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Technical Report

Empire State Mines 2021 NI 43-101 Technical Report

Titan Mining Corporation

Gouverneur, New York, USA

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

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1 Summary

1.1 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) has been engaged by Titan Mining Corporation (Titan) to update the current National Instrument 43-101 (NI 43-101) Technical Report for the Empire State Mine (ESM) operation. This technical report was prepared following the guidelines of NI 43-101.

In September 2017 JDS Energy and Mining Inc. (JDS) prepared a Preliminary Economic Assessment (PEA) for St. Lawrence Zinc Company, LLC (SLZ) a wholly owned subsidiary of Titan. The purpose of that study was to provide a Mineral Resource estimate with mine plan and economics for SLZ's ESM. Subsequently JDS prepared an update to the Technical Report in May 2018 to report increased Mineral Resources and updated mine plan.

SLZ owns the Balmat No. 4 Underground Zinc Mine (the Mine) which is now known as ESM No. 4 Mine or #4 Mine. 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.

The key difference between this Technical Report (2021 PEA) and the PEA completed in 2018 (2018 PEA) is the consideration of near surface Mineral Resources to be extracted by open pit mining. The 2021 PEA considers the economic impact of both underground and open pit mining to be processed through the existing process plant. Some adjustments are planned to include a lead concentrate circuit to treat lead mineralization from the proposed open pits.

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

Throughout this report, words such as orebody, ore shaft and fine ore bins have been used; these refer to standard terms and do not imply the confirmed presence of Mineral Reserves.

1.2 Project description

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 were maintained to good standards during the 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 utilizes a combination of selective longhole stoping, modified or stepped room and pillar and mechanized Cut and Fill as mining methods. 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). The proposed mine plan is expected to reach an initial target production rate of 1,400 t/d for 2021 and ramp up to 1,800 t/d in 2022 for the combined open pit and underground mines. The overall mine life is projected to be seven years with open pit mining completed in year three.

Tailings are being 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 and Dam Safety.

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

1.3 Location, access, and ownership

ESM is located approximately 1.3 mi south-west 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 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.

Cabo was contracted to drill underground in 2018 to 2019 and Boart Longyear was contracted for all surface programs in 2018 to 2020. Prior to ESM's 2018 to 2020 surface and underground drill programs, the drillhole database contained 4,342 drillholes completed at various times in the project's history within the Balmat area. ESM has subsequently added 4,050 historic drillholes to the database through the digitization of original log scans. Drilling during 2018 to 2020 consisted of both surface and underground holes. A total of 128 surface holes and 110 underground holes were drilled in the #2 and #4 Mines as well as near the historic No. 1 and No. 2 shafts, with an aggregate total of 194,755 feet (ft).

The Balmat mine (now ESM) has produced a total of 33.8 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
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 2018.

1.5 Geology and mineralization

ESM is comprised of multiple deposits in and around Fowler, NY. There are ten deposits currently considered as viable economic targets; American, Cal Marble, Davis, Fowler, Mahler, Mud Pond, N2D, Northeast Fowler, New Fold, and Sylvia Lake. Historic mining at these locations has provided a good geological understanding of each, with supporting mapping, sampling, and drilling data.

This Mineral Resource report has been created through a collaboration between ESM and SRK and has been prepared under the Canadian NI 43-101 guidelines. A comprehensive re-modelling effort was undertaken by ESM in 2018 using Leapfrog Geo for all geological models. Mining and grade control experience by ESM geologists have supported that the implicit modelling of the mineralized zones as veins in Leapfrog Geo, results in more accurate geological wireframes.

The 2017 Mineral Resources were in seven mineralized zones between 1,400 ft and 5,500 ft below surface in the #4 Mine; these zones are known as: Mud Pond, Mahler, New Fold, NE Fowler, Davis, Sylvia Lake, and Cal Marble. 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 20° and 35°, with local variations from 10° to 90°. 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. For 2018, follow-up work focused upon the remnant and / or unmined portions in the #2 Mine and #3 Mine areas.

The Balmat-Edwards district deposits are similar to Mississippi Valley-type resources that were deposited in flat lying limestones and subsequently metamorphosed and folded. The mineralized zones are elongated parallel to ancient shorelines and were deposited in porous host rocks. 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.

The 2005 metallurgical test results, and operational results from 2006 to 2008, support a zinc recovery of 96% and a zinc concentrate grade of 58% for the underground operations.

ESM recently discovered two new zones of near-surface mineralization near the existing operation. Metallurgical test work was undertaken on the samples from the new zones to determine the process flowsheet for treating them to produce both lead / silver and zinc concentrates.

The primary objective of the test work undertaken at Resource Development Inc. (RDi) in 2020 was to determine if the ores from the Turnpike and Hoist House prospects can be processed in the existing circuit with minor modifications to produce both lead and zinc concentrates.

Approximately 121 pounds (lbs) or 55 kgs of each sample, some half core samples and existing mill feed samples were sent to RDi for metallurgical test work which consisted of Bond's Mill Work Index and abrasion index determination and flotation test work. Reagents, currently employed in the milling circuit at the mine, were also sent for the study.

The conclusions drawn based on the scoping level study undertaken by RDi were that the recently discovered prospects could be processed using sequential flotation process to produce separate lead and zinc concentrate. Mineralization form Turnpike and Hoist House prospects are slightly harder than the current ore being processed in the plant. The lead recovery and concentrate grade are dependent on the feed grade of the ore. The higher the feed grade, the higher the final concentrate recovery and grade.

Due to the low feed lead grade, one would require a large amount of mineralization to run a locked-cycle test. Since limited ore was available, the optimization can be done once new flotation cells for the lead circuit are incorporated into the flowsheet.

1.7 Mineral Resource estimates

1.7.1 Drillhole database

The drillhole database was provided to SRK through the current Vulcan projects for each zone. Assays and associated composites were extracted from drillholes that were used in estimation, of which there were 1,622 in total.

The complete database for ESM consists of 8,678 surface or UG core holes. There are 68 sets of channel samples, 1,728 surface core holes, 6,872 UG core holes, and 10 core holes identified as other (including monitoring wells). Smaller subsets of this database were used for geologic modelling and / or estimation on a lithological unit basis. Each lithological group was modelled separately in isolated geological and estimation projects.

1.7.2 Geologic model

The ten deposit zones were defined and modelled by ESM geologists. Each one is comprised of multiple veins designating variably oriented and spatially-distinct mineralized zones which were modelled using combinations of explicit and implicit methods. Input data for these models are based on drilling intercepts and years of surface and underground mapping. Some wireframes for these zones were modelled using GEMS software from 2008 - 2017 and have subsequently been modified as new information has become available and modelling software has changed.

All new geological modelling in 2019 - 2020 was conducted in Leapfrog Geo. Each zone has been analyzed and divided where appropriate to facilitate a more accurate estimation of grade. SRK notes that, in some cases, this has resulted in splitting of domains based on morphology or orientation for the purposes of estimation. Mud Pond has been separated into a main zone and an upper Apron lens of mineralization as well, but for the purposes of this report will be discussed collective as Mud Pond. Updates periods for modelling are summarized in Table 1.2.

Table 1.2 Update periods for geological modelling

Zone	Years modelled and updated
American	2019
Cal Marble	2009, 2017, 2019
Davis	2017, 2019
Fowler	2019
Mahler	2009, 2017, 2019
Mud Pond	2008, 2009, 2017, 2019
N2D	2019
New Fold	2009, 2017, 2020
Northeast Fowler	2017, 2019
Sylvia Lake	2017, 2019

Source: ESM 2020.

1.7.3 Block model

Separate block models were created for each zone. The parameters for each consist of origins, rotations (in Maptek rotation convention), parent block parameters and associated sub-block parameters.

Historical mine workings, or as-built solids, were used for sub-blocking during model creation and mined blocks contained in these wireframes were removed from the estimated material. A comprehensive as-built wireframe was updated as of 1 October 2020 and utilized to deplete tonnage within the block models.

Due to the high variability of the ESM deposits and the lack of robust variography, inverse distance squared estimates were used to estimate grade into parent blocks within the block model. The control of each estimate was based on sample selection criteria such as, minimum and maximum number of composites, minimum number of drillholes, and search distances. For each pass, the search distances were either isotropic (spherical) or anisotropic (ellipsoidal) depending on the geometric control and limits in each vein. For isotropic searches, the geometry of the vein was considered adequate to control sample selection. For anisotropic searches, the direction was defined using a variable orientation algorithm in Vulcan called Locally Varying Anisotropy (LVA). This oriented the search ellipse for each block down a plane which paralleled the modelled geologic continuity (i.e., the hangingwall or footwall of the ESM veins). LVA parameters were defined as the mid-point between the vein bounding surfaces, or manually set based on a triangulated surface.

Underground Mineral Resources have been modelled (Leapfrog Geo) and estimated (Maptek Vulcan) by ESM geologists and reviewed for consistency with industry standards by SRK. In some cases, SRK participated in classification or refinement of the estimates based on this review. Matthew Hastings of SRK Consulting (U.S.) Inc. is the Qualified Person (QP) who has reviewed the geological models and estimates and has conducted multiple site inspections. Mineral Resources for the underground Number 4 mine areas have been compiled from ten separate block models including the American, Cal Marble, Davis, Fowler, Mahler, Mud Pond, Number 2 Deeps, North East Fowler, New Fold, and Silvia Lake areas (Table 1.3).

Table 1.3 Underground Mineral Resource estimate as of 1 October 2020

Category	Tons (000's US short tons)	Zn (%)	Contained pounds (000's lbs)
Measured	190	13.56	51.6
Indicated	1,524	11.49	350.3
Measured + Indicated	1,714	11.72	401.9
Inferred	6,551	11.11	1,455.1

Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that any part of the Mineral Resources estimated will be converted into a Mineral Reserves estimate. Resources stated as in-situ grade at a Zinc price of $1.07/oz, with an assumed zinc recovery of 96.3% Resources are reported using a 5.3% Zinc cut-off grade, based on actual break-even mining, processing, and G&A costs from the ESM operation. Numbers in the table have been rounded to reflect the accuracy or the estimate and may not sum due to rounding.
Source: SRK 2020.

Open-pit Number 2 Mine Mineral Resources have also been modelled (Leapfrog Geo) and estimated (Leapfrog EDGE) by ESM geologists and reviewed for consistency with industry standards by SRK. In some cases, SRK participated in classification or refinement of the estimates based on this review. Matthew Hastings of SRK Consulting (U.S.) Inc. is the QP who has reviewed the geological models and estimates, and has conducted one site inspection to the Number 2 Mine surface areas. Mineral Resources for the Number 2 Mine Open Pit area have been taken from a single block model which features the Hoist House, Pump House, and Turnpike areas (Table 1.4).

Table 1.4 Open pit Mineral Resource estimate as of 1 October 2020

Category	Tons (000's US short tons)	Zn (%)	Contained pounds (000's lbs)
Measured	105	3.34	3,190
Indicated	595	3.09	16,675
Measured + Indicated	701	3.13	19,864
Inferred	217	3.37	6,639

Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that any part of the Mineral Resources estimated will be converted into a Mineral Reserves estimate. Resources stated as internal to an optimized pit shell, above a cut-off grade of 1.57% Zn. Cut-off is based on break-even economics at a Zinc price of $1.07/oz, with an assumed zinc recovery of 94%, and actual processing, and G&A costs from the ESM operation. No mining costs were considered in the calculation of this COG, as the pit optimization incorporates the mining costs to develop the shape for reporting. Numbers in the table have been rounded to reflect the accuracy or the estimate and may not sum due to rounding.
Source: SRK 2020.

1.8 Mining

The mine plan tons at the ESM deposit will be extracted using a combination of longitudinal retreat stoping (LGS), Cut and Fill (C&F), Panel Mining – Primary and Secondary (PAP & PAS), and development drifting underground mining methods with rock backfill. Longhole backstopes (BCK) are also used in the design where applicable. The proposed combined underground and open pit mine plan is expected to reach an initial target production rate of 1,200 t/d for 2021 and ramp up to 1,800 t/d in 2022. Open pit mining will be completed in Year three (2024). The overall mine life will be seven years.

The ESM deposit will be accessed from surface via the No. 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 Mineral Inventory is based on the Mineral Resource stated as of February 2020 and is estimated at a 6% Zinc cut-off grade for the underground mine and 1.2% Zn for open pit mining.

For the underground mine, 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 LGS and BCK stopes, two sources of dilution were considered. Sloughing was estimated to be 2.0 ft on both the hangingwall and footwall of LGS stopes. For C&F, planned over break dilution of 0.5 ft was applied to both walls. A dilution grade of 0% Zn was assumed for all dilution. An additional 5% of unplanned dilution at a grade 0% Zn is also included in all mining methods.

Mine recovery was calculated under the following mine assumptions:

- C&F and waste development passing incremental cut-off, assume 95% mine recovery after losses.
- Longitudinal retreat and backstopes assume 95% mine recovery.
- Panel mining assumes 71% mine recovery after losses from pillars left behind.

Provided care is taking during blasting and rigorous ore control and monitoring systems are followed, AMC estimates that dilution and ore losses can be minimized for open pit mining. A mining recovery factor of 95% and dilution of 5% has been applied. The production schedule for both the underground and open pit mines and the combined productions schedule are provided in Table 1.5.

Table 1.5 Mine production schedule

Item	Unit	LOM	2021	2022	2023	2024	2025	2026	2027
Underground ore mined	000s tons	2,650	375	390	390	390	390	390	325
Zinc grade	%	8.5	8.6	8.7	9.2	8.8	8.3	8.1	7.8
Contained zinc	000s lbs	450,371	64,345	67,704	71,575	68,396	64,400	62,965	50,987
Open pit ore mined	000s tons	658	69	275	275	40	-	-	-
Total open pit waste	000s tons	3,262	325	1,450	1,331	156	-	-	-
Stripping ratio		5.0	4.7	5.3	4.8	3.9	-	-	-
Total material moved	000s tons	3,921	394	1,725	1,606	196	-	-	-
Zinc grade	%	3.1	2.5	2.9	3.3	3.3	0.0	0.0	0.0
Lead grade	%	0.9	0.8	1.3	0.6	0.4	0.0	0.0	0.0
Contained zinc	000s lbs	40,364	3,454	15,968	18,321	2,621	-	-	-
Contained lead	000s lbs	11,875	1,146	7,308	3,129	293	-	-	-
Ore processed	000s tons	3,309	444	665	665	430	390	390	325
Zinc grade	%	6.6	7.6	6.3	6.8	8.3	8.3	8.1	7.8
Lead grade	%	0.4	0.1	0.5	0.2	0.0	0.0	0.0	0.0
Contained zinc	000s lbs	490,735	67,799	83,672	89,896	71,016	64,400	62,965	50,987
Contained lead	000s lbs	11,875	1,146	7,308	3,129	293	-	-	-

Source: AMC 2021.

1.9 Recovery methods

Mineralized material mined in the ESM deposits is processed at the existing ESM concentrator that was commissioned in 1970 and last shut down in 2008. The concentrator was refurbished in late 2017 and began processing ore in 2018. The concentrator flowsheet includes crushing, grinding, sequential lead and zinc flotation circuits, concentrate dewatering circuits, and loadout facilities.

The design capacity of the concentrator is 5,000 t/d. Through-out the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines' capacity. The operating strategy is to operate the concentrator at its rated hourly throughput of 200 tons per hour (t/h) to 220 t/h, but for only as many hours as necessary to suit mine production. It currently is processing between 6,500 to 7,000 tons per week operating on a schedule of one shift per day, four days per week. The concentrator suffers no notable losses from intermittent operation.

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. Currently, the concentrator is producing zinc concentrate at an average of 59.0% zinc with 3% iron and 0.50% magnesium.

Lead values in the underground ore will be generally very low, and lead concentrate is not planned to be produced. Lead values in the open pit ore are expected to be higher and it will be possible to produce a lead concentrate from this ore source.

While aged, the concentrator is in good working order and runs efficiently. No modifications are required to continue processing underground ore sources and minimal modifications would be required for processing the mineralized material to be mined from the open pits.

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 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. 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 provides 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.

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.

The ground floor also contains mine offices, a boiler room and lamp room. Hot water for sanitary purposes is provided by quick recovery propane water heater, eliminating the need to operate a steam boiler through the summer months. The second floor contains a warehouse, machine shop, mine rescue room, first aid equipment room and training room.

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 are pumped from the Sylvia Lake pump house to two 100,000 gallon (gal) concrete deluge tanks near the concentrate storage building / rail loadout shed. Water is pumped from the reservoir tanks to the concentrator. Mine water is 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 the New York State Department of Environmental Conservation (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

All permits required to operate the ESM #4 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 this existing fully permitted mine. The site is well managed and is in compliance with all environmental regulatory requirements.

Renewals for State Pollutant Discharge Elimination System (SPDES) Permit and Water Withdrawal Permit were submitted to the NYSDEC in a timely manner. Both permits are on the Department's schedule for technical review due to length of time elapsed since previous review.

Tailings are non-acid generating so conventional reclamation methods can be used to rehabilitate the tailings 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.

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 $1,627,341 certificate of deposit.

1.12 Operating and capital cost estimates

Estimated project capital costs (including closure costs) total $19.1 million (M), consisting of the following distinct areas:

- #2 Mine pre-production
- #4 Mine capital equipment
- #4 infrastructure and process capital

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

Table 1.6 presents the capital estimate summary for each area in 2020 US$ with no escalation.

Table 1.6 Capital cost summary

Area	Cost estimate ($M)
#2 Mine pre-production capital	3.1
#4 Mine capital equipment	5.2
#4 infrastructure and process capital	2.9
Total capital cost	**11.1**
Closure costs	11.9
Salvage value	4.0
Total capital cost (incl. closure costs)	**19.1**

Source: Titan / AMC 2021.

Underground capital costs are estimated to be $5.2M, which include the lease purchase of one bolter and two 6-yard loaders, mobile equipment rebuilds, replacement of one single-boom Jumbo drill, one bolter, one lift truck and service cage, and purchases of a StopeMaster longhole drill, a 40T haul truck, 750 KW transformer and a leaky feeder head.

AMC has assumed that, due to the short life of the pits (three years), a contractor will be used to mine the open pits. Mark-ups on the operating costs have been assumed to cover the contractor's mining equipment and infrastructure capital costs.

Capital item allowance for the open pit includes upgrade of the railway right of way into a haul road, land acquisition, process plant upgrade for lead circuit, and site facility preparation.

Closure costs were estimated based on the SRK cost estimate of a total of $11.9M, this will be offset by the estimated $4M in salvage value. This cost is however not included in the economic model due to ongoing mining discoveries and expansions.

Indirect, owner's, and contingency costs are all incorporated into the capital cost estimates.

Preparation of the site operating cost estimate is based on current underground operation performance. 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. Open pit operating costs were estimated by AMC.

Site operating costs in this section of the report is broken into three major sections, which include mining, processing, and general and administrative (G&A) costs. AMC estimated open pit mining costs assuming a contractor mining operation. The operating cost estimate allows for all labour, equipment, supplies, fuel, consumables, and supervision.

Site operating costs (Table 1.7) are presented in 2020 US$ on a calendar year basis. No escalation or inflation is included.

Table 1.7 Breakdown of estimated site operating costs

Site operating costs	Unit cost ($/t milled)	LOM cost ($M)
Underground		
Mining	43.00	114.0
Processing	14.00	37.1
G&A	22.00	58.3
Underground total	**79.00**	**209.4**
Open pit		
Mining	20.07	13.2
Processing	7.00	4.6
G&A	5.92	3.9
Open pit total	**32.99**	**21.7**
Underground and open pit		
Mining	38.44	127.2
Processing	12.61	41.7
G&A	18.80	62.2
Underground and open pit total	**69.95**	**231.1**

Source: Titan / AMC 2021.

1.13 Economic analysis

An economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and 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.

It must be noted that 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.

Other economic factors include the following:

- Discount rate of 8%.
- Nominal 2021 dollars.
- Revenues, costs, and taxes are calculated for each period in which they occur.
- All costs and time prior to 1 January 2021 are considered sunk costs.
- Results are presented on 100% ownership basis.

The project has been evaluated on an after-tax basis to provide an indicative value of the potential project economics. Corporate income tax was calculated by Titan of $8.4M for the life-of-mine (LOM).

The economic analysis incorporates royalties. A royalty of 0.3% is applied to the NSR for the zinc concentrate. However, it is assumed that there are no royalties for the sale of the lead concentrate.

The results of the economic evaluation indicate that the project is economic under the current assumptions. The pre-tax cash flow is estimated to be $107M, with a pre-tax and post-tax Net Present Value (NPV) at a discount rate of 8% of $88M and $81M, respectively. The results of the assessment are provided in Table 1.8.

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 price, then zinc grade, followed by operating costs and capital costs. The results of the sensitivity analysis are provided in Table 1.9.

Table 1.8 Summary of results

Summary of results	Unit	Value
Mine life	Years	7.0
Resource mined	kt	3,309
LOM throughput rate	t/d	1,294
Average head zinc grade	% Zn	6.6
Average head lead grade	% Pb	0.4
LOM recovered zinc	Mlbs	470
LOM recovered lead	Mlbs	10
LOM payable zinc	Mlbs	400
LOM payable lead	Mlbs	9.5
Revenue by commodity (zinc)	%	98
Revenue by commodity (lead)	%	2
Zinc revenue	$M	460
Lead revenue	$M	8
Total revenue	$M	468
Total offsite charges	$M	113
Royalties	$M	1
NSR (net of royalties)	$M	349
Capital costs (including sustaining)	$M	11
Operating costs	$M	231
Operating costs	$/t processed	69.85
Pre-tax cash flow	$M	107
Taxes	$M	8
After-tax cash flow	$M	98
Pre-tax NPV (8% discount)	$M	88
After-tax NPV (8% discount)	$M	81

Source: AMC 2021.

Table 1.9 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
Zinc price	13	88	162	13	81	144
Zinc grade	31	88	144	31	81	128
CAPEX	90	88	85	83	81	78
OPEX	125	88	50	112	81	48

Source: AMC 2021.

1.14 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.

