6	99%
	40	2	153.4	11.5	156.3	11.4	99%

Table 16.1: Mineral Resource Before and After Sub-blocking (continued)

Model Comparison			Percent Block Model		Sub-blocked Model		Difference
GRADEGROUP	ROCKGROUP	CLASS Grade	Tonnage T x 1000	Zn% Grade	Tonnage T x 1000	Zn% Grade	
	41	2	140.8	10.4	142	10.4	99%
	43	2	13.2	11.9	13.2	11.8	100%
	50	2	592.4	15.1	593.6	15.1	100%
	51	2	87.1	18.6	87.1	18.6	100%
	52	2	34.9	11.3	35.0	11.4	99%
	60	2	0.0	0.0	0.0	0.0	0%
	70	2	263.1	11.6	264.1	11.6	100%
	Total	**2**	**1,377.1**	**13.3**	**1,384.2**	**13.3**	**100%**
INFERRED	**10**	3	0.1	7.4	0.2	8.1	38%
	20	3	438.8	11.5	434	11.6	101%
	30	3	56.2	12.9	58.4	13.0	95%
	40	3	398.2	11.0	410.5	11.0	97%
	41	3	24.4	9.3	24.2	9.2	101%
	43	3	0.6	13.8	0.6	13.8	101%
	50	3	329.1	12.8	329.1	12.8	100%
	51	3	186.2	21.1	186	21.1	100%
	52	3	8.4	9.8	8.4	9.8	100%
	60	3	348.3	14.6	348.5	14.6	100%
	70	3	543.2	13.9	543.3	13.9	100%
	Total	**3**	**2,333.60**	**13.4**	**2,343.30**	**13.4**	**100%**

Source: JDS (2017).

16.4 Mining Method Selection

Given the irregular geometry of the resource, several mine methods were considered and ultimately selected for the ESM.

Sub-level longhole (LH) stoping will be used at ESM as the principal mining method, due to its high productivity, low cost, selectiveness, and successful history of application for deposits of this nature. Alternatively, cut and fill (C&F) and modified room and pillar (MRP) mining will be used where conditions are not suitable for longhole stoping.

Longhole stoping is a semi-selective and productive underground mining method, and well suited for steeply dipping deposits of varying thickness. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralization. In the planned longhole stopes at ESM, a top and bottom drift delineate the stope and a dedicated longhole drilling machine drills blast holes between the two drifts.

The drill holes will be loaded with explosives and the stope is blasted, with broken material falling to the bottom drift for extraction. In longhole stopes, remote controlled load haul dump machines (LHD) are required to remove the blasted material from the stope once blasting commences.

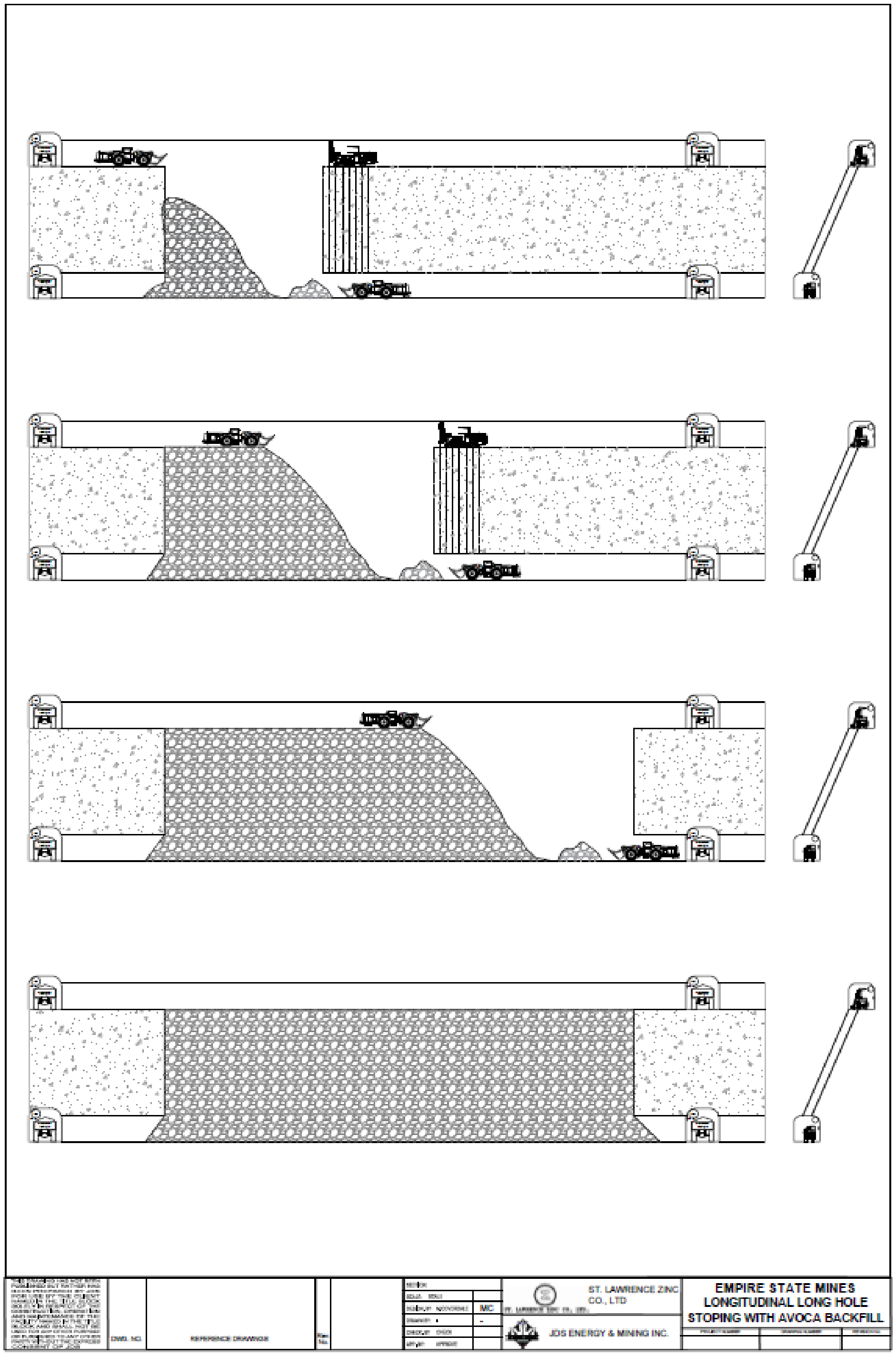
One of the limitations with longhole stoping is that the dimensions of the stope height should not exceed a longhole drilling machine's effective range, which, for small hole, top hammer drill rigs, is generally 80 ft. Another limitation with longhole stoping is the stopes must remain open long enough to remove the mineralized material and then filled with an engineered backfill material (where support pillars are not used). These limitations generally restrict level spacing at ESM to 70 ft or less.

Longitudinal stoping will be the primary method at ESM, whereby a central sub-level is driven along strike through the mineralization to provide access for drill and mucking equipment. This method is beneficial for minimizing waste development as the bulk of mining activities stays within the mineralized zones. The shortfall of longitudinal longhole mining is that production is limited to one stope at a time as the level is mined in retreat.

Stope structural support will be provided through a combination of rib pillars and un-cemented rock fill. Pillars will be left where there is limited access to the sub-level. Where there is access to the backside of the sub-level during mining, an Avoca backfill program will be utilized where backfill is deposited along strike while the level is mined. Long hole stoping with rib pillars and with Avoca backfill is shown in Figures 16.4 and 16.5.

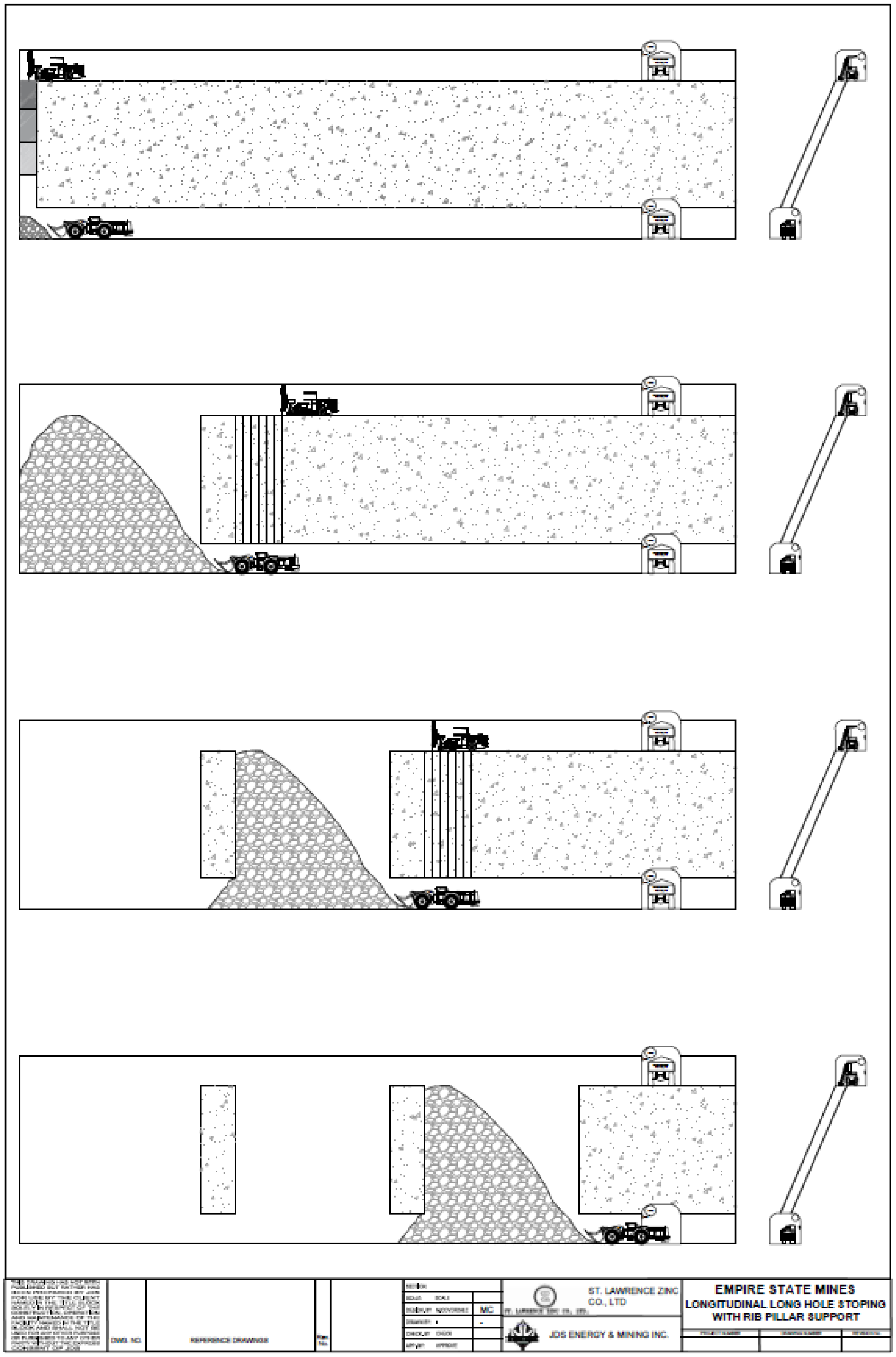
Longhole stoping is planned for use in Mahler, New Fold, Mud Pond and NE Fowler mineralized zones.

Figure 16.4: Longitudinal Longhole Stoping with Avoca Backfill (Typical Layout)



Source: JDS (2017)

Figure 16.5: Longitudinal Longhole Stoping with Pillar Support (Typical Layout)

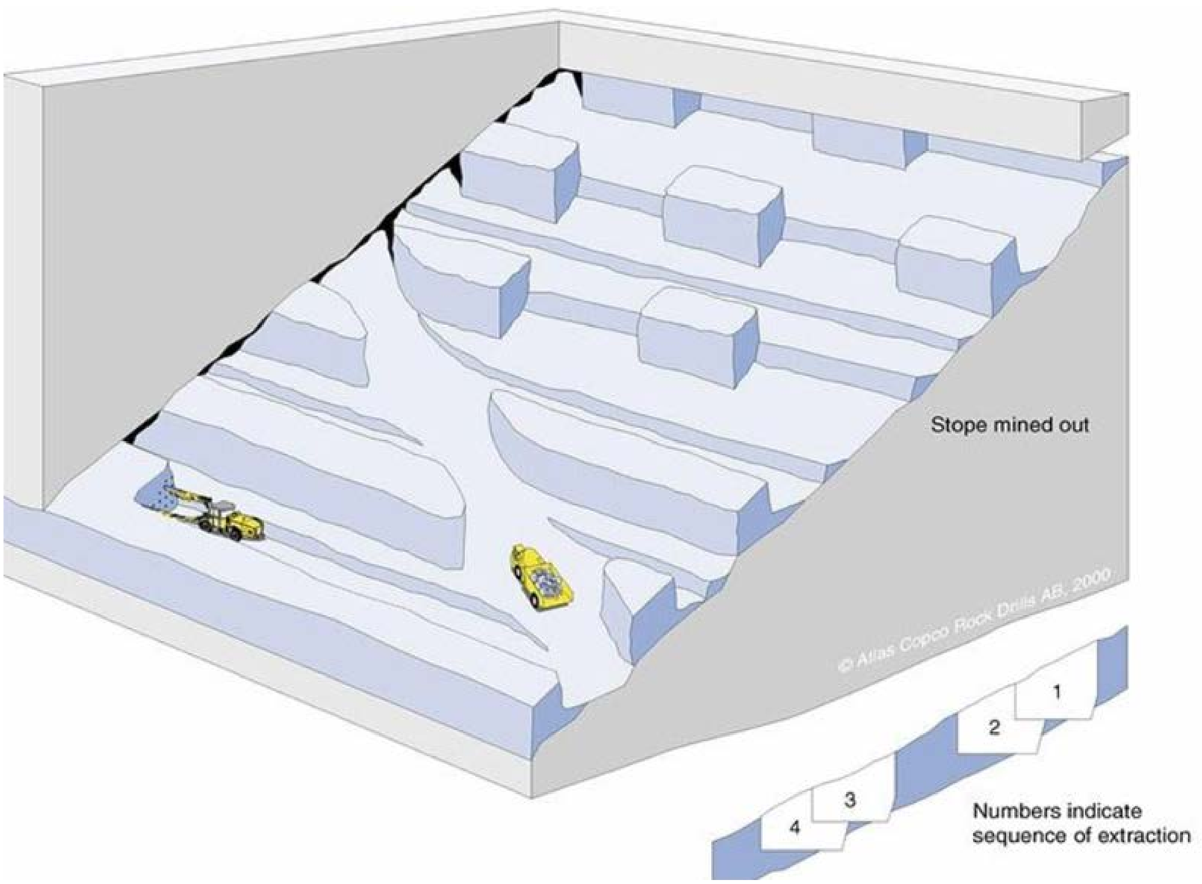


Source: JDS (2017)

Modified room and pillar (MRP), and post pillar (PP) cut and fill will be used at ESM for areas of the deposit that flatten out and do not support multiple panel extraction without addition of cemented backfill. MRP has been extensively used at ESM in the past and allows for the selective extraction of resources while maintaining the majority of development in mineralization, and permits mining top down rather than bottom-up as required in overhand C&F or long hole stoping. MRP utilizes the mineralization as an internal ramp, with cross-cuts spaced along drift, and subsequent rooms driven perpendicular to the cross-cuts to form rooms and pillars. In MRP, an assumed 25% pillar loss is accounted for, which provides for a 13 ft x 13 ft pillar between 13 ft x 13 ft rooms. Figure 16.6 below depicts a typical modified room and pillar layout.

Modified room and pillar is planned for use in Mahler and Cal Marble mineralized zones.

Figure 16.6: Modified Room and Pillar (Typical Layout)

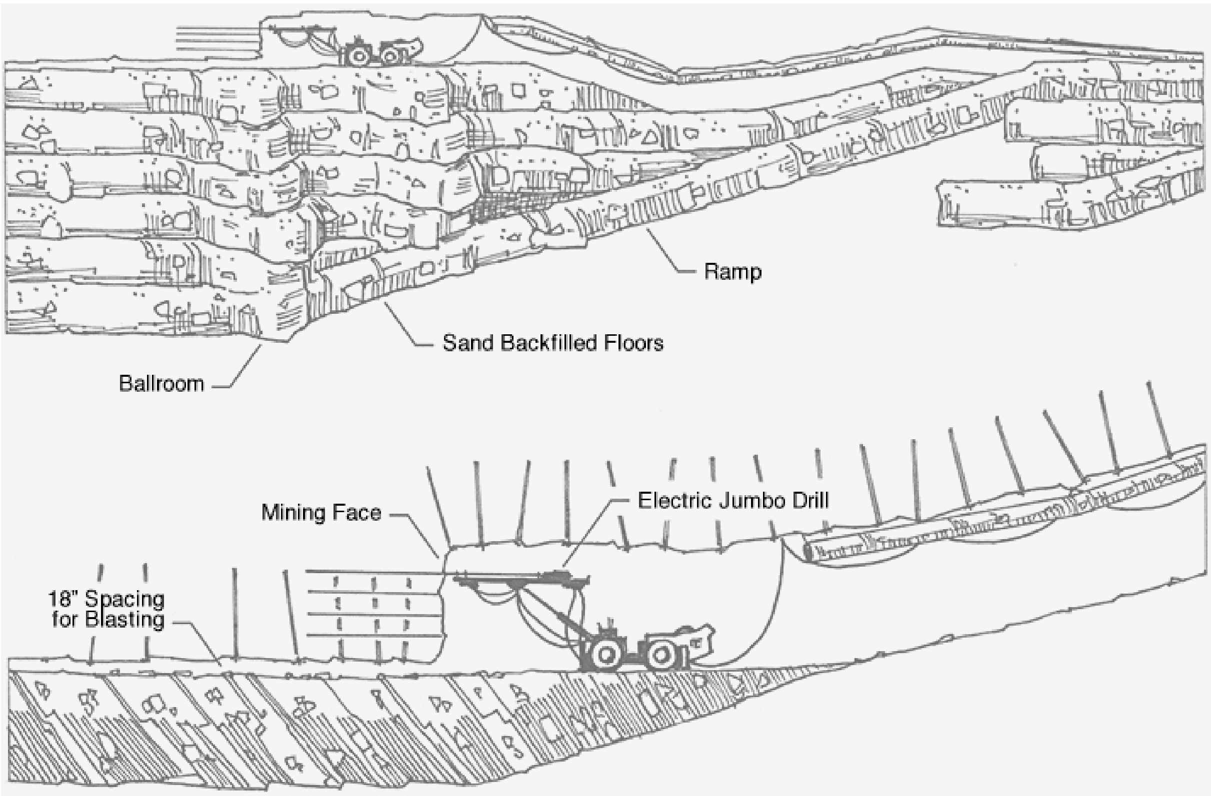


Source: Atlas Copco (2000)

C&F mining will be used at ESM for areas of the deposit which fall below an allowable dip for longhole stoping, or where more selective mining are required. The method will be an overhand C&F whereby drifts are driven across strike on level, backfilled with un-cemented fill, and then the next level above mined. This method is well suited for narrow, gently dipping zones. A typical layout for C&F is shown in Figure 16.7.

C&F is planned for use in Cal Marble, Mud Pond, and Sylvia Lake mineralized zones.

Figure 16.7: Cut and Fill (Typical Layout)



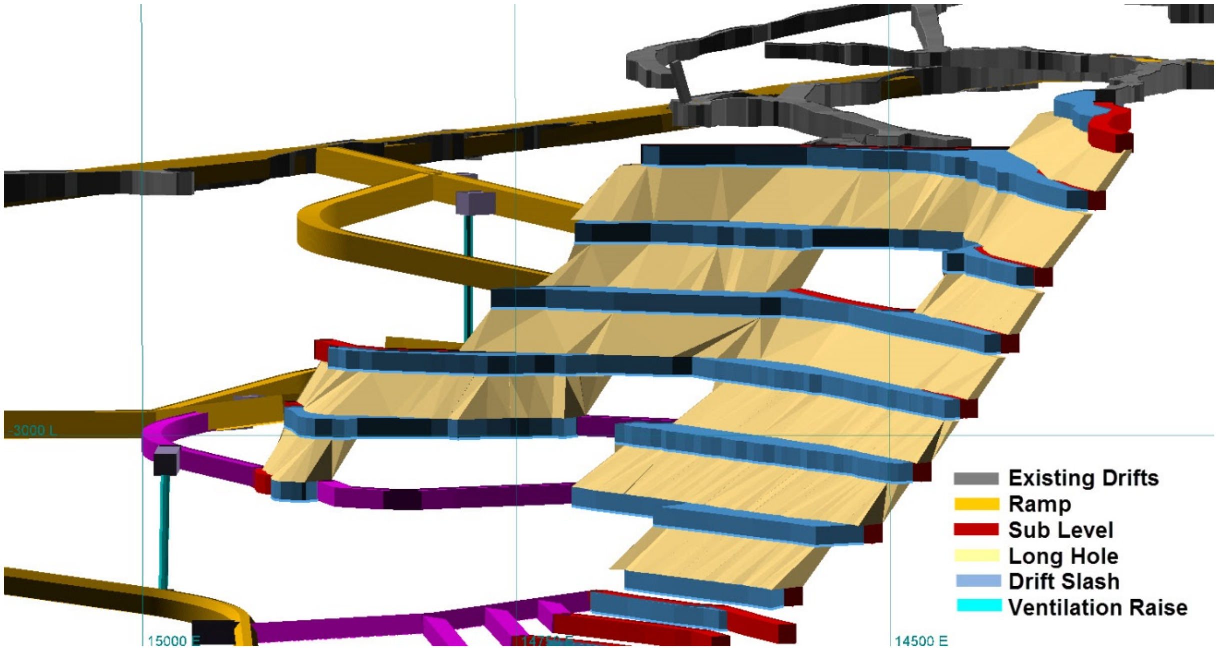
Source: Atlas Copco (2000)

Drift slashing (SLS) will be used at ESM to extend sub-level drifts laterally to provide drill access for long hole stopes. Slashing will also be performed to extract portions of remnant pillars left between existing development drives throughout the mine.

Resue mining will be utilized throughout MRP and SLS areas to selectively mine waste separately from the mineralized material. In resue mine methods, either the mineralized material or the waste material is drilled and blasted independently from each other. On an underground tour of the mine, the QP observed the black and white nature of the mineralization and host rock respectively. Drifts inspected showed 5 ft bands of mineralization crossing the 13 ft drift laterally, with mineralization dipping approximately 20° across the drift. It is the opinion of the QP that resue mining will be achievable where the resource is gently dipping and the mineralization is kept against either the back or floor of the drift. It has been assumed that up to 75% of the waste contained within the MRP and SLS stopes may be extracted by resue mining, accounting for a 25% internal dilution of waste to accompany the mineralization extraction.

A representative view of the mining methods used in Mud Pond is shown in Figure 16.8.

Figure 16.8: Mud Pond Mining Methods



Source: JDS (2017)

16.5 Geotechnical Parameters

The majority of geotechnical parameters as written in this section have been referenced from a 2005 geotechnical review of the ESM by Itasca Consulting Canada (Itasca), in addition to ground support measures utilized in the most recent mining campaign.

Ground conditions at the ESM are considered very good, and estimated to be RMR of 80 or greater. The underground shop on 2,500 level has a span of 50 ft and length of 200 ft, with a calculated RMR of 87, supported by a combination of SP33 split sets, dywidag resin rebar, and woven chain link mesh. There is no visible loose in the mesh or opening joints (Itasca, 2005).