1.14.1 Risks

The most significant risks associated with the project are commodity prices, uncontrolled dilution, mineral recovery, operating and sustaining capital cost escalation, ventilation limitations, and Inferred Mineral Resource confidence.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

1.14.2 Opportunities

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. The replacement of ore reserves depended heavily on the ability to follow the mineralized zones through mine development. Additional exploration drilling may yield high returns in the discovery and upgrade of additional Mineral Resources.

Dilution is important to manage in any mining operation, particularly where mineralization occurs in narrow zones. The implementation of grade control by equipping geologists on shift with electronic survey and mapping software is an opportunity to improve control of the excavations and follow the mineralization more closely.

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. In addition, a sorted mill feed may permit a lower mine cut-off grade which could increase the Mineral Resources within the PEA mine plan, without requiring additional exploration.

1.14.3 Recommendations

The items shown in Table 1.10 are recommended for ESM to improve confidence and performance of the PEA mine plan and economics.

Table 1.10 Project recommendations and cost

Item	Cost ($)
Infill drilling and conversion of Inferred Mineral Resources	1,500,000
Geotechnical review	50,000
Sorting test work and integration study	100,000
Contractor quotes for open pit cost assumptions	15,000
Total estimate	**1,665,000**

Source: AMC 2020.

Contents

Tables

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2 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) has been engaged by Titan Mining Corporation (Titan) to update the current National Instrument 43-101 (NI 43-101) Technical Report for the Empire State Mine (ESM) operation. This technical report summarizes the results of this 2021 Preliminary Economic Assessment (2021 PEA) study and was prepared following the guidelines of NI 43-101.

In September 2017 JDS Energy and Mining Inc. (JDS) prepared a PEA for St. Lawrence Zinc Company, LLC (SLZ) a wholly owned subsidiary of Titan. The purpose of that study was to provide a Mineral Resource estimate with mine plan and economics for SLZ's ESM. Subsequently JDS prepared an update to the Technical Report in May 2018 to report increased Mineral Resources and updated mine plan.

ESM or the Property, is an underground zinc mine near the town of Gouverneur, New York State. It is located approximately 1.3 miles (mi) south-west of Fowler, in St. Lawrence County. Titan 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. Titan holds 100% ownership.

ESM is comprised of a group of high-grade mines, the ESM #4 Mine which is an underground mine that is in production, and six historic mines. ESM #4 Mine restarted mining operations in January 2018 and began producing zinc concentrate in March 2018. The ESM #1, #2, and #3, Hyatt, Pierrepont and Edwards mines are all within a 30-mile radius of the 5,000 tons per day (t/d) mill. Open pit potential has been identified in three areas named Hoist House, Turnpike, and Pump House.

2.1 Basis of technical report

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

- Overall report authors: AMC
- Geology and Mineral Resource: SRK Consultants Ltd (SRK)
- Open pit and waste dumps: AMC
- Open pit geotechnical assessment: Knight Piésold Ltd (KP)
- Underground mine plan and production schedule: Jackleg Consulting, LLC (Jackleg)
- Metallurgical testwork and mineral processing: RDI Minerals (RDI)

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).

The key information used in this report is listed in Section 27, References.

This Technical Report has been produced in accordance with the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 utilizes the definitions and categories of Mineral Resources and Mineral Reserves as set out in the May 2014 edition of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIM Definition Standards).

A draft of the Technical Report was provided to Titan to check for factual accuracy. The Technical Report is effective as of 20 December 2020.

Table 2.1 Persons who prepared or contributed to this technical report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of Titan?**	**Date of last site visit**	**Professional designation**	**Sections of report**
D. Warren	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	15 to 18 February 2020	P.Eng. (BC)	1 (part), 15 (part), 16 (part), 21 (part), 25 (part) 26 (part), 27 (part)
G. Methven	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC, YT)	1 (part), 2, 3, 4, 5, 18, 19, 20, 21 (part), 22, 23, 24, 25 (part), 26 (part), 27 (part)
D. Malhotra	Principal / Director	RDI Minerals Ltd.	Yes	2016	SME registered Member	1 (part), 13, 17, 25 (part), 26 (part), 27 (part)
D. Vatterodt	Underground Mine Engineer	Jackleg Consulting, LLC	Yes	3 to 7 November 2019	SME registered Member	1 (part),15 (part), 16 (part), 21 (part), 25 (part), 26 (part), 27 (part)
B. Peacock	Senior Engineer	Knight Piésold Ltd.	Yes	No Visit	P.Eng. (ON, NFLD, NU / NWT)	1 (part), 16 (part), 25 (part), 26 (part), 27 (part)
M. Hastings	Practice Leader (Resource Geology)	SRK Consulting (Canada) Inc.	Yes	9 to 13 March 2020	AusIMM registered Member	1 (part), 6, 7, 8, 9, 10, 11, 12, 14, 25 (part) 26 (part), 27 (part)
Other Experts who assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of Titan?**	**Visited site**	**Sections of Report**	
C. Austin	Leases and tenements	Titan	No	Yes	4.2	
S. Trader, PG, CPG	Environmental & Hydrogeology	Alpha Geoscience	Yes	Yes	20	
M. McClelland, CPA, CA	Chief Financial Officer	Titan	No	No visit	22 (part)	

Source: AMC.

2.2 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 the table of contents. This report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3 Reliance on other experts

The QPs opinions contained herein are based on information provided by Titan and others throughout the course of the update. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Technical Report.

The following disclosure is made in respect of this Expert:

- Scott Burkett, Vice President Exploration, Titan Mining Corp, Vancouver, BC, Canada.
- Report, opinion, or statement relied upon: Information on mineral tenure and status, title issues, royalty obligations, etc.
- Extent of reliance: Full reliance following a review by the QP(s).
- Portion of Technical Report to which disclaimer applies: Section 4.

The QPs have relied, in respect of environmental aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Technical Report.

The following disclosure is made in respect of this Expert:

- Ryan Schermerhorn. Production Manager, Empire State Mines, Gouverneur, NY, USA
- Report, opinion, or statement relied upon: Information on permitting, environmental, social, and community factors.
- Extent of reliance: Full reliance following a review by the QP(s).
- Portion of Technical Report to which disclaimer applies: Section 20.

The following disclosure is made in respect of this Expert:

- Michael McClelland, CPA, CA, Chief Financial Officer, Titan Mining Corp., Vancouver, BC, Canada.
- Report, opinion, or statement relied upon: Information on taxation regarding the Property.
- Extent of reliance: Full reliance following a review by the QP(s).
- Portion of Technical Report to which disclaimer applies: Section 22.

4 Property description and location

4.1 Location

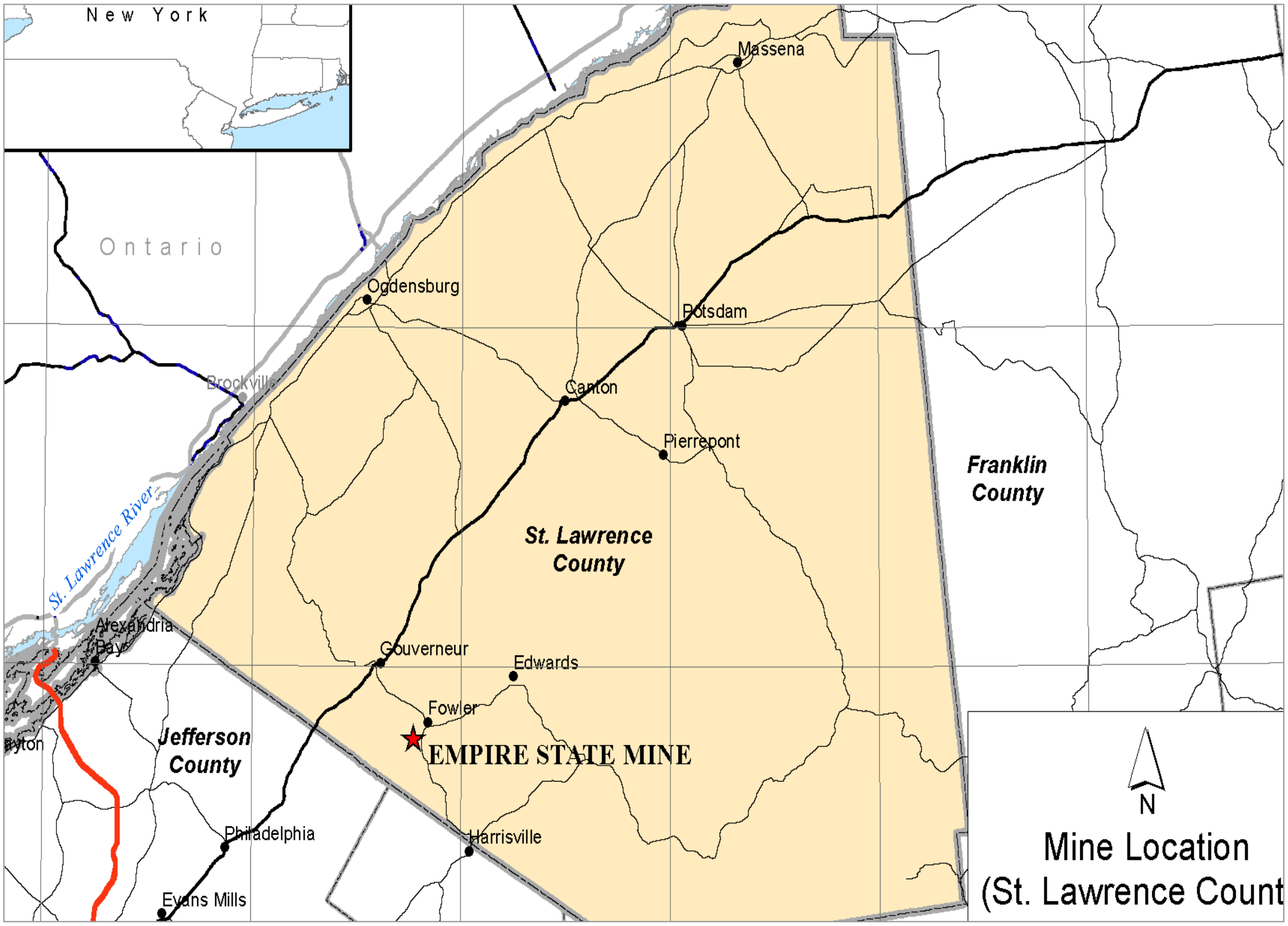
The ESM mine is located 7 mi south-east of Gouverneur, New York at 44°14'51” N latitude, 75°23’50” W longitude, and 710' ASL. The site is 38 mi via State Road #812 from the St. Lawrence Seaway at Ogdensburg, NY (Figure 4.1 and Figure 4.2).

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

Figure 4.1 Regional project location

Source: Titan 2020.

Figure 4.2 Local project location



Source: Titan 2020.

4.2 Mineral tenure

The 2,699 acres of surface rights owned by Titan 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 10 leases are held in the Pierrepont mine area, covering 515 and 985 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 $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 and assumes mining will be able to proceed as a result. The current resource and subsequent planned mining areas are not impacted in any way by the expired leases. Legal consultation should be obtained before any mining occurs on expired leases as it cannot be assumed that the lease agreement will extend beyond the expiration date, despite acceptance of payment by leasers.

Table 4.1 Lease list with expiration dates

Name	Type	Expiration date	Payment anniversary	Acres	Term	NSR	Notes
Warriner Lease	Lease	18/01/2031	18/01/2017	80.82	20-year lease: renewable	4%	Lease Agreement signed 1/18/2011.
St. Lawrence Ore Lease	Lease	25/01/2010 see note	25/01/2017	135	20 years: NOT renewable	4%	Expired 1/25/2010 however minimum annual payments have been made including for 2020.
Whitman Lease	Lease	10/2/2018 see note	10/2/2017	30	20 years: renewable for additional 20 years	4%	Per annual royalty paid in 2018 this lease renewed for another 20 years will expire 10/2/2038.
Gilbert Option	Option	3/3/2016 see note	3/3/2017	96.4	5-year option	4%	Option with escalator. Phillip Crundall now deceased. Still waiting for attorney handling his estate to let us know who Mr Crundall's MR interests go to. Have contacted attorney numerous times.
Brian Tripp Lease (90Ac)	Lease	22/03/2021	22/03/2017	90	20 years: renewable for additional 20 years	4%	
Gilbert Lease	Lease	22/03/2031	22/03/2017	96.4	20-year lease: renewable	4%	The lease portion of the agreement was signed.
Jenne Lease	Lease	7/4/2000 see note	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. Will expire 7/4/2020. New Lease Agreement sent in 2019 all parties have signed and returned except for three individuals (9 heirs total on lease) Delays caused by COVID-19 restrictions.
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 7/23) used for all Wells leases taken directly from original index file cards.
St. Lawrence County Option	Option	11/3/2024	20/04/2017	85.5 & 30	5-year option	4%	Option payment with escalator schedule.
Hull Lease	Lease	30/04/2017 see notes	30/04/2017	20	20 years: renewable for additional 20 years	4%	Renewed 30/04/2017 will now expire 4/30/2037.
Kelly Freeman Lease	Lease	2/5/2015 see note	2/5/2017	310	20 years: renewable for additional 20 years	4%	First 20-year expired 2015 however lease renewable for 20 years and payments made on time. New expiration date 2 May 2035.
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.

Name	Type	Expiration date	Payment anniversary	Acres	Term	NSR	Notes
Cromwell Heir Option	Option	16/6/2016 see note	16/06/2017	369	5-year option	4%	Option payment with escalator schedule, new agreements sent 2019. No response from Irwin Cromwell. Per call from Margaret Cromwell's son she is interested in possibly selling her MR to ESM.
Edwards Lease	Lease	3/6/2039	3/6/2020	96	20 years: renewable for additional 20 years	4%	
Cole Lease	Lease	20/6/2000 see note	20/06/2017	94	20 years: renewable for additional 20 years	4%	First 20-year term expired. Lease renewable for 20 years and payments made on time each year. Will expire 6/20/2020. New Lease Agreement sent in 2020. All parties have signed and returned new agreement except for two individuals. (15 heirs total on the lease) Delays caused by COVID-19 restrictions.
Aleta Billings Heirs Leases	Lease	26/6/2039 (Gary E. Wight) 12/6/2039 (Joann A. Whitaker) 5/7/2039 (Lee H. Wight) 13/6/2039 (Linda M. Love)	6/26/2021 6/12/2021 7/5/2021 6/1/2020	157.5	20 years: renewable for additional 20 years	4%	Each heir had their own Lease Agreement all signed in 2019.
Alan Latimer Lease	Lease	7/7/2023	7/7/2017	20	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%	
Barrigar Lease (Larry P. & Elaine P.) (part of former Lloyd & Lillian Barrigar Lease)	Lease	2/7/2039	7/2/2021	122.4	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 7/2/2019, part of former Lloyd & Lillian Barrigar Lease, on west side of road.
Pusateri-Linda, Etal Lease (part of former Lloyd & Lillian Barrigar Lease)	Lease	29/7/2039	7/29/2021	158.4	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 7/29/2019, part of former Lloyd & Lillian Barrigar Lease, on east side of road.
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.
Zira Lease	Lease	27/07/2027	25/07/2017	0.93	20 years: renewable for additional 20 years	4%	

Name	Type	Expiration date	Payment anniversary	Acres	Term	NSR	Notes
Webb Lease	Lease	18/9/2039	9/18/2021	46	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 9/18/2019 by Alan J. Webb.
Van Brocklin Lease	Lease	27/07/2002 see note	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. Will expire 7/27/2022.
Davis, Daniel Lease (formerly Barkley Lease)	Lease	25/7/2040	7/25/2021	78	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 7/25/2020 by Daniel Davis.
Brown Lease	Lease	9/9/2039	9/9/2021	165	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 9/9/2019 with current heirs.
Lawrence Emrich Heirs Options	Option	11/1/2022	11/1/2020	229.04	3-year options	4%	Option payment (each heir has her own option agreement signed 11/1/2019.
Thivierge Lease	Lease	27/8/2002 see note	27/08/2017	66	20 years: renewable for additional 20 years	4%	First 20-yr term has expired; however, lease is renewable for 20 years and payments have been made on time each year. Will expire 8/27/2022.
Bogardus Lease (Peter & Penny Bogardus)	Lease	11/12/2039	11/12/2021	162.2	20 years, renewable in 20 years	4%	New Lease signed 11/12/2019.
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%	
				162.2	5-year option	4%	EXPIRED Ryan Bogardus did not renew NO LONGER ACTIVE.
Lansing-Dodge Lease	Lease	10/8/2039	10/8/2021	~ 22,000	20 year: renewable for additional 20 years	4%	New Lease signed 10/8/2019 (Coral Coleman).
Emery Webb Lease	Lease	22/9/2029	22/09/2017	181.46	20 year: renewable for additional 20 years	4%	
Stiles Lease EXPIRED	Lease	27/9/2002 see note	27/09/2017	32	20 years: renewable for additional 20 years	4%	EXPIRED Owner did not wish to renew NO LONGER ACTIVE.
Hutchinson Lease	Lease	1/10/2002 see note	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. Will expire 10/1/2022.
Manning Lease	Lease	1/10/2027	1/10/2017	0.65	20 years: renewable for additional 20 years	4%	

Name	Type	Expiration date	Payment anniversary	Acres	Term	NSR	Notes
Cromwell Heir Option	Option	21/10/2016 see note	21/10/2017	369	5-year option	4%	Option payment with escalator schedule. New lease sent 2019. No response from Irwin Cromwell. Per call from Margaret Cromwell's son, she is interested in possibly sell her MR to ESM.
Steven A. Sullivan Option EXPIRED	Option	28/10/2012	28/10/2017	158.8 (98.45 [60.00+38.45] + 60.35)	3-year option	4%	New Lease Agreement sent 2019, per call Mr Sullivan wants to sign new lease however he still has not returned agreement to us. EXPIRED.
Caswell Lease	Lease	5/11/2002 see note	5/11/2017	98	20 years: renewable for additional 20 years	4%	First 20-yr term has expired; however, lease is renewable for 20 years and payments have been made on time each year. Will expire 11/5/2022.
Walter Planty Lease (64.39 Ac)	Lease	30/10/2039	10/30/2020	64.39	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 10/30/2019.
Marjory Tyler Lease	Lease	6/11/2039	11/6/2020	183	20 years: renewable for additional 20 years	4%	New Lease Agreement signed 11/6/2019.
Brian Tripp Lease (0.79Ac)	Lease	6/12/2026	6/12/2017	0.79	20 years: renewable for additional 20 years	4%	
Brian Tripp (formerly Robert G., Sr. and Phyllis J. Tripp) Lease (19 Ac)	Lease	20/5/2039	5/20/2021	19	20 years: renewable for additional 20 years	4%	New Lease w/Brian Tripp signed 5/20/2019.
Davis (Stanley and Carol) Lease (14.4 Ac)	Lease	12/6/2026	6/12/2017	12.28 & 2.12	20 years: renewable for additional 20 years	4%	
Gouverneur Talc Co Lease	Lease	28/6/2030	None	0	20-year lease	4%	Renewed for an additional 20 years 6/28/2010 - 06/28/30.

Land surface rights for the purpose of construction of buildings and for other purposes, are purchased from landowners; Titan 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 4.2 Mineral tenure information

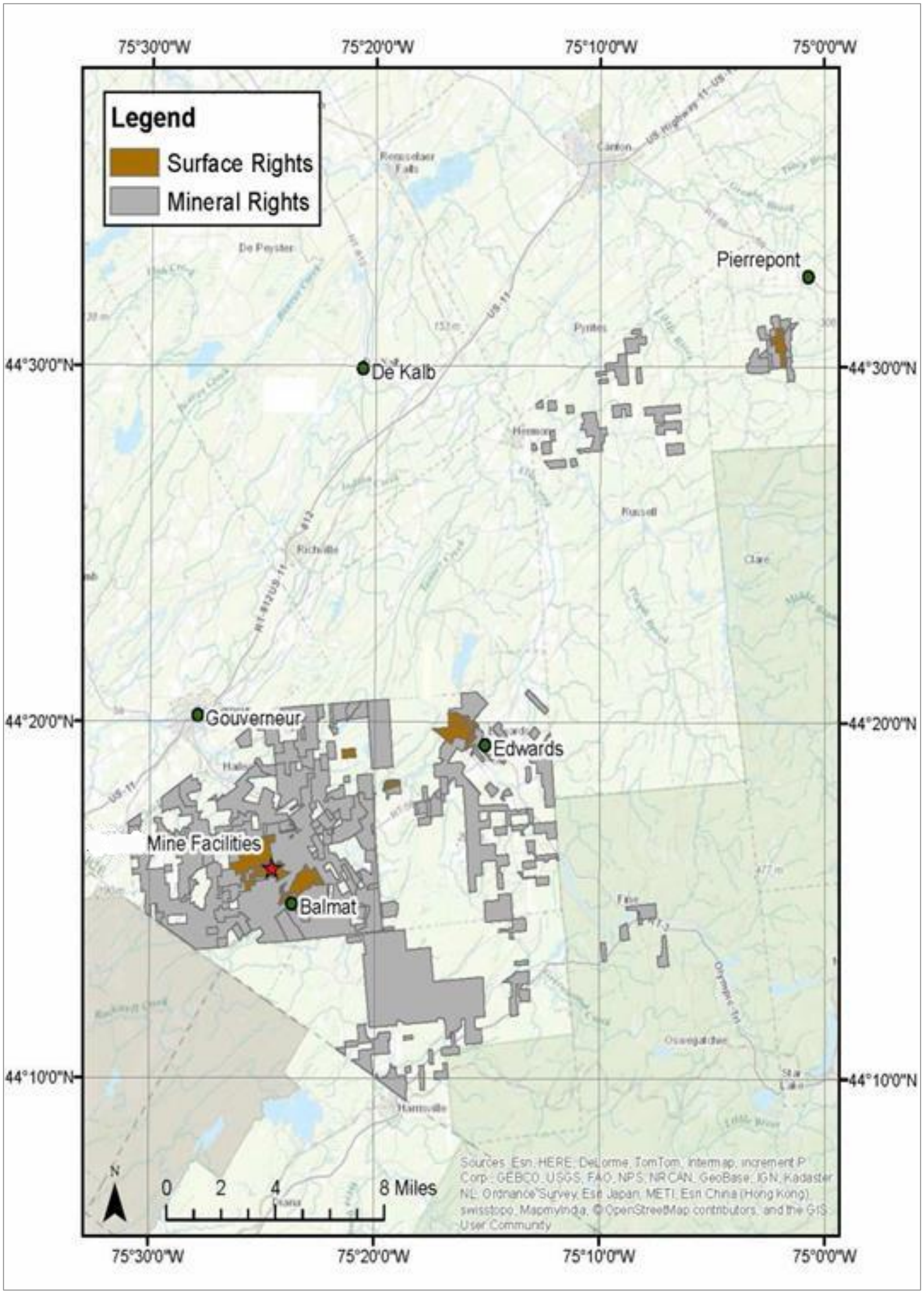
Assessor parcel number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2019 taxes ($)
119.001-1-8	Pierrepont	80.4			322	391.35
119.001-1-10	Pierrepont	102.1			330	496.93
119.001-1-11	Pierrepont	0.52			720	1.63
119.001-1-12	Pierrepont	59.3			720	337.36
119.001-1-18./1	Pierrepont		1.4		720	40.61
174.004-3-2	Edwards	0.85			314	42.11
174.004-4-2	Edwards	10.37			720	174.48
174.004-4-1	Edwards	1.35			314	76.21
175.003-3-1.1	Edwards	71.6			720	541.50
175.003-3-19.1	Edwards	3.4			720	104.29
175.002-1-5.1	Edwards	370.2			323	2,338.43
175.002-1-33	Edwards	161.7			323	1,084.98
175.002-1-34.1	Edwards	72.2			330	545.51
175.002-1-32.1	Edwards	11.7			330	182.49
175.002-1-34./1	Edwards		74		720	142.39
1.044-18	Edwards		100		720	140.39
175.002-1-25./1	Edwards		92.2		720	132.36
175.001-1-4./1	Edwards		165		720	142.39
175.002-1-5./1	Edwards		1044		314	525.44
175.003-1-1./2	Edwards		72		720	132.36
175.003-1-1./4	Edwards		18.8		720	132.36
175.003-3-1.1/1	Edwards		70		720	415.14
175.003-3-1.1/4	Edwards			Electrical	720	1,163.19
175.003-3-10./1	Edwards		115		720	132.36
175.003-3-13./2	Edwards		53.1		720	132.36
175.004-1-3./1	Edwards		58		720	132.36
175.004-1-6./1	Edwards		20		720	132.36
175.004-1-7./1	Edwards		63.8		720	132.36
175.004-1-11./1	Edwards		97.4		720	212.59
175.004-1-14./2	Edwards		62		720	132.36
187.002-2-1./1	Edwards		30		720	132.36
187.002-2-1./2	Edwards		80.9		720	132.36
188.001-1-15./2	Edwards		25		720	132.36
188.001-1-15./3	Edwards		169.1		720	132.36

Assessor parcel number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2019 taxes ($)
188.001-1-17./1	Edwards		65.6		720	132.36
188.001-1-27./1	Edwards		73.8		720	132.36
188.002-1-2./1	Edwards		36		720	132.36
174.004-1-18	Fowler	89.3	89.3		720	325.40
187.001-1-5	Fowler	2.5			720	108.47
187.001-1-21.2	Fowler	44.49			720	224.52
186.004-1-44	Fowler	705.3			720	1,084.65
186.004-1-33.11	Fowler	86.5			720	1,100.18
186.004-1-31	Fowler	61.6			720	1,003.32
187.003-1-2	Fowler	82.3			720	216.93
187.003-1-1	Fowler	1.6			720	3,742.07
187.069-1-38	Fowler	0.7			720	1,403.34
187.003-1-4.11	Fowler	63.8			720	906.24
187.003-1-4.121	Fowler	124.7			720	379.63
187.003-2-1.1	Fowler	45.2			720	216.93
199.001-2-52	Fowler	445			720	1,084.65
186.002-1-14.11/3	Fowler		146.6		720	10.85
186.002-1-14.11/4	Fowler		144		720	10.85
187.003-1-3./1	Fowler		0.01		720	108.47
187.003-1-4.11/2	Fowler			Shaft 4	720	26,718.85
187.003-1-4.11/3	Fowler		0.01		720	9,355.19
187.003-1-4.11/5	Fowler			Shop	720	2,826.62
187.003-1-4.11/7	Fowler			Electric	720	23,986.60
187.003-1-4.11/9	Fowler			Buildings	720	58,686.58
187.003-1-4.11/11	Fowler			Paint, oil storage building	720	2,095.57
187.003-1-4.11/12	Fowler			Timber storage	720	2,245.25
187.003-1-4.11/17	Fowler			Railroad #4	720	5,613.10
187.003-1-4.11/18	Fowler			Mill	720	71,942.22
187.003-1-4.11/20	Fowler			Storage buildings	720	11,237.07
187.003-1-4.11/21	Fowler			Storage	720	6,643.54
199.001-2-43.1/2	Fowler			Pipe shop 2	720	299.37
Owned Fee Parcels		2699	2967			244,794

Source: St. Lawrence County Government 2019.

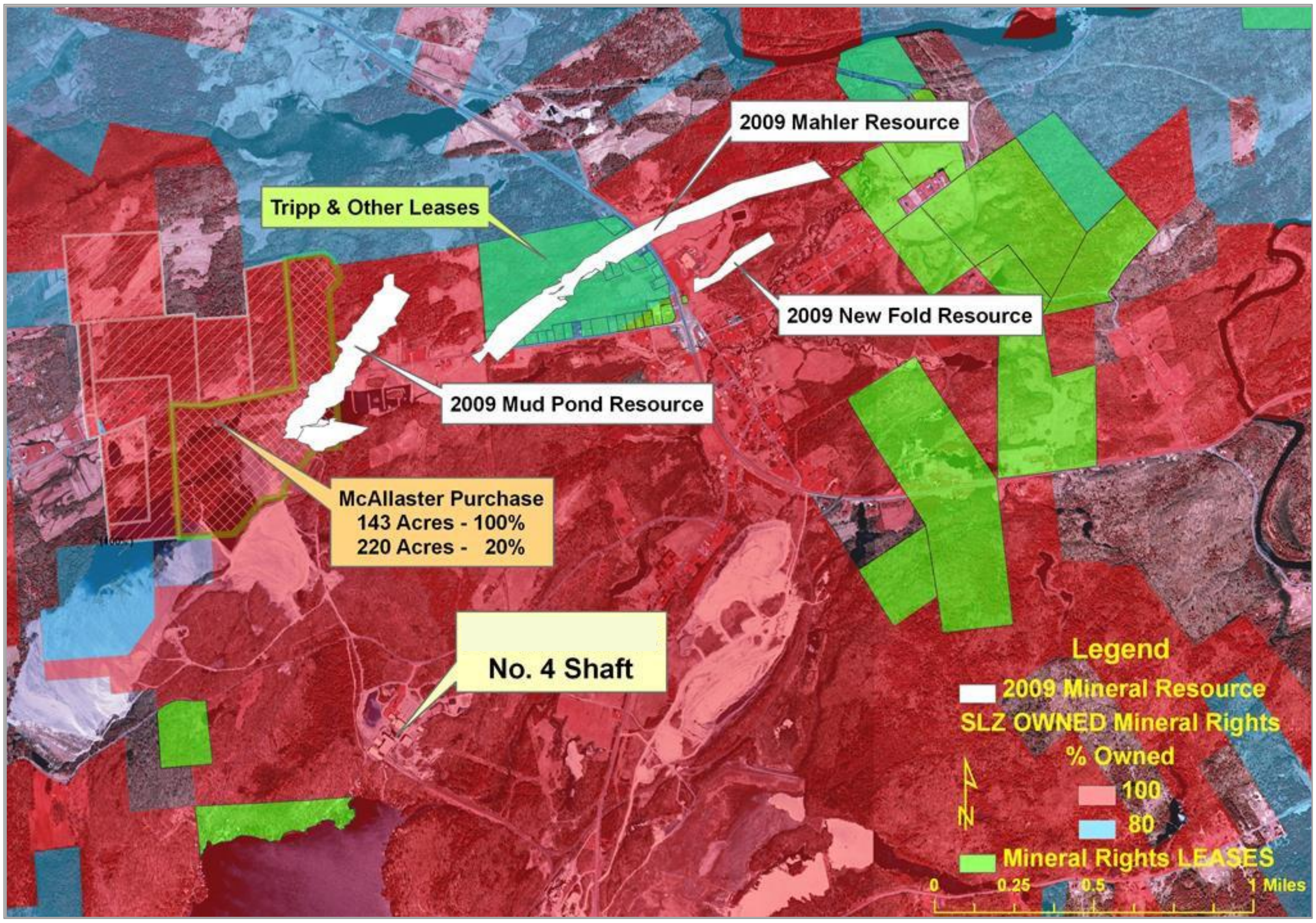
All property listed in Table 4.2 matches the St. Lawrence County 2019 tax rolls and are fully paid and current as of 1 March 2020.

Figure 4.3 Mineral tenure map



Source: Titan 2018.

Figure 4.4 Mineral tenure map



Source: Titan 2018.

4.3 Mining rights

Real property in New York State was originally granted to the owner to include both surface and mineral rights. However, mineral rights can subsequently be reserved or sold (severed) separately. Titan 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 may also be subject to the payment of royalties to the landowner. Average royalties for ESM mineral production are estimated to average 0.3% over the life of the mine (Titan 2020).

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, 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 Titan. 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 Inc. (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 (DEC) 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 4.3 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	30 Sep 2024
Water	SPDES Water Discharge Permit	NY0001791	31 May 2019[1]
Water	Water Withdrawal Permit	6-4038-00024/02001	31 May 2019[2]
Mining	Mining Permit	6-4038-00024/00006	31 Jul 2025
Storage	NYDEC Chemical Bulk Storage	CBS#6-000122	1 Oct 2021
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770	26 Sep 2023
Radiation	Certificate for Density Gauge	44023174	15 Sep 2022

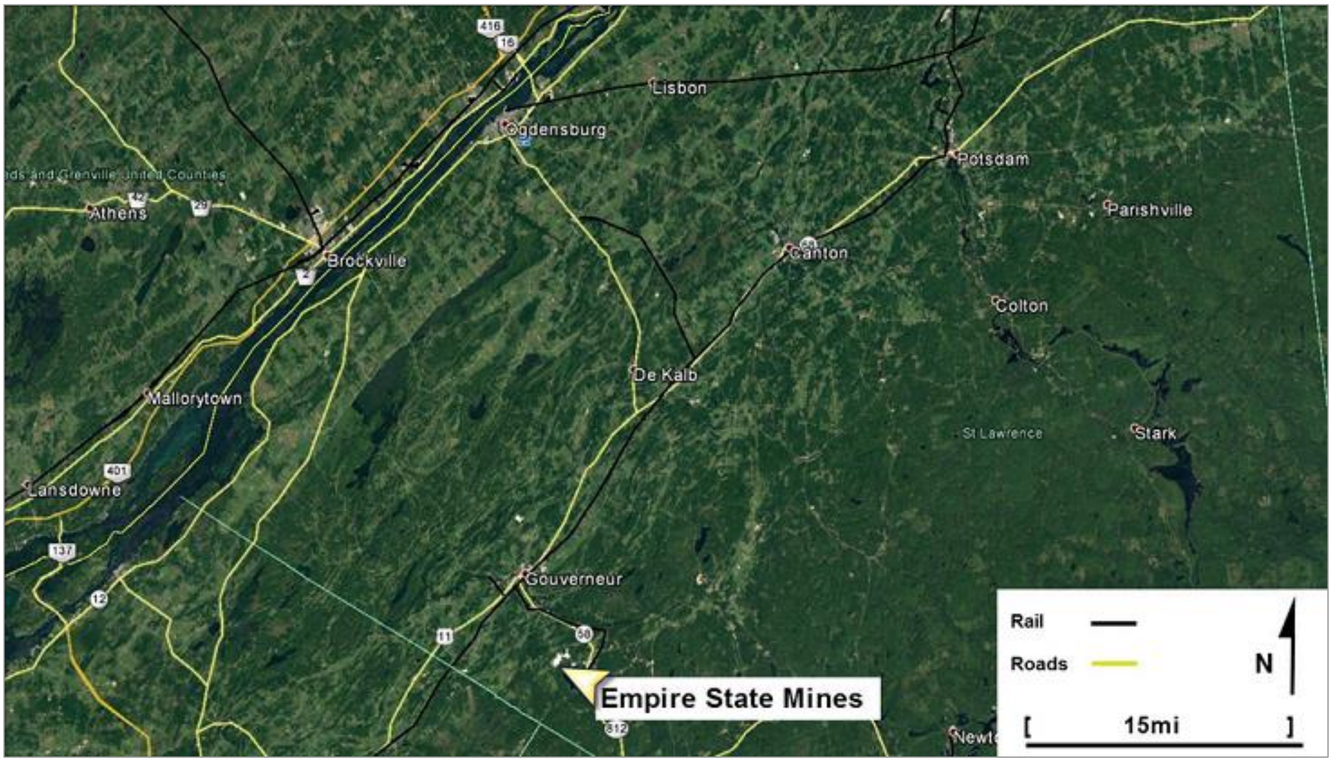
Notes:
[1] SPDES = State Pollutant Discharge Elimination System.
[2] SPDES AND Water Withdrawal permits are under Technical Review by the New York State DEC and are still valid despite the expiration dates. Source: ESM 2020.

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Accessibility

The property is reached by traveling south-east 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 2018 Report.

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 south-west. Ottawa, Ontario, Canada lies 90 mi to the north.

The mine is located in a desirable area to live. While a large portion of the workforce was non-local during the 2018 restart, the current workforce is nearly 100% local to Gouverneur and the surrounding communities.

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 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 north-west 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 ft to 1,106 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.

5.6 Surface facilities and rights

The existing operation is located on lands owned or leased by Titan. 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.

The site facilities have been maintained and the company has re-established surface infrastructure including office buildings, shops, mill, headframe, tailings, and ventilation facilities. During the start-up of the mine, labour that was not available locally has been sourced from outside of the region. A training program has commenced to provide miner basic training, to establish a source of trained local personnel.

Figure 5.2 Empire State Mine aerial view

Source: Titan.

6 History

6.1 Management and ownership

The ESM operation is wholly owned by 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 30 December 2016, Titan US purchased the shares of Balmat Holding Corporation, which in turn holds the shares of SLZ. Titan was a privately held company which had ESM as its primary asset. 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 & Predecessors	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 Inc. 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 ESM.

Source: SLZ 2018.

6.2 Exploration history

In 1838, zinc was discovered in a prospect pit on the Balmat farm which is located near the current location of Balmat No. 1 shaft. Further zinc mineralization was discovered in the Balmat-Edwards-Pierrepont district from road excavations that was developed into the Edwards mine (1908) and Hyatt (1917) mine. 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 zone. 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.

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 (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. In September 2003 OntZinc, renamed Hudbay in December 2004, purchased the idle Balmat assets. 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 have produced 33.8 million tons (Mt) at 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 reserve estimation 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 by mine

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 2018.

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	55.0
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
2018	ESM	187	7.9	0	0	23.9	58.1
2019	ESM	218	8.3	0	0	29.9	58.7

Source: SLZ 2018.

6.4 Historical Mineral Reserves

A list of most recent Mineral Reserve estimates is presented in Table 6.4. Hudbay's Reserve estimates concluded in 2008, with the 2015 reserves prepared by Star Mountain Resources. ESM is not treating these historical estimates as a current Mineral Reserve. The QPs are unaware of the methods, parameters or assumptions used to generate these historic estimates and cannot comment to their accuracy.

Table 6.4 Historical 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	1,159	11.52%	598	9.81%	1,758	10.94%
2005	686	10.60%	1,023	11.40%	1,709	11.00%
2006	912	10.10%	1,163	11.40%	2,075	10.80%
2007	1,000	9.50%	890	10.80%	1,891	10.20%
2015	152	9.00%	394	9.20%	531	9.20%

Source: SLZ 1985, Hudbay 2005-2009, Star Mountain 2015.

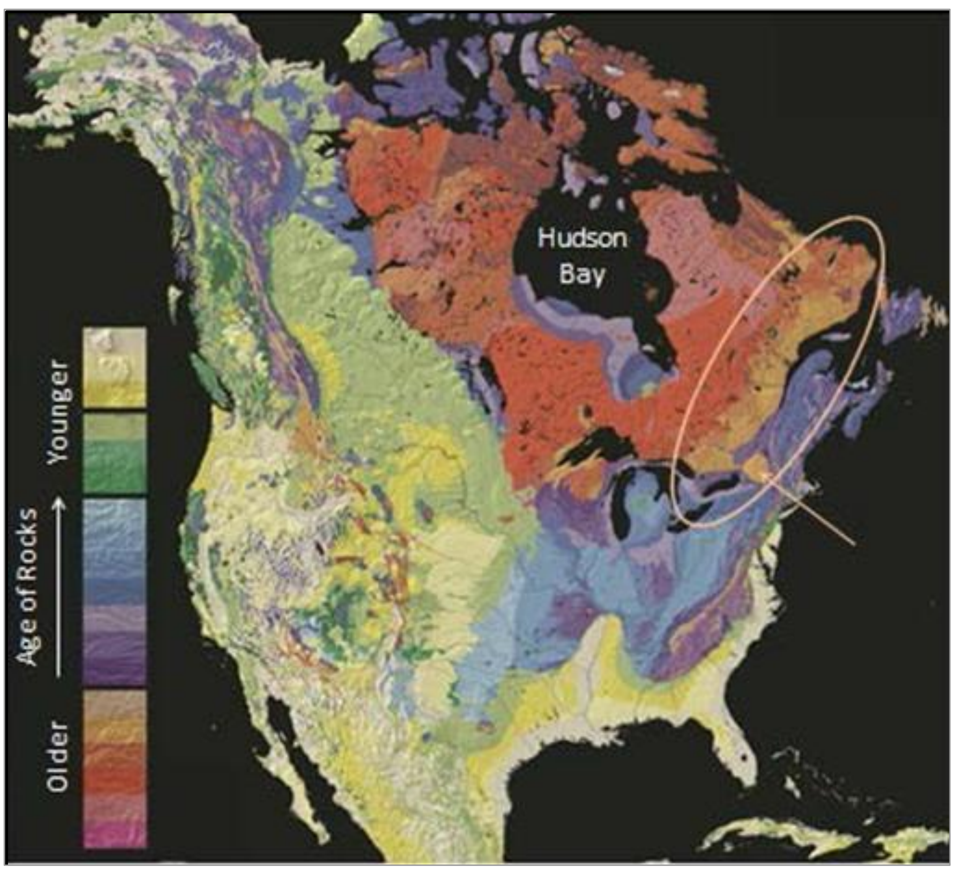
7 Geological setting and mineralization

7.1 Geological setting

The host rocks at ESM were deposited during the mid-Proterozoic era between roughly 1,300 to 1,000 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 forces that, over a billion years, assembled and broke up into two supercontinents at different times: Rodinia in the late Proterozoic, and Pangaea in the late Paleozoic to early Mesozoic. Sulphide deposition is interpreted to have occurred contemporaneously with deposition of the rock units. The originally tabular sulphide deposits were intensely deformed and metamorphosed along with their host rocks through eons of varying tectonic forces. The primary mineral of interest in the district is sphalerite.

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 is the red and orange band encircling Hudson Bay.

Figure 7.1 Regional geology setting



Source: SLZ 2018.

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 sulphide 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.

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).

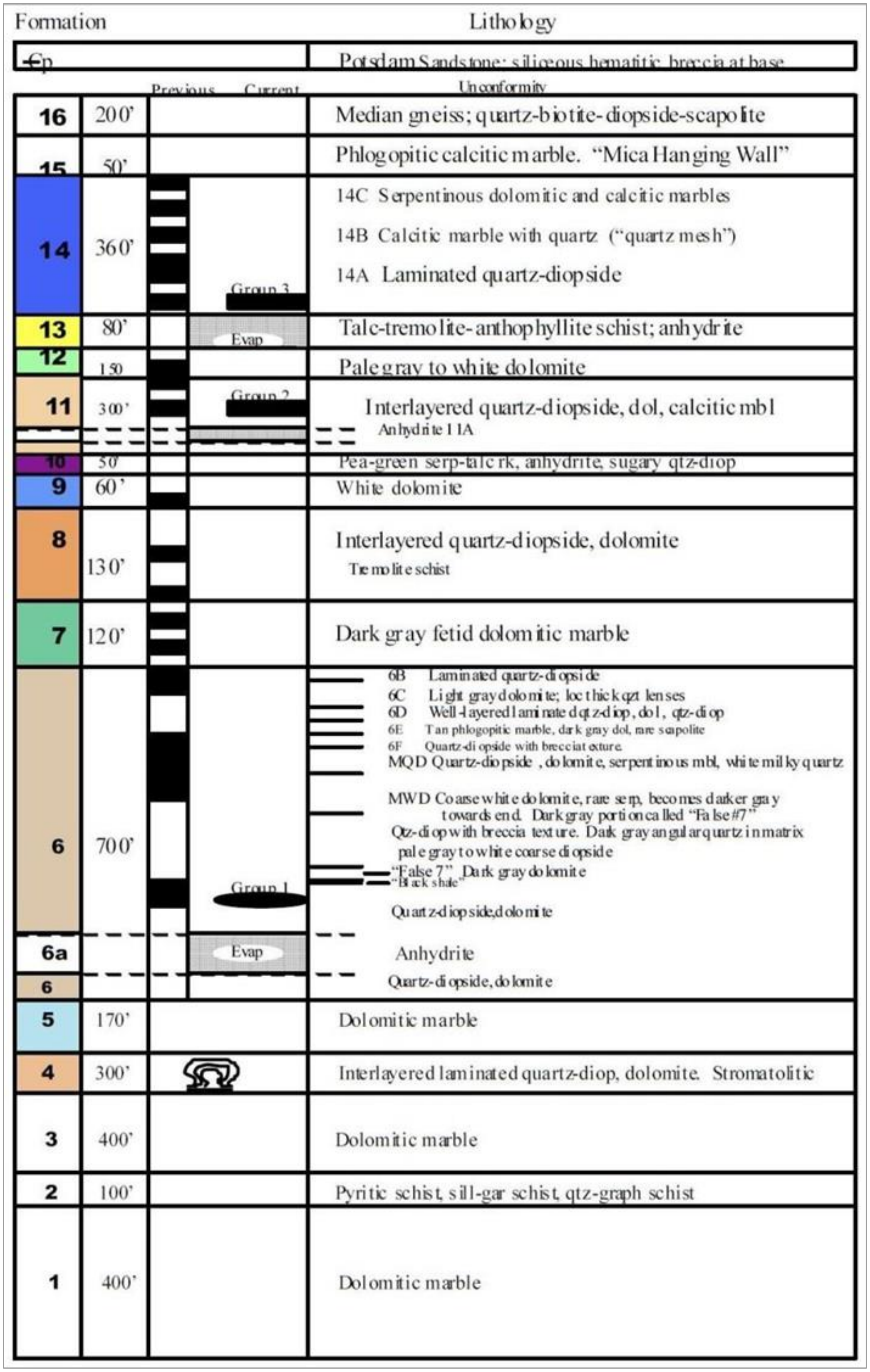
7.2 Regional geology

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 north-western 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 north-western 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-Colton 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 sulphide mineralization is hosted in the Upper Marble Formation.