Figure 16.9: 2,500 Level Shop Ground Conditions

Source: Itasca (2005)

Prior to mine shut down in 2001, the underground workings were supported on an as needed basis using minimal support. Pattern bolting and mesh application was not used, as evident when traveling through historical workings. Fall of ground (FOG) accidents total 50 between the years 1994 through 2000, 46 of which involved workers being struck by falling rock (Ibid). The majority of these incidents were during scaling and loading the face, suggesting that insufficient or improper installation of ground support was not root cause for these incidents. It was noted that previous contractors were permitted to work under unsupported ground provided they deemed it safe, which is a practice not permitted or recommended in today's mining environment.

From 2006 to 2008, when the mine was re-opened and operated by Hudbay, a minimum ground support standard was established for all new development, which primarily includes the continued use of SP33 split sets. Depending on the dimension of the drift and depth within the mine, split sets are increased in length and the application of welded wire mesh is incorporated. Nearly all future development in the mine will be driven below the 3,100 level, suggesting all future development will be fully bolted and screened on the back and shoulders.

Results from pull tests conducted in 2007 were reviewed to show 86% of installed bolts passing manufacture strength of 3-6 ton. Table 16.10 below provides results from the 14 five-foot SS39 pull tests conducted from August through September in 2007.

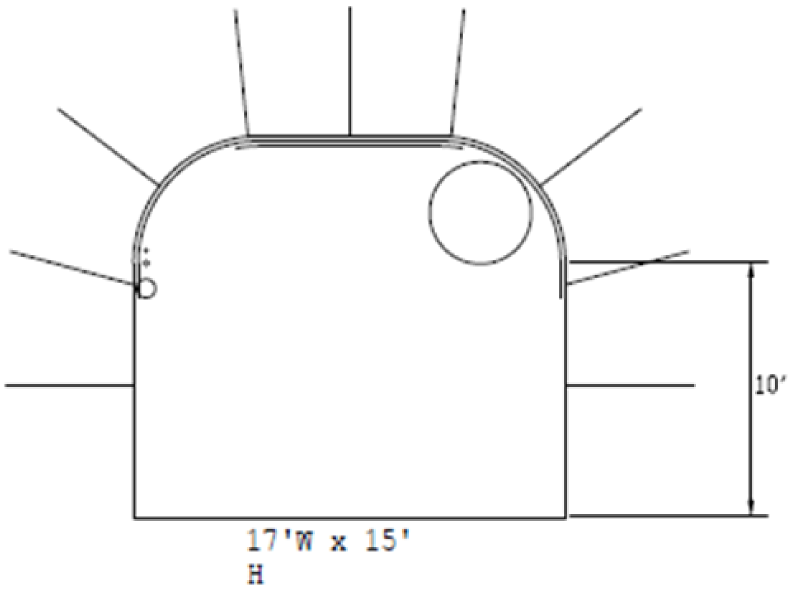
Table 16.2: 2007 Pull Test Results

Heading	Date	Mineral / Waste	Wall / Back	Bit Size	Pullout Strength
Mahler Decline	August 10, 2007	Waste	Back	35 mm	6.5 tons
Mahler Decline	August 10, 2007	Waste	Wall	35 mm	5 tons
Mahler 3493	August 10, 2007	Waste	Back	35 mm	7 tons
Mahler 3493	August 10, 2007	Waste	Wall	35 mm	6 tons
Mahler 3470	August 15, 2007	Mineral	Back	35 mm	2.5 tons
Mahler 3470	August 15, 2007	Mineral	Wall	35 mm	2.5 tons
Mud Pond 37 DEC	August 29, 2007	Waste	Wall	36 mm	7 tons
Mud Pond 37 DEC	August 29, 2007	Waste	Wall	36 mm	6 tons
Mud Pond 37 DEC	August 29, 2007	Waste	Wall	37 mm	6 tons
Mud Pond 37 DEC	August 29, 2007	Waste	Wall	37 mm	3.5 tons
Mud Pond 3347	September 4. 2007	Mineral	Wall	35 mm	7 tons
Mud Pond 3347	September 4. 2007	Mineral	Wall	35 mm	7 tons
Mud Pond 3347	September 4. 2007	Mineral	Wall	36 mm	6 tons
Mud Pond 3347	September 4. 2007	Mineral	Wall	36 mm	3 tons

Source: SLZ (2007)

The ground support minimum requirements currently in use at ESM were reviewed by JDS and deemed appropriate for continued use in future lateral development. Figure 16.11 below outlines support requirements for three primary heading types used in the LOM design.

Figure 16.10: Minimum Ground Support Profiles



Notes for drifts over 12 feet in height:
1. At intersections, utilities are to have a minimum clearance of 12' above sill
2. Vent tubing to have minimum height of 10.5 feet above sill
3. Lifters to be positioned to subdrill floor a minimum of 1/2 foot.
4. Initial ground support includes welded wire to ten foot above the sill and 5 or 6 foot friction stabilizers on 4 foot collar spacing.
5. Final ground suport consistes of chain link or cyclone fence attached to the back with 2' 33mm friction bolt in existing bolts and spot bolting as required with 5 foot bolts in the crown a minimum of 8 feet wide to a max of 16 feet wide depending upon ground conditions

Source: HBMS (2006)

16.6 Stope Design Parameters

Stope design criteria are summarized in Table 16.3.

Table 16.3: Production Stope Design Criteria

Mine Method	Stope Width (ft)	Stope Height (ft)	Stope Length (ft)	Dip (°)
Cut and Fill	13	13	N/A	0-45
Room and Pillar	13	13	N/A	0-30
Typical Long Hole Stope	10-40	20-70	Max 150	45-90

Source: JDS (2017)

Lateral stope dimensions are designed with consideration of existing equipment on-site to be used in production. Larger stopes may be possible, and in the mine plan the sub-levels are often slashed on the walls to provide drill access for planned longhole stope dimensions.

Geotechnical reports and recommendations have not been reviewed for long hole stoping mine methods. The geotechnical report prepared by Itasca in 2005 suggests that high in-situ stresses at depth may limit resource extraction to 60%, if stope support is relied upon pillars alone. Several long hole stopes have been mined during the last production campaign and some were still open and not backfilled at the time JDS toured the facility in February 2017. For stope design basis, the mine shop was used as a baseline comparable for maximum void that could be developed.

Longhole stope dimensions are variable to accommodate the geometry of the resource. A minimum 6 ft true width was used for stope design, along with a minimum 45° footwall and maximum 30° hanging wall. Level spacing of stopes generally resides between 30 to 50 ft and is dependent of the dip and thickness of the resource. A maximum drill depth of 80 ft can be achieved using the top hammer drills on-site and stope height was calculated by measuring the average dip of each mining zone and converting the hypotenuse of an inclined 80 ft hole to the vertical. 4 below outlines the maximum level spacing calculated for each mining zone.

Table 16.4: Level Spacing by Dip and Drill Depth

Mining Zone	Average Dip (degrees)	Sill Height (ft)	Maximum Level Spacing (ft)
Cal Marble	23	13	45
NE Fowler	45	13	71
Mahler Main	30	13	54
Mahler QD	25	13	48
Mahler WD	20	13	41
Mud Pond Apron	18	13	38
Mud Pond Main	25	13	48
Mud Pond QD	60	13	84
New Fold	53	13	79
Sylvia	20	13	41

Source: JDS (2017)

16.7 Mine Dilution & Recovery

Dilution was estimated based on typical stope dimensions to calculate unplanned over break experienced during mining operations. The rock quality at ESM is considered to be very good geotechnically, so overbreak is considered to be minimal. For long hole stopes three sources of dilution were considered. Sloughing estimated to be 1.5 to 2.0 ft on both the hangingwall and footwall of longhole stopes. Two typical stopes were designed in detail. The smaller stope was designed with a true width of 10 ft, while the larger with a true width of 40 ft. For C&F, MRP, SLS, and sub-level drifts over break dilution of 0.5 ft was applied to the floor, back, and walls. A dilution grade of 0% Zn was assumed for all overbreak. Overbreak dilution parameters are in Table 16.5.

Table 16.5: Overbreak Dilution Parameters

Typical Profiles	Units	Sub-level Mining	Cut and Fill	Room and Pillar	Slashing	LH Stope Small	LH Stope Large
Height	ft	13.0	13.0	15.0	15.0	50.0	50.0
Width	ft	15.0	10.0	15.0	15.0	10.0	40.0
Wall Over break	ft	0.50	0.50	0.50	0.50	1.50	2.00
Back Over break	ft	0.50	0.50	0.50	0.50	0.00	0.00
Fill Undercut	ft	0.50	0.50	0.50	0.50	0.00	0.00
External Dilution	%	**13**	**16**	**12**	**12**	**23**	**13**

Source: JDS (2017)

Mine recovery was calculated under the following mine assumptions:

- Room and pillar zones are subject to 25% loss from in-situ structural pillars left behind;
- 50% of longhole stoping zones are subject to 20% loss from in-situ rib pillars left behind, calculated based on a pillar lengths 1.5 times the average thickness of the LH stope;
- 50% of longhole stope production will be backfilled using Avoca method and will not require rib pillars for structural support;
- Resue mining is to be utilized in sill slashing and room and pillar zones to mine 75% of contained waste independently of the mineralized material;
- Remnant pillars between existing development that will be slashed in retreat upon mine closure are subject to a 75% mine recovery; and
- All lateral drifts in sub-level development, slashing, room and pillar, C&F, and waste development passing incremental cut-off, assume 95% mine recovery after losses from pillars.

Approximately 965,000 t of waste is planned to be effectively resue mined in the plan, accounting for 15% of the fully diluted mine plan tonnage before removal of pillar material.

Approximately 540,000 t will be left behind as structural pillars, accounting for 8% of the fully diluted mine plan tonnage.

16.8 Cut-off Grade Criteria

Zinc cut-off grade calculation criteria are summarized in Table 16.6.

Table 16.6: Cut-off Grade Parameters

Parameter	Unit	Value
Zn Price	US$/lb	1.00
Mill Recovery	%	96.0
TC/RC/Transport	$/dt Zn	238
Payable Metal from Refinery	%	85
Royalties	%	0.3
Operating Costs	$/t milled	70.00
Calculated Cut-off (%Zn)	%Zn	5.9
Cut-off Utilized (%Zn)	**%Zn**	**6.0**
Incremental Cut-off (%Zn)	%Zn	2.0
Incremental Cut-off Utilized (%Zn)	**%Zn**	**2.0**

Source: JDS (2017)

Incremental cut-off accounts for the cost of crushing, hoisting, milling, and general services incurred per ton of milled material. Incremental cut-off is applied to any waste development that crosses mineralization in order to access stopes designed with the primary cut-off of 6.0% Zn. Approximately 18% of all tons reporting to the mill are classified as incremental. Cut-off grade parameters may not reflect those used for economic modelling and were assumed to contain the most accurate information available at the time of preparation.

16.9 Mine Plan Tons and Grade

The PEA mine plan tons for ESM is a product of stope optimizations performed by Vulcan Stope Optimizer© software, manually designed stopes, and selective shelling of the geologic resource in areas deemed to have extraction potential without modification. All stopes were designed based on the applicable stope shapes, geological boundaries, and grade extents, ensuring the final stopes' shapes meet cut-off criteria. Table 16.7 outlines the diluted, recoverable, mine plan tons used for mine planning purposes.

Longhole stoping will contribute 50% of the mine plan tons, 19% from slashing, 11% from room and pillar, 13% from sub-level development, and 7% from cut and fill.

Table 16.7: Mine Plan Tons Contained in Mine Plan

Zone	Diluted Tons (kt)	Diluted Zn Grade (%)
Cal Marble	303	8.09
Sylvia Lake	77	9.18
Mud Pond	573	8.56
Mud Pond Apron	174	6.20
Mud Pond QD	50	5.80
Mahler Main	1,363	9.06
Mahler WD	353	12.19
Mahler QD	15	7.27
NE Fowler	465	8.92
New Fold	904	9.98
Total	**4,278**	**9.21**

Source: JDS (2017)

Table 16.8: Mine Plan Tons by Class

Resource Class	Diluted Tons (kt)	Diluted Zn Grade (%)
Measured	768	8.60
Indicated	1,406	8.80
Inferred	2,104	9.70

Source: JDS (2017)

16.10 Mine Design Criteria

16.10.1 Mine Access

The ESM deposit consists of a mining resource extending nearly 4,500 vertical feet. Multiple shafts extend from surface to the existing underground workings. Extensive underground workings exist from previous mining operations. Digitized underground survey suggest there are more than 50 miles of development in the No. 4 mine alone. Fresh air shafts and secondary egress paths are already in place at ESM. Existing development ranges from 10 ft wide by 10 ft tall to over 17 ft wide by 15 ft tall. The maximum gradient of the existing workings is 20%.

The ESM is situated on moderately flat lying terrain.

Existing workings will need to be rehabilitated to ensure a safe work area. When accessing new deposits, a ramp will be driven at a maximum grade of 15% at a 17.5 ft by 15 ft profile. Mineralized zone development will be as small as a 13 ft by 13 ft profile.

16.10.2 Production Rate Selection

The ESM mine plan has been sized to ramp up from 800 t/d to a sustained maximum of 1,800 t/d. Cycle times of the different mining methods were considered along with the existing mine hoist capacity and existing equipment fleet in determining the production rate.

The mine schedule was created using Minemax iGantt© software. The scheduling rates used are shown in Table 16.9.

Table 16.9: Scheduling Rates Used for Mine Scheduling

Scheduling Rates			
Development	**Unit**	**Regular Rate**	**Multiple Headings**
Ramp	ft/day	9	
Auxiliary	ft/day	9	
Sub-Level Waste	ft/day	9	
Sub-Level Mineralized	ft/day	9	18
Cut and Fill	ft/day	7.2	14.4
Room & Pillar	ft/day	9	27
Slashing	tday	478	
Rehab existing workings	ft/day	100-200	
Vertical Development			
Drop Raise	ft/day	20	
Raiseboring	ft/day	20	
Stoping			
Longhole Large	t/day	788	
Longhole Small	t/day	324	

Source: JDS (2017)

16.10.3 Production Sequencing

Production in longhole stoping zones will be mined with a bottom-up sequence in which loose rock backfill is used. Where necessary in situ sill pillars will be left to separate mining horizons.

C&F zones will be mined in a bottom-up fashion from a main access drift. From the main ramp, a drift will access the production area with a +/-18% attack ramp. Once the production drift is mined out on that level, it will be backfilled and the access cross-cut slashed along the back and backfilled on the floor to allow access to the next level above, where the mining process is to be repeated.

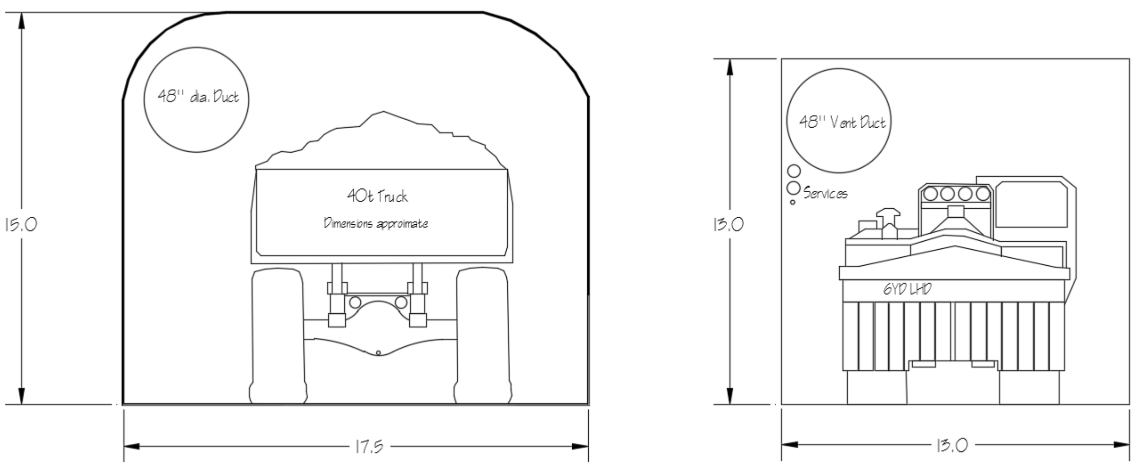
16.11 Underground Mine Development

16.11.1 Lateral Development

Ramp are envisioned to be driven at a 15 ft x 17.5 ft arched profile to accommodate fully loaded 40 t haul trucks and 48" round vent ducting. Cross-cuts and sub-level development will be driven flat back style 13 ft x 13 ft to accommodate remote LHD entry.

Figure 16.11 depicts a typical development ramp and cross-cut cross-sections.

Figure 16.11: Typical Development Cross-sections



Source: JDS (2017)

16.11.2 Vertical Development

Muck passes at a 6 ft x 6 ft profile are planned to bring mined material to the 3,100 level from within the Mahler and Mud Pond zones where mining activity takes place above the main haulage route. A grizzly will be installed at the top of each muck pass to remove oversize blasted material. LHDs will load trucks at the bottom of the muck pass for transport to the crusher.

Ventilation raises at 6 ft x 6ft profile will be established to provide fresh air for each of the mining zones. All raises will be driven with the use of contract raise bore or method fit for purpose.

Total lateral and vertical development over the mine life is summarized in Table 16.10.

Table 16.10: Development Schedule

Development Type	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Rehab	ft x 1000	**59**	35.7	8.3	15.4	-	-	-	-	-
	ft/day	**73**	97.8	22.8	42.1	-	-	-	-	-
Lateral Capital Development	ft x 1000	**44**	8.2	7.1	10.7	3.8	6.0	3.5	3.7	1.2
	ft/day	**19**	22.5	19.4	29.3	10.3	16.5	9.5	10.0	3.3
Lateral Waste Development	ft x 1000	**20**	0.9	4.7	5.0	2.1	2.9	2.0	1.6	0.5
	ft/day	**9**	2.5	12.9	13.8	5.8	8.0	5.5	4.4	1.3
Mineral Development	ft x 1000	**101**	2.5	9.4	10.4	17.5	12.4	12.9	29.2	7.2
	ft/day	**47**	6.7	25.7	28.6	47.7	33.9	35.4	80.0	19.7
Total (excl. Rehab)	**ft x 1000**	**165**	12	21	26	23	21	18	34	9
	ft/day	**64**	32	58	72	64	58	50	94	24
Jumbo Productivity	**ft/day/jumbo**	**26**	16	19	24	32	29	25	31	12

Source: JDS (2017)

16.12 Unit Operations

16.12.1 Drilling

Development headings are planned to be driven with electro-hydraulic single and dual boom jumbos. Twelve foot steel is planned in C&F zones where single boom jumbos are required to make quick turns to follow the mineral. The advance per round is assumed to be 12 ft for 14 ft steel and 10 ft for 12 ft steel. One jumbo has the capacity to drill between two to three rounds per shift, however, cycle productivities are as listed in Table 16.10.

Production drilling for the longhole stopes will be performed by longhole drills. Blast holes with 3.5” diameter will be drilled in a fan pattern from the overcut to the undercut.

16.12.2 Blasting

Development rounds will be charged by a bulk explosives tractor. Lifter holes will be loaded with packaged emulsion. Blasting is planned to be initiated by non-electric (NONEL) detonators.

For longhole production blasting, bulk emulsion will be used together with NONEL detonators and 60 g boosters.

16.12.3 Ground Support

After mucking and scaling is complete, ground support will be installed by a mechanized bolter or manually by experienced operators using jacklegs and stopers. Typical ground support in access development is planned to consist of 5 ft and 6 ft split-set bolts in the back and in the walls at a spacing of 4 ft x 4 ft. Welded wire mesh will be installed in all ground conditions. In intersections, 22 ft cable bolts will be installed on a 6 ft x 6 ft pattern for deep ground support.

Cable bolts will be installed into the hangingwall prior to long hole stope firing with an average pattern of six bolts per ring and 10 feet between rings.

16.12.4 Mucking

Blasted material from development headings will be mucked with either 4.0 yd^3 (7 t) or 6.0 yd^3 (10 t) LHD directly to a haul truck, remuck bay or muck-pass. Broken material from longhole stopes will be mucked by remote control LHD.

16.12.5 Hauling

A fleet of 40 t and 26 t haul trucks will drive underground and haul mineralized material from the active production areas and internal muck-passes to the shaft loading station. The same haul trucks will be used for waste material transport to areas requiring backfill.

Haulage profiles for each of the mineralization zones were generated to calculate equipment hours for the fleet.

16.12.6 Backfill

The selected mining methods require the placement of backfill for an increased extraction ratio of the mineralized zones. Stopes require the use of Avoca backfill to provide stability to the active stope when mining along strike. This necessitates having access on both ends of the stope. Alternatively, rib pillars are used when Avoca backfill is not practical. No cemented backfill will be used at ESM.

Underground development waste may be placed as backfill in attack ramps and remote stopes to minimize waste haulage to surface.

16.13 Mine Services

16.13.1 Mine Ventilation

Minimum airflow requirements were based on expected diesel emissions of the underground mining fleet required at peak mine production. Additional airflow is used underground to improve air quality. The power rating of each piece of equipment was determined, and the utilization factors representing the equipment in use at any time, were applied to estimate the amount of air required. Equipment specified for site has undergone testing by MSHA to determine the ventilation requirements to dilute the engine emissions to a safe working level. The volume of air required for ventilating the diesel emissions is 282,000 cfm.

Air flow measurements from August 2008 indicate 221,000 cfm exiting the mine and 337,000 CFM circulating below the 3100 level. The added air flow in the bottom of the mine is the result of recirculation through the old workings, primarily in the Upper Fowler area.

During the last operational year in 2008, the mine received eight citations for exceeding the allowed Diesel Particulate Matter (DPM) exposure levels as a result of this recirculation of ventilation.

In 2016, the ESM ventilation network was modelled using Ventsim® Visual software by Practical Mining LLC (Practical Mining) under contract by Star Mountain Resources to prepare a plan to eliminate recirculation below the 3100 level and provide enough ventilation to safely re-start operations (Practical Mining, 2016).

The network was created from as-built surveys and uses the correct airway length, cross sectional area and elevations. Resistance factors measured by SLZ engineers during a 2006 ventilation survey were utilized for the #2 Mine, and #4 Shaft. These airways represent the majority total mine head. Generally accepted friction factor vales were used for the remainder of the workings. These values were typically higher than those measured by the 2006 survey.

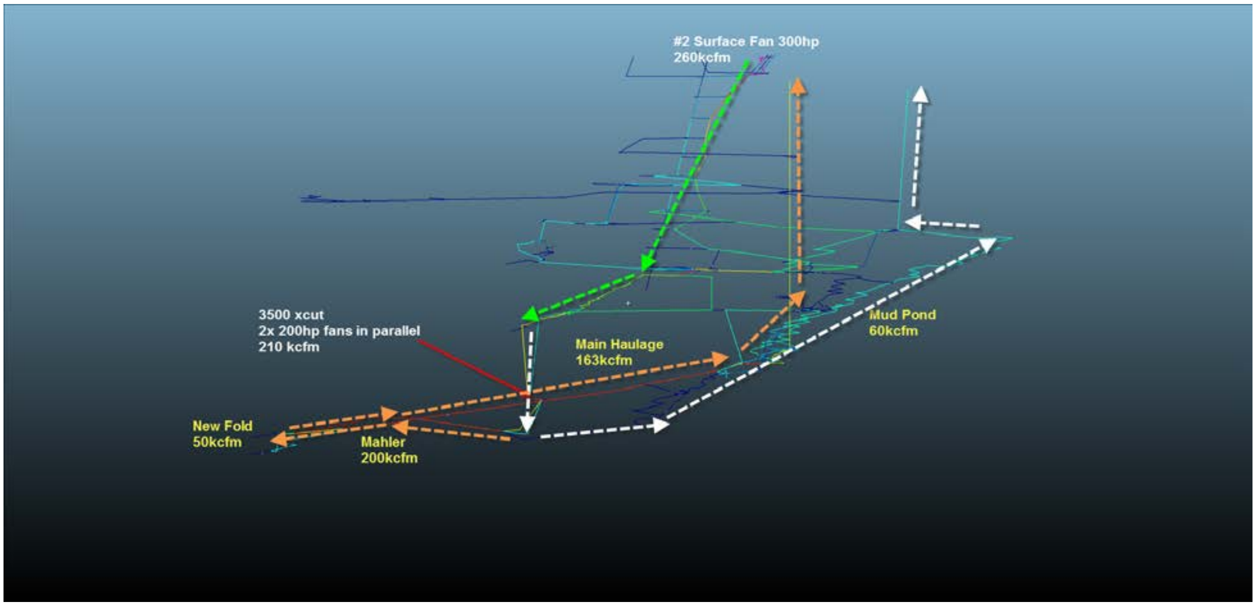
The ventilation network prepared in 2016 was provided to JDS and used to generate an updated vent design for the 2017 PEA mine plan.