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. Figure 7.2 shows the mine stratigraphic column which is divided into 16 units.

Figure 7.2 Empire State Mines stratigraphic section

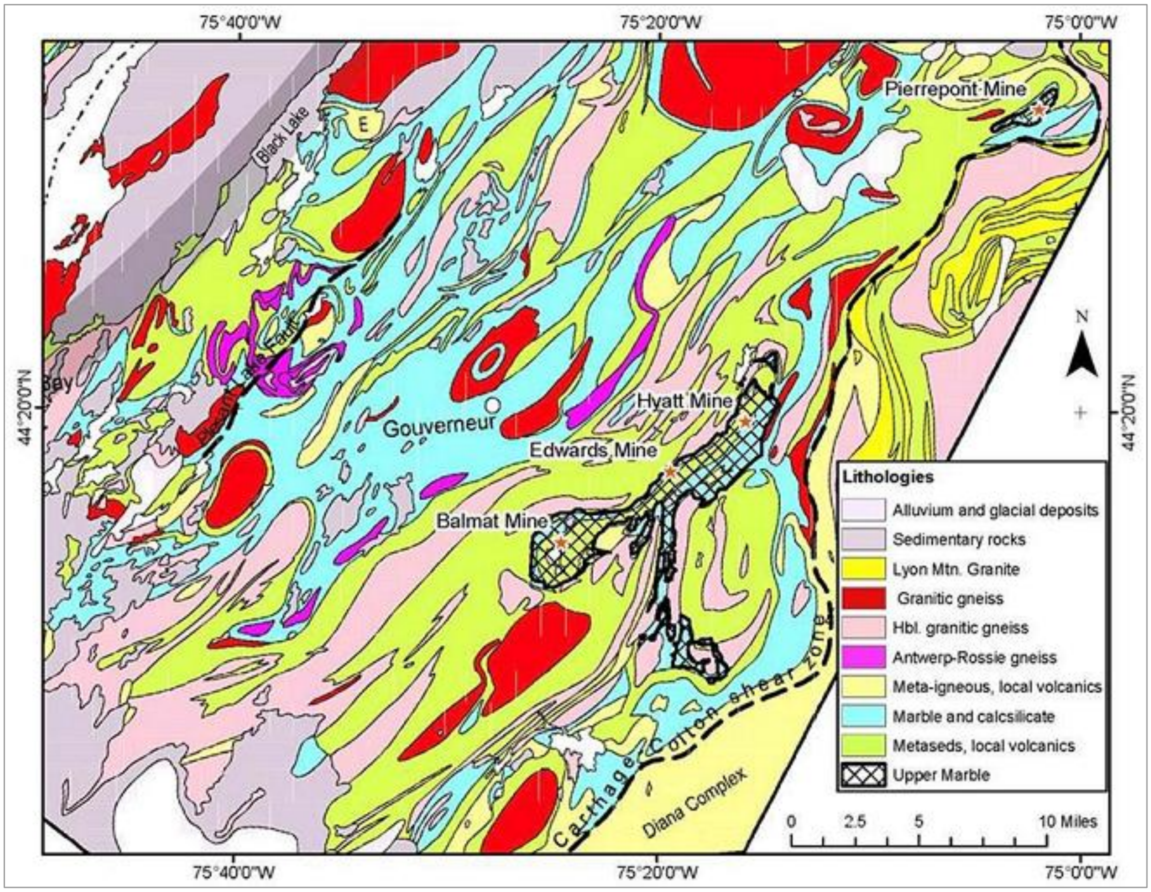


Source: SLZ 2018.

7.3 Property geology

As a result of intense tectonism, the Upper Marble Formation is extensively deformed. The predominant structure is the Sylvia Lake Syncline, a major south-west to north-east 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 mineralization 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 Balmat, Edwards, and Hyatt mines mark the axial trace of the Sylvia Lake Syncline.

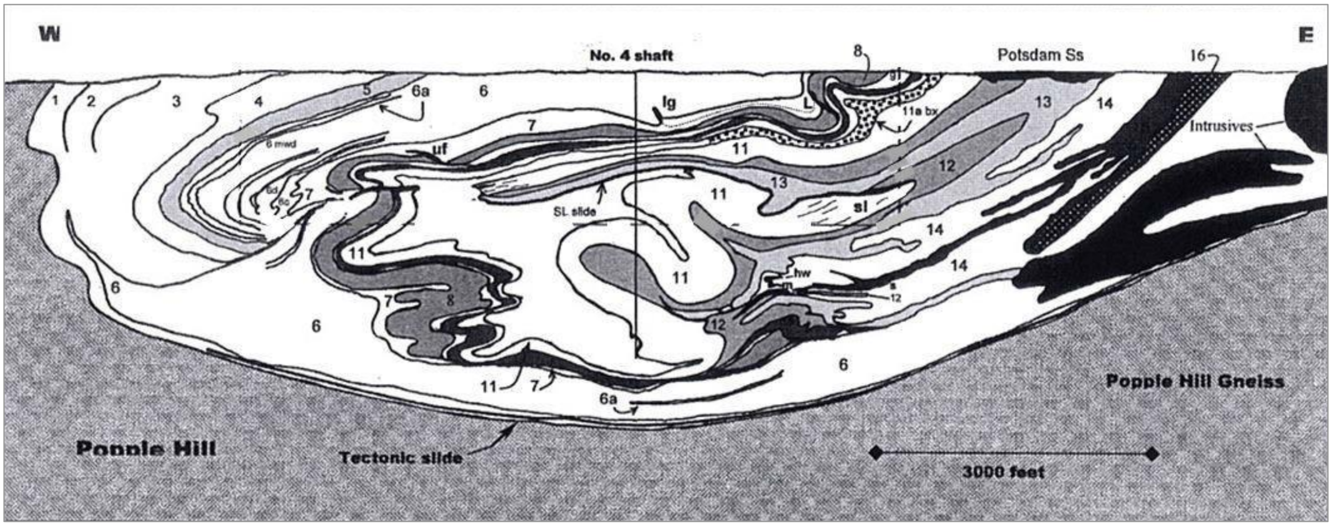
Figure 7.3 Local geologic setting



Source: SLZ 2018.

The sulphide deposits 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 experienced during the orogeny's mentioned in Section 7.1. Extreme contrasts in ductility exist in the Upper Marble Formation, ranging from very ductile anhydrite and sulphide beds to brittle silicious interlayered quartzite and diopside. These rheologic contrasts in the rocks drove complex large (miles) to small (tens of feet) scale structural processes during compression. Large scale fold interference patterns resulted in broad north-eastern trending arc-like structures that trend with the axial trace of the Sylvia Lake Syncline. Figure 7.4 is a cross section through the Sylvia Lake Syncline which illustrates the extent of deformation of the Upper Marble Formation.

Figure 7.4 Section through the No. 4 Shaft



Source: SLZ 2018.

7.4 Mineralization

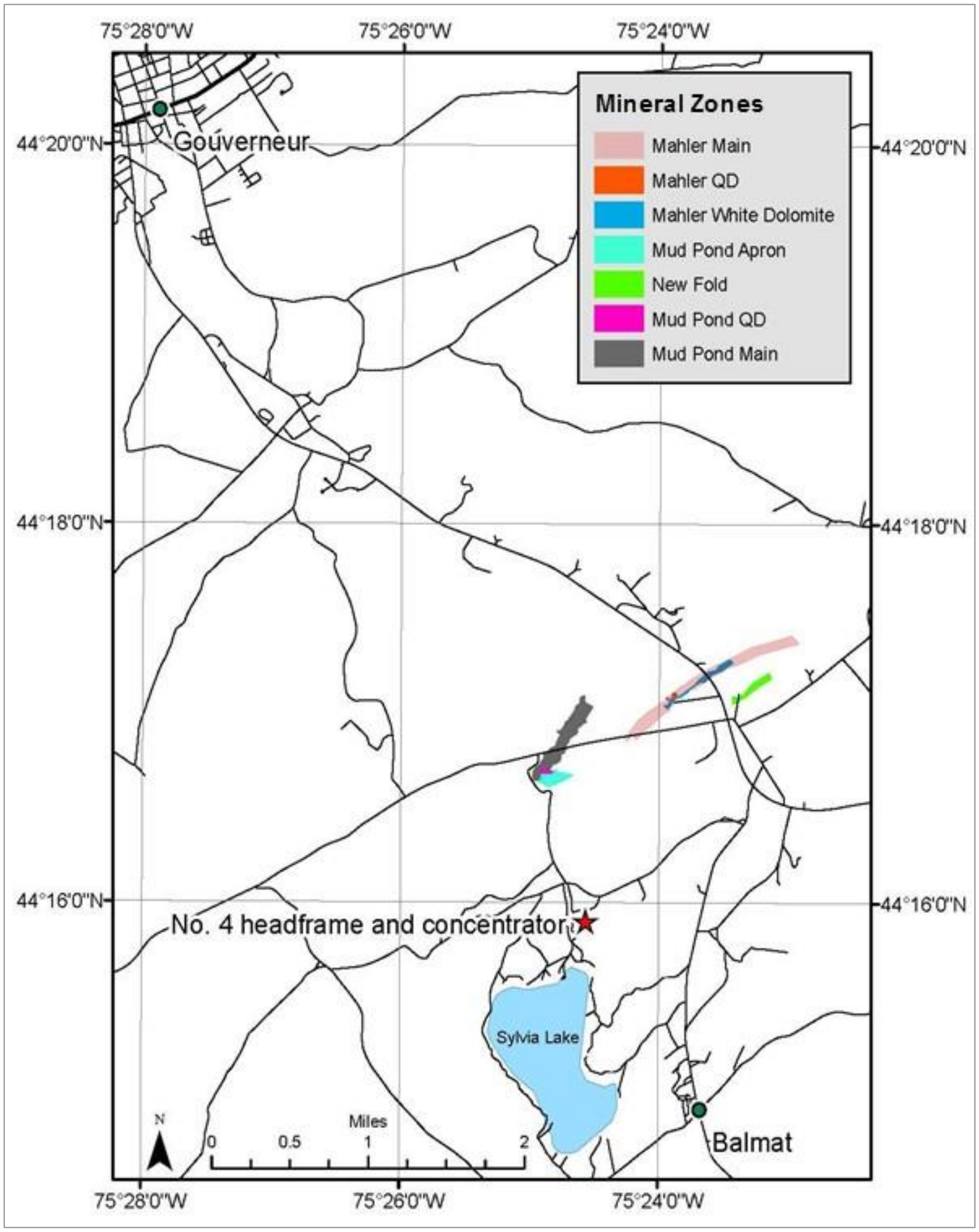
The mineralization at ESM has been classified as sedimentary exhalative (Sedex) in origin. The composition of the mineralization is unique, composed of primarily massive sphalerite and only minor galena and pyrite. Massive and semi-massive sphalerite-bearing deposits occur in siliceous dolomitic and 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 property contains 14 known zones of sphalerite mineralization. Three clusters have been defined consisting of three to five deposits each. Geometry of mineralization varies, ranging from tabular to podiform and shallow to steeply dipping. Areas defined to date contain tonnages ranging from roughly 0.5 Mt to over 10 Mt. Typical thickness ranges from 2 ft to 12 ft thick. Mineralization tends to be very continuous along strike, ranging from 50 ft to 800 ft Plunge-lengths may exceed 6,000 ft. Figure 7.5 shows the locations of sphalerite mineralized bodies currently being considered for production.

There are two mineralization styles recognized in the district. Stratiform high-grade massive sphalerite is interpreted as primary mineralization contemporaneous with deposition of the Upper Marbles. Discordant breccia-like "durchbewegung" textured sphalerite is considered to be secondary and remobilized along brittle-ductile shear zones. Mine geologists conceptualize a primary-secondary relationship, where the stratiform mineralization is the primary source and the crosscutting zone, locally called "durch", is the secondary. The structural model suggests that secondary resources are formed from sphalerite remobilized during metamorphism. The sphalerite migrates along structural conduits laterally from their source. The remobilized zones share similar trace element geochemical signatures with the interpreted primary zones. The durch often contains significant quantities of occluded wall rock material which imparts a distinctive texture. Previous workers have experienced exploration success using the structural model, defining four new zones in the 1990's.

The zinc-lead ratio is approximately 35:1 in most mineralized areas. 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. Galena is characteristic of primary stratiform mineralization with the secondary deposits exhibiting very minor amounts. Mine geologists have hypothesized that intense metamorphism may have concentrated the sphalerite, perhaps fractionating zinc sulphide (sphalerite) from lead and silver sulphides (galena) and remobilizing them to different locations leading to the high zinc grades observed at ESM. Galena and pyrite are occasionally observed within an aureole adjacent to some resources concentrated as fine veinlets or disseminations on the order of a few inches to feet particularly within the more brittle lithologies.

Figure 7.5 Location of zinc mineralized zones



Source: SLZ 2018.

8 Deposit types

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

The term Sedex is derived from the words sedimentary and exhalative 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.

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).

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 to 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 9,800 ft (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.

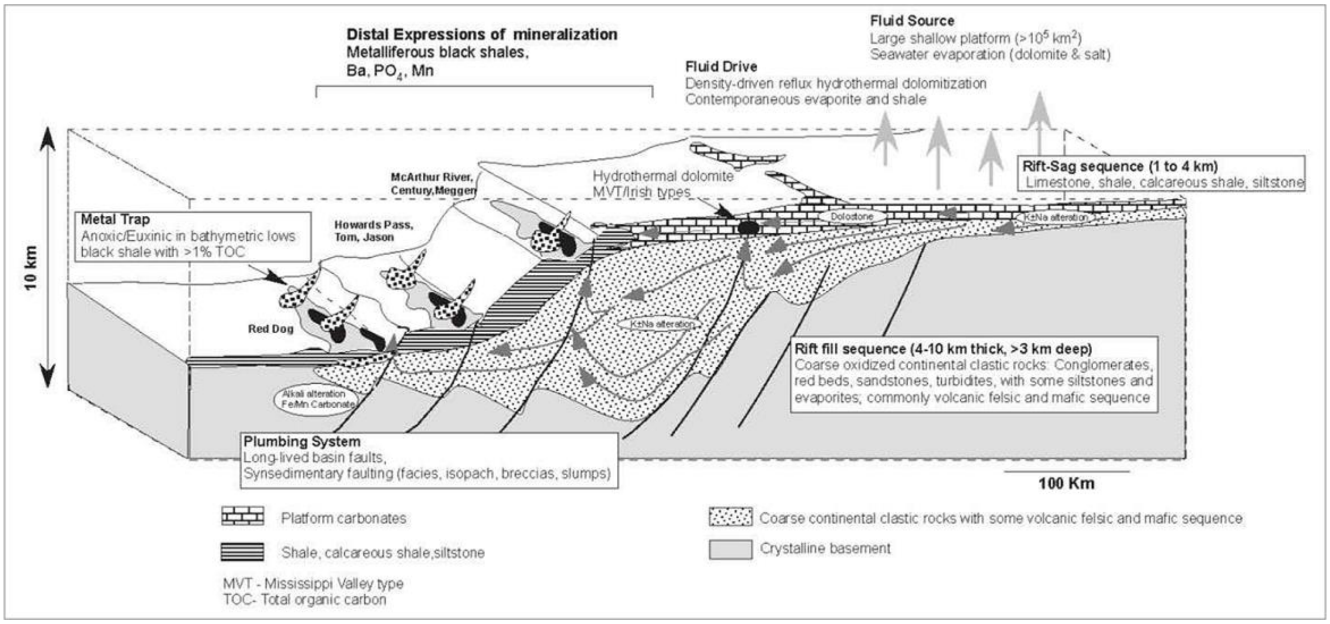
Warm, metal-laden migrating brines may eventually encounter extensional fault surfaces and migrate up the faults to the basin floor. Workers describing sedimentary basins have divided the basins into three orders of scale. First-order sedimentary basins which host Sedex deposits are greater than 328,000 ft (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.

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 are 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. Fluid inclusion studies indicate that sediment-hosted lead-zinc deposits, both Sedex and MVT (Mississippi Valley-type), originate from similar brines.

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: ESM 2018.

9 Exploration

Current exploration activities, in the Balmat-Edwards district, include surface exploration drilling, digitization of historic exploration and mine data (drill logs, geological mapping, cross sections, and mine workings) that are utilized to generate 3D geologic models in Leapfrog, and a surface geochemistry sampling program to identify "blind" deposits using indicator minerals. Exploration activities in the north-west Adirondacks include prioritizing high quality exploration areas, identified from the 2013 VTEM survey, followed by field reconnaissance (mapping and sampling). Drill targets for the Balmat-Edwards district and the north-west Adirondacks are generated using 3D geologic modelling, surface geochemistry data and geophysical surveys and ultimately tested.

Regional zinc exploration in the Balmat-Edwards marble belt, as well as the north-west Adirondacks resulted in the discovery of five new mineralized bodies within the last 25 years (three in the Balmat mine and two in the Hyatt mine).

All major resources exist on a trend between the original Balmat mines and the Pierrepont mine, called the Balmat-Pierrepont trend. Resource exploration is divided into three categories: near-mine, Balmat-Pierrepont trend, and district wide. Near-mine exploration focuses on developing extensions of existing resources within the Sylvia Lake Syncline and re-analyzing historic drilling for opportunity. Balmat-Pierrepont trend exploration seeks to discover on-trend untested pockets of mineralization similar in style to the existing resources between Balmat and Hyatt. District wide exploration has potential to discover a separate yet-to-be discovered trend of mineralization. The last three discoveries were all located near-mine in the Sylvia Lake Syncline.

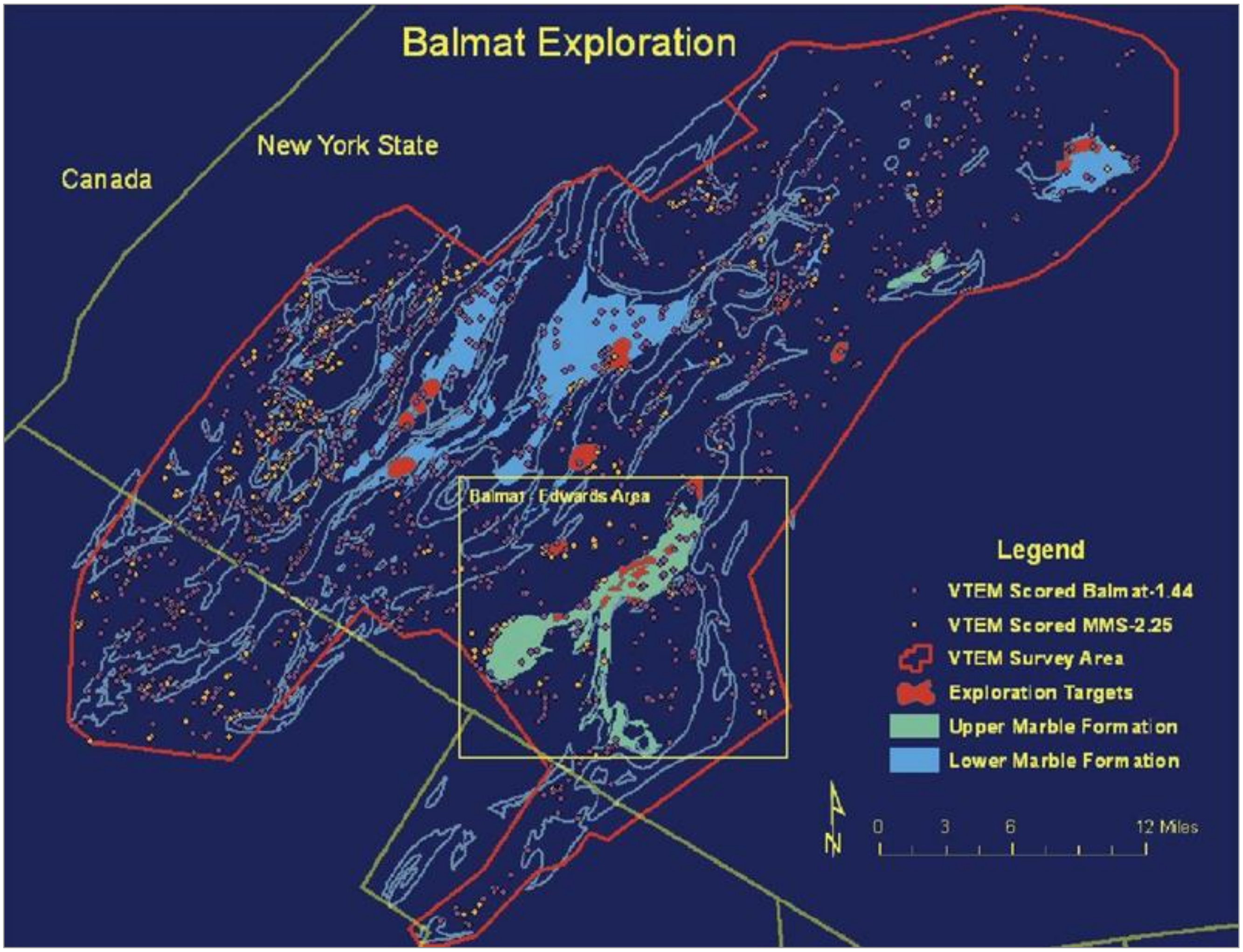
In 2013, Geotech Ltd. of Aurora, Ontario flew a helicopter borne VTEM (versatile time domain electromagnetic) geophysical survey over the Adirondack Lowlands of northern New York on behalf of Hudbay. The survey area covered a nominally rectangular area of 47 mi x 22 mi, including the greater Balmat mining 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 deposit 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. Ten high quality exploration areas were identified outside the Balmat mining district.