The generalized strategy for ventilating the ESM mine is to use the #2 Mine inclined shaft, stopes and associated workings as intake. Air will be exhausted through the #4 Shaft and #4 Bore Hole. This will be accomplished using a push/pull configuration. The existing 300 hp ABC centrifugal fan located in the pit west of the #2 hoist house will pressurize the #2 Mine with 260 kcfm. Approximately 15% losses to unknown connections to surface through the #2 mine are expected.

On the 3500 level ventilation cross-cut, two booster fans installed in parallel will draw air from the two mines and feed it to the lowest levels of the Mahler and New Fold areas, with some air exhausting through the main haulage ramp and up the #4 Shaft, and the rest through Mud Pond and out the #4 Bore Hole. These fans will provide 200 to 210 kcfm after allowing for compression. Flow losses, primarily through unknown connections to surface at the #2 Mine, account for 15% of the total air mass provided by the #2 fan (Practical Mining, 2016).

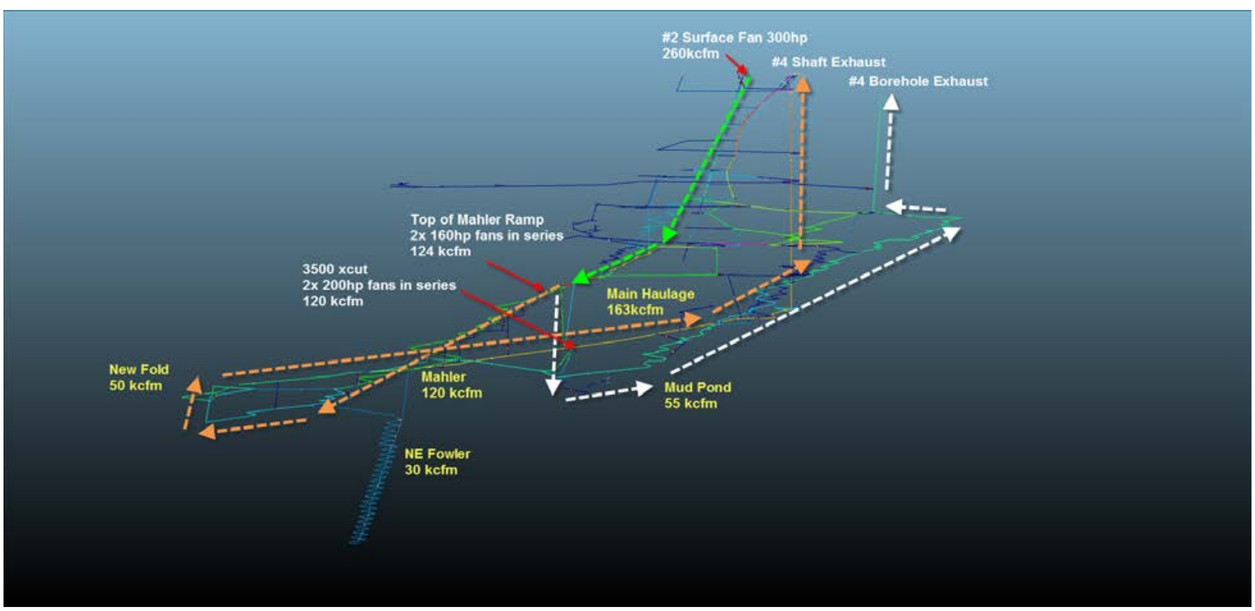
As the mine develops two more booster fans will be installed in series at the top of the Mahler ramp to downcast fresh air through the Mahler workings and into New Fold and NE Fowler. The booster fans installed in the 3500 level ventilation cross-cut will be adjusted from a parallel mount to series to accommodate the additional pressure. With these booster fans in place, the unknown connections to surface through the #2 workings will provide additional fresh air to the circuit to supply a total of 280,000 cfm.

Figure 16.12: Initial Ventilation Installations



Source: JDS (2017)

Figure 16.13: Life of Mine Ventilation Installations



Source: JDS (2017)

16.13.2 Mine Air Heating

Existing 8.5M BTU direct fired propane heater(s) will be used to heat to a minimum 34°F in the No. 4 shaft. Estimated propane consumption is 46,000 gallons annually.

16.13.3 Electrical Power

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

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

High-voltage cables enter the mine via the existing shafts and are distributed to electrical sub-stations near the mining zones. High-voltage power are delivered at 4160 V and reduced to 480 V at electrical sub-stations.

Total electrical power consumption for underground mining is estimated at 2.4 MW during operations.

16.13.4 Compressed Air

Compressed air will be required for longhole drills, jacklegs, and face pumps. Compressed air will be provided by stationary compressors on surface. Reticulation of compressed air through the mine will utilize the existing pipes in addition to new 6” pipes as development advances.

16.13.5 Service Water Supply

Service water for drilling, dust control, washing and fire suppression will be sourced from surface and distributed in 2” diameter steel piping.

16.13.6 Dewatering

Water-bearing fracture zones at ESM generally occur above a depth of 900 ft, diminish with depth, and become nearly nonexistent in the deeper portions of the mines below 1,300 ft. Most of the fresh water encountered in the mines enters from the upper levels. This water enters through fractures connected to the surface water features and the water table.

All the water entering the mine is collected at the sumps near the No. 4 shaft. Most of the water collects at the 1300’ level sump and a small percentage makes its way to the 3100’ sump. The water at 3100’ is stage pumped to the 1300’ sump, then to surface.

The mine has been plugged at 900 elevation, which prevents the majority of ground water from entering the mine and descending to the bottom at 3100 level. What small quantities are encountered are picked up at the 1300 sump.

The mine neighbors onto a talc operation, which hosts a flooded pit. There is an excavation between the ESM and the talc pit and SLZ has been pumping inflow from the talc mine out through the 1300 sump pump to prevent inflow from reaching the lower levels of the mine. Historically during operation, total water discharge from the mine has varied between 223,000 gallons per day (g/d) to a high of 727,000 gallons per second (g/s), and fluctuations appear to correlate with periods of high rainfall or snowmelt (Hudbay, 2005).

During periods of care and maintenance, an average 270 kW has been required to keep the mine fully pumped out (SLZ, 2017). Additional pumping requirements estimated for the life of mine include small sump pumps to be installed in new working areas to collect and remove water brought underground for equipment consumption. Sumps will be designed down ramp of the entry to each mining level to collect water. Remuck bays no longer in use may be slashed in the floor to provide small sumps in which portable submersible pumps will be used.

Water will be pumped from sump pumps in the mine through 6-inch steel pipes.

16.13.7 Explosives Storage and Handling

Primary explosives storage magazines will be located on surface. Secondary magazines will be located underground to provide explosives storage for up to seven days. Bulk explosives and detonators will be stored in two separate facilities.

Bulk and bagged ANFO will be used as the major explosives for mine development and production.

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

Typically, underground operations of this rock type require powder factors of approximately 1.9 lb/t for development and 0.7 lb/t for LH stoping with good fragmentation.

16.13.8 Fuel Storage and Distribution

Mobile equipment will be re-fueled at underground fueling stations currently in place.

16.13.9 Underground Transport of Personnel and Materials

The existing shafts and hoists will be used for moving materials and personnel in and out of the mine. Underground Kubota style personnel carriers will be used to shuttle workers to the active development and production areas. Supervisors, mechanics, engineers, geologists and surveyors will also use Kubota ATVs as transportation underground. A boom truck, flat deck truck and forklift will be used to transport supplies and consumables from shaft station to active underground workplaces.

16.14 Underground Mine Equipment

The required underground mobile equipment was based on the existing fleet at ESM. Equipment hours were constrained in the schedule as to not exceed the availability and utilization of the current fleet. Scheduled quantities of work in combination with cycle times, productivities, availabilities, and efficiencies formed the basis to limit the fleet size to the existing numbers on the property.

Table 16.11 summarizes the underground mobile fleet.

Table 16.11: Existing Mobile Mine Equipment Fleet

Description	On-Site	Utilized
Drill Jumbo – 2 Boom – Garden Denver MK-65	1	2
Drill Jumbo – 1 Boom – Garden Denver MK-35	4	-
Drill Jumbo 1 Boom – MTI VR II	2	2
Longhole – Boart Longyear Stopemate	2	2
Bolter – Secoma Pluton-17	3	3
LHD (10t/6yd) Wagner ST 6-C	3	1
LHD (10t/6yd) Atlas Copco ST 1000	3	3
LHD (6.5t/3.5yd) Wagner ST-3.5	3	-
LHD (7.0t/4yd) MTI 650	5	2
LHD (3t/2.5yd) MTI 270	1	-
Haulage Truck – 40 Ton – Tamrock 40 D	6	6
Haulage Truck – 26 Ton – Wagner MT 426	4	4
Powder Tractor – John Deere JD-210C – PT 0003	5	2
Personnel Carrier – Kubota L5030GST	1	-
Scissor Lift – Getman A-64	5	4
Flatdeck – Elmac 975	1	1
Shotcrete Manual – Aliva – AL 257	1	-
Transmixer – Maclean TM3, 6 m3	1	-
Grader – Champion C80-A27 – GR0002	1	1
Backhoe – John Deere JD-210C	1	1
Utility Vehicle – John Deere JD-210C	1	1
Telehandler – GENI GTH5519 – FDL-0016	1	1
Mechanics Truck – Kubota L2350	14	2
Personnel Trucks – Kubota RTV 900	16	6
Grout Pump	1	1
Jacklegs/Stopers	18	5

Source: JDS (2017)

Haulage requirements for LHDs and trucks were estimated for mineralized material, waste and backfill. Mineralized material is hauled to a remuck, loaded into trucks or dropped into muck-passes, where it is re-handled and loaded into haul trucks for transportation to the shaft loading station.

A development crew with dedicated drill jumbo, LHDs and bolter will drive the critical path development during production ramp up. Some development equipment will be used for C&F mining later in the mine life, when the critical path access development is completed.

Mine development is split between single and twin boom jumbos. Bolting will be performed with a Secoma Pluton-17 bolter in addition to jacklegs working off muck piles or scissor decks.

Two Boart Longyear Stopemate longhole drills are to be used for longhole production stoping.

16.14.1 Mine Equipment Maintenance

Mobile underground equipment will be maintained at the existing underground mine shop. Major rebuild work will be performed off-site. Minor maintenance and repairs will be done underground with use of a mechanics truck to minimize tramming of equipment to the shop.

16.15 Mine Personnel

The ESM mine department will employ 107 people during mine development and ramp up. Once in full production there will be a maximum of 176 mine employees between the different rosters. For the first year, a contractor on a 7-day roster will provide the labour for the underground operations. Once ESM hires operations and maintenance staff, they will work an alternating schedule which provides two 10-hour shifts, seven days per week, less night shifts on Friday and Saturday, which will be unmanned, except for the critical activities for which overtime has been allowed in the labour costs. The roster for the three rotating crews is listed in Table 16.12 below.

Table 16.12: Hourly Labour Roster

Crew	M	T	W	T	F	S	S	M	T	W	T	F	S	S	M	T	W	T	F	S	S
Crew A	D	D	D	D	O	O	O	O	N	N	N	O	O	N	N	O	O	O	D	D	D
Crew B	N	O	O	O	D	D	D	D	D	D	D	O	O	O	O	N	N	N	O	O	N
Crew C	O	N	N	N	O	O	N	N	O	O	O	D	D	D	D	D	D	D	O	O	O

Source: SLZ (2017)

Mine personnel will reside in nearby towns and will be responsible for transportation to and from the site on a daily basis.

Table 16.13 below outlines the anticipated mine labour force quantities, and rotation schedules.

Table 16.13: Mine Personnel Summary

Position	Roster	Rotation	LOM Average	LOM Max
Mining Management				
Mine Superintendent	Salary	5x2	1	1
Mine Maintenance General Foreman	Salary	5x2	1	1
Mine Foreman	Salary	5x2	1	1
Mine Clerk	Salary	5x2	1	1
Subtotal – Mining Management			***4***	***4***
Mining Operations (Production)				
Shift Supervisor	Staff	7/4 5/2-3	3	3
Trainer	Staff	5x2	4	4
Production Drill Operator	Hourly	7/4 5/2-3	5	6
Jumbo Operator	Hourly	7/4 5/2-3	10	12
Bolter Operator	Hourly	7/4 5/2-3	7	9
Stoper/Jackleg Ground Support Miner	Hourly	7/4 5/2-3	11	15
Development Services	Hourly	7/4 5/2-3	10	15
Blaster	Hourly	7/4 5/2-3	5	6

Table 16.13: Mine Personnel Summary (continued)

Position	Roster	Rotation	LOM Average	LOM Max
Scooptram Operator	Hourly	7/4 5/2-3	13	18
Haul Truck Operator	Hourly	7/4 5/2-3	26	30
Grader Operator	Hourly	7/4 5/2-3	3	3
Nipper/Equipment Operator	Hourly	7/4 5/2-3	3	6
Drift Maintenance	Hourtly	7/4 5/2-3	6	6
Dry/Lapman/Bitman	Hourly	7/4 5/2-3	2	2
Subtotal – Mining Operations (Operations)			***107***	***132***
Crushing and Hoisting				
Hoistman	Staff	7/4 5/2-3	3	3
Skip Tender	Staff	5x2	3	3
Crusher Operator	Hourly	7/4 5/2-3	2	2
Lead Shaft Miner	Hourly	7/4 5/2-3	1	1
Shaft Miner	Hourly	7/4 5/2-3	1	1
Subtotal – Crushing & Hoisting			***10***	***10***
Mine Maintenance				
Maintenance Supervisor	Staff	5x2	1	1
Maintenance Planner	Staff	5x2	1	1
Maintenance General Foreman	Staff	5x2	1	1
Master Electrician	Staff	5x2	1	1
Heavy Equipment Mechanic	Hourly	7/4 5/2-3	9	9
Apprentice	Hourly	7/4 5/2-3	3	3
Electrician	Hourly	7/4 5/2-3	3	3
Subtotal – Mine Maintenance			***19***	***19***
Mining Technical Services				
Chief Mine Engineer	Staff	5x2	1	1
Senior Mine Engineer	Staff	5x2	1	1
Junior Mine Engineer	Staff	5x2	1	1
Ventilation / pumping Technician	Staff	5x2	1	1
Surveyor	Staff	5x2	1	1
Technician	Staff	5x2	1	1
Chief Geologist	Staff	5x2	1	1
Senior Geologist	Staff	5x2	1	1
Junior Geologist	Staff	5x2	3	3
Subtotal – Technical Services			***11***	***11***
Grand Total			**151**	**176**

Source: JDS (2017)

16.16 Mine Production Schedule

Mine scheduling for the ESM project was conducted by JDS using Minemax iGantt software. The scheduler seeks to optimize the Net Present Value (NPV) of the operation subject to constraints of development rates, production rates, and backfill rates, and other engineering constraints such as ventilation or equipment congestion.

Underground production was considered to have started as soon as first mineralization is mined. Mining blocks with higher profitability (net $/t) mineralization were targeted in the early stages of the mine life to optimize project economics. Resulting optimized schedules were reviewed and modified where necessary to account for a logical mining approach. One such modification includes placing Mud Pond into production sooner given the high indicated content, proximity to existing development, and ability to quickly mine stopes that were drilled but never fired before mine shut down in 2008.

Annual mine production statistics are provided in Table 16.14.

Table 16.14: Annual Mineralized Material, Waste and Backfill Schedule

Zone	Unit	TOTAL	Year							
			1	2	3	4	5	6	7	8
Mineralized Material Mined										
Mahler Main	ktons	**1,363**	113	343	231	48	141	294	172	22
Mahler QD	ktons	**15**	1	14	-	-	-	-	-	-
Mahler WD	ktons	**353**	35	68	103	93	34	7	13	-
Mud Pond Main	ktons	**573**	50	140	87	94	100	81	22	-
Mud Pond QD	ktons	**50**	48	2	-	-	-	-	-	-
Mud Pond Apron	ktons	**174**	18	59	-	-	-	-	97	-
New Fold	ktons	**904**	12	7	234	348	304	-	-	-
Cal Marble	ktons	**304**	-	-	-	-	42	120	104	38
NE Fowler	ktons	**465**	-	-	-	-	36	155	111	163
Sylvia	ktons	**77**	-	-	3	74	0	-	-	-
Total Mill Feed	**ktons**	**4,278**	**276**	**633**	**657**	**657**	**657**	**657**	**518**	**224**
Production Rate	**t/d**	**1,614**	**756**	**1,735**	**1,800**	**1,795**	**1,800**	**1,800**	**1,418**	**611**
Contained Zinc	**ktons**	**394**	**26**	**48**	**71**	**69**	**59**	**72**	**34**	**14**
Zn Grade	**%**	**9.2**	**9.5**	**7.6**	**10.8**	**10.5**	**9.0**	**10.9**	**6.6**	**6.1**
Waste Balance										
Waste Mined Loose Volume	ft^3 x 10^6	**35.6**	4.7	7.7	6.6	3.7	4.4	3.4	4.0	1.1
Backfill Required	ft^3 x 10^6	**35.6**	2.9	6.7	6.1	4.9	4.8	4.2	4.2	1.9
Waste Rock Backfill Placed	ft^3 x 10^6	**32.4**	2.7	6.5	6.1	4.2	4.4	3.4	4.0	1.1

Source:JDS(2017)

16.17 Mine Development Schedule

The development schedule was based on estimated cycle times for jumbo development.

All waste development during pre-production is shown as capital development.

During the production phase, the decline, ventilation drifts and raises are considered sustaining capital development, but cross-cuts and drifting on the levels were included in the operating costs.

Annual development metres are summarized below in Table 16.15.

Table 16.15: Annual Development Schedule

Development	Units	Total	Year							
			1	2	3	4	5	6	7	8
Rehab Slashing	ft x 1000	**0.7**	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Rehab Bolting	ft x 1000	**58.7**	35.0	8.3	15.4	0.0	0.0	0.0	0.0	0.0
Ramp	ft x 1000	**34.0**	6.3	5.2	7.9	2.8	5.0	2.8	3.1	0.9
Aux	ft x 1000	**10.1**	1.9	1.9	2.8	1.0	1.0	0.7	0.6	0.3
Sub-Level Waste	ft x 1000	**19.9**	0.9	4.7	5.0	2.1	2.9	2.0	1.6	0.5
Sub-Level Mineral	ft x 1000	**25.6**	2.5	6.2	8.6	1.0	1.7	1.1	3.6	0.9
Cut and Fill	ft x 1000	**24.0**	0.0	3.1	0.2	4.2	0.0	0.0	13.3	3.1
Room and Pillar	ft x 1000	**51.8**	0.0	0.0	1.6	12.2	10.7	11.8	12.3	3.2
Total Lateral Development	**ft x 1000**	**224.8**	**47.3**	**29.5**	**41.5**	**23.4**	**21.3**	**18.4**	**34.5**	**8.9**

Source:JDS (2017)

17 Process Description/Recovery Methods

17.1 Introduction

Mineralized material mined in the ESM deposits will be processed at the existing ESM concentrator that was commissioned in 1970 and last shut down in 2008. The concentrator flowsheet includes crushing, grinding, sequential lead and zinc flotation circuits, concentrate dewatering circuits, and loadout facilities. The flowsheet for crushing, grinding and lead flotation is shown in Figure 17.1. The flowsheet for zinc flotation and tailings disposal is shown in Figure 17.2.

The design capacity of the concentrator is 5,000 t/d. Throughout the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines. The traditional operating strategy has been to operate the concentrator at its rated hourly throughput of 200 to 220 t/h, but for only as many hours as necessary to suit mine production. In the last full year of production (2008), the concentrator was operated for 25% of the total available hours in the year.

Brief descriptions of the concentrator circuits, equipment condition assessments, design criteria, and recommendations for work prior to re-starting the concentrator follow below.

17.2 Plant Design Criteria

From a metallurgical perspective, the best way to operate a concentrator is on a continuous basis to minimize the usual occurrences of sub-standard metallurgy on start-up and product losses on shut-downs.

While the mill has a capacity of 5,000 t/d, mine production is typically less than 1,800 t/d. The mill is operated for eight to 10 hours per day. This inherently introduces instability during start-up and shut-downs. A better mode of operation would be to stockpile mineralized material on surface, and operate the mill continuously for periods of at least one week.