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 and areas where low resistivity was recorded (Rivard, Stephens, Beaufield Resources 2013).

Figure 9.1 Geophysical survey area



Source: SLZ 2018.

10 Drilling

10.1 Drilling summary

A total of 8,645 diamond drillholes have been completed historically, totaling 3,475,744 ft, as shown in Table 10.1. All known historic drilling was digitized and incorporated into the digital drillhole database in a major update in 2018, so the numbers in this document will be significantly greater than earlier reports (JDS 2017). As far as ESM is aware, no additional significant groups of historical drilling remain to be digitized.

The primary focus of the 2018 – 2020 drilling programs were to further definition of the New Fold, NE Fowler, and #2D resources. Overall, 67 definition holes were drilled in areas with relatively low confidence in support of mine planning.

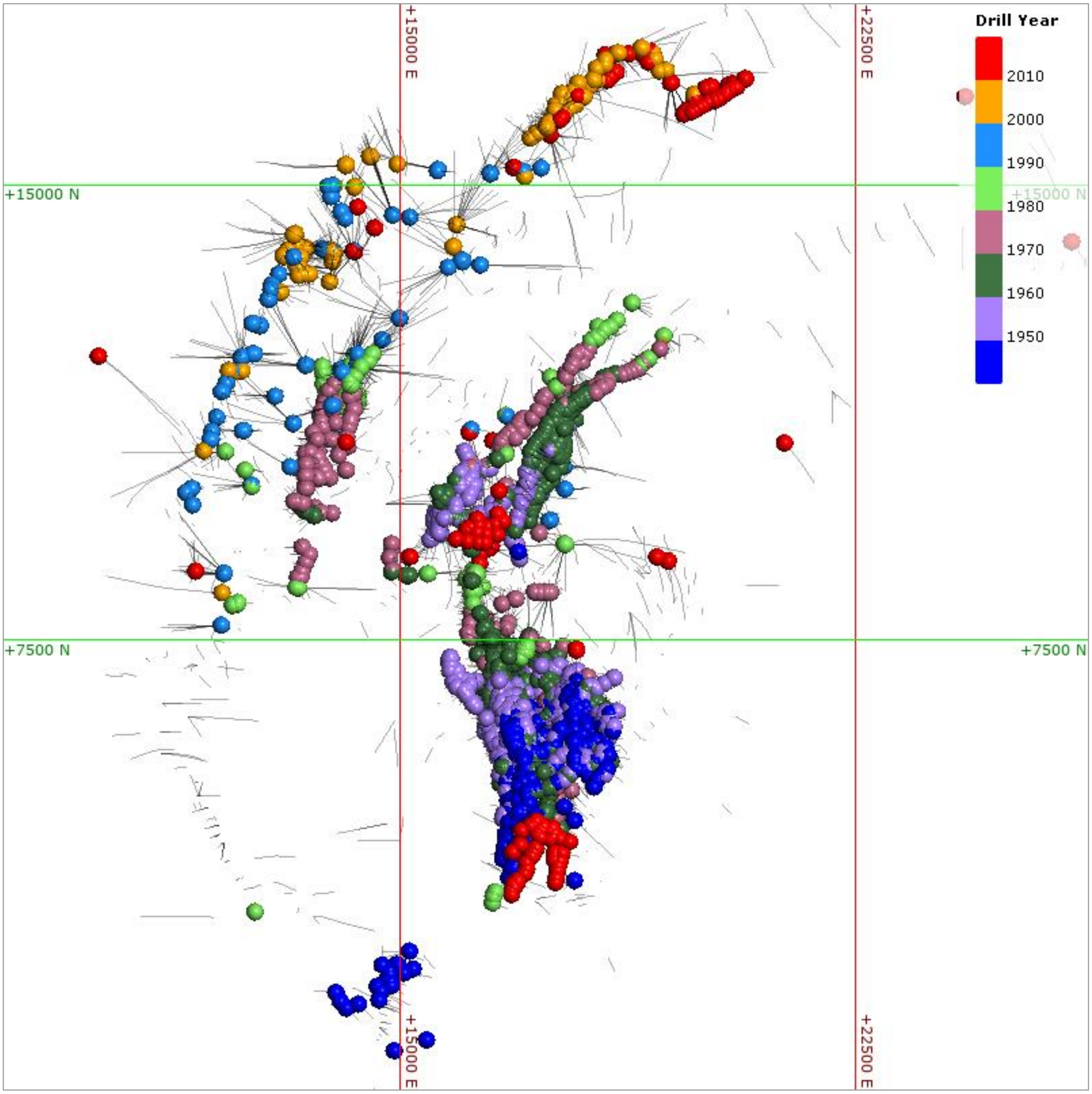
Additional drilling was carried out at the historic #1 and #2 Mines, referred to as the Turnpike and Hoist House areas, respectively. The purpose of this program was to confirm historic mineralization and to define Mineral Resources for potential open pit mining.

Table 10.1 Project drilling by year

Year	Surface core		UG core		Total holes	Total length (ft)	Total length (m)
	Holes	Length (ft)	Holes	Length (ft)			
Pre-ESM	1,650	1,561,591	6,758	1,700,012	8,408	3,261,603	994,137
2017	9	16,079	16	9,019	25	25,097	7,650
2018	28	81,353	43	42,129	71	123,482	37,637
2019	73	39,475	68	26,088	141	65,563	19,984
Total	**1,760**	**1,698,498**	**6,885**	**1,777,247**	**8,645**	**3,475,744**	**1,059,407**

Source: ESM 2020.

Figure 10.1 Map showing the distribution of drilling by decade



Source: ESM 2020.

10.2 Drilling procedures

Drilling at ESM has been exclusively core drilling. The mine owns two Diamec 262 underground drills which drill AW size core, which were utilized for the current definition programs. Three contract Longyear underground drills that drill BQ size core were utilized from 2005 - 2008. Cabo was contracted to drill underground in 2018 - 2019 and Boart Longyear was contracted for all surface programs in 2018 - 2020. Prior to ESM's 2018 - 2020 surface and underground drill programs, the drillhole database contained 4,342 drillholes completed at various times in the project's history within the Balmat area. ESM has subsequently added 4,050 historic drillholes to the database through the digitization of original log scans. Drilling during 2018 – 2020 consisted of both surface and underground holes. A total of 128 surface holes and 110 underground holes were drilled in the #2 and #4 Mines as well as near the historic No. 1 and No. 2 shafts, with an aggregate total of 194,755 ft.

10.3 Core handling and sampling

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 wax impregnated cardboard or plastic core box. Wooden blocks were used to mark the ends of individual core runs. The core was then transported to the surface where geologists logged the core and selected and marked the intervals to be prepared for assay samples.

Surface drill core is transferred from the core barrel to the core box. The core is then collected from the rig by an ESM core technician and brought to ESM's core shed where it is photographed, logged, and sampled. All core is cut in half, lengthwise, using a diamond saw with a diamond-impregnated blade. Typical sample intervals lengths range from 1-ft to 5-ft depending on areas of mineralogical or geological interest.

After a sample is cut, one half of the core was returned to the original core box for reference and long-term storage. The second half of the core was placed in a plastic or cloth sample bag, labelled with the corresponding sample identification number, along with a sample tag. All sample bags were secured with staples or a draw string, weighed and packed in shipping boxes. They are transported by UPS courier to ALS Minerals' laboratory in Sudbury, ON, Canada for sample preparation and then to ALS's lab in Vancouver, BC, Canada for analysis.

Drillholes are logged directly into the GeoSpark digital database and all assays are imported upon receival from the analytical lab. Drilling conditions in the Upper Marble Formation are generally very good, and core recovery is typically excellent. Average core recovery from the most recent drilling programs was 97%. Sphalerite mineralization is readily identified, and sample intervals are chosen by trained geological staff. Samples are shipped off-site for analysis by a reputable independent assay laboratory.

10.4 Downhole surveying

Downhole survey methodology on the property has evolved significantly over the last century as industry technology changed. The first surface exploration drillholes to develop the Number 2 resource relied on acid-etch tubes for some form of control, but the bulk of the drilling completed in the first half of the 19th century have no downhole survey information and holes were assumed to be straight. In the mid 1960's the Pajari Directional Survey Instrument, aka. Tro-Pari, became the primary source of downhole directional data if it was collected at all. The Tro-Pari was used until 2018. The device is susceptible to numerous sources of error and as such any hole known to be surveyed with the instrument is now considered to be low confidence and flagged as such in the database. Surface drilling, since 2017, has exclusively used the REFLEX EZ-SHOT instrument while underground drilling has relied on the DeviShot.

Other than the downhole surveying in the historical drillholes, the QP are not aware of any issues that would negatively impact the accuracy and reliability of drill sample results at ESM.

11 Sample preparation, analyses, and security

11.1 Historical assaying

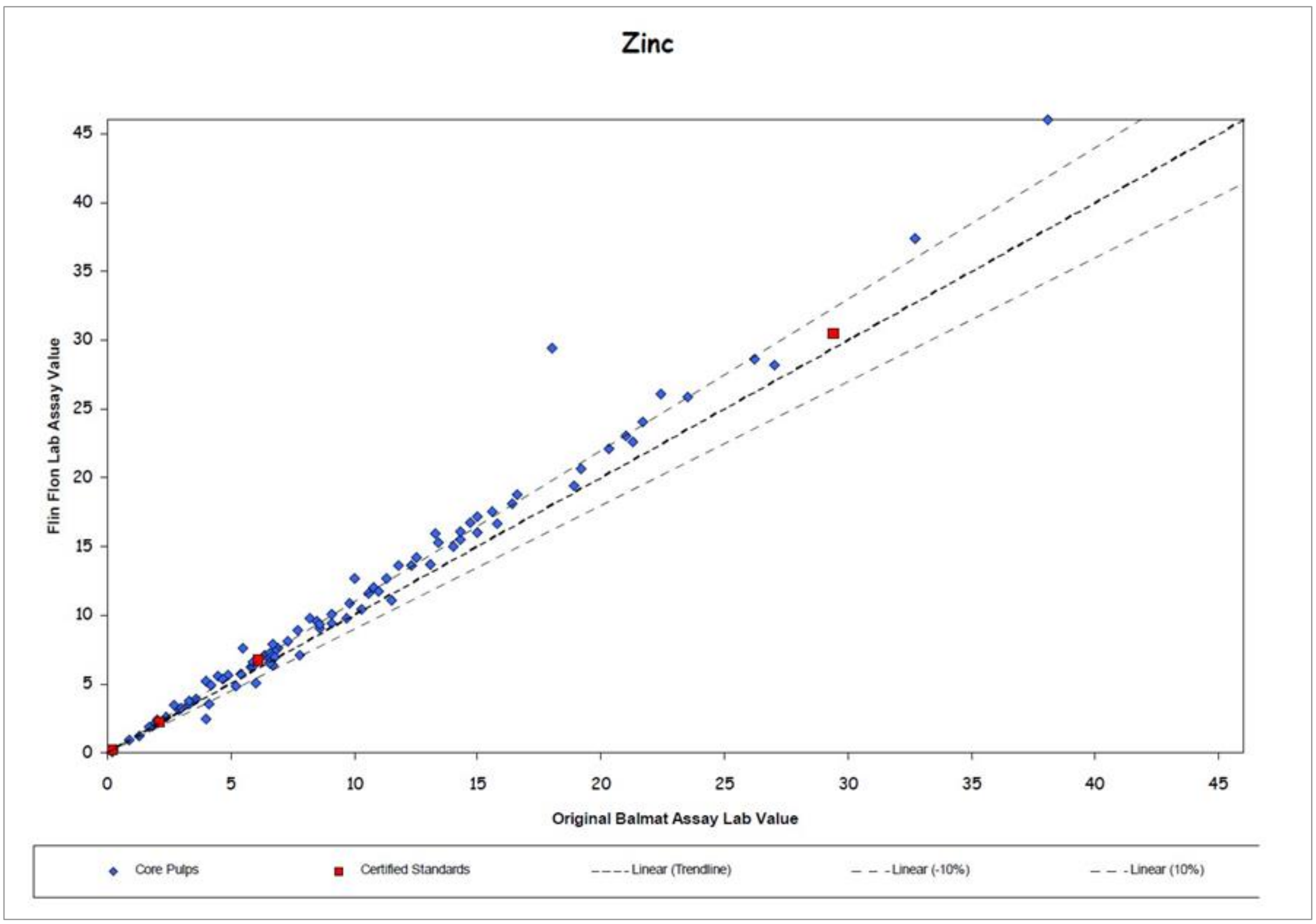
11.1.1 Pre Hudbay and checks

Prior to the 2003 acquisition of the Property by Hudbay, 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 drillhole identification and assay interval.

Assays from these years were not supported by a defined quality assurance / quality control (QA/QC) protocol. Hudbay selected 86 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 gal 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 particle size. Zinc assays were completed for each sample.

The Flin Flon laboratory reported consistently higher results than those obtained by the ESM lab. The Flin Flon laboratory reported zinc assays more than 10% higher than the ESM laboratory for zinc assays greater than 25%. The certified reference standards were all within acceptable limits.

Figure 11.1 Hudbay Flin Flon Lab check assays of ESM 1995 to 2000 pulps



Source: SLZ 2018.

There are a limited number of check assays performed at Hudbay's in-house laboratory; these indicate that the ESM assays prior to 2003 may underestimate zinc concentrations.

11.1.2 Hudbay post-2005 assaying

All drillhole core samples from the 2005 to 2010 diamond drilling programs were sent to the ALS Chemex Laboratory in Sudbury, Ontario. The QA/QC program initiated by Hudbay included:

- Insertion of a barren material (Blank) for one in 50 samples.
- Insertion of one in-house reference material for one in 20 samples.

The materials used as Blanks were sourced from different local material and were not consistently barren of zinc. There was no evidence of systematic zinc contamination.

In 2004, Hudbay supplied five different grades of material (grab samples) from the mines in the Flin Flon camp that represented the grades encountered at the mines. Ore Research and Exploration Pty. Ltd. (OREAS) prepared packets of certified reference materials (CRMs) based on a "round robin" and used the average of assays from eight independent laboratories.

Table 11.1 Hudbay QA/QC standards certified by OREAS Hudbay

	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 2018.

All standards come finely crushed in foil packages clearly labelled with the standard type (A-4, B-4, C-4, or E-4). These reference materials are no longer in use.

In 2008, two new CRMs (G-5 and H-5), were prepared by OREAS using sulphide material from the Balmat mine. The CRMs were certified with round robin assaying at 15 laboratories. All of the laboratories performed analysis using an aqua regia digest and mostly ICP-OES instrumental finishes.

Table 11.2 ESM QA/QC certified standards supplied by OREAS 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 2018.

No check assay data were located from the Hudbay drill programs.

There is no documentation to suggest that Hudbay found systematic errors for the assays performed at ALS, Sudbury.

11.2 ESM 2017 assaying

The 2017 ESM drilling program included 25 drillholes; nine surface drillholes totaling 16,071 ft and 16 underground drillholes totaling 9,009 ft. A total of 561 samples (148 surface and 413 UG) were assayed by ALS Geochemistry (ALS), a laboratory accredited to the ISO/IEC 17025-2005 standard by the Standards Council of Canada.

The average turnaround time was 21 days from leaving ESM to receipt of the results (between 14 to 36 days).

Analysis was conducted by ALS (Quote Number ALSM-CE16-125-LAWZIN). The samples were shipped to ALS in Sudbury by either UPS parcel or YRC Freight. A master pulp was prepared at the Sudbury preparation laboratory with a sub-sample of the pulp sent to the ALS in the Vancouver laboratory for analysis.

Samples were prepared and assayed using the same protocols as described for drill core samples described for 2018 and 2019.

Quality assurance samples were submitted into the sample stream in a regular sequence; every sample ending in a zero (0) was selected as either a blank, CRM, or core duplicate. A total of 61 quality assurance samples were submitted consisting of 12 Blanks, 46 CRM’s, and three duplicates. The G-5 and H-5 standards prepared by OREAS were utilized for the 2017 to 2019 programs.

11.3 2018 and 2019 sample preparation and assaying

A total of 10,541 drill core samples were submitted to ALS Geochemistry between June 2018 and December 2019. The quality control data for these sample submittals are discussed in Section 11.3 for zinc, lead, copper, silver, gold, and iron.

11.3.1 Sample preparation and analysis

For the 2018 to 2020 drilling campaign, sample preparation (crushing and pulverizing) has been performed at ALS, an ISO/IEC 17025 accredited lab located in Sudbury, Ontario, Canada. ALS prepares a pulp of the sample and a portion (usually 100 grams) is forwarded to their laboratory in Vancouver, BC, Canada, for analysis.

All samples were prepared using ALS Method Core Prep-31 that includes:

1 Air dry if possible (maximum 120°C if oven drying is necessary).

2 Crush entire sample to at least 70% passing 0.1” (2 mm).

3 Riffle split 8 oz (250 g).

4 Pulverize approximately 8 oz (250 g) to at least 85% passing 75 microns.

As required, high grade samples are flagged on the ALS submittal form for an extra wash in sample preparation. Crushers and pulverizers are cleaned using quartz or other barren material after each sample that is flagged as being high grade.

The analytical methods are summarized in Table 11.3.

Table 11.3 Summary of assay methods

Analyte	Method code	Detection limit	Digest	Instrumentation
35 elements, see below	ME-ICP41	Varies; see below	0.25 g two-acid: HNO3 + HCl digest plus HCl leach	ICP-AES
Au	Au-ICP21	0.001 ppm	30 g fire assay	ICP-AES
Ag	Ag-OG46	1 ppm	0.25 g two-acid: HNO3 + HCl	ICP-AES
Pb	Pb-OG46	0.001%	0.25 g two-acid: HNO3 + HCl	ICP-AES
Zn	Zn-OG46	0.001%	0.25 g two-acid: HNO3 + HCl	ICP-AES
Zn	Zn-VOL50	0.01%	1 g Titration	Titration

Reference to metric units of g = grammes.

High grade samples, for silver greater than 100 g/tonne and base metals over 1 percent, are analyzed a second time using inductively coupled plasma methods optimized for high grade samples (Method Codes with OG). The same sample weight and acids are used for the repeat analysis. All samples in which zinc is greater than 30% are re-run once more using titration (Method Code Zn-VOL50) and reported in percentage.

The lower and upper limits for the aqua regia digest method (ME-ICP41) are shown in Table 11.4.

Table 11.4 Upper and lower limits for aqua regia ICP method

Analyte	Lower limit	Upper limit	Analyte	Lower limit	Upper limit	Analyte	Lower limit	Upper limit
Ag (ppm)	0.2	100	Fe (%)	0.01	50	S (%)	0.01	10
Al (%)	0.01	25	Ga (ppm)	10	10,000	Sb (ppm)	2	10,000
As (ppm)	2	10,000	Hg (ppm)	1	10,000	Sc (ppm)	1	10,000
B (ppm)	10	10,000	K(%)	0.01	10	Sr (ppm)	1	10,000
Ba (ppm)	10	10,000	La (ppm)	10	10,000	Th (ppm)	20	10,000
Be (ppm)	0.5	1,000	Mg (%)	0.01	25	Ti (%)	0.01	10
Bi (ppm)	2	10,000	Mn (ppm)	5	50,000	Tl (ppm)	10	10,000
Ca (%)	0.01	25	Mo (ppm)	1	10,000	U (ppm)	10	10,000
Cd (ppm)	0.5	1,000	Na (%)	0.01	10	V (ppm)	1	10,000
Co (ppm)	1	10,000	Ni (ppm)	1	10,000	W (ppm)	10	10,000
Cr (ppm)	1	10,000	P (ppm)	10	10,000	Zn (ppm)	2	10,000
Cu (ppm)	1	10,000	Pb (ppm)	2	10,000			

11.3.2 Security

Core is photographed and split in half with one-half retained in a secured facility for verification purposes.

Core and samples are stored in secure shipping containers, owned by ESM, on the mine site located in Gouverneur, New York. The on-site storage location also has facilities for core logging, core cutting, and core sampling. Core is stored in wax cardboard boxes and organized in shipping containers by drillhole number.

11.3.3 Quality assurance/quality control

To ensure reliable sample results, ESM has a rigorous QA/QC program in place that monitors the chain-of-custody of samples and includes the insertion of blanks and CRMs at consistent intervals within each batch of samples.

The assays for QA/QC samples are reviewed as certificates are received from the laboratory. Failures are identified on a batch basis and followed up as required. Quarterly QA/QC reports are prepared internally to monitor overall laboratory performance.

Barren coarse-grained silica blanks were inserted after high grade (visual estimate over 10% zinc) samples. Low, medium, and high grade (with respect to zinc) CRMs were inserted every 20th sample by random selection.

Elevated values for blanks may indicate sources of contamination in preparation, in the analytical procedure (contaminated reagents or test tubes) or sample solution carry-over during instrumental finish. Barren samples were purchased from Analytical Solutions Ltd. and certified by ALS in Vancouver, BC. The source of the material is carboniferous sedimentary rocks of the Maritimes Basin in New Brunswick from deposit of nearly pure silica.

The threshold levels for blanks are defined in Table 11.5.

Table 11.5 Blank failure threshold

Blank	Zinc (ppm)	Lead (ppm)	Silver (ppm)	Copper (ppm)	Iron (%)
Blank (ASL)	400	400	5	400	0.7

The threshold levels were applied based on observations of past results and understanding of the risks to the project. The weight of the blanks is approximately 200 grams or usually less than 10% of the weight of the sample; metal concentrations are enhanced in the smaller blank samples relative to what would be potentially carried-over in sample preparation to larger drill core samples.