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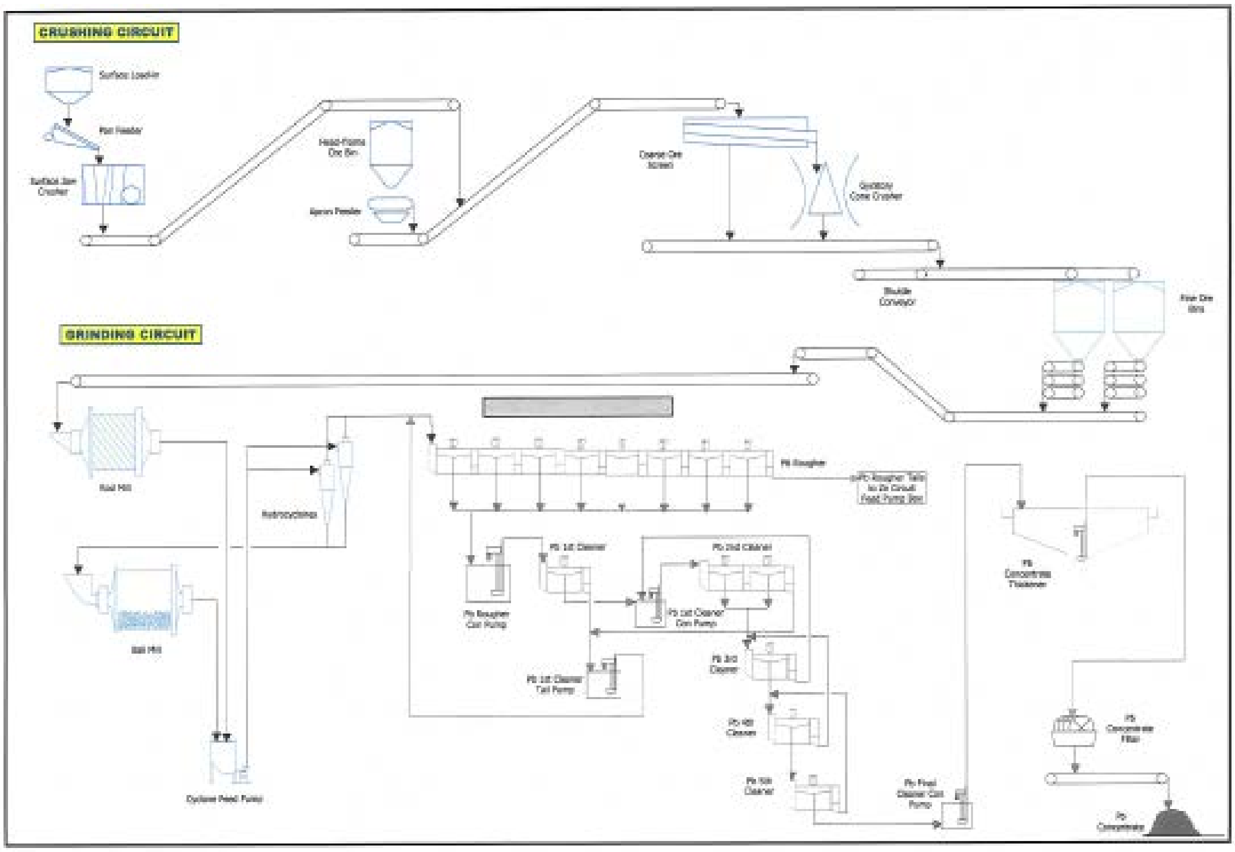
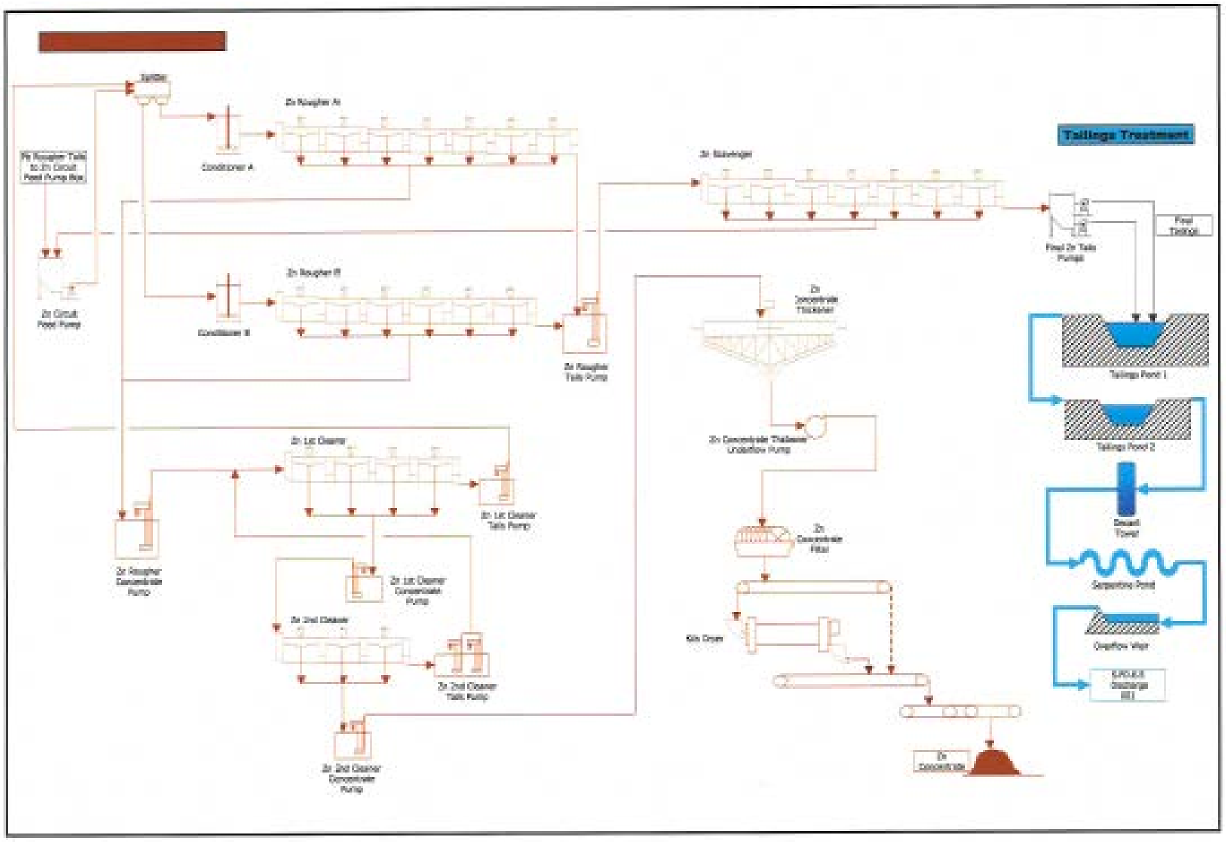
Figure 17.1: Crushing, Grinding and Lead Flotation Flowsheet



Source: SLZ (2017)

Figure 17.2: Zinc Flotation and Tailings Disposal



Source: SLZ (2017)

17.2.1 Crushing Circuit

Primary crushing will either be done underground by a 36” by 48” jaw crusher, or on surface by a 30” by 42” jaw crusher set up outside the concentrator.

Coarse material from the surface crusher or the shaft hoist will be conveyed to the secondary crusher by a 36” conveyor, equipped with an electromagnet for tramp removal. A Corrigan metal detector is situated near the top end of the conveyor and is interlocked with the conveyor. There is a picking station at the top of the conveyor for observation and removal of scrap by an operator.

Coarse material from the above conveyor will be discharged into the feed chute of a 6’ by 14’ Tyler Tyrock Screen, Model F-900. The screen undersize reports to the #2 conveyor and the screen oversize reports to the crusher. Records indicate that the screen deck opening size is 1.5”.

The crusher is an Allis Chalmers Hydrocone, Model 1084 EHD (84” diameter, extra heavy duty) equipped with a 300 hp motor. The crusher will operate in open circuit, discharging to the #2 conveyor, to be combined with the screen undersize.

In a Hydrocone crusher with an intermediate chamber, the close-side setting can be set between ½” and 2” with corresponding capacities in the order of 275 to 400 t/h. The total circuit capacity will be greater than this by an amount equal to the fines in the feed that are screened out before entering the crusher.

The cone crusher has not been rotated or bumped since shutdown in 2008. A thorough inspection will be required prior to recommissioning.

Conveyor #2 is equipped with a four-idler Merrick weightometer, and discharges via a transfer chute to the #3 conveyor that runs to the top of the fine muck bins. An automatic sampler is installed on this belt. Discharge from the #3 conveyor will be distributed between the two fine muck bins by a shuttle conveyor. Each fine muck bin has a rated capacity of 2,000 t.

Historic production records show that the operating hours on the crushing plant were approximately the same as that of the grinding circuit, i.e., crusher throughputs were the same as mill throughputs. Undoubtedly the actual capacity of the crusher would be higher than indicated by the records, and in any case should be more than adequate for future requirements. The crusher may have been operated at low capacity (with a tight gap setting) by choice, given that the crusher operates in open circuit and the product size from the crusher will have a direct impact on the feed size to the rod mill and on the final grind size. The crushing circuit design criteria are shown in Table 17.1.

Table 17.1: Crushing Circuit Design Criteria

Design Criteria	Units	Value
Crushing Circuit Operating Time	hours/day	10 – 12
Crushing Circuit Operating Time	days/week	4 – 5
Design Throughput	t/h	220
Muck Feed Size to Secondary Crusher, 80% Passing (estimated)	in.	4
Type of Screen	Vibrating Single Deck	
Aperture Size	in.	1.5
Screen Dimensions	ft.	6 x 14
Installed Motor on Screen	hp	30
Type of Secondary Crusher	Cone	
Secondary Crusher Bowl Diameter	ft.	7
Installed Motor on Secondary Crusher	hp	300
Secondary Crusher Discharge Size, 80% Passing (estimated)	in.	1

Source: TR (2017)

17.2.2 Fine Muck Bin

There are two bins with a nominal capacity of 2,000 t each. It was not possible to inspect the interior of the bins during the site visit. The condition of liners and the live capacity of the bins could not be estimated, but there were no indications of any particular problems. Some repairs to the steel were made approximately 20 years ago.

Each bin is fitted with three slot feeders and DC variable speed drive conveyors. The DC drives have been left energized and this will likely have kept the motors dry and in good condition.

17.2.3 Grinding Circuit

Fine crushed mill feed is conveyed to the rod mill on a 36” conveyor equipped with a four-idler Merrick weightometer.

The rod mill is an 11.5’ by 16’ Allis Chalmers mill with a 1,000 hp Allis Chalmers synchronous motor. The mill will operate in open circuit, and will be charged with 4” diameter rods.

The ball mill is a 12.5' by 14' Allis Chalmers mill with a 1,000 ”p motor (identical to the rod mill motor). The mill will be charged with 2” diameter balls, and operated in closed circuit with two Warman 26” cyclones.

Typical mill feed rates were in the range of 200 to 220 t/h. The final grind size was normally 80-85% passing 65 mesh.

The media charges were left in the mills on shutdown. The lubrication systems have been operated regularly and the motor switchgear has been kept energized. The mills have not been rotated periodically since shutdown in 2008. The mills were thoroughly ground out prior to shutdown which minimizes the probability of the charges becoming frozen or stuck together.

A frozen charge will lift and move as a single mass upon start-up and will crash as the mill rotates, causing possible damage to the mill and its bearings and drive.

The rod mill needs to be relined prior to recommissioning. The liners are at site ready for installation once the decision to re-start is made.

The existing grinding circuit will be adequate for future requirements. Laboratory test work on the proposed mill feed has indicated that there is no benefit in grinding any finer than was done in the past. If future plant test work does show that finer grinding improves metallurgical performance, this could be accomplished simply by reducing throughputs and increasing operating time.

Table 17.2: Grinding Circuit Design Criteria

Design Criteria	Units	Value
Grinding Circuit Operating Time	hours/day	10 – 12
Grinding Circuit Operating Time	days/week	4 – 5
Design Throughput	t/h	200
Balmat Mill Feed Material Work Index	kWh/ton	8.3
Rod Mill Diameter	ft.	11.5
Rod Mill Length	ft.	16
Installed Motor on Rod Mill	hp	1000
Required Power on Rod Mill	hp	1000
Grinding Rod Size	in.	4
Estimated Charge Volume	%	35
Rod Mill Feed Size, 80% Passing	µm	25,000
Rod Mill Discharge Size, 80% Passing	µm	650
Ball Mill Diameter	ft.	12.5
Ball Mill Length	ft.	14
Installed Motor on Ball Mill	hp	1000
Required Power on Ball Mill	hp	1000
Grinding Ball Size	in.	2
Estimated Charge Volume	%	34
Ball Mill Feed Size, 80% Passing	µm	1000
Cyclone Diameter	In	26
Number of Operating Cyclones		2
Cyclone O/F, 80% Passing Size	µm	150

Source: TR (2017)

17.2.4 Lead Flotation Circuit

Cyclone overflow reports by gravity to the head end of the lead circuit. The lead rougher circuit consists of a single bank of eight Wemco 300 ft^3 cells.

All of the air inlet ports on the Wemco cells are wide open; it appears that control valves or slide gates were not in use. This is not unusual for Wemco cells.

The current geologic model suggests that ESM mill feed will have lead values in the order of 0.02%. At this low level, it will not be necessary to run the lead circuit. During the last period of operation, the lead flotation circuit was used to pre-float talc during periods of low zinc head grades (<3%). Excessive talc in the final concentrates results in high magnesium content and will incur penalties.

Two options for utilizing the existing lead circuit are put forward for consideration:

- Maintain the circuit in serviceable condition in case there are short-term lead spikes in the feed, i.e., when the mill is treating a high proportion of Type 2 mill feed. It is unlikely that a marketable lead concentrate would be produced, and the concentrate could simply be pumped to the final tails pumpbox. A splitter box should be installed at the head end of the zinc circuit to divert feed either to the zinc circuit or to the lead roughers as needed; and
- Use lead rougher as a talc “pre-float” to remove excessive talc from low grade pulps.

17.2.5 Zinc Flotation Circuit

The zinc rougher circuit consists of two parallel banks of Wemco 300 ft^3 cells. There are seven cells in the east bank and six cells in the west bank.

At the end of the west rougher bank is a tails box equipped with a vertical sump pump that pumps tailings from both rougher banks to the scavenger bank.

All motor stands on these cells have been reinforced.

The scavenger circuit consists of a single bank of seven Wemco 300 ft^3 cells. All motor stands on these cells have been reinforced.

The zinc cleaner circuit consists of four Denver 300 ft^3 cells as first cleaners and three Denver 300 ft^3 cells as second cleaners. These cells appear to be in good condition.

Design criteria for the zinc rougher/scavenger flotation circuit are shown in Table 17.3. The lead circuit was not included, at this point it is assumed that the lead circuit will be by-passed the majority of the time.

The retention times in roughing and scavenging stages are 15 minutes and eight minutes respectively. The retention times in the first and second cleaner stages are nine and 11 minutes. Normal design practice would be to provide approximately the same retention times in cleaning as in roughing. Given the fast kinetics of ESM mill feed, this may not be an issue. However, if it becomes evident in operation (from high circulating loads) that the cleaner capacity is too low, the mill feed rate could be lowered as necessary to reduce the load on the cleaners. Design criteria for the zinc first cleaner and zinc second cleaner flotation circuits are shown in Table 17.4 and Table 17.6, respectively.

Table 17.3: Zinc Rougher/Scavenger Flotation Circuit Design Criteria

Design Criteria – Zinc Roughers	Units	Value
Solids Feed Rate into Zinc Circuit	t/h	200
Zinc 1st Cleaner Tails to Zinc Roughers	t/h	53
Feed Pulp Density	% w/w	39
Feed Flowrate into Zinc Circuit	g/m	1,940
Existing Zinc Rougher Cells:		
- type (Wemco self-aspirated)		
- individual cell size	ft^3	300
- number of cells		13
- installed motor size in each cell	hp	30
Total Zinc Flotation Rougher Retention Time	min	15
Zinc Rougher Concentrate:		
- grade	% Zn	28
- zinc recovery	%	112
- solids to zinc rougher concentrate	t/h	94
- % solids	% w/w	35
- flowrate	g/m	640
Existing Zinc Scavenger Cells:		
- type (Wemco self-aspirated)		
- individual cell size	ft^3	300
- number of cells		7
- installed motor size in each cell	hp	30
Total Zinc Scavenger Flotation Retention Time	min	8

Source: TR (2017)

Table 17.4: Zinc First Cleaners Design Criteria

Design Criteria – Zinc First Cleaners	Units	Value
Solids Feed Rate into Zinc First Cleaners	t/h	102
Feed Pulp Density	% w/w	31
Feed Flowrate into Zinc First Cleaners	g/m	1008
Existing Zinc First Cleaner Cells:		
- type (Denver forced air)		
- individual cell size	ft^3	300
- number of cells		4
- installed motor size in each cell	hp	30
Total Zinc First Cleaner Retention Time	min	9
Zinc First Cleaner Concentrate		
- grade	% Zn	49
- zinc recovery	%	103
- solids flow rate zinc cleaner concentrate	t/h	49
- % solids	% w/w	25
- volume	g/m	640

Source: TR (2017)

Table 17.5: Zinc Second Cleaners

Design Criteria – Zinc Second Cleaners	Units	Value
Solids Feed Rate into Zinc Second Cleaners	t/h	49
Feed Pulp Density	% w/w	25
Feed Flowrate into Zinc Second Cleaners	g/m	640
Existing Zinc Second Cleaner Cells:		
- type (Denver)		
- individual cell size	ft^3	300
- number of cells		3
- installed motor size in each cell	hp	30
Total Zinc Second Cleaner Retention Time	min	11
Zinc Second Cleaner Concentrate:		
- grade	% Zn	55.5
- zinc recovery	%	96
- solids to zinc second cleaner concentrate	t/h	41
- % solids	% w/w	36
- flowrate	gpm	326

Source: TR (2017)

17.2.6 Lead Dewatering Circuit

The lead thickener is 40’ in diameter, and may have been modified extensively from the original design. There are no rakes, and overflow pipes have been installed in the tank walls at a level several feet lower than the original overflow. There is no underflow pump, it appears that a submersible pump may have been used to extract solids from the bottom of the thickener and pump directly to the vacuum filter.

The lead filter is an 8’ 10” Eimco disc type unit with four of the five possible rows of discs installed. The filter appears to be in good condition. Filtered lead concentrate is conveyed to the concentrate loadout. The concentrate conveyor is equipped with a four-idler Merrick weightometer.

None of the equipment in the lead dewatering circuit was operated during the last production run and will not be required unless high lead grades are discovered and mined in future years.

17.2.7 Zinc Dewatering Circuit

The zinc thickener is a 50’ diameter conventional Eimco unit. The steel in the center well shows signs of corrosion damage. The thickener appears to have been properly cleaned out on shutdown. Thickener underflow is pumped directly to the vacuum filter.

The zinc filter is an 8’10” Eimco disc type with seven of eight possible discs installed. The filter appears to be in good condition and it was flushed out on shutdown.

The vacuum pumps were not seen on the site visit. An equipment list indicates that there are two Nash pumps, one is 100 hp and the other is 125 hp.

Zinc concentrate is conveyed to a 90” diameter by 45’ Koppers oil-fired dryer. It is also possible (with a reversible conveyor) to bypass the dryer. It was reported that the dryer can be by-passed during routine operations. The filter cake typically has high moisture during daily start-up and shut down, requiring operation of the dryer. Mechanically, the dryer appears to be in reasonable condition. The inside of the dryer could not be seen to determine if it was cleaned out on shutdown.

Dried zinc concentrate is conveyed to the loadout. The front end loader used to load trucks is fitted with a load cell in the bucket which is used to weigh shipments.

17.2.8 Ancillary Equipment

Reagent Distribution – There are mixing tanks on the upper floor of the concentrator for copper sulphate, sodium cyanide, sodium sulphide and xanthate as well as storage tanks for the neat reagents (e.g., Cytec 3477, 3418, and MIBC). There are three 12’ diameter copper sulphate storage tanks on the bottom floor of the mill. All tanks appear to have been cleaned out on shutdown, but will need to be inspected and cleaned out as required.

Eco-Gearchem pumps (variable speed) with Krone magnetic flowmeters are used for reagent distribution.

Lime Mixing – the design capacity of the lime silo is 150 t. A drag chain conveyor delivers lime from the silo to a 4’ by 3’ Denver ball mill for slaking. Lime is being used for water treatment at present so the lime slaker is fully operational.

Process water pumps – There are three water pumps installed on the process water lagoon inside the mill.

During the last operating run, lower sections of many steel columns were replaced due to extensive corrosion in the flotation area.

17.2.9 Start-Up Recommendations

In general, the recommended approach to re-starting the ESM concentrator is to:

- Address all obvious and potential safety issues first;
- Test all electrical systems for integrity. Many of the electrical control components are likely obsolete and this is an area of risk;
- Replace the rod mill liners plus any parts that were known to be worn out when the concentrator was shut down;
- Disassemble the cone crusher and fully inspect it prior to recommissioning;
- Do a complete lubrication tour and preliminary mechanical inspection of all equipment;
- Do a water test to check operation of pumps and flotation cells, etc.;
- Do one or more waste runs to test crusher and grinding circuit operation;
- Start concentrator on feed when all critical problems identified in testing have been fixed; and
- Bring in technicians from FLSmidth & Co (FL Smidth) / PERI Group (PERI) to recommission the on-stream analyzer.

17.3 Metallurgical Balance

The concentrator mass balance in Table 17.6 shows estimated stage recoveries and zinc grades based on the locked cycle test results and operating data, extrapolated to the estimated average zinc head of 8.5% for the life of mine.

Table 17.6: Concentrator Mass Balance

Stream	Distribution (%)	Mass Flow (t/h)	Assay (% Zn)	Recovery (%Zn)
Heads	100	200	8.5	100
Zinc Concentrate	14.6	28.1	56	96
Tails	85.4	170.8	0.38	4

Source: TR (2017)

17.4 Water Balance

Overall water balances for the ESM site are summarized in Table 17.7 and Table 17.8 for the following scenarios:

- Plant operating, summer;
- Plant operating, winter;
- Plant not operating, summer; and
- Plant not operating, winter.

The corresponding detailed flowsheets, as well as chemical analyses on a sample of concentrator feed water taken in July 2005 are shown in Appendix 6 of the Hudbay 2005 Feasibility Study (Hudbay, 2005). Water flowrates on these flowsheets were provided in US gallons per day, as submitted in 2005 to the New York State Department of Environmental Conservation in compliance with State Pollutant Discharge Elimination System (SPDES) permits. Flowsheet data was provided by ESM personnel.

Table 17.7: ESM Water Balance, Plant Operating

Water Inflow	US gal/d		Water Outflow	US gal/d	
	Summer	Winter		Summer	Winter
Mill Feed Moisture	12,000	12,000	Concentrate Moisture	10,000	10,000
Lake Pumps	851,000	889,000	Plant Water to Tailings	1,577,000	1,716,000
Mine Water	379,000	491,000			
Run-off and Drain Water	345,000	334,000			
Total Inflow	1,587,000	1,726,000	Total Outflow	1,587,000	1,726,000

Source: SLZ (2017)

Table 17.8 ESM Water Balance, Plant Not Operating

Water Inflow	US gal/d		Water Outflow	US gal/d	
	Summer	Winter		Summer	Winter
Mill Feed Moisture	-	-	Concentrate Moisture	-	-
Lake Pumps	45,000	73,000	Plant Water to Tailings	426,000	483,000
Mine Water	279,000	335,000			
Run-off and Drain Water	102,000	75,000			
Total Inflow	426,000	483,000	Total Outflow	426,000	483,000

Source: SLZ (2017)

17.5 Opportunities for Metallurgical Improvement

The ESM concentrator will be required to operate for approximately 30% of the time to handle the proposed mining rates. If ways can be found to increase mine production, the additional tonnage could be handled with no modifications to the plant.

Locked cycle tests produced zinc concentrate grades of 60%. The metallurgical forecast grade was reduced to 56%, in part from operating results from 2006 to 2008. However, it may be possible to produce higher grades than forecast, and future plant test work should be directed towards this. As examples, retention times in the cleaner flotation stages are lower than typical design values of today, and an expansion of cleaner capacity may be warranted. It is possible that a forced air type of cell would deliver superior performance to the Wemco cells in the rougher stage, and replacement of these cells is another potential way of improving performance.

The current zinc dewatering equipment consisting of a disc filter and rotary dryer are now largely obsolete. Currently best practice uses vertical pressure filters to produce filtered concentrate with moisture content sufficient for transport. The investment in new vertical pressure filters is usually offset from savings in operating costs. Filtration testing should be completed to determine equipment requirements and provide capital and operating cost estimates.

17.6 Assumptions

- The samples used for the metallurgical test work are representative of the mineralized material planned to be mined in the Mud Pond and Mahler deposits;
- The results of the metallurgical test work conducted at ESM, in conjunction with Lakefield, are representative of the metallurgical results that are anticipated to be produced by the concentrator while in operation;
- Lead values in the mill feed will be generally very low, and lead concentrate is not planned to be produced;
- The recovery of zinc to zinc concentrate is planned to be 96%; and
- The forecast zinc concentrate grade of 56% was reduced from the locked cycle test grade based on:
- Iron in sphalerite increasing from 3% in Type 1 mineralization to 5% in Type 2 mineralization;
- Iron in heads increasing from 0.85% in the locked cycle test to 3.5% based on geological estimates;
- Expected plant inefficiency relative to the locked cycle test; and
- Operating data from the last production run from 2006 to 2008.
- Moisture content will be 6.5% based on historical data.

17.7 Conclusions

Minimal modifications to the ESM concentrator are required for processing the mineralized material to be mined. Mill feed will be similar to that processed during the last production run from 2006 to 2008. Lead concentrations will not be high enough to require operation of the lead flotation circuit.

All major circuits in the ESM concentrator have been reviewed to ensure they are suitable to process the planned design throughput, i.e., up to 635,000 t/a of ESM feed at a rate of 200 t/h upon recommissioning. Appropriate process flow diagrams are included showing a mass balance for design throughput conditions. The following areas of the ESM concentrator have not been reviewed, as future service conditions will be similar to past periods of operation:

- Grinding media storage and charging;
- Reagent mixing capacities;
- Fresh, process and gland water pumps and sewage pumps;
- Slurry pumps;
- Compressed air supply;
- Flotation air blowers;
- Vacuum pumps;
- Building heating; and
- Metallurgical Sampling/accounting.