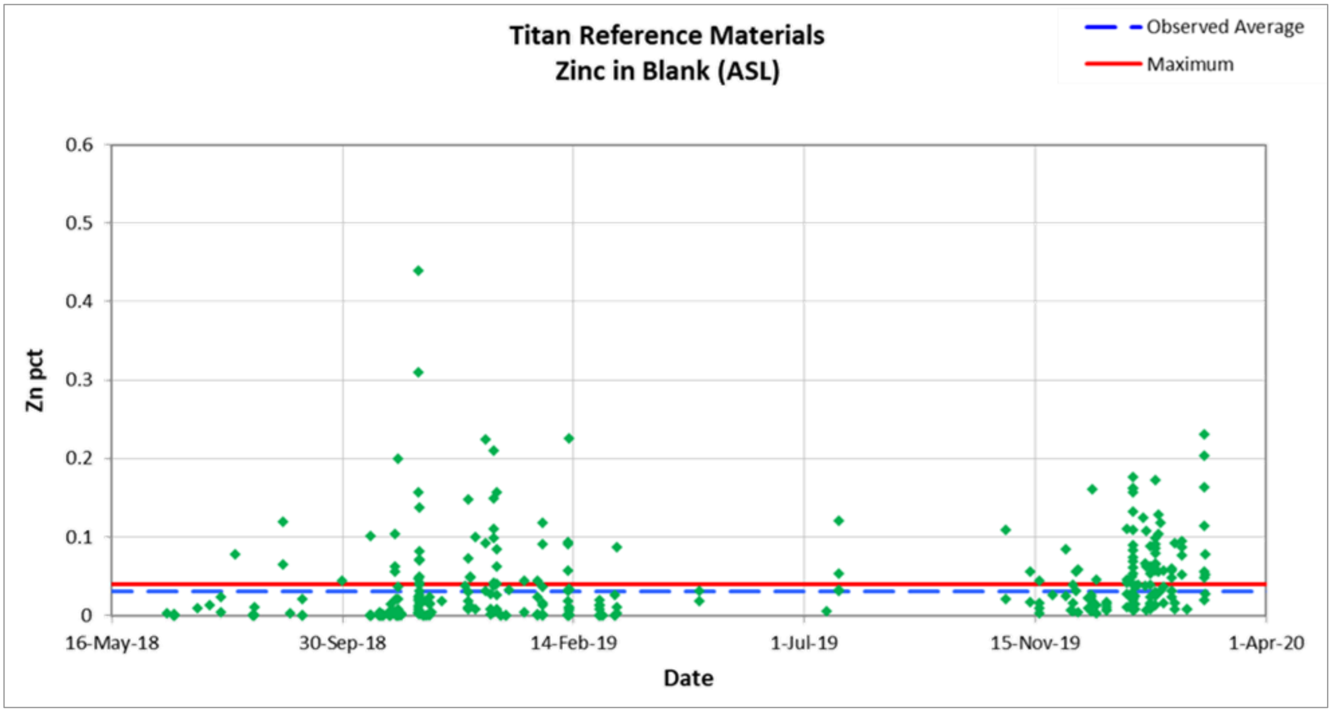
For the 439 blanks inserted with samples, all silver values were less than 1 ppm and copper values were less than 50 ppm.

There was a total of four cases where lead values exceeded 400 ppm and reported up to 0.057% Pb.

Blanks are inserted after samples expected to report more than 10% zinc as well as approximately every 20^{th} sample. As a result, there were cases of sample cross-contamination with 165 out of 439 cases reporting over 0.04% zinc.

Figure 11.2 is the control charts for zinc in blanks. In October to December 2018, there were a series of zinc values reporting over 0.04% Zn. The higher values for blanks were consistently found to be associated with preceding high grade drill core samples prepared before the blank. Similarly, there is a period in January and February 2020 where zinc values in blanks were reporting over 0.04% zinc.

Figure 11.2 Zinc in blank control chart



Source: Graph generated by Analytical Solutions Ltd. in QC Mine Software.

The potential for zinc contamination is moderated by ESM's practice of flagging sulphide-rich samples and requesting that the laboratory carry out additional quartz washes at crushing and pulverizing stages. Differences of 0.1 to 0.2% Zn within the high grade mineralized zones, with over 5% Zn, is not material for the project and does not constitute a risk.

When zinc reports over 0.4%, there are also reported cases of iron over 0.7%. The elevated iron values are also associated with high mineralized sulphide-rich zones and, again, do not constitute a risk to the project.

In cases where there appears to be a higher than expected carry-over, repeat assays have been requested at ALS. In general ALS responds that the carry-over was less than 1% which is within its method expectations.

The results for reference materials are summarized in Table 11.6.

Table 11.6 Summary tables of results for reference materials

RM	N	Outliers excluded	Failures excluded	Zn pct		Observed Zn pct		Percent of accepted
				Accepted	Std. Dev.	Accepted	Std. Dev.	
OREAS-H5	138	-	2	22.900	1.400	24.803	0.498	108.3%
OREAS-G5	173	-	-	9.970	0.590	10.333	0.216	103.6%
OREAS-135	201	1	1	2.800	0.104	2.773	0.061	99.0%
Total	**512**				**Weighted average**			**103.1%**

RM	N	Outliers excluded	Failures excluded	Cu ppm		Observed Cu ppm		Percent of accepted
				Accepted	Std. Dev.	Accepted	Std. Dev.	
OREAS-H5	139	-	1	433.000	20.000	453.345	14.860	104.7%
OREAS-G5	173	-	-	601.000	38.000	614.058	18.745	102.2%
OREAS-135	205	1	1	282.000	12.000	284.888	9.146	101.0%
Total	**517**				**Weighted average**			**102.4%**

RM	N	Outliers excluded	Failures excluded	Pb pct		Observed Pb pct		Percent of accepted
				Accepted	Std. Dev.	Accepted	Std. Dev.	
OREAS-H5	139	-	1	0.075	0.006	0.082	0.003	108.7%
OREAS-G5	173	-	-	0.076	0.006	0.080	0.003	104.8%
OREAS-135	183	1	1	1.700	0.062	1.726	0.047	101.6%
Total	**495**				**Weighted average**			**104.7%**

RM	N	Outliers excluded	Failures excluded	Ag ppm		Observed Ag ppm		Percent of accepted
				Accepted	Std. Dev.	Accepted	Std. Dev.	
OREAS-H5	137	-	-	3.810	0.510	4.328	0.213	113.6%
OREAS-G5	166	-	-	3.500	0.550	3.787	0.179	108.2%
OREAS-135	199	-	-	54.900	2.170	55.069	4.014	100.3%
Total	**502**				**Weighted average**			**106.5%**

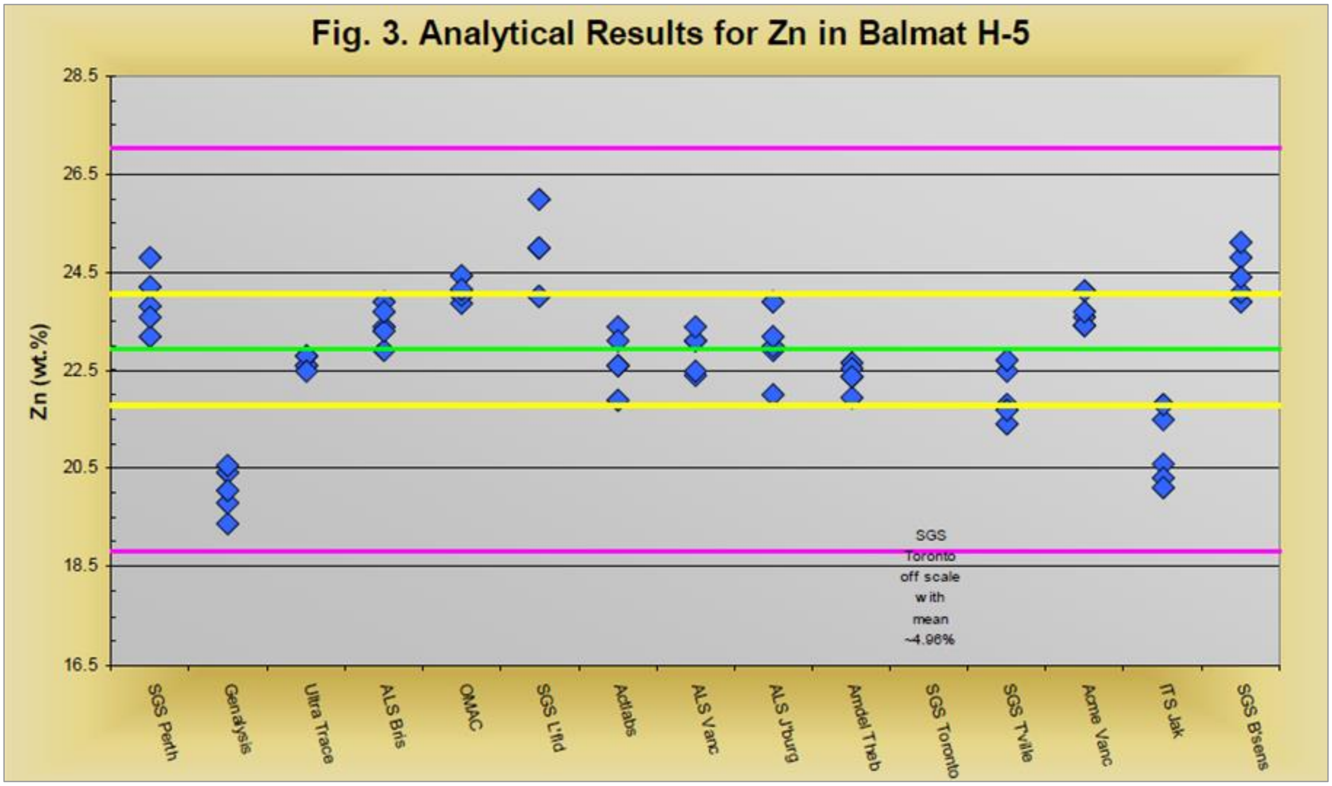
RM	N	Outliers excluded	Failures excluded	Fe pct		Observed Fe pct		Percent of accepted
				Accepted	Std. Dev.	Accepted	Std. Dev.	
OREAS-H5	140	-	-	1.590	0.100	1.587	0.052	99.8%
OREAS-G5	173	-	-	1.490	0.090	1.473	0.042	98.8%
OREAS-135	204	1	2	9.130	0.376	8.834	0.249	96.8%
Total	**517**				**Weighted average**			**98.3%**

An Outlier is defined as being outside five standard deviations from the accepted value. These are cases that are most likely sample mis-labels. Failures are defined as lying outside ± three standard deviations from the accepted values. There is a very low failure rate for reference materials in the database primarily quality control failures were followed up with requests for repeat assays. The fewer than 1% of the reference material insertions resulted in requests for repeat assays.

ALS performed well for all five metals for reference material ORE-135 prepared by OREAS. ORE-135 is a commercially available reference material created in 2017 and analyzed by 24 recognized laboratories.

The reference materials G-5 and H-5 tend to report 2 to 8% higher for zinc, copper, and lead than the accepted values. These materials were also prepared by OREAS. but were prepared in 2008 and analyzed at 15 laboratories. Figure 11.3 is a control chart for the reference material H-5 that is included in the certificate.

Figure 11.3 Analytical results for Zn in H-5



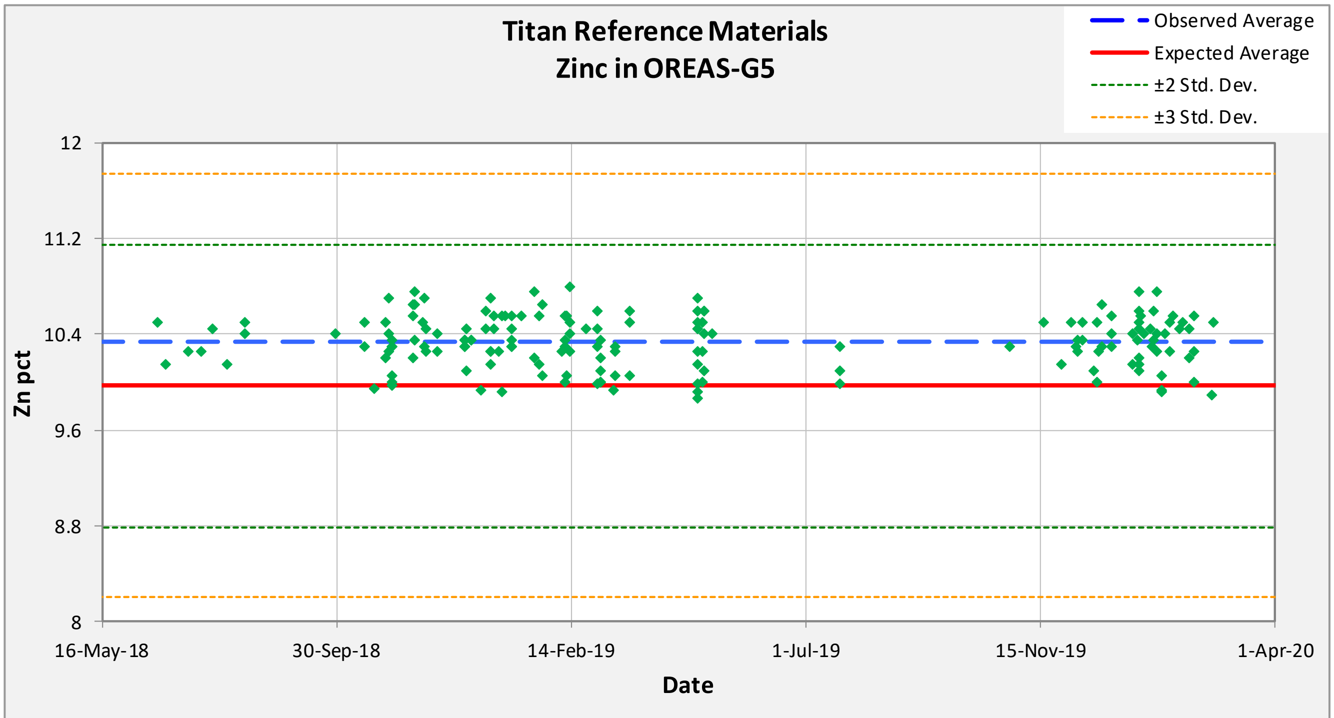
Source: Certificate of Analysis for Standard H-5 prepared by OREAS.

All of the laboratories were instructed to use an aqua regia digest for the analyses. The aqua regia digestion is not applied consistently at all laboratories. This often results in discrepancies for base metal assays between laboratories. This is evident from the collaborative study (round robin) for reference material H-5. For example, three SGS laboratories (Lakefield, Booysens, and Perth), ACME, and OMAC all report zinc values around 24%. The remaining laboratories tend to report lower with Genanalysis, for example, reporting an average value of 20.1% Zn.

The apparent high bias for zinc values in reference materials H-5 and G-5 may be attributable to differences in digestions between laboratories and difficulties in defining accepted values for these methods. The assumption that differences may be due to the relative strength of digestions is compelling because the other elements determined on the same digest (copper, lead, and silver) also tend to show a high bias.

Figure 11.4 shows that the zinc results reported for G-5, for example, have been consistent and reported within a narrow range.

Figure 11.4 Control chart for zinc in reference material G-5



Summary Statistics

Expected Values	
Mean	9.970
Standard Deviation	0.590
2 x RSD	11.84%

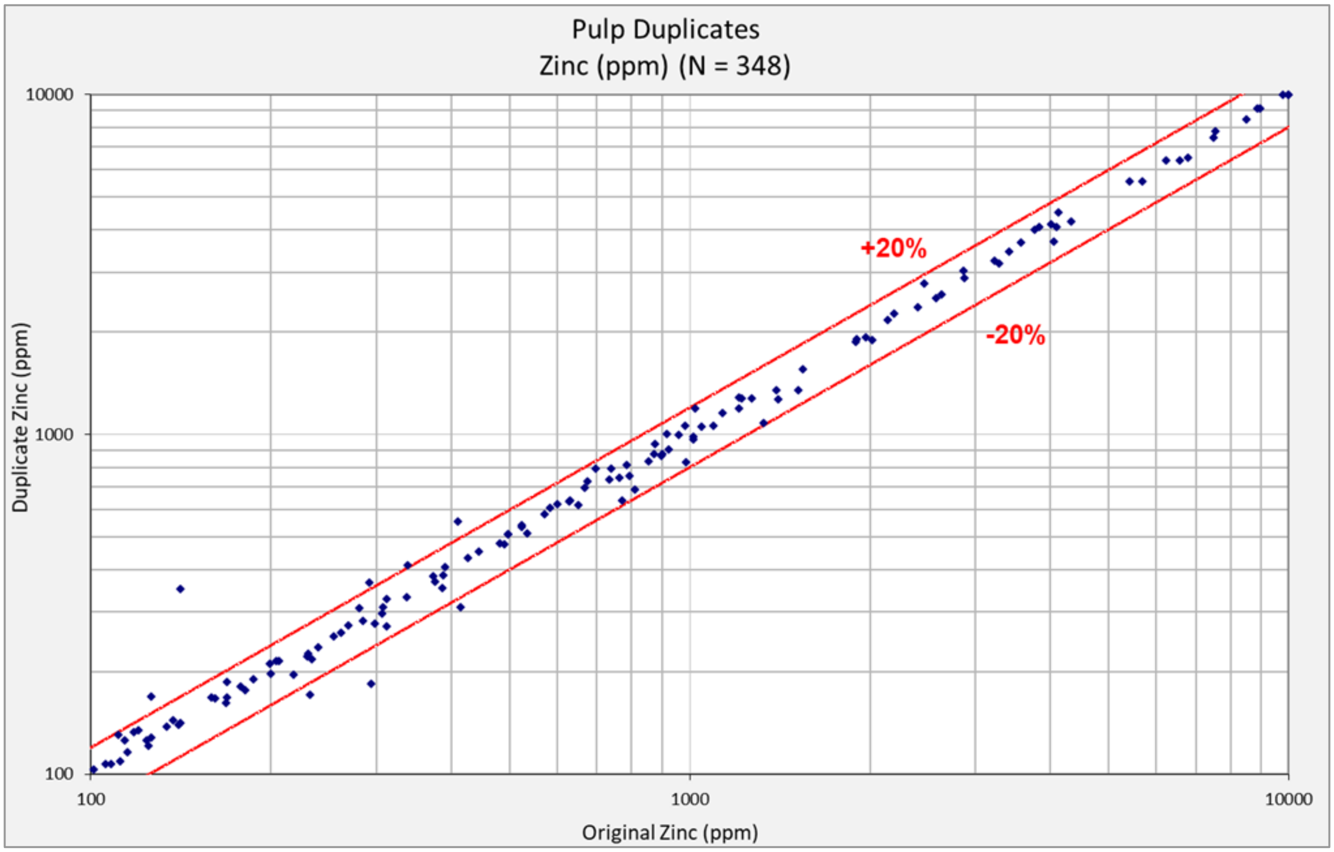
Observed Values	
Number of Samples	173
Mean	10.333
Standard Deviation	0.216
2 x RSD	4.19%
Falls Within 3 SD of Certified Mean	100%
Falls Within 2 SD of Certified Mean	100%
Falls Within 1 SD of Certified Mean	88%

Source: Graph generated by Analytical Solutions Ltd. in QC Mine Software.

Data for routine pulp duplicates and preparation duplicates were retrieved from ALS Geochemistry.

ALS Geochemistry includes pulp duplicates routinely to monitor its' internal quality control. There was a total of 348 pulp duplicates recovered. For zinc, copper, lead, and iron pulp duplicates, over ten times the detection limit, the precision is 5 to 7%. An example of the comparison of pulp duplicates is shown in Figure 11.5 for zinc. Over-range duplicates were not provided but are expected to have similar precision.

Figure 11.5 Comparison of pulp duplicates for zinc

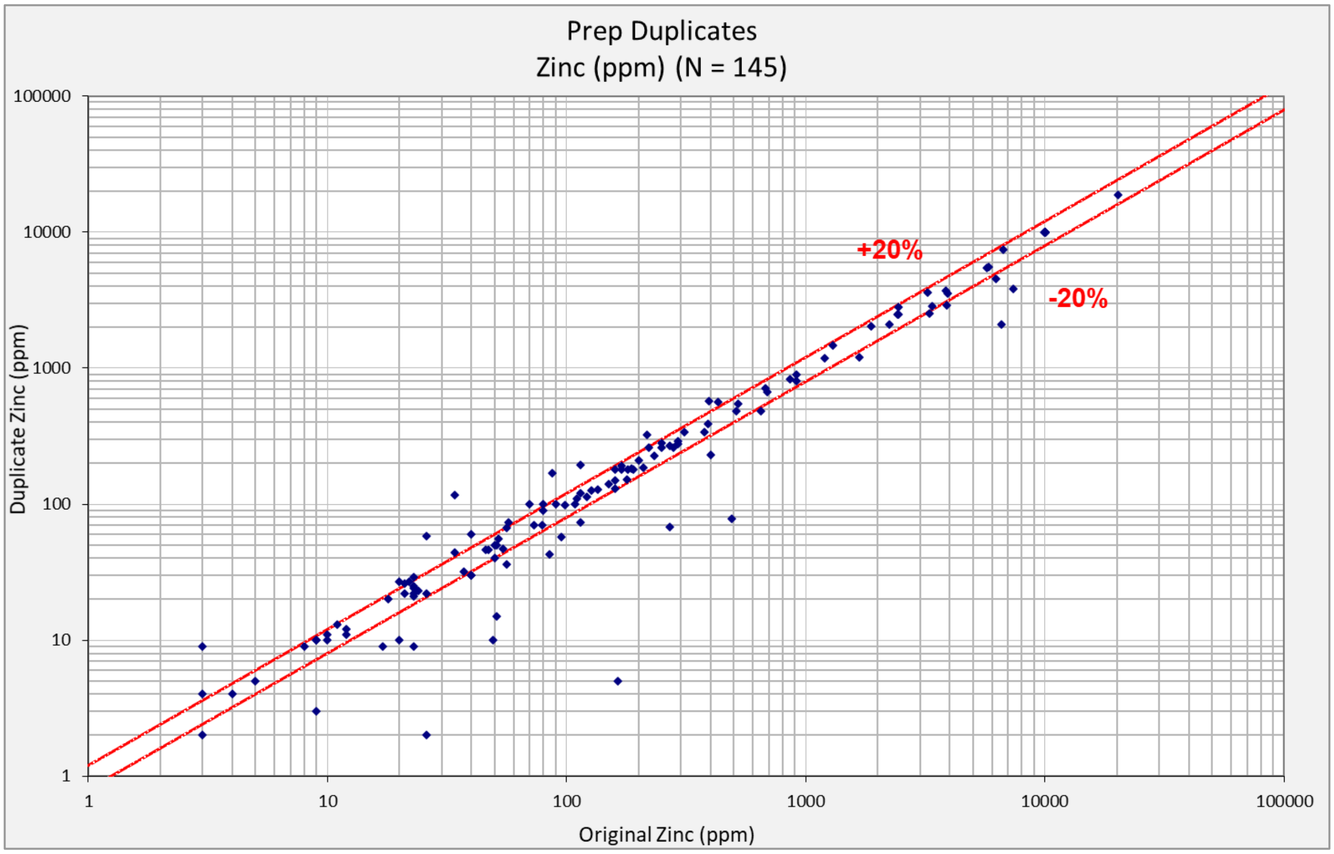


Source: Graph generated by Analytical Solutions Ltd.

The precision for silver results, greater than 5 ppm, for pulp duplicates is 10% based on 116 duplicate pairs. The precision for silver is not as good as for the base metals primarily as expected for the analytical method.

ALS Geochemistry includes preparation duplicates (a second 250 gram split of the crushed sample) routinely to monitor its' internal quality control. There was a total of 348 preparation duplicates recovered. For zinc, copper, lead, and iron pulp duplicates, over ten times the detection limit, duplicate assays generally repeat within ±20%. An example of the comparison of preparation duplicates is shown in Figure 11.6 for zinc. There were 18 cases of over-range duplicates for zinc (greater than 1%) that were not provided but are expected to have similar precision or better precision.

Figure 11.6 Comparison of preparation duplicates for Zinc



Source: Graph generated by Analytical Solutions Ltd.

The precision for copper, lead, silver, and iron preparation duplicates is similar to that for iron.

There were no check assays reported for the period.

It is the opinion of the author that the sample preparation, security, analytical procedures, and quality control practices meet or exceed industry standards and are, therefore, acceptable for the estimation of Mineral Resources.

12 Data verification

12.1 Verifications in previous Technical Reports

The QPs reviewed the drillhole data set provided which at the time consisted of 4,317 holes from which a subset of 633 were used for the previous Mineral Resource estimate. The assay data was reviewed 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 ESM digital database and some of the corresponding raw data files (source data) for the verification. Independent consultant geologists, Kim Tyler P.Geo. and Brett Armstrong, were retained by SLZ to work with site staff to clean the resource databases of errors and review the sampling data prior to delivery. The authors reviewed all relevant data and recommended corrections and additions prior to preparing the Mineral Resource estimate.

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 Reserve estimate.

Data categories reviewed 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. The only notable instances of this were in selected very old holes (1920's) where typographical errors were noted in the database in comparison to the logs. None of these were relevant to the model areas.
- **Downhole surveys:** Raw downhole survey reports were unavailable for some historical holes prior to the 1960's. These collars would have been surveyed for drill orientation and 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 five 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 to 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. SLZ consultant geologists compared assay values in the database to original drill logs and assay certificates to rectify obvious errors. 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. In 2018 ESM geologists

thoroughly audited the assay database for additional 'visual' grade estimates and purged records as necessary for recoding as "no sample".

12.2 Verifications by the authors of this Technical Report

ESM staff continually validate collar locations, downhole surveys, assay values, assay intervals, and geologic logging as new data is appended to the database. Drillhole information used in the resource models are checked against their original source, which is typically typed geologic paper logs for drilling conducted prior to 2017.

Staff also followed up on the observation of visual zinc grade estimates in the assay table by broadly sorting and searching the assay table for suspicious values. Values considered suspicious were integer values with no accompanying Fe, Pb, or Cu value. Once flagged, these values were then compared against the geologic log, and removed from the assay table if confirmed as a visual estimate. The impact to the database was minimal and outside the scope of the resources being considered for production in this report.

12.3 Limitations

Neither SRK nor ESM has completed a 100% validation of the entire database to original source data. Focus has been placed in the previous five years on those portions of the database relevant to the public disclosure.

12.4 Adequacy

The current and historical verification of these data sets has shown minor inconsistencies to source data, with uncertainty in the type or generation of data dealt with using classification of the Mineral Resource. SRK is of the opinion that the verification process is appropriate, and that the drilling database is adequate for the purposes of Mineral Resource estimation.

13 Mineral processing and metallurgical testing

Empire State Mines is a currently operating mine, processing underground mineralization to produce zinc concentrate. They have recently discovered two new zones of near-surface mineralization near the existing operation. Metallurgical test work was undertaken on the samples from the new zones to determine the process flowsheet for treating them to produce both lead / silver and zinc concentrates.

13.1 Metallurgical test work at Resource Development Inc. (RDi)

The primary objective of the test work undertaken at RDi in 2020 was to determine if the mineralization from the Turnpike and Hoist House prospects can be processed in the existing circuit with minor modifications to produce both lead and zinc concentrates.

Approximately 121 lbs (55 kgs) of each sample, some half core samples and existing mill feed samples were sent to RDi for metallurgical test work which consisted of Bond's Mill Work Index and abrasion index determination and flotation test work. Reagents, currently employed in the milling circuit at the mine, were also sent for the study.

13.1.1 Sample preparation and characterization

Turnpike and Hoist House half core samples received for comminution testing were crushed to minus 3/4 inch and submitted for Bond Abrasion Index testing. The comminution samples were then crushed to P_{100} passing 6 mesh for Bond Ball Mill Work Index (BWi) testing. A current mill feed sample was also received for comminution testing for comparison purposes.

The metallurgical composite samples were crushed to P_{100} passing 6 mesh, blended, and split into 2.2 lb (1 kg) charges for testing. A representative sample of each composite was pulverized and submitted for head analysis. A summary of the assay results is given in Table 13.1.

The composite samples contained significant levels of zinc and sulphide sulphur. The Turnpike composite assayed 4.04% Zn and 5.4% $S_{sulphide}$, while the Hoist House assayed 2.86% Zn and 5.2% $S_{sulphide}$. The Turnpike sample contains more lead and silver than the Hoist House sample (1.97% Pb and 20.2 g/tonne Ag compared to 0.36% Pb and 11.7 g/tonne Ag). Both samples contained trace amounts of gold.

Table 13.1 Head analyses of composite samples including ICP

Element	Turnpike	Hoist House
Au, g/tonne	0.022	0.010
Ag, g/tonne	20.2	11.7
Sulphide S %	5.37	5.22
Sulphate S %	3.74	2.38
Total S %	9.11	7.60
%		
Al	0.17	0.48
Ca	15.58	12.83
Fe	7.02	6.32
K	0.09	0.36
Mg	6.57	8.50
Na	0.07	0.28
Pb	1.97	0.36
Ti	0.01	0.04
Zn	4.04	2.86
ppm		
As	38	148
Ba	143	323
Bi	<10	<10
Cd	98	61
Co	1	5
Cr	97	85
Cu	46	127
Mn	1,180	1,811
Mo	2	6
Ni	6	7
Sr	167	352
V	3	20
W	226	152

13.1.2 Bond's Ball Mill Work Index / Bond Abrasion Index

Bond's BWi was determined for the Turnpike, Hoist House, and Rod Mill Feed samples at a closed size of 100 mesh (150 microns). In addition, samples were submitted for Bond Abrasion Index testing. The comminution results are summarized in Table 13.2. The results indicate that the samples would be considered medium hardness and low abrasion. The Turnpike and Hoist House mineralization are slightly harder than the currently processed underground mineralization.

Table 13.2 Bond's Ball Mill Work Index

Sample	BWi (kWh/st)	Ai
Turnpike	11.93	0.0346
Hoist House	12.11	0.0687
Rod Mill feed	10.03	0.0723

13.1.3 Rougher flotation testing

Initial rougher flotation tests were completed with 1-kilogram charges of each composite sample. Testing utilized a sequential flotation approach to produce separate lead and zinc concentrates. The primary grind was varied between P_{80} 65 mesh and P_{80} 100 mesh. Reagent types and dosages employed in these tests were the ones currently used in the plant. The samples were ground with sodium sulphide. The zinc was depressed with a combination of sodium cyanide and zinc sulphate while the lead was floated. Aerophine 3418A promoter was used to collect the lead and silver minerals. Additional tests were completed with Aerofloat 31 promoter to determine if lead / silver recovery could be increased. After the lead flotation, zinc was activated with copper sulphate and then collected with Aero 5100 promoter. All test products were submitted for assay of silver, lead, and zinc. The sequential flotation results are summarized in Table 13.3 and Table 13.4.

Table 13.3 Sequential rougher flotation results - Turnpike

Product	Recovery %				Product grade		
	Wt	Ag	Pb	Zn	Ag (g/tonne)	Pb (%)	Zn (%)
FT-1 (65 mesh, Standard Reagents)							
Pb Rougher Concentrate	13.7	72.7	91.8	10.1	106	13.35	3.05
Zn Rougher Concentrate	10.2	18.5	2.1	86.4	36.4	0.41	35.05
Rougher Tail	76.1	8.7	6.1	3.5	2.3	0.16	0.19
Calculated Feed	100	100	100	100	20.0	2.00	4.15
FT-2 (100 mesh, Standard Reagents)							
Pb Rougher Concentrate	14.0	72.2	91.6	9.9	106	11.57	2.84
Zn Rougher Concentrate	11.2	19.0	2.4	86.9	35.1	0.39	31.25
Rougher Tail	74.9	8.7	5.9	3.2	2.4	0.14	0.17
Calculated Feed	100	100	100	100	20.6	1.76	4.01
FT-5 (65 mesh, AP31 Collector)							
Pb Rougher Concentrate	10.9	69.1	88.5	6.6	126	14.04	2.54
Zn Rougher Concentrate	12.3	21.7	4.0	89.8	35.1	0.57	30.71
Rougher Tail	76.7	9.2	7.5	3.6	2.4	0.17	0.20
Calculated Feed	100	100	100	100	20.0	1.74	4.22

Table 13.4 Sequential rougher flotation results - Hoist House

Product	Recovery %				Product grade		
	Wt.	Ag	Pb	Zn	Ag (g/tonne)	Pb (%)	Zn (%)
FT-3 (65 mesh, Standard Reagents)							
Pb Rougher Concentrate	11.0	32.2	81.7	9.3	24.3	2.77	2.51
Zn Rougher Concentrate	8.5	38.7	5.2	87.2	37.7	0.23	30.49
Rougher Tail	80.5	29.2	13.0	3.5	3.0	0.06	0.13
Calculated Feed	100	100	100	100	8.3	0.37	2.97
FT-4 (100 mesh, Standard Reagents)							
Pb Rougher Concentrate	12.3	33.4	83.9	8.9	21.4	2.38	2.14
Zn Rougher Concentrate	8.6	39.5	4.8	88.2	36.3	0.20	30.38
Rougher Tail	79.1	27.0	11.3	2.9	2.7	0.05	0.11
Calculated Feed	100	100	100	100	7.9	0.35	2.96
FT-6 (65 mesh, AP31 Collector)							
Pb Rougher Concentrate	11.5	33.7	80.5	9.9	21.7	2.46	2.57
Zn Rougher Concentrate	8.7	43.5	5.8	86.8	33.9	0.23	29.65
Rougher Tail	79.9	22.7	13.7	3.2	2.1	0.06	0.12
Calculated Feed	100	100	100	100	7.4	0.35	2.97

The scoping level rougher flotation test results indicated the following:

- The sequential flotation approach floated over 80% of the lead and zinc into their respective concentrates. Approximately 73% of the silver and 92% of the lead reported to the rougher lead concentrate of the Turnpike sample. Maximum lead rougher concentrate grade was 13.35% Pb. The lower lead and silver grade Hoist House sample recovered approximately 33% of the silver and 83% of the lead in the lead rougher concentrate. The rougher concentrate grades were lower due to the lower head grade at approximately 22 g/tonne Ag and 2.7% Pb. Zinc recovery to the zinc concentrate was similar for both samples, averaging approximately 87% with grades of over 30% Zn.
- Grinding the samples finer to P_{80} 100 mesh did not significantly improve metal recovery or grade. The use of Aerofloat 31 did not provide better results than Aeropine 3418A.

13.1.4 Cleaner flotation testing

Initial cleaner flotation tests were completed with lead and zinc rougher concentrates produced from each composite sample. Testing utilized three stages of cleaners for the lead flotation and two stages of cleaners for the zinc flotation. The lead rougher concentrate was cleaned with and without regrind prior to flotation. The zinc rougher was not reground prior to cleaner flotation. The reagent types and dosages were kept similar to the rougher flotation process. All test products were submitted for assay of silver, lead, and zinc. The cleaner flotation results are summarized in Table 13.5 and Table 13.6.

Table 13.5 Cleaner flotation results - Turnpike

Product	Recovery %				Product grade		
	Wt.	Ag	Pb	Zn	Ag (g/tonne)	Pb (%)	Zn (%)
FT-7a (Lead Cleaner without Regrind)							
Pb Cleaner 3 Conc	14.3	66.5	92.2	8.5	438	56.1	2.08
Pb Cleaner 2 Conc	16.7	68.3	98.1	9.6	385	51.1	2.01
Pb Cleaner 1 Conc	19.2	72.6	98.1	9.7	356	44.4	1.76
Rougher Conc	100	100	100	100	94	8.71	3.50
FT-7b (Lead Cleaner with Regrind)							
Pb Cleaner 3 Conc	14.8	61.0	78.9	12.5	442	56.6	1.26
Pb Cleaner 2 Conc	18.4	67.2	87.0	17.1	392	50.3	1.39
Pb Cleaner 1 Conc	22.9	70.7	87.2	24.2	332	40.6	1.58
Rougher Conc	100	100	100	100	108	10.7	1.50
FT-7c (Zinc Cleaner without Regrind)							
Zn Cleaner 2 Conc	55.8	76.0	43.1	92.0	34.5	0.23	37.9
Zn Cleaner 1 Conc	65.7	83.2	57.3	96.9	32.0	0.26	33.9
Rougher Conc	100	100	100	100	25.3	0.30	23.0

Table 13.6 Cleaner flotation results - Hoist House

Product	Recovery %				Product grade		
	Wt.	Ag	Pb	Zn	Ag (g/tonne)	Pb (%)	Zn (%)
FT-8a (Lead Cleaner without Regrind)							
Pb Cleaner 3 Conc	8.1	39.2	64.6	3.4	126	19.2	1.62
Pb Cleaner 2 Conc	17.2	67.5	86.0	12.4	103	12.1	2.81
Pb Cleaner 1 Conc	26.5	73.1	86.7	12.9	72.2	7.93	1.90
Rougher Conc	100	100	100	100	26.2	2.42	3.89
FT-8b (Lead Cleaner with Regrind)							
Pb Cleaner 3 Conc	10.3	55.6	21.6	7.4	142	23.7	1.32
Pb Cleaner 2 Conc	17.7	65.5	24.9	15.7	97.4	15.9	1.63
Pb Cleaner 1 Conc	25.6	70.6	28.2	30.0	72.5	12.5	2.16
Rougher Conc	100	100	100	100	26.3	11.3	1.84
FT-8c (Zinc Cleaner without Regrind)							
Zn Cleaner 2 Conc	64.8	83.6	12.6	95.0	37.7	0.22	35.9
Zn Cleaner 1 Conc	69.8	87.0	13.7	96.5	36.4	0.22	33.8
Rougher Conc	100	100	100	100	29.2	1.13	24.5

The scoping level open-circuit cleaner flotation test results indicate the following:

- Lead cleaner flotation tests with the Turnpike rougher concentrate produced lead grades ranging from 40.6% Pb to 56.1% Pb with one to three stages of cleaning. Lead recovery ranged from 92.2% to 98.1% without regrind. In addition, silver recovery ranged from 66.5% to 72.6%. Two stages of lead cleaners are sufficient to produce a ±50% Pb concentrate.
- Lead cleaner flotation tests with the Hoist House rougher concentrate produced lead grades ranging from 7.9% Pb to 23.7% Pb with one to three stages of cleaning. Lead recovery ranged from 64.6% to 86.7% without regrind. In addition, silver recovery ranged from 39.2% to 73.1%.
- The zinc cleaner results were similar for both composite samples. Two stages of cleaners produced a zinc concentrate grade of 35.9% Zn at 95.0% recovery for the Hoist House composite, and 37.9% Zn at 92.0% recovery for the Turnpike composite.
- Regrind of the lead rougher concentrate did not significantly improve lead cleaner concentrate grades and was detrimental to lead recovery.

13.1.5 Conclusions

The following conclusions can be drawn based on the scoping level study undertaken by RDi:

- The recently discovered prospects can be processed using sequential flotation process to produce separate lead and zinc concentrate.
- The mineralization from Turnpike and Hoist House prospects are slightly harder than the current mineralization being processed in the plant.
- The lead recovery and concentrate grade are dependent on the feed grade of the mineralization. The higher the feed grade, the higher the final concentrate recovery and grade.
- Due to the low feed lead grade, one would require a large amount of mineralization to run a locked-cycle test. Since limited mineralization was available, the optimization can be done once new flotation cells for lead circuit are incorporated into the flowsheet.

13.2 Projected lead recovery and process flowsheet

The author has projected the lead recovery in the final concentrate based on extensive polymetallic processing experience and reasonable assumptions.

The following recovery and concentrate grade are projected based on scoping level test work:

- The lead rougher recovery will be ±92% at a concentrate grade of ±10% Pb as long as the feed grade is higher than 1% Pb.
- Two stages of cleaners are sufficient for production of lead concentrate assaying ±50% Pb. The lead concentrate will assay 350 g/tonne to 450 g/tonne Ag. However, if the feed grade is lower than 1% Pb, three to four stages of cleaners may be needed to produce marketable-grade lead concentrate.
- The cleaner flotation circuit will recover ±95% of lead recovered in the rougher flotation stage. Hence, the overall recovery of lead is projected to be 80% to 85%.
- The zinc recovery will be similar to that obtained with the underground mineralization.

Additional test work should be undertaken with the projected blend of underground mineralization and near-surface mineralization to determine the lead concentrate recovery and grade in the proposed process flowsheet.

13.3 Underground 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 used 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 supported 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. Current process plant recovery reflects the underground test work, with assumptions for the economic assessment of 96% Zn recovery with a zinc concentrate grade of 58%.

14 Mineral Resource estimates

This section of the report describes the preparation and creation of the geologic and grade block model for the ESM deposits. A representation of the geological interpretation is constructed by assigning geologic zones to small space-filling rectangular blocks within a larger rectangular volume (the block model). Grades are assigned to the blocks from the drillhole samples or composites, and the blocks within the block model are tabulated at various cut-off grades (COG). Due to the nature and geometry of the deposit, not all blocks have the same degree of certainty in their grade assignment, nor mining potential; therefore, a classification of certainty is assigned. Tabulated grade and tonnage results segregated by confidence levels are the final product of this effort.

ESM is comprised of multiple deposits in and around Fowler, NY. There are ten deposits currently considered as viable economic targets; American, Cal Marble, Davis, Fowler, Mahler, Mud Pond, N2D, Northeast Fowler, New Fold, and Sylvia Lake. Historic mining at these locations has provided a good geological understanding of each, with supporting mapping, sampling, and drilling data.

This Mineral Resource report has been created through a collaboration between ESM and SRK and has been prepared under the Canadian NI 43-101 guidelines. A comprehensive re-modelling effort was undertaken by ESM in 2018 using Leapfrog Geo for all geological models. Mining and grade control experience by ESM geologists have supported that the implicit modelling of the mineralized zones as veins in Leapfrog Geo results in more accurate geological wireframes.

14.1 Drillhole database

The drillhole database was provided to SRK through the current Vulcan projects for each zone. Assays and associated composites were extracted from drillholes that were used in estimation, of which there were 1,622 in total. The number of drillholes used for each zone is listed in Table 14.1.

Due to active drilling being implemented while each individual deposit model was created and updated, each of these databases are slightly different from the others. The ultimate data for each project was finalized on the effective date of the report.

This data has been continually checked for errors by ESM geologists and any errors that have been discovered were corrected in real time. Beyond checking for data loss in compositing, SRK did not independently validate the drilling database as part of the current scope of work but has relied on a review of ESM's verification work as summarized in Section 12 of this report. SRK notes that there are historic drillholes with uncertainty in survey or analytical methodology as well as other drillholes that are drilled at poor angles to the relevant geological zone which are not ideal for use in estimation. These drillholes were locally necessary to model the geology and, in certain cases, were used for estimation. The low confidence in these particular drillholes is addressed in the classification of the resource. ESM noted in their verification work that they did remove samples or entire holes from the database if there was insufficient confidence for their use in driving interpretation or grade assumptions.

The complete database for ESM consists of 8,678 surface or UG core holes. There are 68 sets of channel samples, 1,728 surface core holes, 6,872 UG core holes and 10 core holes identified as other (including monitoring wells). Smaller subsets of this database were used for geologic modelling and / or estimation on a lithological unit basis. Each lithological group was modelled separately in isolated geological and estimation projects.

Table 14.1 Core holes used in estimation of each zone

Zone	Number of core holes used
American	43
Cal Marble	26
Davis	45
Fowler	16
Mahler	245
Mud Pond	282
N2D	148
New Fold	57
Northeast Fowler	25
Sylvia Lake	89

Source: SRK 2020.