18 Project Infrastructure and Services

18.1 General Site Arrangement

The general site arrangement is depicted below in Figure 18.1. No modifications to the site layout have been made since mine closure in 2008.

Figure 18.1: Empire State Mines General Site Arrangement

Source: JDS (2017)

18.2 Roads/Barging/Airstrip/Rail

Access to the ESM facility is by existing paved state, town and site roads. All access to the mine/mill facility as well as concentrate haulage from the facility is by paved public roads and/or an existing CSX rail short line. The existing facilities at ESM are well established and will generally meet the requirements of the planned operations with practically no modifications.

The ESM site is located adjacent to State Highway 812, approximately 1.5 miles from the junction with State Highway 58. A mile long stretch of Sylvia Lake Road currently handles traffic to and from the site, including truck haulage of concentrate to the Port of Ogdensburg should overseas shipping be used. Road maintenance is carried out by the Town and State Government Department of Highways.

There are currently two entries from Sylvia Lake Road providing access to the site. The main entry gives access to the parking lot and the approach to the office complex and the tailings line entry is the waste truck haulage route to the tailings impoundment. These accesses are adequate and no improvements are planned.

18.3 Buildings and Structures

Northeast Construction was the primary contractor for the No. 4 mine shaft and main office facilities. The No. 4 mine shaft was completed in the spring of 1972.

The office complex was completed in the fall of 1971. The mill facility was constructed by Northeast Construction Company starting in April 1970 until its completion in August 1971. The new mill started operations in the spring of 1972. Building construction details are available in Table 18.1.

The quality of construction is very good. Much of the steel is galvanized and the corrugated siding is heavy and has weathered the elements well. The buildings were well-maintained during the 8-year care and maintenance period between 2008 and 2017.

Table 18.1: List of Buildings and Structures

Building Name	Dimensions (ft)	Area (ft^2)	Construction Type	Year Built
#2 Mine Office Complex, Maintenance Office, Change Room (all unheated)	142 x 60	8,520	Steel frame, Steel sheet on gorts, steel sheet roof, roof trusses	1929
	80 x 47	3,760		
#2 Mine Switch Gear (unheated)	62, 47	2,604	Steel frame, steel sheet on girts, steel sheet roof, roof trusses	1929
	25 x 19	475		
#2 shaft warehouse (unheated)	28 x 100	2,800	Steel frame, steel sheet on girts, steel sheet on purlins	1929
#2 Electrical substation (unheated)	25 x 58	1,450	Concrete block, built up roof on concrete, plant on trusses	1929

Table 18.1: List of Buildings and Structures (continued)

Building Name	Dimensions (ft)	Area (ft^2)	Construction Type	Year Built
Headframe	26 x 51 8 x 70	3,362	Steel frame, galbestos insulation panel & galbestos sheet, membrane on conc. Plank upper roof, galbestos sheet lower roofs.	1969
Hoist House	135 x 138	18,630	Steel frame, conc. Block lower, galbestos insulated panel lower, insulation on membrane	1969
Maintenance and Warehouse	125 x 273	34,125	Steel Frame, galbestos insulation panel, Built up roof on conc. Plank w/ steel joists	1970
Maint Vehicle Storage, Boiler Room, Change Room	60 x 273	16,380	Steel Frame, Conc. Block	1970
Concentrator 4A	133 x 267	35,511	Steel Frame, Conc. Block Lower, Galbestos	
4B	46 x 80	3,680	Insulation panel lower, membrane roof on	1970
4C	67 x 97	6,499	conc.W3 steel joists	
Maintenance Shop 2-story (heated)	36 x 104	3,672	Steel Frame, Conc. Block, Built up roof on conc.	1970
Storage	70 x 140	9,800	Steel Frame, Steel sheet w/fiberglass sheet	1970
Concentrate Storage	60 x 98	5,880	Steel Sheet roof on steel purlins	1970
Concentrate Storage 2-story (unheated)	94 x 161	15,134	Steel Sheet roof on steel purlins	1970
Timber Storage Building (unheated)	29 x 118	3,422	Steel frame, conc.block lower, galbestos on upper, galbestos roof on steel joists	1970
Elec. And Tire Storage (heated)	24 x 40	960	Steel frame, conc.block lower, galbestos on upper, galbestos roof on steel joists	1970
Pine Oil Storage (heated)	22 x 32	704	Steel frame, conc.block lower, galbestos on upper, galbestos roof on steel joists	1970
Booster Pumphouse (heated)	25 x 33	825	Steel frame, conc.block lower, galbestos on upper, galbestos roof on steel joists	1970

Table 18.1: List of Buildings and Structures (continued)

Building Name	Dimensions (ft)	Area (ft^2)	Construction Type	Year Built
Lake Pumphouse (heated)	20 x 22	440	Steel frame, conc.block lower, galbestos on upper, galbestos roof on steel joists	1970
Fuel Oil Pumphouse (heated)	10 x 10	100	Steel frame, conc.block lower, galbestos on upper, galbestos roof on steel joists	1970
Oil Storage (heated)	30 x 60	1,800	Steel Frame, Steel sheet on steel girts, Steel	1970
Mine Lagoon Pumphouse	14 x 20	280	Conc. Block, built up roof on conc. Plank	1970
Office Complex	64 x 103	13,184	Steel Frame Concrete Block 2ith brick face. Built up roof on conc. Plank w/ steel joists	1970
Warehouse Storage (unheated)	70 x 120	8,400	Steel Frame, Steel sheet on steel girts, Steel sheet roof on steel purlins	1976
Electrical Storage (unheated)	60 x 100	6,000	Steel Frame, Steel sheet on steel girts, Steel sheet roof on steel purlins	1976
Security Gatehouse (heated)	8 x 8	64	Wood frame, vinyl siding, asphalt shingle roof	1985

Source: JDS (2017)

18.3.1 Office Complex

The existing mine office complex, is a two-story steel frame and concrete block/galbestos-sided building with steel joist/concrete plank built up roof system. As part of the first floor, the maintenance vehicle storage garage, the boiler room and the dry/lamp room is a 60 ft x 273 ft area. The dry, located on the ground floor, accommodates 125 men with individual lockers for clean clothes and hanging baskets for working clothes for all personnel, as well as the appropriate number of showers and toilet facilities.

A foreman's locker room is located near the front of this floor and can accommodate 25 supervisors and visitors. Females can use the locker near the mine rescue room which can hold 15 people.

The ground floor also contains mine offices, a boiler room and lamp room. The boiler room houses two Cleaver Brooks 250 HP boilers and one Cyclotherm 100 HP boiler.

The second floor (125 ft x 273 ft) contains a warehouse, machine shop, mine rescue room, first aid equipment room and training room. The warehouse has a 15 t overhead gantry crane and the machine shop has a 25 t crane. For the ESM operation, shipping/receiving will continue to be done from the existing surface warehouse. A second warehouse is located on the 2500 level underground, as part of the mine maintenance shop complex, for the storage of mechanized equipment parts. One warehouse person will work largely underground, except for the receiving of freight on surface.

The first and second floor of the northwestern brick-faced extension of the building (64 ft x 103 ft each floor) is used for office space and currently is organized to provide space for the following personnel and requirements:

- Manager;
- Mine superintendent;
- Mine clerk and surveying;
- Engineering and geology personnel;
- Conference room; and
- Accounting, purchasing and human resources.

The entire office/dry/shop complex is protected by a wet sprinkler system.

18.3.2 Hoisting Facility

The existing hoisting facility is a two-story steel frame and concrete block/galbestos-sided hoist building with steel joist/concrete plank built up roof system and a headframe building of similar construction (26 ft x 51 ft + 8 ft x 70 ft + 26 ft x 51 ft). The headframe is 145 ft high and fully clad. The hoistroom is a 135' x 138' area and contains a 15 t overhead gantry crane. An adjoining compressor room houses (2) Joy 600 hp WN-114-C10 air compressors. There is a bundle-type aftercooler in the discharge line. The compressor room has a 10 t Load Lifter crane. Next to the compressor room is the electrical shop. This is equipped with a 5 t Shaw Box crane.

18.3.2.1 No. 4 Shaft

18.3.2.1.1 Headframe

The 140' tall galvanized structural steel headframe was built in 1972 by Northeast Construction. The upper sheave deck supports two 15' diameter head sheaves grooved for 2 ¼" wire rope which services the production skip compartment. The lower sheave deck supports two 12' diameter head sheaves grooved for 1 ¾" wire rope designed to service the man and material cage, and a counter weight.

The headframe is equipped with a skip discharge structure consisting of two skip dump scrolls, a chute, a diversion gate to separate mineralized material from waste, an muck bin and a waste crib. The muck bin feeds an inclined mill conveyor over a 48" wide by 14' 6" long 20 hp Portec apron feeder.

Details of the structure and condition of the No. 4 headframe and production hoist are in the GL Tiley & Associates (Tiley) report in Appendix 9 of the 2005 Hudbay FS (Hudbay, 2005). The report has been updated by site staff to identify items that have been addressed (Tiley, 2005).

18.3.2.1.2 Production Hoisting Plant

The production hoist is a Nordberg double-drum, double clutch mine hoist with Lebus grooving. The production hoist features two 15' diameter by 8' wide drums each with capacity to handle 3,300' of 2 ¼" head rope. The hoist system is driven by two 1,250 hp 500 rpm DC motors and is capable of hoisting at a speed of 1,750’ per minute. The resultant hoisting rate is 200 t/h. Shaft and hoist related maintenance tasks that affect production hoisting (and hence daily capacity) are shown below.

Table 18.2: No. 4 Shaft Availability

Critical Tasks that Interfere with Skip Hoisting	Hours Per Week
Hoisting Compartment Maintenance	5
Cage & Counterweight Compartment Maintenance	1
Crusher Bin & Flopgate Maintenance	1
Rope Maintenance	0.75
Headframe scrolls & Flopgate Maintenance	2.5
Shaft Mucking	1.75
Hoist Inspections	3
Powder Delivery – 1300	1
Powder Delivery – 2100	2
Powder Delivery – 2500	2
Powder Delivery – 3100	1
Total non-hoist hours per week	**21**
Smoke time hours per week	**14**
Hours per week that hoist is not available	**35**
Hours per day that hoist is not available	**5**

Source: SLZ (2017)

Assuming a hoisting rate of 200 t/h and an average availability of 19 h/d, the resulting daily hoist capacity is 3,800 t of material.

DC power is provided to the hoist from a three-unit motor-generator set which includes a 2,240 hp synchronous motor and two DC generators rated at 1,000 kW.

The hoist controls are 1970 vintage, using relay logic and printed circuit boards. The safety devices are single governor Model Lilly C controllers.

Obsolete field supplies and analog controls were replaced in 2001.

18.3.2.1.3 Service Hoisting Plant

A Nordberg, Lebus grooved, double-drum, single clutch mine hoist transports personnel, equipment and materials into and out of the mine. The service hoist features two 12’ diameter by 91” wide drums each holding 3,300’ of 1 ¾” head rope and driven by a single 900 hp 400 rpm DC motor. The maximum hoisting speed is 1,190’ per minute. When the hoist is used for mine equipment moving operations, it can handle a maximum piece weight of 13 t. The cage rope is new in December 2014, and the counter rope new in March 2017.

DC power is provided to the hoist from a two-unit motor-generator set which includes a 920 hp synchronous motor and 1 DC generator rated at 720 kW.

Details of the hoisting system are in the Tiley report in Appendix 9 of the 2005 Hudbay FS (Tiley, 2005). A list of capital improvements made to the hoisting facility is in the Hudbay 2010 AFE document (Hudbay, 2010).

18.3.2.2 No. 2 Shaft

18.3.2.2.1 Headframe

The hoist building and headframe is a brick and steel structure which supports two headsheaves and houses the skip loadout facility. The headropes are supported by an intermediate set of two idler sheaves located between the hoist room and headframe.

The steel in the headframe is not in very good condition but is capable of continued emergency service until repairs can be completed.

Details of the structure and condition of the No. 2 headframe are in the Tiley report in Appendix 9 of the 2005 Hudbay FS.

18.3.2.2.2 Hoisting System

An Ottumwa Iron Works double-drum, double clutch mine hoist lifts and lowers personnel, equipment and materials out of the mine. The service hoist features two 84” diameter by 76” wide drums each holding 3,300’ of 1 ¼” head rope and driven by a single 700 hp 514 rpm wound rotor induction motor. The maximum hoisting speed is 1,150’ per minute.

The hoist controls are very basic including a speed lever, two brake and two clutch levers, emergency stop and hoist speed indicators. The safety devices are two Model D Lilly controllers.

The hoist is in adequate condition and has all the safety equipment to operate within the MSHA code 30 CFR 57 regulations.

18.3.3 Concentrator and Support Facilities

The existing mill and support facility is a steel frame and concrete block/galbestos sided building with steel joist/concrete plank built up roof system. The concentrate mill is a three section, four-story heated building (133' x 267' + 46' x 80' + 67' x 97') complete with a raised mill control room, physical and analytical labs, offices and x-ray room.

A two-story heated pipe shop (36' x 104') has full facilities with a 2 ton Demag bridge crane is contiguous. Three, two-story cold storage (70' x 140' + 60' x 98' +94' x 161') areas give plenty of room for storage of critical spares.

18.3.4 No. 2 Mine Escape Shaft Complex

The escape hoist facility is a steel frame hoist building and a headframe building of similar construction. The hoist room is 62' x 42' with a 25' x 19' switch gear room. A mine office/shaft complex (60' x 142' + 80' x 47') is unheated.

18.3.5 Storage and Miscellaneous Facilities

The following building list in Table 18.3 makes up the rest of the facility.

Table 18.3: Facility Building List

Building	Dimensions
Timber Storage Building	29' x 118'
Electrical & Tire Storage	24' x 40'
Pine Oil Storage	22' x 32'
Booster Pumphouse	25' x 33'
Lake Pumphouse	20' x 22'
Fuel Oil Pumphouse	10' x 10'
Warehouse Storage	70' x 120'
Electrical Storage	60' x 100'
Oil Storage House	30' x 60'
Mine Lagoon Pumphouse	14' x 20'
Security Gate House	8' x 8'

Petroleum and chemical storage tankage at ESM are currently registered by the New York State Department of Environmental Conservation (NYSDEC). All tanks and tank farms have containment areas. A list of all surface tanks follows below.

18.4 Power

The primary feed for the ESM is 115kV originating from Niagara Mohawk's substation at Battle Hill-Balmat #5 circuit. Downstream from the main power supply are two (2) 7500 kVA General Electric transformers that feed the ESM plant. Secondary voltage of 4,160 volts feeds sub-feeders to mill, mine, the No. 4 vent fan, lake pumps and booster pumps.

At the ESM No.4 main vent fan location, there is a 1,000 kVA 4,160 volt to 480 volt step-down transformer substation. The substation switchgear is General Electric Magne Blast.

The primary feed for the No.2 hoist fan unit is the Niagara Mohawk 23 kV Balmat-Emeryville circuit #24. Downstream from the main power supply are two (2) 3750 KVA General Electric transformers (23000-2200) feeding the surface plant with secondary voltage of 2300 V for sub-feeders.

The No. 2 vent fan feeder is part of the mine feeder vent fan transformer 300 kVA in the substation by the vent fan. Substation switchgear is General Electric Magne Blast. There will be plenty of power to run the proposed 300 hp fan on the surface as well as the mine air heater, if required.

There are three small miscellaneous electrical services around the main property. Other services from Niagara Mohawk (National Grid) are:

- Lighting for the No. 2 mine entrance; and
- South dam pumphouse at the tailings area.

SLZ owns two portable generators for emergency use. One is a125 kVA portable used for general 480V / 220 V / 110 V applications. The other is a 100 kVA portable generator which will run the No. 2 emergency egress hoist.

Niagara Mohawk supplies the transmission and energy, although SLZ has the option to go to other energy suppliers. In any case, ESM would continue to use Niagara Mohawk as the transmission company.

18.5 Water

18.5.1 Water Supply

The current non-potable water supply system will be adequate to supply the ESM project for shower, boiler make up, toilet facilities, etc. with no modifications envisaged at this time. Non-potable water will be supplied by a 6 hp, 9-stage, 460 V, Goulds Model 55 GS 30 well pump which is capable of 50 gallons per minute (g/m) at 65 psi. This well is located near the fence line at the front gate location. The water will run through an underground 2" Sclairpipe (HDPE) to the vehicle storage building where it will be treated by a Magnum CY 962 water softener before it will enter one of two 1,000 gal. holding tanks. A chlorinator injection system (Pulsatron metering pump) injects 0.5 to 1.5 mg of chlorine per litre of water throughput. A Burks 5 hp pump will deliver 65GPM at 70 psi to feed a series of three bladder tanks (total drawdown capacity of 94 gal. between 40 and 60 psi) to be used for toilets and showers.

When the total number of employees on-site will reache over 50 people, the chlorine residual will be monitored on a daily basis and the result recorded as per NYS Dept. of Health code 360. The Department of Health will review this report monthly. A monthly water sample will be submitted for a coliform bacteria test.

Mill process and cooling water (non-potable) for the site will be pumped from the Sylvia Lake pumphouse with (3) Worthington 14-135-2, 75 hp pumps rated @ 1,500 g/m. The third pump will constitute excess capacity and the other two cycle off and on. Pump discharge will be through a 10" pipe to (2) 100,000 gal. Each of the concrete deluge tanks (a concentrator water tank and a fire pump storage tank) are near the concentrate storage building/rail loadout shed. Water is pumped from the reservoir tanks to the concentrator. Mine water will be pumped from the mill basement sump down the 4" shaft water line to the various mine levels.

Grey water from the surface facilities, surface run-off, water from the facility catch basins, and overflow from the reservoir tank will be directed to the mill holding pond. Waste water from the holding pond will be either recycled in the mill or pumped to the tailings dam through a pipeline comprising of 5,000' of 14" diameter Sclairpipe. From the tailings area, it will flow northeast through a series of settling and polishing ponds before it will be discharged to the environment.

18.5.2 Water Treatment

Water from the tailings area polishing pond can be treated with a reagent dosing system to precipitate metals and suspended solids. The dosing system consists of a variable speed auger which meters sodium sulphide into the effluent. The zinc and iron will be precipitated out of the water at this point. There will be no need to run the dosing system for eight months per year due to the warmer temperatures. The warmer water promotes biomass activity that will help filter metals and other solids. The treated water will drain by gravity over the State Pollutant Discharge Elimination System (SPDES) discharge point #0001 for discharge to the environment. The discharge water at this point meets all environmental regulations. Since January 2009, all treatment of mine dewater has been successfully accomplished with lime.

18.5.3 Water Balance

Mine water balances are calculated seasonally for May to October (summer) and November to April (winter) conditions. During the operating summer months, a total of 851,000 gallons per day (g/d) of fresh water will be drawn from Sylvia Lake. ESM underground workings will produce 379,000 g/d of inflow. The mine inflow and process water will be collected and pumped through the tailings pipeline to the tailings at a rate of 1,577,000 g/d. Also, tailings area run-off will add to this volume so that the water treatment plant will see an average discharge at the SPEDES outfall of 2,350,000 g/d.

During winter months, it is estimated that the water inflows into ESM will increase to 491,000 g/d. Also during winter, the fresh water intake from Sylvia Lake will be increased to 889,000 g/d average. The tailings line discharge will see an average flow increase over the warmer months of 1,716,000 g/d. Tailings area run-off will add to this volume so that the water dosing system will see an average discharge at the SPEDES outfall of 2,640,000 g/d.

The full operation water balance is predicated on a 362-day operating year, 1,750 t/d of mill feed production and 110,000 to 115,000 t/a of concentrate production.

18.6 Waste Rock Management

The mineralized material and waste rock from the development and operation of the mine is nonacid-generating due to the alkaline nature of the host rock. The designated surface pads were designed such that any run-off will drain to the concentrator pond.

Waste rock from the mine will be hoisted in 10 t bottom dump skips and dumped over a diversion gate to an outdoor storage crib. Waste will be mucked from the crib to surface stockpiles. The maximum size of the stockpile will be 15,000 t. No special permit is required to stockpile waste.

Waste from the surface stockpile will be loaded by a Michigan L-320 FEL to dump trucks and utilized at the tailings for impoundment construction or sold to an aggregate company. The tailings area is 5,000 to 6,000 ft from the stockpile area via a private haul road.

18.7 Tailings Management Facility

Tailings from the mill are pumped to the Tailing Management facility (TMF) where it will be permanently stored.

The TMF is an existing 260 acre conventional impoundment that is fully permitted. The TMF is categorized as low-risk by New York State Bureau of Flood Protection & Dam Safety. In addition to tailing, mine impacted water is also pumped to the TMF at a rate approximately 500 gpm. The TMF is permitted as a discharge facility and continuously operates within compliance limits. Slaked lime and sodium sulphide will be added to achieve water quality discharge standards for an average of five months per year.

The ultimate capacity of the entire 260-acre TMF footprint has been estimated in an undated letter report contained within the 2005 FS study (Hudbay, 2005 – page 296-299) at 20 Mt of tailing at an embankment crest elevation of 675 ft amsl. This would require additional staged construction to raise the containment embankments.

The first embankment raise will be needed to contain to fully contain the 4.2 Mt within the current resource. This stage of construction will require approximately 445,000 yd^3 of fill to be sourced from either mine waste or other local sources. A preliminary estimate of remaining capacity within the active Tailing Pond #1 and without further embankment construction, will approximately be three years of production at 1,600 t/d (1.6 Mt).

While the TMF is classified as a Class D – No Hazard, and there is no visible evidence to suggest otherwise, no design or as-built information exists with the exception of a relatively recent topography map and Google Earth Imagery. It is unknown how the native surface was prepared, what design features were included, what sub-surface conditions existed prior to construction, or the material properties of fill used for construction. At the time of writing this report, the ground surface was covered by 1.5 ft of snow so it was not possible to see the embankment surfaces to establish what types of fill were used during construction. It is assumed to be a combination of waste rock and tailing as reported by the site manager during the visit.

A geotechnical assessment and engineering design is recommended to establish both of the above capacity estimates along with static and seismic stability. Establishing written tailing management plans and systems is also recommended to ensure consistency with design goals and industry best management practices.

The area where tails were last deposited shows that the tailing beach is relatively steep with an average slope of ~3.5%. This suggests that the tails dewatered and consolidated rapidly. During past operating periods, tails were discharged directly from the open-ends of two elevated pipelines. The tails surface reaches an elevation of about 968 feet amsl which is 18 ft above the South Dam crest elevation. This demonstrates an ability to “stack” tails due to rapid dewatering. There will be ample space in this area for drainage and continued tailing containment.

The TMF and discharge water quality management facilities consist of four contiguous areas:

- Tailing Pond #1 (TP1) 190 acres;
- Tailing Pond #2 (TP2) 30 acres;
- Reclaimed Tails Area 40 acres; and
- Polishing Ponds 25 acres.

Tailings Pond 1 (TP1) will be the active area for tailing placement. The South Dam is on the upstream side with a crest elevation of 650 feet amsl. It is 55 ft high with 4h:1v or flatter outside slope. The east embankment crest averages 630 ft in elevation and was constructed from waste rock. The present height of fill is approximately 5 ft above the native ground elevation. The west side abuts rising terrain. The north side is separated from TP2 by a low embankment with a crest elevation of 620 ft. The north end of TP1 is utilized as a settling pond as well as the entirety of TP2. Water will flow from TP1 to TP2 through a culvert in the north embankment.

Tailings Pond 2 (TP 2) will be used as a clarifying pond. It is bounded on the east and west sides by existing topography. The North Dam forms the downstream containment structure with a crest elevation of 618 ft. The downstream toe is submerged beneath a water surface elevation of approximately 595 ft. Flow from TP2 will overflow via a decant tower and pipeline to a series of polishing ponds that make up the rest of the TMF.

The Reclaimed Tails Area abuts TP2 to the east and as the name implies is an area of consolidated and reclaimed tailing.

The polishing ponds will allow additional time for solids to settle and for natural attenuation to improve water chemistry by flow through a passive wet lands system. Water flow will be diverted by a system of dikes that increase flow distance to approximately 4,800 ft. Flow will exit the property boundary at a State Pollutant Discharge Elimination System (SPDES) discharge point where flow measurements and compliance water quality samples will be taken. To achieve discharge standards, slaked lime will be added at the mill to the combined tailing and mine water flow. At times, sodium sulphide may be added to the flow at TP2.

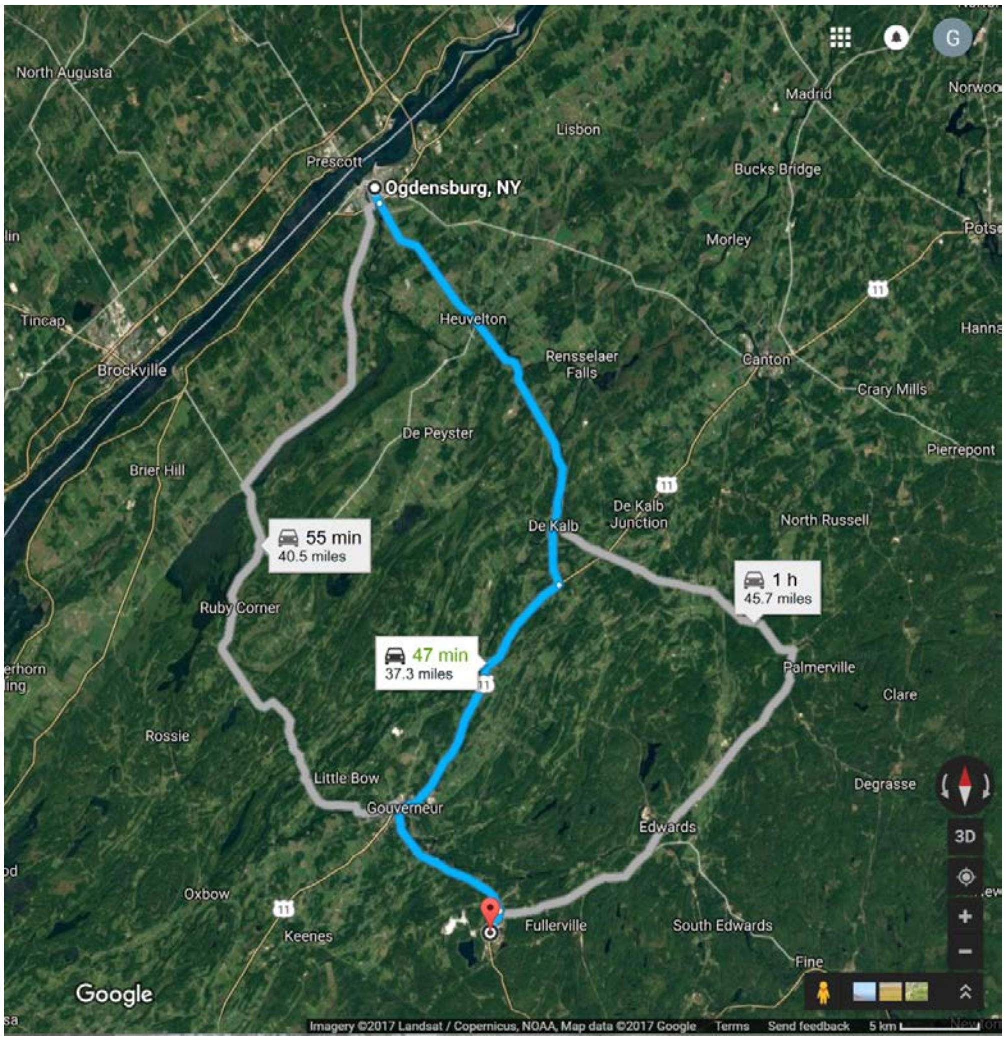
Tailing and waste rock materials at the TMF are non-acid generating due to the high carbonate content of the host rocks. Volunteer vegetation is evident and continues to naturally revegetate inactive areas of the TMF.

18.8 Concentrate Transportation

18.8.1 Roads

A well-maintained system of paved state and county roads surrounds the ESM, providing a year-round option to transport concentrate to a port or smelter by truck if required. The concentrate loading shed at the ESM is designed to accommodate truck loading under cover. The existing railcar scale can be reconfigured to weigh trucks to prevent overloading. Traffic on-site can be routed away from the main compound on a dedicated system of haul roads. Delivery of concentrate to the Port of Ogdensburg, a distance of 38 miles, would be undertaken following highways NY-812 N, NY-58 N, US-11 NE and NY-812 N.

Figure 18.2: Road Access between Empire State Mine and Port of Ogdensburg



Source: JDS (2017)

18.8.2 Rail Lines

The facilities at the ESM were originally constructed using rail as the primary transportation mode for delivering zinc concentrate to market. A four track siding as well as a railcar weigh scale are available. A fiberglass gondola cover lift crane is used to place car covers before shipment. A front end loader would be utilized to load gondola rail cars with capacities of 90 t per car.

The primary rail provider from the siding is the CSX. A CSX – New York Ogdensburg Railroad short line arrangement can be utilized for shipment to the Port of Ogdensburg.

The ESM also has the ability to connect to the rest of the North American rail network, providing access to a number zinc smelters and port facilities.

18.8.3 Sea Ports

The Port of Ogdensburg is located 38 mi northwest of the ESM and is accessed by paved road or by a short line rail system. The Port of Ogdensburg is the only U.S. port on the St. Lawrence Seaway. The facility can receive ships of up to a 27-foot draft over a shipping season between April to December. Owned and operated by the Ogdensburg Bridge and Port Authority, the port provides weighing, covered storage, stevedoring services and ship loading services of bulk cargo. There is also 52,000 ft^2 of bulk warehousing. The dock also offers extensive outside storage for bulk concentrate if required. Inside storage is available for approximately 20,000 t of concentrate.

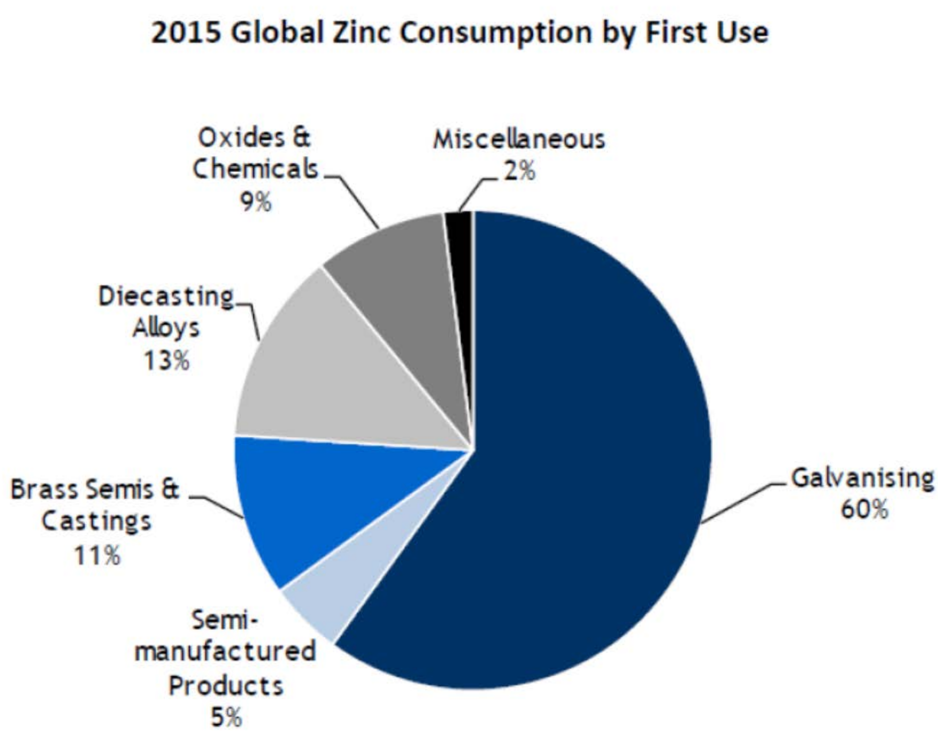
The Ports of Trois Rivieres and Quebec City *are* located approximately 230 mi and 310 mi, respectively, northeast of the ESM and can be easily accessed by truck or rail. These two ports are the primary year-round ports on the St. Lawrence which handle bulk concentrate imports and exports. The facilities can receive ships of up to a 35 ft draft allowing for larger ships for a more efficient transatlantic crossing. The ports provide the full scope of services for bulk cargo trade.

19 Market Studies and Contracts

19.1 Zinc Market

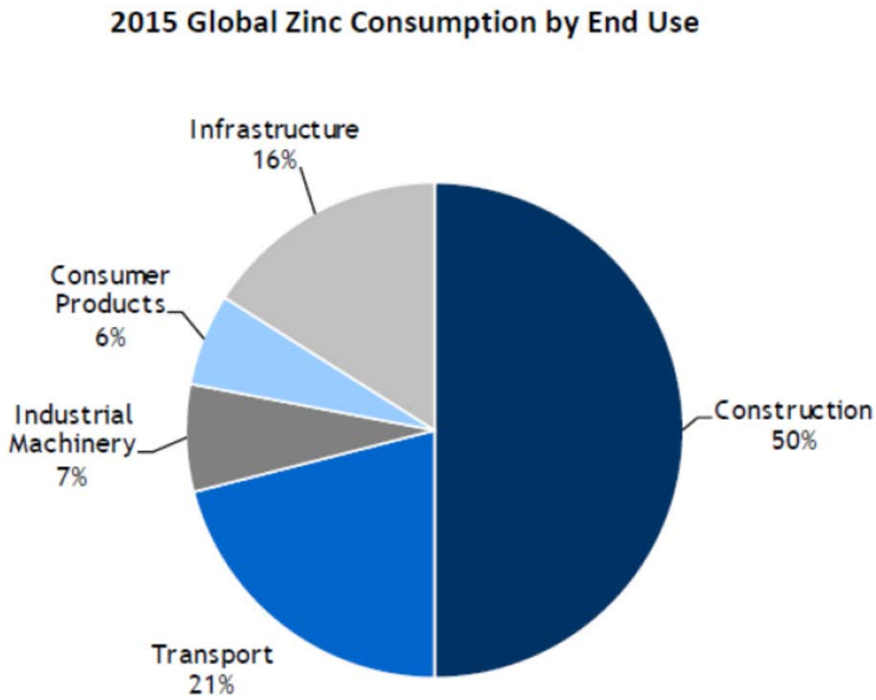
Zinc is one of the most widely used base metals in the world, known for its ability to resist oxidation and corrosion. Global consumption totalled nearly 14 million tons (Mt) in 2016 and is growing at approximately 3% per year. The primary use for zinc is for galvanizing steel, a process of applying a zinc metal coating to steel to prevent rust and corrosion. Approximately 60% of the global zinc consumption is first used for galvanizing, with the remaining 40% used in a number of more specialized industrial processes (Figure 19.1). In terms of end uses for zinc, the construction industry accounts for 50% of the total global production of zinc, with transportation and infrastructure the other major users at 21% and 16% respectively (Figure 19.2).

Figure 19.1: Global Zinc Consumption by First Use



Source: Wood Mackenzie (2017)

Figure 19.2: Global Zinc Consumption by End Use

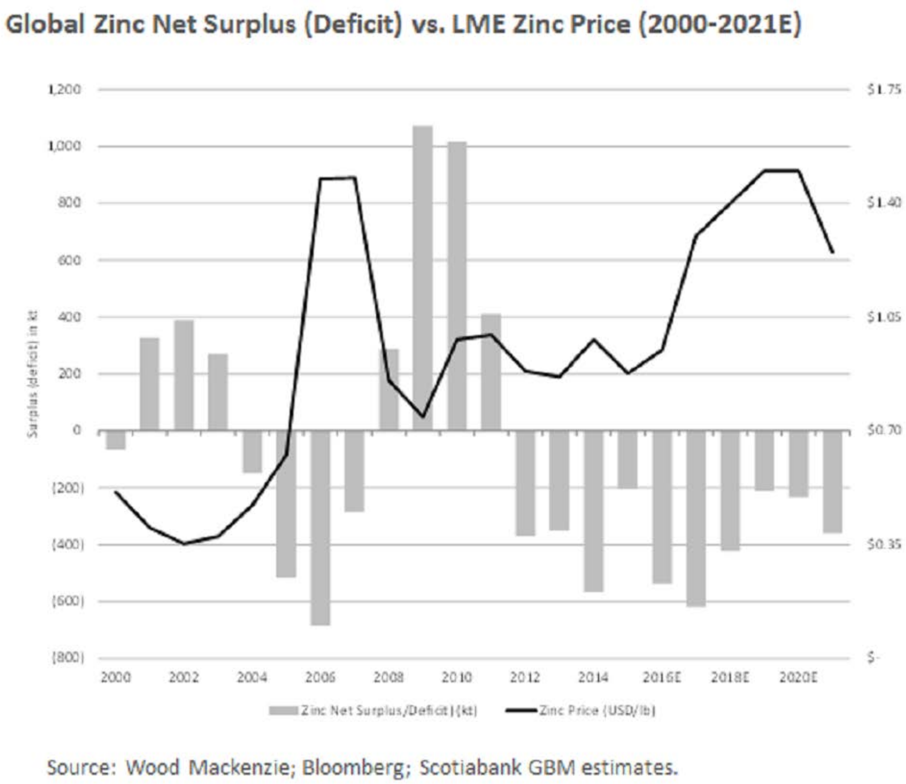


Source: Wood Mackenzie (2017)

19.1.1 Supply / Demand

Since 2012, the global zinc market has been in a state of imbalance, with consumption outpacing supply resulting in a net deficit in the market between 200 – 500 kt (Figure 19.3). In the last five years, stocks of zinc on the London Metal Exchange (LME) peaked in 2013 at just over 1,200,000 t (Figure 19.4). Since then, inventories have dropped nearly 70% to approximately 370,000 t today (Figure 19.5). This supply imbalance is expected to continue in the coming years with China continuing to be the largest global consumer, accounting for nearly 50% of all global zinc consumption.

Figure 19.3: Global Zinc Net Surplus (Deficit) vs. LME Zinc Price

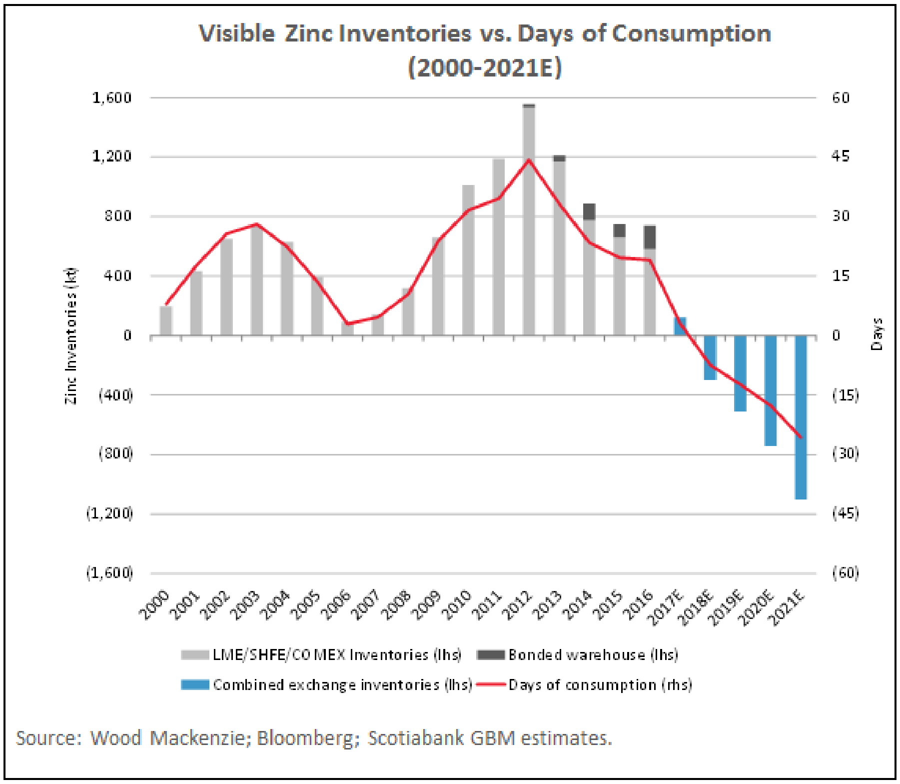


Source: Wood Mackenzie, Bloomberg, Scotiabank (2015)

Figure 19.4: 5-Year LME Zinc Warehouse Stocks Level

Source: Kitco (2016)

Figure 19.5: Visible Zinc Inventories vs. Days of Consumption



Source: Wood Mackenzie, Bloomberg, Scotiabank (2016)

19.1.2 Metal Price History

Over the past five years, zinc prices have ranged from a low of US$0.65/lb to a high of US$1.30/lb. Prices are driven by supply and demand fundamentals, as observed in Figure 19.3, where expanding deficits have been driving prices up in the past few years (Figure 19.6), with shrinking deficits forecast to moderate prices to the US$1.20/lb range over the next few years (Figure 19.3). Metal price history and forecasted future needs were used to establish the metal prices used in this PEA.

Figure 19.6: 5-Year Zinc Spot Price

Source: Kitco (2016)

19.2 Smelter Market

There are a number of operating zinc smelters around the world, including four in North America (Table 19.1) and several overseas smelters in Europe, Asia and Latin America.

Table 19.1: North American Zinc Smelters

Company	Plant Name	Location	Zinc Capacity (kt)
Glencore	Valleyfield	Valleyfield, QC	265
Nyrstar	Clarksville Zinc	Clarksville, TN	124
Hudbay	Flin Flon Zinc	Flin Flon, MB	115
Teck	Trail Zinc Plant	Trail, BC	290

Source: JDS (2017)

19.2.1 International Zinc Smelters (Partial List)

Table 19.2: International Zinc Smelters

Company	Plant Name	Country	Zinc Capacity (kt)
Glencore	San Juan de Nieva	Spain	486
Glencore	Nordenham	Germany	150
Glencore	Portovesme	Italy	Not operating
Nyrstar	Balen	Belgium	260
Nyrstar	Budel	Netherlands	291
Nyrstar	Auby	France	172
Nyrstar	Hobart	Australia	271
Boliden	Kokkola	Finland	290
Boliden	Odda	Norway	170
Korea Zinc	Onsan	South Korea	550
Hindustan Zinc	Chanderiya, Debari, and Dariba	India	747
Votorantim	Cajamarquilla	Peru	300
Shaanxi Nonferrous Metals	Mianxian Operations	China	340
China Minmetals	Zhuzhou	China	450

Source: JDS (2017)

19.2.2 Concentrate Terms

Although there have been efforts to adjust the industry standard zinc payable formula to better reflect actual recoveries, zinc smelters pay for 85% of the value of contained zinc metal in concentrates, subject to a minimum deduction of eight units, applicable when the concentrate grade is less than 53.33% zinc. Additional payable byproducts may include gold and silver when levels are sufficiently high enough.

Spot treatment charges for zinc concentrate have dropped steadily since 2015 as global supplies of concentrate have been reduced due to several mine closures. Current spot treatment charges are in the \$30 to \$50/dmt range compared to 2017 annual contract terms in the \$170/dmt range. Price participation “escalators” and “de-escalators” have traditionally applied to treatment charges which typically vary under the annual ‘benchmarks’ from -4 to +8% relative to the base case price.

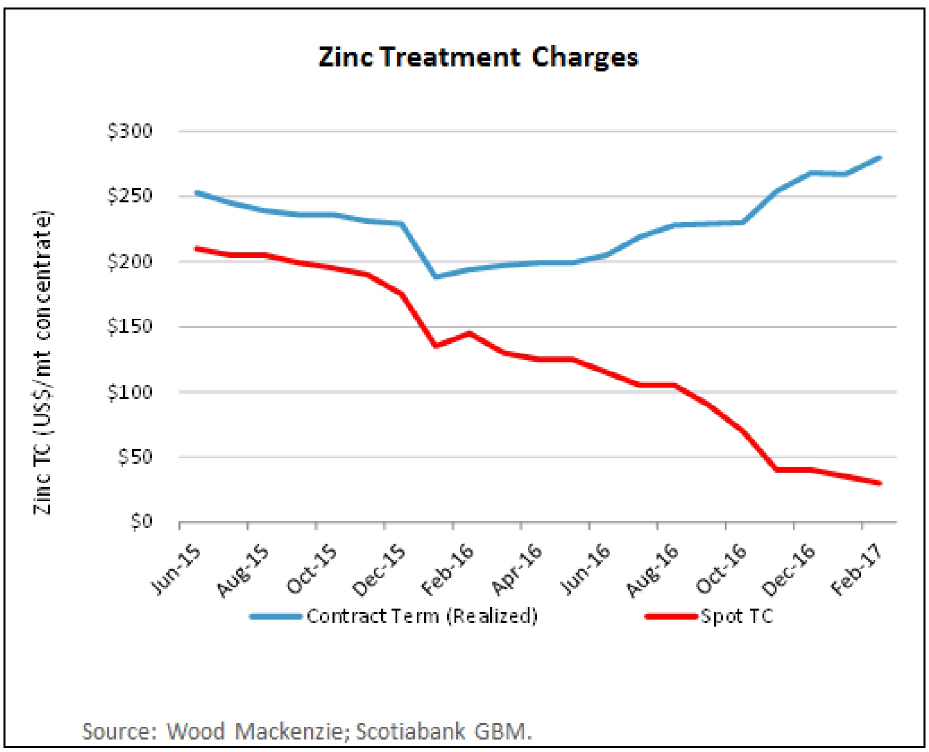
Penalties may be assessed to concentrates containing impurities such as iron, cadmium, lead, manganese, cobalt, magnesia and/or mercury above threshold values.

Table 19.3: Empire State Mine – Zinc Pricing and PEA Concentrate Terms*

Term	Value
Zn Price**	2017 - $1.25/lb 2018 - $1.45/lb 2019 - $1.40/lb 2020 - $1.35/lb 2021 - $1.20/lb 2022 - $1.05/lb LT - $1.05/lb
Zn Concentrate Treatment Charge	US$150.00/dT
Zn Payable	85%
Zinc concentrate grade (% Zn)	56%
Freight costs	US$85/wT
Losses and Penalties	US$15/dT
Concentrate Moisture Content	6.50%

*All terms to be negotiated
**""Year" runs from June - May
Source: JDS (2017)

Figure 19.7: Zinc Smelter Treatment Charges



Source: Wood Mackenzie, Bloomberg, Scotiabank (2016)

19.3 Contracts

No contract for concentrate sales are in place as of the publishing date of this report.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Studies

Since 1915, six zinc mines have operated in the Balmat-Edwards district. Zinc was first produced from the Edwards mine in 1915 and from the Balmat No. 2 Mine in 1930. The other mines in the district are the Balmat No. 3, Balmat No. 4, Hyatt, and Pierrepont. The only remaining operating mine is No. 4. No. 2 is used for ventilation and as an alternate mine escape route. The other sites are successfully reclaimed and no longer subject to permit or financial assurance obligations. The company does monitor the sites routinely as part of their ongoing management practices.

The waste rock and tails is non-acid generating so there are no issues or concerns with material reactivity. As stated in Section 18.7, a geotechnical review and designs for expansion are recommended for the TMF. Also, a tailing management plan should be developed in conjunction with the expansion design to ensure future water quality discharge parameters remain in compliance as additional tailings are planned to be deposited in the TMF and to ensure continuity of operation due to management succession.

Water is discharged from the TMF as a point source to surface waters under a New York State Pollutant Discharge Elimination System permit (SPDES). Water quality parameters are in compliance with surface water discharge permits.

20.2 Permitting

To the extent known, all permits required to operate the ESM mine are active and in place. Additionally, there are not any other significant factors or risks that may affect access, title, or the right or ability to perform work on the ESM properties.

Permits have remained active for mining at No. 4 since the previous operating periods. No environmental studies are underway at this time, nor are any required for the re-start of this existing fully permitted mine. The site is well managed and is in compliance with all environmental regulatory requirements.

Environmental permits required for operation of the No. 4 mine are listed in Table 20.1.

Table 20.1: Environmental Permits

Permit Type	Permit	Permit Number	Expiration
Air	Registration to Operate a Zinc Mining and Milling Complex (amended)	6-4038-00024/02001	9/30/2024
Water	SPDES Water Discharge Permit	NY0001791	5/31/2019
Water	Water Withdrawal Permit	6-4038-00024/02001	5/31/2019
Mining	Mining Permit	6-4038-00024/00006	8/1/2020
Storage	NYDEC Chemical Bulk Storage	CBS#6-000122	10/1/2017
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770	9/26/2018
Radiation	Certificate for Density Gauge	44023174	9/15/2018
Public Water Supply	No permit required, but regulated by NYS Dept. of Health. Registered ID #NY4430004	Registered ID #NY4430004	None
Haz Mat Transport	US Department of Transportation Registration – Pipeline and Haz Mat Safety Administration	072216 550 004Y	06/30/2018
Explosives	US Bureau of Alcohol, Tobaco, Firearms and Explosives (ATF – FEL) – (issued to individuals)	Hance Lic# 2-41368-5	
Blasting	NY State Certificate of Competence – (issued to individuals)	Hance 08-4885, Baderman 01-4709	

Source: SLZ (2017)

Tailings storage and management is discussed in detail in Section 18.7 of this report. Tailing is non-acid generating so conventional reclamation methods can be used to rehabilitate the tailing area. Currently, surface water discharge is in compliance with a SPDES permit and is expected to remain so for operating, closure, and post-closure periods.

20.3 Groundwater

The No. 3 underground mine contains water seal plugs below the water table to minimize groundwater inflow to the lower levels of the mine. The static water level at No. 3 is approximately 30 ft below the surface collar elevation. Planned operation levels at No. 4 mine are currently dry. During operations between 2005 through 2008, the majority of water pumped from the mine was fresh water brought underground for drilling activities. Presently, the No. 4 mine also receives some water flow from the No. 2 & 3 mines, plus flow from upper levels of Gouverneur Talcs' abandoned underground workings. The majority of flow reporting to No. 4 is from the No. 2 mine.

Water quality sampling data from the ESM No. 3 mine indicates that as the mine floods oxygen deficiency in the mine water will reduce its ability to react with host rock mineralization (Personal communication with Mr. Ryan Schermerhorn, Site Manager SLZC, February 17, 2017).

However, water quality samples taken from No. 3 indicated that zinc concentrations are above surface water quality discharge limits.

For final mine closure, the pumps will be turned off and the mine allowed to flood. Estimates of the recharge rate suggest it will take between 18 to 26 years for the water level to reach equilibrium (Personal communication with Mr. Ryan Schermerhorn, Site Manager SLZC, 2017). The water table elevation is estimated to return to an elevation of approximately 652 ft asml. Mine openings intersecting the ground surface are all above that elevation with the lowest being the No. 2 mine vent fan portal at an elevation of 660 ft amsl. This portal intersects the ground surface within a small open pit. The open pit floor elevation is 649 ft amsl so mine water could pond within this pit.

An August 2012 internal HudBay memo (Hair, 2012) discusses the possibility that once the mine water levels rebound, a portion of mine flood waters may need to be pumped and treated to maintain an inflowing hydraulic gradient that would prevent potential groundwater contamination. It should also be pointed out that no historical baseline water quality information exists for comparison so it is not possible to differentiate between existing conditions and what the naturally occurring impacts from the mineralized zone were prior to development.

Prior to final mine closure, further investigation should be considered to evaluate the potential for groundwater impacts and to determine what, if any, mitigation measures can be employed underground, prior to water levels returning to the upper mine levels.

Should pumping and water treatment be a future requirement, it appears that the cost would be relatively low. A combination of lime dosing and passive treatment options, such as biological treatment methods are successfully in use for water discharge treatment at ESM, and at other mine sites with similar chemistry. Since it is uncertain if treatment would be required and the cost component would be relatively low, especially when considered on a Net Present Value basis, no closure costs are included in this Technical Report for pumping, treatment, or groundwater monitoring.

20.4 Closure

The New York State Department of Environmental Conservation (NYSDEC) has accepted the reclamation completed at four of the sites and released them from the permit requirements as of November 2003. The NYSDEC has reviewed the reclamation at the Hyatt mine tailings and mine sites and the Pierrepont mine site and has released the reclamation bonds posted for these areas. No further work is required.

The ESM No. 2 Mine site has been partially reclaimed. ESM No. 2 shaft serves as secondary access to the underground operations at the No. 4 mine and will be included in the final reclamation of the No. 4 mine and concentrator complex. No. 4 mine and mine tailings reclamation is assured with a $ 1,662,870 certificate of deposit.

Final closure will commence when it is determined by the company that the mine and plant will no longer support future economic recovery of any remaining or undiscovered resource. Past history demonstrates that ESM and its predecessors have continued to discover economic resources intermittently since operations began circa 1910.

At the time of final site closure, beyond any ongoing care and maintenance programs, demolition and salvage of surface infrastructure would occur. Remaining equipment will be sold for reuse or scrap. Surface structures will be demolished with suitable materials such as steel being recycled. Other materials would be disposed of in an approved landfill.

Due to the age of the facility, some buildings may contain asbestos, so an appropriate asbestos program will be needed to identify those affected materials and a mitigation plan established to ensure proper handling, transportation, and disposal. Remaining concrete slabs are typically perforated in place to promote water drainage and covered or buried with sufficient soil for native vegetation to reestablish.

The TMF surface would be contoured as needed to promote surface run-off and aid in vegetation reestablishment. Cover soils may be needed if the tailing surface generates dust during windy periods. Tails stabilization by use of fast-growing plants may reduce the need for these cover soils however, as the tails themselves are a suitable plant growth media, as demonstrated by the amount of volunteer vegetation growing unaided on the exposed tails surface.

Removal of building’s and concrete structures such as the reagent dosing system, decant tower, and water sampling station would be removed when appropriate during closure, or during the post-closure monitoring period.

Post-closure vegetation and water quality monitoring would continue until such time as it can be demonstrated that site conditions, reclamation, and water chemistry is stable and no further monitoring is required. Any remaining financial assurances not used for closure and reclamation costs would be released back to the owner at that time. In the case of ESM, this final financial assurance release would likely occur after a 5 to 10-year successful post-closure monitoring period.

A Closure Plan and Cost Estimate update was completed by SRK Consulting in 2011 (SRK, 2011). It is a comprehensive report that discusses in more detail and provides costs for the closure of:

- Buildings and process plants;
- Tailings impoundment area;
- Material stockpiles;
- Contaminated soils;
- Landfills;
- Surface water management;
- Miscellaneous infrastructure; and
- Mine openings.

The SRK report reasonably represents the activities and cost for site closure, although it has attached actual calendar years for activities. Those dates are no longer relevant; however, the relative time periods for closure activities to occur are reasonable estimates.

Table 20.2: Post-Closure Water Quality Monitoring Frequency

Duration	Frequency	Sites
Years 1 – 5	monthly	SPDES permit station, South Dam
		discharge ditch, interception*
	annual	ditch, North Dam spillway, run-off pond
		Sylvia Lake
Years 6 – 10	quarterly	SPDES permit station, South Dam
		discharge ditch, interception
	annual	ditch, North Dam spillway, run-off pond
		Sylvia Lake
Years 11 – 15	bi-annual	South Dam discharge ditch, North Dam spillway, interceptor ditch, run-off pond, SPDES permit
		station
	annual	Sylvia Lake
Years 16 – 25	annual	Run-off pond, interception ditch, SPDES permit station, South Dam discharge ditch, North Dam spillway, Sylvia Lake

* Five year period including closure to monitor performance of new construction.

Source: SRK (2011)

Table 20.3: Schedule of Closure Activities

Closure Component	Closure Year 1				Closure Year 2			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Project Management/Admin	x	x	x	x	x	x	x	x
Demolition		x						
Shaft capping			x					
Contaminated Soils Removal			x					
Tailings Impoundment & Pile			x			x		
Surface Water Diversions		x	x					
Landfills		x	x			x		
Environmental Management	x	x	x	x	x	x	x	x

Source: SRK (2011)

20.5 Social and Community Factors

The ESM is an established facility; it is well accepted in the surrounding community. Business encountered during the site visit (community hotels, restaurants, groceries) had a positive view on the mine and its economic benefits. There are no known issues with social or community relations that currently would affect mining operations.

Many local families have benefited historically, and continue to do so through royalties, leases, and direct employment. SLZ is also a large tax payer in St. Lawrence County.

Over the years, housing development has increased in the area. Sylvia Lake, adjacent to the No. 4 property, is surrounded by homes. Many are used as vacation properties. As the ownership of these properties declines, new owners could be less appreciative of the benefits the mine has historically provided to the community.

In the interval since the mine suspended operations in 2008, much of that labour force has left the area, so skilled mine workers will need to be hired from outside the region. This could put a strain on local infrastructure but also brings the benefit of increased economic activity to an area that has limited employment opportunities.

There are no known social or community relations issues that would adversely impact the ESM.

21 Capital Cost Estimate

21.1 Capital Cost Summary & Estimate Results

Estimated project capital costs total $69.2M, consisting of the following distinct stages:

- **Initial Capital Costs** – includes all pre-production costs to replace, repair and upgrade the infrastructure and resource to an 1,800 t/d operation. Initial capital costs total $10.7M and are expended over a 5-month construction and commissioning period;
- **Sustaining Capital Costs** – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations.

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

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 the PEA will be realized.

Table 21.1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q1 2017 US dollars with no escalation.

Table 21.1: Capital Cost Summary

Area	Pre-Production (M$)	Production (M$)	LOM (M$)
Mining	5.3	40.4	45.7
Mineral Processing	1.1	0.7	1.8
Tailings Management	0	4.7	4.7
Infrastructure	0.8	0	0.8
Indirect Costs Incl. EPCM	0.4	0.2	0.5
Owners Costs	0.1	0.1	0.2
Closure Costs	0	11.9	11.9
Salvage Value	0	-4	-4
Subtotal Pre-Contingency	**7.6**	**53.9**	**61.6**
Contingency	1	4.6	5.6
Subtotal	**8.6**	**58.5**	**67.2**
Capitalized OPEX	7.6	0	7.6
Revenue Credit	-5.5	0	-5.5
Total	**10.7**	**58.5**	**69.2**

Source: JDS (2017)

Table 21.2 presents the capital cost distribution for the pre-production phase.

Table 21.2: Distribution of Initial Capital Costs

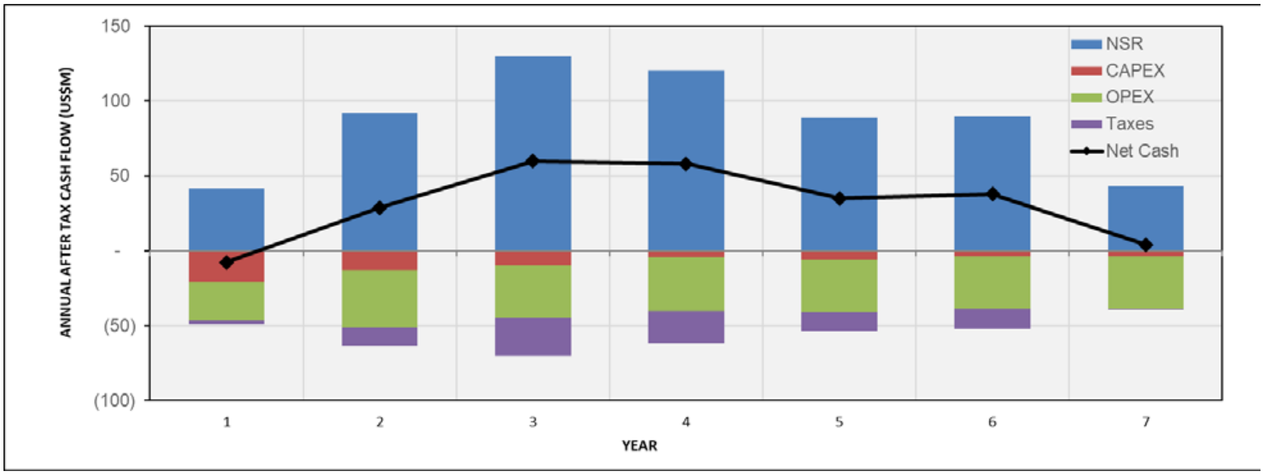
Capital Expenditures – Pre-production	$ (x1,000)
Infrastructure Capital	**3,813**
Headframe Repairs & Upgrades	187
Crusher Repairs & Upgrades	362
Compressor System Repairs	343
Ventilation Upgrades and Improvements	193
Mill Repairs & Upgrades	246
Facility Electrical	411
Long Hole Drill	515
Equipment Repairs and Modifications	451
No. 4 Shaft Utilities Rehab	157
First Fills / Stores	300
General Infastructure Repairs & Upgrades	167
ERP/Computers	150
Engineering & EPCM	225
Owners Costs	105
Mining Capital	***3,835***
Mobile Equipment Purchases	28
Mobile Equipment Rebuilds	6
Fixed Equipment	19
Drift Rehabilitation	2,469
Capital Lateral Development	1,313
Capitalized Pre-Commercial Production	***2,058***
Mining OPEX	4,236
Process OPEX	730
G&A OPEX	2,614
Capitalized Revenue (Credit)	-5,523
Contingency at: 10%	***971***
TOTAL	**10,677**

Source: JDS (2017)

21.2 Capital Cost Profile

All capital costs for the project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21.1 presents the monthly capital cost profile.

Figure 21.1: Capital Cost Profile (Closure Years not Shown)



Source: JDS (2017)

21.3 Key Estimate Parameters

The following key parameters apply to the capital cost estimates:

- **Estimate Class**: The capital cost estimates are considered Class 4 estimates (-20%/+30%);
- **Estimate Base Date**: The base date of the estimate is February 1, 2017. 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 estimated in US Dollars (US$)

21.4 Basis of Estimate

21.4.1 Mine Capital Cost Estimate

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, and in-house cost databases. Table 21.3 summarizes the underground mine capital cost estimate.

Table 21.3: Mine Capital Costs

Capital Costs	Initial ($x1000)	Sustaining/ Closure ($x1000)	Total ($x1000)
Mobile Equipment Purchases	28	1,581	1,608
Mobile Equipment Rebuilds	6	662	668
Fixed Equipment	19	31	54
Drift Rehabilitation	2,487	1,463	3,933
Capital Lateral Development	1,313	35,556	38,870
Capital Vertical Development	-	1,105	1,105
Capital Period OPEX	4,236	-	4,236
Total Mining (excl. contingency)	**8,072**	**40,399**	**48,471**

Source: JDS (2017)

21.4.1.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotations or database costs have been carried and applied to the required quantities. Hour meters on-site equipment were used to estimate remaining equipment life and schedule rebuilds and replacements accordingly.

21.4.1.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.4.1.3 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes, and underground infrastructure.

21.4.1.4 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, stoping, mine maintenance, and mine general costs) incurred prior to the introduction of feed to the processing facilities and the commencement of project revenues. They are included as an initial capital cost.

The basis of these costs is described in Section 22, operating cost estimate, as they are estimated in the same manner. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.4.2 Processing Cost Estimate

Processing pre-production capital costs include some equipment repairs, inspections and relining of the rod mill. The cone crusher control system will be replaced with a modern PLC based control system. The costs are based on quotations.

21.4.3 Infrastructure Cost Basis of Estimate

Infrastructure costs include repairs, replacements, inspections, electrical, mechanical and instrumentation. These cost estimates are primarily based on provided labour rates 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: Basis of Cost Estimate

Commodity	Basis
Headframe Repairs & Upgrades	Budget quotations were solicited from qualified contractors in the local region. A proposal accuracy of +/- 15% was requested from the contractor.
Crusher Repairs and Upgrades	Budget quotations were solicited from qualified contractors in the local region. A proposal accuracy of +/- 15% was requested from the contractor. In addition, budgetary unit rates were obtained from local contractors.
Compressor System Repairs	Budget quotations were solicited from qualified contractors in the local region. A proposal accuracy of +/- 15% was requested from the contractor.
Ventilation Upgrades and Improvements	Budget sourced from existing estimates provided by the client, and elemental factors.
Mill Repairs and Upgrades	Budget quotations were solicited from qualified contractors in the local region. In addition, budgetary unit rates were obtained from local contractors.
Facility Electrical	Budget quotations were solicited from qualified contractors in the local region. A proposal accuracy of +/- 15% was requested from the contractor.
Long Hole Drill	Budget quotations were solicited from qualified suppliers for the major equipment.
Equipment Repairs and Modifications	Budget quotations were solicited from qualified contractors in the local region.
Mine Rehabilitation	Quantities were developed from 3D Model. Budgetary unit rates were obtained from local contractors in conjunction with in-house cost estimates.
Construction Equipment Rentals/Usage	Construction equipment costs are included according to the tasks performed and the crew hours involved. This account is used for rentals and any purchase of commonly shared equipment, scaffolding, and subcontractor equipment charges.

Source: JDS (2017)

21.4.3.1 Surface Construction Sustaining Capital

With the age of the process facilities much of the mill equipment, including electrical equipment, is likely obsolete. In case of a failure, replacement would be difficult and time consuming to find a suitable alternate. A long term plan to replace obsolete electrical equipment (such as motor control centers) is recommended. A fund of 1% of processing direct costs is recommended to replace obsolete equipment. The 4-day on and 3-day off process schedule provides time for major equipment replacement.

21.4.4 Indirect Costs

Indirect costs are those that are not directly accountable to a specific cost object. Table 21.5 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21.5: Indirect Cost Basis of Estimate

Commodity	Basis
Construction Support Services	Time based cost allowance for general construction site services (temporary power, contractor support, etc.) applied against the surface construction schedule
Contractor Indirect Costs	Factored allowance (1.5%) of direct costs for contractor mobilization/demobilization (exclusive of freight costs)
	Factored allowance (1.0%) of direct costs for contractor facilities and auxiliary expenses
Detailed Engineering	Allowance of $375k (4%) of direct costs for engineering and procurement support activities
Project & Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration
	Database unit (hourly) rates

Source: JDS (2017)

21.4.5 Owners Costs

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- **Pre-production milling:** Costs of the Owner's processing labour, power, and consumables incurred before declaration of commercial production;
- **Pre-production general & administration**: Costs of the Owner's labour and expenses (safety, finance, security, purchasing, management, etc.) incurred prior to commercial production.

21.4.6 Closure Costs

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine. Activities include:

- Buildings and process plants;

- Tailings impoundment area;
- Material stockpiles;
- Contaminated soils;
- Landfills;
- Surface water management;
- Miscellaneous infrastructure; and
- Mine openings.

Closure costs were estimated based on the SRK cost estimate adjusted for the Consumer Price Index from 2014 to 2017 dollars and now total $11,930,000. The majority of the physical closure work will occur over a 2-year period. Monitoring and environmental management costs would continue for another 23-years, as estimated by SRK, totalling $1,147,000 in 2017. For this Technical Report, and the included economic model calculation, those costs are treated as lump-sum in Year 3 of closure. Year 1 of closure is assumed to begin the year after processing ceases.

Table 21.6: Closure Cost Summary

Closure Costs (2017 dollars)	Total ($x1,000)	Closure Year 1 ($x1,000)	Closure Year 2 ($x1,000)	Closure Year 3 ($x1,000)
Demolition and Miscellaneous Infrastructure	3,786	3,786		
Tailings	5,058	506	4,552	
Surface Water Diversions	1,034	1,034		
Contaminated Soils	125	125		
Landfills	74	37	$37	
Closure Project Management Administration and Environmental Management Costs	706	353	353	
Subtotal	10,783	5,841	4,942	
Post-Closure Costs				
Earthworks Inspection and Maintenance	292			292
Environmental Management	855			855
Subtotal	1,147			1,147
Total	**11,930**	**5,841**	**4,942**	**1,147**

Source: JDS, updated to 2017 dollars from SRK 2014

21.4.7 Contingency

An overall contingency of 10% was applied to the LOM capital costs of the project. LOM project contingency amounts to $5.6 M.

21.4.8 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any project sunk costs (studies, exploration programs, etc.);
- State sales tax;
- Closure bonding; and
- Escalation cost.

22 Site Operating Cost Estimate

Preparation of the site operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined proven project execution strategies. The estimate was developed using first principles and applying direct applicable project experience, and avoiding the use of general industry factors. The site operating cost is based on Owner-owned and operated mining/services fleets, and minimal use of permanent contractors except where value is provided through expertise and/or packages efficiencies/skills.

Virtually all of the estimate inputs were derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

Site operating costs in this section of the report include mining, processing, and G&A costs.

Site operating costs are presented in 2017 US dollars on a calendar year basis. No escalation or inflation is included.

The site operating cost estimate is broken into three major sections:

- Mining;
- Processing; and
- G&A.

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 the PEA will be realized.

22.1 Site Operating Cost Summary

Table 22.1: Breakdown of Estimated Site Operating Costs

Site Operating Costs	Unit Cost ($/t milled)	LOM Cost (M$)
Mining	42.27	180.9
Processing	8.89	38.0
G&A	9.60	41.1
Total	**60.77**	**260.0**

Source: JDS 2017

Table 22.2: Summary of Personnel

Position	Staff/Hourly	Quantity
Mining		
Mine Management	4/0	4
Mine Operations	7/125	132
Crushing & Hoisting	6/4	10
Mine Maintenance	4/15	19
Technical Services	11/0	11
Total Mining Personnel	32/144	176
Process Plant		
Operations & Maintenance	1 / 10	11
Process Technical Services	2 / 1	3
Supervision	1 / 0	1
Total Process Plant Personnel	4 / 11	15
G&A		
Surface & Infrastructure Maintenance	6 / 0	6
Environment	1 / 0	2
Administration	8 / 0	
Health & Safety	2 / 0	2
Human Resources	1 / 0	3
IT & Communications	0 / 0	0
Security	0 /3	3
Other*	2 /0	2
Subtotal General & Administration**	20 / 3	7
Total Personnel – All Areas* **	**56 / 158**	**214**

*contract shift electricians

**1 Scheduler/Cost Control for 3 months

***IT, payroll: outside contractors used

Source: JDS (2017)

22.2 Mine Operating Cost Estimate

Costs for the mining activities for the ESM project, will be undertaken by a contractor labour force for the first year of operations and transition to owner operated in the second year. Operating costs were built up from first principles, as well as JDS experience of similar-sized operations and local conditions. Mining costs for both mineralized and waste material take into account variations in haulage profiles and equipment selection. Local and contract labour rates, and local fuel and power pricing estimates were utilized for estimation purposes.

Mining operating costs listed in Table 22.3 are averaged over the life of mine. During the mine life, operating costs on a per ton basis range from a high of $62.67/t to a low of $36.04/t in Year 1 and 6, respectively. The higher operating cost in Year 1 is due to lower production rate and increased operating expenses during ramp up.

Table 22.3: Mining Operating Cost Summary

Area	Average Annual (M$/year)	Life of Mine (M$)	Unit Cost ($/t milled)
Waste Development	1.4	10.5	2.46
Production	13.4	107	25
Backfill	1	8	1.87
Crushing and Hoisting	1.5	11.9	2.78
Mine Maintenance	2,2	17.5	4.1
Mine General	3.2	25.9	6.06
Total Mining Operating Cost	**22.6**	**180**	**42.27**

Source: JDS (2017)

22.3 Processing Operating Cost Estimate

The ESM project process plant estimated operating costs in Table 22.4 are based on 2016 known US Zinc mine costs.

Table 22.4: Processing LOM Average OPEX Estimate by Area

Process Plant Operating Cost	Unit Cost ($/t milled)
Process Labour	2.06
Power	1.61
Consumables	3.41
Supplies and Services	1.81
Total	**8.89**

Source: JDS (2017)

22.4 General and Administrative Site Operating Cost Estimate

General and administrative (G&A) costs comprise the following categories:

- Labour; and
- On-site items as such health and safety, environmental, human resources, legal, insurance, external consulting, communications and office supplies.

The total G&A unit operating cost is estimated at $9.56/t of plant feed processed. Table 22.5 summarizes the annual G&A site operating costs.

Table 22.5: G&A Average OPEX Estimate by Area

G&A Site Operating Cost Category	Unit Cost ($/t milled)
G&A Labour	3.41
Surface Support Equipment	1.55
Infrastructure	0.28
Other G&A Items	4.37
Total	**9.60**

Source: JDS (2017)

23 Economic Analysis

23.1 Introduction

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in grade, metal price, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 and 22 of this report. The economic analysis has been run with no inflation (constant dollar basis).

The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations.

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 the PEA will be realized.

23.2 Life of Mine Summary and Assumptions

Table 23.1 summarizes parameters and assumptions pertinent to the eight year LOM that were used in the economic analysis.

Table 23.1: LOM Plan Summary

Parameter	Unit	Value
Mine Life	Years	8
Plant Feed Material	Mt	4.3
Throughput Rate	t/d	1,465
Average Head Grade	%Zn	9.2
LOM Recovered Zinc	LOM, Mlbs	756
LOM Payable Zinc	LOM, Mlbs	643
Average Annual Zinc Production	Mlbs	95
Average Annual Zinc Payable	Mlbs/yr	80

Source: JDS (2017)

Other economic factors include the following:

- Discount rate of 8%;
- Closure cost of $11.93 M were included;
- Nominal 2017 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- All costs and time prior to construction decision considered sunk;
- Results are presented on 100% ownership basis; and
- No management fees or financing costs (equity fund-raising was assumed).

23.3 Revenues and New Revenue Parameters

Mine revenue is derived from the sale of zinc concentrate into the international marketplace. No contractual arrangements for zinc concentrate sales exist at this time. Details regarding the terms used for the economic analysis can be found in the Market Studies (Section 19) of this report.

Table 23.3 indicates the Net Revenue (NR) parameters that were used in the economic analysis.

Table 23.2: Net Revenue Parameters

Parameter	Unit	Value
Mine Operating Days	Days/a	365
Zinc Recovery from Process Plant	%	96

Source: JDS (2017)

23.4 Taxes

The project has been evaluated on an after-tax basis in order to provide an indicative value of the potential project economics. A preliminary tax model was prepared by JDS, with input from ESM's accounting firm. The tax model contains the following assumptions:

- 35% federal income tax rate;
- 4.9 % state income tax; and
- Total taxes for the LOM $88.6M.

23.5 Royalties

The economic analysis incorporates royalties.

23.6 Results

At this preliminary stage, the project is economically viable with an after-tax IRR of 121% and a Net Present Value (NPV) of $150 M at an 8% discount rate using the prices and exchange rates described in Section 19.

Table 23.3 summarizes the economic results. Table 23.4 shows the pre-tax projected cash flows for the project.

Table 23.3: Summary of Results

Summary of Results	Unit	Value
Mine Life	Years	8
Resource Mined	Mt	4.278
Throughput Rate, LOM	t/d	1,465
Average Head Grade	%	9.2
Zinc Recovered	LOM Mlbs	756
	Mlbs/a	95
NSR (net of Royalties)	LOM US$M	622.0
Operating Costs	LOM US$M	260.0
	$/payable lb zinc	0.69
	$/T processed	60.77
Pre-Production Capital (excluding contingency)	US$M	9.7
Pre-Production Contingency	US$M	1.0
Total Pre-Production Capital	US$M	10.7
Sustaining & Closure Capital	US$M	53.9
Sustaining & Closure Contingency	US$M	4.6
Total Sustaining & Closure Capital	US$M	58.5
Total Capital	US$M	69.2
Working Capital	US$M	6.5
Pre-Tax Cash Flow	LOM US$M	299.4
	US$M/a	37.6
Taxes	US$M	88.6
After-Tax Cash Flow	US$M	210.7
	US$M/a	26.5
Pre-Tax $NPV_{8\%}$ Discount	**US$M**	**216**
Pre-Tax IRR	**%**	**153**
Pre-Tax Payback	**Years**	**1.2**
After-Tax $NPV_{8\%}$ Discount	**US$M**	**150**
After-Tax IRR	**%**	**121**
After-Tax Payback	**Years**	**1.3**

Source: JDS (2017)

Table 23.4: Cash Flow Model

Titan Mining (US) Corporation
Empire State Mines
PEA Economic Model

Item	Unit	Pre-Production Total	Production Total	Life of Mine Total	Year										
					Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
METAL PRICE AND FX RATE															
Zinc Price	US$/lb			1.25	1.25	1.45	1.40	1.35	1.20	1.05	1.05	1.05			
FX Rate	US$:C$			0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	-		
UNDERGROUND MINING															
Total Mine Production															
Extracted Resource	ktons	42	4,236	4,278	276	633	657	657	657	657	518	224			
Average Zn Grade	%	8.3%	9.2%	9.2%	9.5%	7.6%	10.8%	10.5%	9.0%	10.9%	6.6%	6.1%			
Total Contained Zn	ktons	4	390	394	26	48	71	69	59	72	34	14			
	Mlbs	7	781	788	52.4	96.7	142.5	138.1	118.6	143.6	68.6	27.1			
MINERAL PROCESSING															
Processing Schedule															
Operating Days	*days*	153	2,767	2,920	*365*	*365*	*365*	*365*	*365*	*365*	*365*	*365*			
Average Plant Throughput Rate	tpd	457	1,496	1,465	756	1,735	1,800	1,800	1,800	1,800	1,418	613			
Metal Recovered															
Zn Recovery	*%*			*96.0%*	*96.0%*	*96.0%*	*96.0%*	*96.0%*	*96.0%*	*96.0%*	*96.0%*	*96.0%*			
Zn Recovered	Mlbs	6.7	749.5	756.2	50.3	92.9	136.8	132.6	113.8	137.9	65.9	26.1			
	US$M	8.4	937.6	946.0	62.9	134.7	191.5	179.0	136.6	144.8	69.1	27.4			
Concentrate Grade	%			56.0%	56.0%	56.0%	56.0%	56.0%	56.0%	56.0%	56.0%	56.0%			
Moisture Content	%			6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%			
Zn Concentrate Produced	dton	6,251	697,079	703,329	46,793	86,381	127,236	123,344	105,865	128,233	61,246	24,232			
	wton	6,685	745,539	752,224	50,046	92,386	136,081	131,918	113,224	137,147	65,503	25,917			
SALES & NSR															
Payables															
Zn Payable	Mlbs	5.7	637.1	642.8	42.8	78.9	116.3	112.7	96.8	117.2	56.0	22.1	-		
	US$M	7.1	796.9	804.1	53.5	114.5	162.8	152.2	116.1	123.1	58.8	23.3	-		
Total Payable Metals	US$M	**7.1**	**796.9**	**804.1**	53.5	114.5	162.8	152.2	116.1	123.1	58.8	23.3	-		
Total Treatment & Transport Charges	US$M	**$(1.6)**	**$(178.6)**	**$(180.2)**	(12.0)	(22.1)	(32.6)	(31.6)	(27.1)	(32.8)	(15.7)	(6.2)	-		
Royalties Payable	%			**0.30%**	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	-		
	US$M	**$(0.0)**	**$(1.9)**	**$(1.9)**	(0.124)	(0.277)	(0.391)	(0.362)	(0.267)	(0.271)	(0.129)	(0.051)	-		
Net Smelter Return (Including Royalties)	**US$M**	**5.5**	**616.5**	**622.0**	**41.3**	**92.1**	**129.8**	**120.2**	**88.7**	**89.9**	**43.0**	**17.0**	**-**		
SITE OPERATING COSTS															
Mining	US$/ton	100.87	41.69	42.27	62.67	43.54	37.32	38.24	36.59	36.04	48.82	59.81	-		
	US$M	(4.2)	(176.6)	(180.9)	(17.3)	(27.6)	(24.5)	(25.1)	(24.0)	(23.7)	(25.3)	(13.4)	-		
Processing	US$/ton	17.39	8.81	8.89	11.19	8.73	8.65	8.65	8.65	8.65	9.22	8.61	-		

Table 23.4: Cash Flow Model (continued)

Item	Unit	Pre-Production Total	Production Total	Life of Mine Total	Year										
					Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
	US$M	(0.7)	(37.3)	(38.0)	(3.0881)	(5.5)	(5.7)	(5.7)	(5.7)	(5.7)	(4.8)	(1.9)	-		
General & Administration	US$/ton	62.81	9.08	9.60	20.91	7.98	7.46	7.84	7.81	7.81	9.71	22.06	-		
	US$M	(2.6)	(38.5)	(41.1)	(5.8)	(5.1)	(4.9)	(5.1)	(5.1)	(5.1)	(5.0)	(4.9)	-		
Total Operating Costs	**US$/ton**	**181.06**	**59.58**	**60.77**	**94.77**	**60.24**	**53.43**	**54.72**	**53.05**	**52.49**	**67.74**	**90.47**	**-**		
	US$M	**(7.6)**	**(252.4)**	**(260.0)**	**(26.2)**	**(38.1)**	**(35.1)**	**(36.0)**	**(34.9)**	**(34.5)**	**(35.1)**	**(20.2)**	**-**		
INCOME		*Pre-production NOI capitalized*													
Net Operating Income	**US$M**	**-**	**364.1**	**362.1**	**15.2**	**53.9**	**94.7**	**84.3**	**53.9**	**55.4**	**7.9**	**(3.2)**	**-**		
	US$/ton	**-**	**85.95**	**84.62**	**55.03**	**85.15**	**144.15**	**128.26**	**81.98**	**84.39**	**15.26**	**-**	**-**		
CAPITAL EXPENDITURES															
Initial & Sustaining Capital Costs															
Infrastructure & Process Capital	US$M	**(3.8)**	**(5.6)**	**(9.4)**	(3.9)	(5.0)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	-	-	-
Mining Capital	US$M	**(3.8)**	**(40.4)**	**(44.2)**	(12.6)	(6.5)	(8.8)	(3.5)	(5.0)	(3.4)	(3.3)	(1.2)	-	-	-
Capitalized Pre-Commercial Production	US$M	**2.1**	**-**	**(2.1)**	(2.1)	-	-	-	-	-	-	-	-	-	-
Contingency	US$M	**(1.0)**	**(4.6)**	**(5.6)**	(1.9)	(1.2)	(0.9)	(0.4)	(0.5)	(0.4)	(0.3)	(0.1)	-	-	
Subtotal - Capital Costs	**US$M**	**(10.7)**	**(50.6)**	**(61.3)**	**(20.4)**	**(12.7)**	**(9.8)**	**(4.0)**	**(5.6)**	**(3.9)**	**(3.7)**	**(1.3)**	**-**	**-**	
Closure Costs and Salvage															
Progressive & Final Closure, Monitoring, Salvage	US$M	**-**	**(7.9)**	**(7.9)**	-	-	-	-	-	-	-	-	(5.8)	(4.9)	2.9
Subtotal - Closure Costs	**US$M**	**-**	**(7.9)**	**(7.9)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**(5.8)**	**(4.9)**	**2.9**
Total															
Total Capital Expenditures	**US$M**	**(10.7)**	**(58.5)**	**(69.2)**	**(20.4)**	**(12.7)**	**(9.8)**	**(4.0)**	**(5.6)**	**(3.9)**	**(3.7)**	**(1.3)**	**(5.8)**	**(4.9)**	**2.9**
WORKING CAPITAL															
Working Capital															
Working Capital	US$M	**(6.5)**	**6.5**	**6.5**	-	-	-	-	-	-	-	-	6.5	-	-
CASH FLOWS															
Pre-Tax															
Net Pre-Tax Cash Flow	US$M	**(17.2)**	**312.1**	**299.4**	(5.2)	41.3	84.9	80.3	48.3	51.6	4.2	(4.6)	0.7	(4.9)	2.9
Post-Tax															
Income Taxes	US$M	**-**	**(88.6)**	**(88.6)**	(2.4)	(12.4)	(25.0)	(22.1)	(13.1)	(13.7)	(0.0)	-	-	-	-
Net Post-Tax Cash Flow	US$M	**(17.2)**	**223.5**	**210.7**	(7.5)	28.8	59.9	58.2	35.2	37.9	4.2	(4.6)	0.7	(4.9)	2.9

23.7 Sensitivities

A sensitivity analysis was performed to determine which factors most affected the project economics. The analysis revealed that the project is most sensitive to zinc grade, then price, followed by capital costs and operating costs. Table 23.4 outline the results of the sensitivity tests performed on pre-tax and after-tax NPV@ 8%.

The project was also tested under various discount rates. The results of these tests are demonstrated in Tables 23.5 and 23.6.

Table 23.5: Sensitivity Results

Variable	Pre-tax NPV @ 8% (M$)			Post-tax NPV @ 8% (M$)		
	-20% Variance	0% Variance	20% Variance	-20% Variance	0% Variance	20% Variance
Price	98	216	335	65	150	232
CAPEX	227	216	205	161	150	139
OPEX	253	216	179	176	150	122
Grade	108	216	324	75	150	223

Source: JDS (2017)

Table 23.6: Discount Rate Sensitivities

Discount Rate (%)	Pre-Tax NPV (M$)	After-Tax NPV (M$)
0	294.9	206.3
8	216.2	149.8
10	200.9	138.8

Source: JDS (2017)

24 Adjacent Properties

There are no adjacent properties relevant to the scope of this report.

25 Other Relevant Data and Information

There is no other relevant data or information relative to the scope of this report.

26 Interpretations and Conclusions

ESM began operating over 100 years ago from 1915, and has a proven track record of replacing reserves with continued exploration efforts; it is also a past producer with demonstrated production rates and recoveries well within the planned re-start parameters.

Stated reserves in 1985 contained 13 kt of zinc, and during production from 1998 to 2008, over 17 kt of zinc was mined. Like most underground mines, the resource is development limited, and as additional development work is completed, the resources grow accordingly. The PEA mine plan tonnage is estimated at 4.3 Mt at 9.2 % Zn, with a mine life of eight years.

Mine and mill refurbishments require minor capital expenditure, and much of this work has already commenced including underground drift rehabilitation and shaft servicing.

JDS is not aware of any significant risks and uncertainties that could be expected to affect the reliability or confidence of the resource and production estimates contained herein.

26.1 Risks

The main risks to the project are higher than planned dilution and depressed commodity prices.

Table 26.1: Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Dilution and grade control	Higher than expected dilution can have a severe impact on project economics. The mine must ensure accurate drilling and blasting practices are implemented to minimize dilution from wall rock, backfill and other low grade mineralized zones.	A well planned and executed grade control plan is necessary immediately upon commencement of mining. Mine designs need to be customized to the mineralization geometry to minimize external dilution. On shift grade control geologists to follow the mining. Focussed grade control efforts have been successful in the past.
Resource Modelling	All Mineral Resource estimates carry some risk and are one of the most common issues with project success. 50% of the resources in the PEA mine plan are classified as Inferred.	Infill drilling and increased sampling is recommended in order to provide a greater level of confidence in certain areas. Infill drilling required with Inferred resources to convert them to reserves.
	No density data exists for zones 10, 20, 21, 30 or 70.	Increase bulk density testing, especially in zones with no sampling at all.
	High sample length variability needs to be better controlled.	Sample lengths should be a uniform length (~5ft) with shorter intervals along contacts.
Geotechnical	Geotechnical criteria not available for long hole stope and pillar design. Limitations in stope dimension may cause additional development for shorter levels or increased quantity of pillars.	Geotechnical assessment of the mine design to evaluate and optimize stope and pillar dimensions.
Metal Prices	Lower than expected zinc prices can have a negative effect on project economics.	Hedging some portion of the mine's production may be an option to guarantee zinc pricing.
Consumable Prices	Prices for major consumables such as power, fuel,	Consider long term contracts for major

Table 26.1: Main Project Risks (continued)

Risk	Explanation/Potential Impact	Possible Risk Mitigation
	mill reagents, liners and explosives could be higher than planned. This will negatively affect operating costs.	consumable items to minimize the impact of pricing fluctuations on operating costs.
Ventilation	Poor ventilation in the extremities of the mine could limit or prevent production in those areas. Losses to unknown sources as well as air door and bulkhead leaks may cause air lower than required ventilation in the mine.	Further detailed analysis of ventilation design and potential upgrades to ventilation system including booster fans, construction of a new ventilation raise to surface or the use of electric (or battery) mine equipment to reduce ventilation requirements.
Capital and Operating Costs	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
	If OPEX increases then the NSR cut-off would increase and, all else being equal, the size of the mineral resources within the PEA mine plan would reduce yielding fewer tonnes.	
Tailings Management Facility (TMF)	An embankment raise is needed to contain to fully contain the 4.3 Million tons within the current mine plan. It is unknown how the native surface was prepared, what design features were included, what sub-surface conditions existed prior to construction, or the material properties of fill used for construction.	A geotechnical assessment and engineering design is recommended to establish a capacity estimate along with static and seismic stability of the facility.
Equipment Reliability	The mine has been on care & maintenance since 2008. Some equipment may be at risk of reduced reliability in a re-start of operations. Much of the mill equipment, including electrical equipment, is likely obsolete. In case of a failure, replacement would be difficult and time consuming to find a suitable alternate.	Review of historic maintenance records, design and implement program to refurbish equipment, hold additional spares in inventory for start-up. Review inventory of mill spares and determine critical areas where no replacement equipment (electrical equipment) is held and determine alternate replacements.
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the project. The ESM has been on care & maintenance since 2008. Sourcing local skilled labour may be a challenge as people may have moved or changed careers.	The early search for professionals as well as competitive salaries and benefits identify, attract and retain critical people. The Company may need to implement an extensive training program for new hires.
	High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	Utilization of contract labour will aid in start-up activities.

26.1 Opportunities

There are several opportunities to improve the project's economics through a combination of resource expansion, productivity enhancements and the use of new technology to lower mine operating costs.

Table 26.2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Resource Expansion	The Mineral Resource has not been fully delineated and there is an opportunity to expand the Mineral Resource.	Increased mine life and increased project Net Present Value.
Plant Feed Sorting	The use of sorting technology could reject waste rock dilution in the mineralized plant feed.	Rejecting waste rock dilution would increase the head grade entering the mill.
Railveyor	The use of the Railveyor technology could simplify material handling in the mine.	Reduced mine operating costs by eliminating or reducing the need for truck haulage for mill feed material.
Mine Material Transportation	Improve the haulage efficiency by grading haul roads, slashing tight areas or corners.	Improved truck speeds and mechanical availability will lead to lower operating costs.
Drill Core Sampling	Resampling core for holes that were excluded from the study due to lack of verification data.	Potential to increase mineral resources within the PEA mine plan grade, and classification without additional drilling.
Metallurgical Testing	Locked cycle test proved concentrate grades of 60%, while budget is set to 56%. Investigate retention times in cleaner flotation stages and forced air type cells in rougher stage.	Potential to increase concentrate grade in processing facility.

Source: JDS (2017)

27 Recommendations

Based on the PEA results, it is recommended that SLZ proceed with project advancement. The following items are recommended for resource upgrade, project optimization, and confirmation of design parameters used in this study:

- Infill drilling from underground, channel sampling, and re-assay of existing drill holes to gain resolution and accuracy of the Mineral Resource and to upgrade the Mineral Resource classification of Inferred material;
- Surface and underground exploration drilling program to add Mineral Resources and convert more of the Inferred Mineral Resource base to Measured & Indicated Mineral Resource classification;
- 3D Litho-stratigraphic modelling of the region and mine areas has been inadequate. Approximately as much as 50% of the historic mine workings and geology mapping remain in the form of linen sections and plans hard copies. It is recommended to digitize these plans into electronic format so that remnant Mineral Resource potential can be evaluated, as well as to be aware of old workings for safety, rehabilitation, security, etc.;
- Establish a true 3D underground mapping and in-field data collection, using photogrammetric surveying to complement and fine-tune diamond drilling mineralized solids, and provide increased ability to perform Mineral Resource and production reconciliations;
- Evaluate geotechnical conditions of long hole stoping to support the stope and pillar dimensions used in this PEA, and to provide guidance on ground support requirements;
- Conduct optical sorting test work to test the ability for separating mineral from waste before entering the mill facility. Perform an integration study to assess how the system would impact the mine and the logistics of application; and
- Investigate alternate haulage methods such as the railveyor for replacement of diesel powered haul trucks.

The 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 the PEA will be realized. Table 27.1 includes the cost for the recommended further definition drilling and engineering field and test programs.

Table 27.1: Definition Drilling and Engineering Field and Test Programs

Item	Cost ($)
Infill drilling (underground)	1,000,000
Surface and underground exploration drilling	4,300,000
3D lithology Model	50,000
Digitize geology maps and survey plans	150,000
Updated mine survey	150,000

Geotechnical review	30,000
Sorting test work and integration study	100,000
Alternate haulage investigation (railveyor)	45,000
Total Estimate	**5,825,000**

Source: JDS (2017)

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29 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
‘	minute (plane angle)
“	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
AA	Atomic adsorption
ac	acre
Acfm	actual cubic feet per minute
ADA	Azimuth, Dip, and Azimuth
ALT	active layer thickness
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
B	billion
Balmat No. 1 shaft	First shaft at the mine
BD	bulk density
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
BWI	Ball Mill Work Index
bya	billion years ago
C$	dollar (Canadian)
C&F	Cut and Fill
Ca	calcium
cfm	cubic feet per minute
CHP	combined heat and power plant
CIM	Canadian Institute of Mining and Metallurgy
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
COG	Cut-off grades
cP	centipoise
CSX	Chessie-Seaboard Consolidated Corporation
Cr	Chromium
CRM	Certified Reference Materials

Symbol/Abbreviation	Description
Cu	copper
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DGPS	differential global positioning system
DMS	dense media separation
dmt	dry metric ton
DPM	Diesel Particulate Matter
ESM	Empire State Mines
°F	Degrees Fahrenheit
Fe	Symbol for iron
FOG	Fall of ground
FS	Feasibility study
G&A	General and administrative
ICP	Induced-coupled plasma
IDS	Inverse Distance Squared
kg	kilogram
LHD	Load haul dump
LME	London Metal Exchange
LOM	Life of Mine
MIBC	Methyl isobutyl carbinol
MRP	Modified room and pillar
MVCU	Mean Value of Composites Used
NE	North East
NN	Nearest Neighbour
NPV	Net present value
NR	Net Revenue
NS	No-samples
NSR	Net smelter return
NYSDEC	New York State Department of Environmental Conservation
PAX	Potassium amyl xanthate
PEA	Preliminary economic assessment
PP	Post pillar
PPM	Parts per million
QP	Qualified Person
QAQC	Quality Assurance/Quality Control
RP	Room and Pillar
SG	Specific Gravity

Symbol/Abbreviation	Description
SPDES	State Pollutant Discharge Elimination System
TFFE	Targets For Future Exploration
TMF	Tailings Management Facility
UG	Underground
US	United States
USDA	US Department of Agriculture
ZCA	Zinc Corporation of America
Zn	Symbol for zinc

Appendix A
Qualified Person Certificates