14.2 Geological model

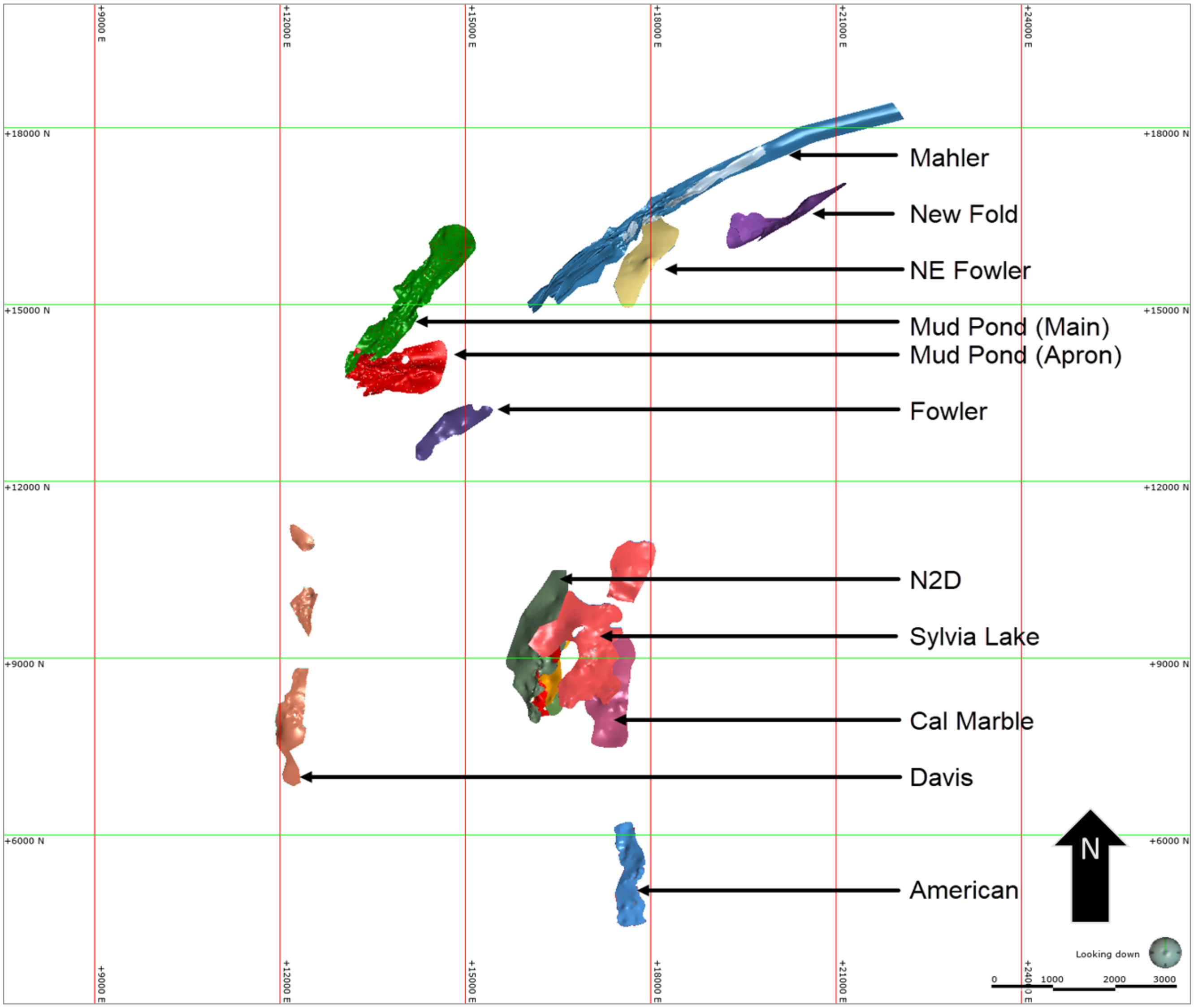
The ten deposit zones were defined and modelled by ESM geologists. Each one is comprised of multiple veins designating variably oriented and spatially-distinct mineralized zones which were modelled using combinations of explicit and implicit methods. Detailed descriptions of the geology of these areas are noted in previous sections of this report. Input data for these models are based on drilling intercepts and years of surface and underground mapping. Some wireframes for these zones were modelled using GEMS software from 2008 - 2017 and have subsequently been modified as new information has become available and modelling software has changed (Table 14.2). All new geological modelling in 2019 - 2020 was conducted in Leapfrog Geo. Each zone has been analyzed and divided where appropriate to facilitate a more accurate estimation of grade. SRK notes that, in some cases, this has resulted in splitting of domains based on morphology or orientation for the purposes of estimation. Mud Pond has been separated into a main zone and an upper Apron lens of mineralization as well, but for the purposes of this report will be discussed collective as Mud Pond. Location and volume of each is demonstrated below in Figure 14.1 and Table 14.2.

Table 14.2 Update periods for geological modelling

Zone	Years modelled and updated
American	2019
Cal Marble	2009, 2017, 2019
Davis	2017, 2019
Fowler	2019
Mahler	2009, 2017, 2019
Mud Pond	2008, 2009, 2017, 2019
N2D	2019
New Fold	2009, 2017, 2020
Northeast Fowler	2017, 2019
Sylvia Lake	2017, 2019

Source: ESM 2020.

Figure 14.1 Locations of each zone



Source: SRK 2020.

Table 14.3 Core holes used in estimation of each zone

Zone	Volume (ft^3)
American	4,585,996
Cal Marble	6,420,100
Davis	4,982,555
Fowler	3,372,746
Mahler	22,479,565
Mud Pond	17,014,250
N2D	37,170,697
New Fold	10,615,213
Northeast Fowler	6,852,584
Sylvia Lake	10,997,333

Source: SRK 2020.

14.3 Assay capping and compositing

14.3.1 Outliers

Neither assays nor composites were capped. However, higher-grade outlier samples were limited when necessary within grade estimation. The high-yield limit restrictions are listed below in Table 14.4. The High Yield Limit is the zinc (Zn) percent value limit and the High Yield Distance is the distance from the estimated block allowed for full unrestricted values. Beyond the distance specified, composite grades are still used but at the truncated high-yield value.

Table 14.4 High yield limits and distances for grade truncation, bypass, and zone

	Area	Pass	High yield limit (%)	High yield distance (ft)
American	ALL	1	19	50
Cal Marble	ALL	1	-	-
Cal Marble	ALL	2	-	-
Davis	DAU	1	20	50
Davis	DAM	1	20	50
Davis	DAL	1	10	50
Fowler	ALL	1	-	-
Fowler	ALL	2	-	-
Mahler	MAM1	1	40	25
Mahler	MAM2	1	-	-
Mahler	MAM2	2	40	25
Mahler	MAM2	3	40	25
Mahler	MAM3	1	-	-
Mahler	MWD1	1	-	-
Mahler	MWD2	1	-	-
Mahler	MWD2	2	-	-
Mahler	MWD3	1	45	50
Mahler	MWD4	1	-	-
Mud Pond	10	1	-	-
Mud Pond	10	2	-	-
Mud Pond	11	1	-	-
Mud Pond	20	1	-	-
Mud Pond	20	2	-	-
Mud Pond	22	1	-	-
Mud Pond	23	1	-	-
Mud Pond	24	1	-	-
Mud Pond	25	1	-	-
Mud Pond	26	1	-	-
Mud Pond	29	1	-	-
Mud Pond	99	1	-	-
N2D	V1	1	-	-
N2D	V2	1	-	-
N2D	V3	1	-	-
N2D	V4	1	-	-
N2D	V6	1	-	-
N2D	V8	1	-	-
N2D	V10	1	-	-

	Area	Pass	High yield limit (%)	High yield distance (ft)
N2D	V99	1	-	-
N2D	V99	2	-	-
N2D	V99	3	-	-
N2D	V99	4	-	-
N2D	V99	5	-	-
N2D	V99	6	-	-
N2D	V99	7	-	-
New Fold	NF1	1	25	25
New Fold	NF2	1	-	-
New Fold	NF3	1	25	25
New Fold	NF5	1	-	-
New Fold	NF6	1	-	-
New Fold	NF7	1	-	-
North East Fowler	ALL	1	30	50
Sylvia Lake	SLM1S	1	12	150
Sylvia Lake	SLM2S	2	12	300
Sylvia Lake	SLM1N	1	-	-
Sylvia Lake	SLM2N	2	-	-
Sylvia Lake	SLN	1	-	-
Sylvia Lake	SLN	2	-	-

Source: SRK 2020.

14.3.2 Compositing

Composites were created using a variety of methods that were appropriate to the individual block models used for estimation. Generally, these were variable run-length composites of approximately 5 ft or 10 ft, honouring the modelled geological boundaries. Interval lengths were adjusted to incorporate small composites and standardize composites across vein widths. Each compositing method is listed below in Table 14.5.

Table 14.5 Compositing method by zone

Zone	Composite
American	Vein length honoring vein boundaries
Cal Marble	5’ variable length honoring vein boundaries
Davis	5’ variable length honoring vein boundaries
Fowler	5’ variable length honoring vein boundaries
Mahler	10’ variable length honoring vein boundaries
Mud Pond	5’ variable length honoring vein boundaries
N2D	5’ variable length honoring vein boundaries
New Fold	Vein length honoring vein boundaries
North East Fowler	Straight compositing
Sylvia Lake	10’ variable length honoring vein boundaries

Source: SRK 2020.

14.4 Density

Density was assigned to the models based on the values listed below in Table 14.6. The values were calculated based on a mean of measured density from 140 samples. Density was calculated by ESM geologists using the conventional immersion technique of weighing samples in air and in water and examining the displacement of the water in a controlled environment. All specific gravity (SG) measurements were converted to bulk density using an assumption of equal relationship of SG to grammes per cubic centimetre (g/cm^3), and a unit conversion to a tonnage factor (tf) represented in short tons/ft^3.

Table 14.6 Density by zone and material type

Zone	SG	tf
American	3.123	0.0975
Davis	3.123	0.0975
Cal Marble	3.123	0.0975
Sylvia Lake	3.123	0.0975
Mud Pond Main	3.159	0.0986
Mud Pond - Apron	3.144	0.0981
Mud Pond - V99	3.123	0.0975
Mahler - Main	3.073	0.0959
Mahler - White Dolomite	3.065	0.0956
NE Fowler	3.137	0.0979
New Fold	3.123	0.0975
#2D – anhydrite / qddm waste		0.091
#2D - tremolitic waste		0.093
#2D		0.103
HH, TP, PH		0.1
HH, TP, PH waste		0.09
Fowler	3.123	0.0975
Waste	2.8	0.0874

Source: Titan 2020.

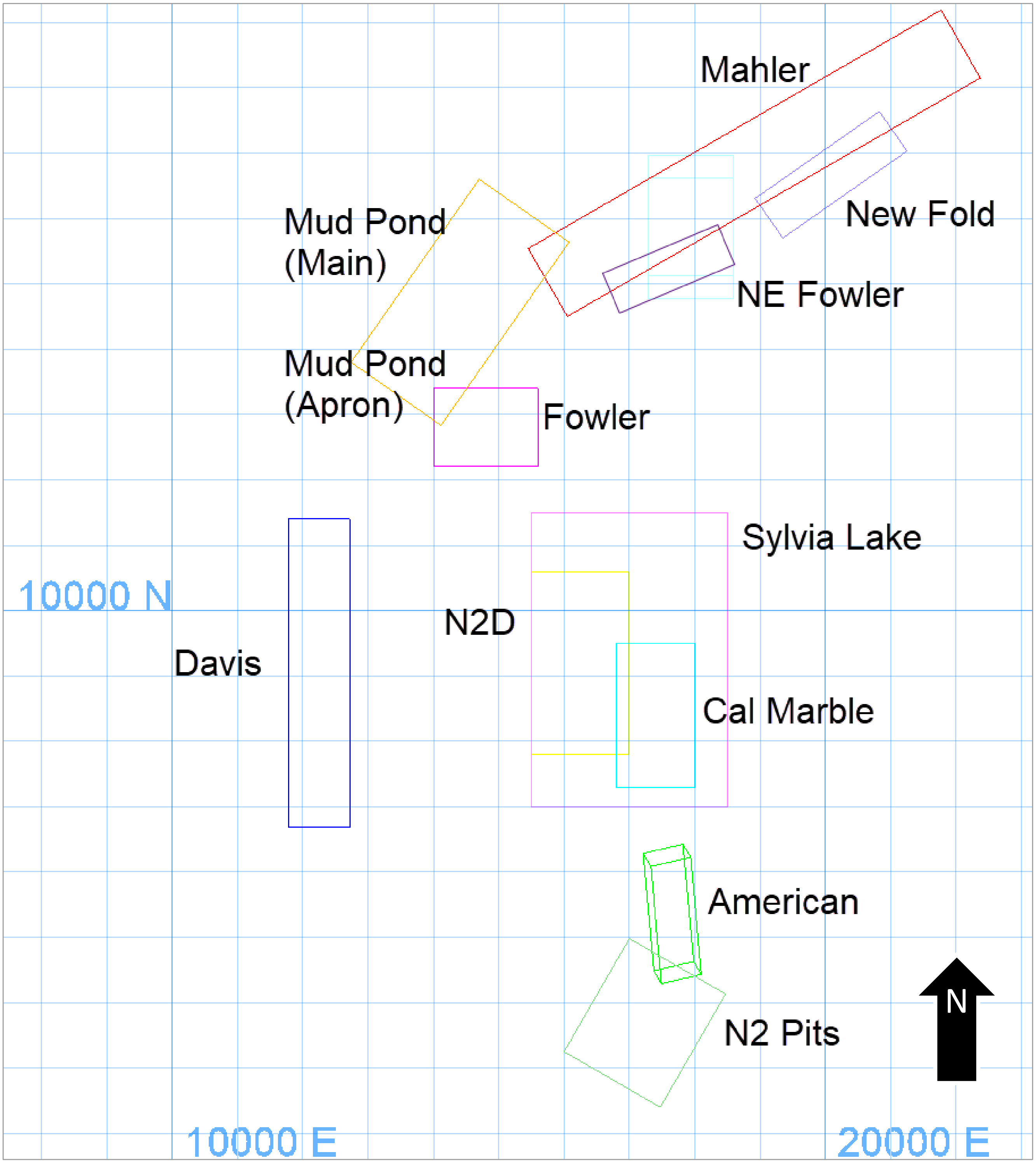
14.5 Variogram analysis and modelling

The highly variable nature of the grade and the geometry of these deposits created poor variograms. The geometry of the modelled vein domains provides a reasonable amount of control to the estimates and any grade anisotropy in the veins is considered during estimation.

14.6 Block model

Separate block models were created for each zone. The parameters for each are listed below in Table 14.7. They consist of origins, rotations (in Maptek rotation convention), parent block parameters and associated sub-block parameters. A plan view of block model extents is shown in Figure 14.2 by zone.

Figure 14.2 Plan view of block model extents



Source: SRK 2020.

Table 14.7 Block model size and location by zone

American		
Bearing = 77.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 33.5 Plunge = 12.0 X origin = 17490.0 Y origin = 4290.0 Z origin = -335.0 #blocksx = 32 #blocksy = 107 #blocksz = 20	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 10.0 Block Size min y = 10.0 Block Size min z = 2.5
Cal Marble		
Bearing = 90.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 0.0 Plunge = 0.0 X origin = 16800.0 Y origin = 7300.0 Z origin = -1900.0 #blocksx = 60 #blocksy = 110 #blocksz = 55	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 2.5 Block Size min y = 2.5 Block Size min z = 2.5
Davis		
Bearing = 90.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 0.0 Plunge = 0.0 X origin = 11780.0 Y origin = 6680.0 Z origin = -1400.0 #blocksx = 47 #blocksy = 236 #blocksz = 90	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 10.0 Block Size min y = 10.0 Block Size min z = 2.5
Fowler		
Bearing = 90.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 0.0 Plunge = 0.0 X origin = 14000.0 Y origin = 12200.0 Z origin = -2700.0 #blocksx = 80 #blocksy = 60 #blocksz = 25	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 2.5 Block Size min y = 2.5 Block Size min z = 2.5

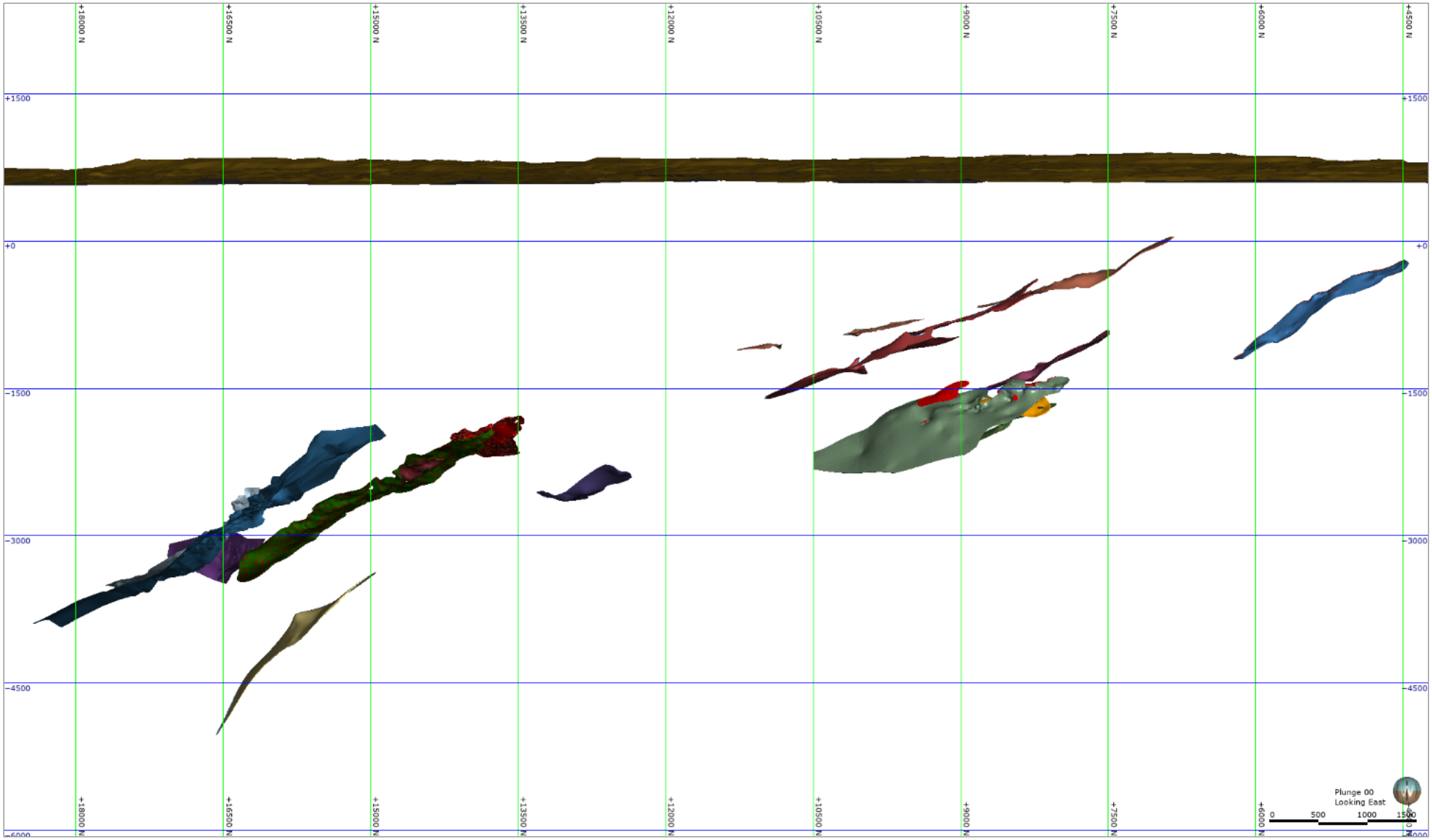
Mahler		
Bearing = 60.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = -0.0 Plunge = -0.0 X origin = 16050.0 Y origin = 14500.0 Z origin = -4100.0 #blocksx = 73 #blocksy = 12 #blocksz = 24	Block Size max x = 100.0 Block Size max y = 100.0 Block Size max z = 100.0 Block Size min x = 100.0 Block Size min y = 100.0 Block Size min z = 100.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 10.0 Block Size min y = 2.5 Block Size min z = 2.5
Mud Pond		
Bearing = 35.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = -0.0 Plunge = -0.0 X origin = 14115.0 Y origin = 12835.0 Z origin = -3555.0 #blocksx = 57 #blocksy = 28 #blocksz = 31	Block Size max x = 60.0 Block Size max y = 60.0 Block Size max z = 60.0 Block Size min x = 60.0 Block Size min y = 60.0 Block Size min z = 60.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 10.0 Block Size min y = 2.5 Block Size min z = 2.5
N2D		
Bearing = 90.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 0.0 Plunge = 0.0 X origin = 15500.0 Y origin = 7800.0 Z origin = -2500.0 #blocksx = 75 #blocksy = 140 #blocksz = 50	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 2.5 Block Size min y = 2.5 Block Size min z = 2.5
New Fold		
Bearing = 55.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 0.0 Plunge = 0.0 X origin = 19350.0 Y origin = 15700.0 Z origin = -3530.0 #blocksx = 116 #blocksy = 37 #blocksz = 36	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 2.5 Block Size min z = 2.5

North East Fowler		
Bearing = 90.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 45.0 Plunge = 0.0 X origin = 17285.0 Y origin = 14775.0 Z origin = -3355.0 #blocksx = 65 #blocksy = 130 #blocksz = 25	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 2.5 Block Size min y = 2.5 Block Size min z = 2.5
Sylvia Lake		
Bearing = 90.0	**Parent block dimensions**	**Sub-block block dimensions**
Dip = 0.0 Plunge = 0.0 X origin = 15500.0 Y origin = 7000.0 Z origin = -2000.0 #blocksx = 150 #blocksy = 225 #blocksz = 90	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 20.0 Block Size min y = 20.0 Block Size min z = 20.0	Block Size max x = 20.0 Block Size max y = 20.0 Block Size max z = 20.0 Block Size min x = 2.5 Block Size min y = 2.5 Block Size min z = 2.5

Source: SRK 2020.

All models were considered below topography as seen in Figure 14.3 below.

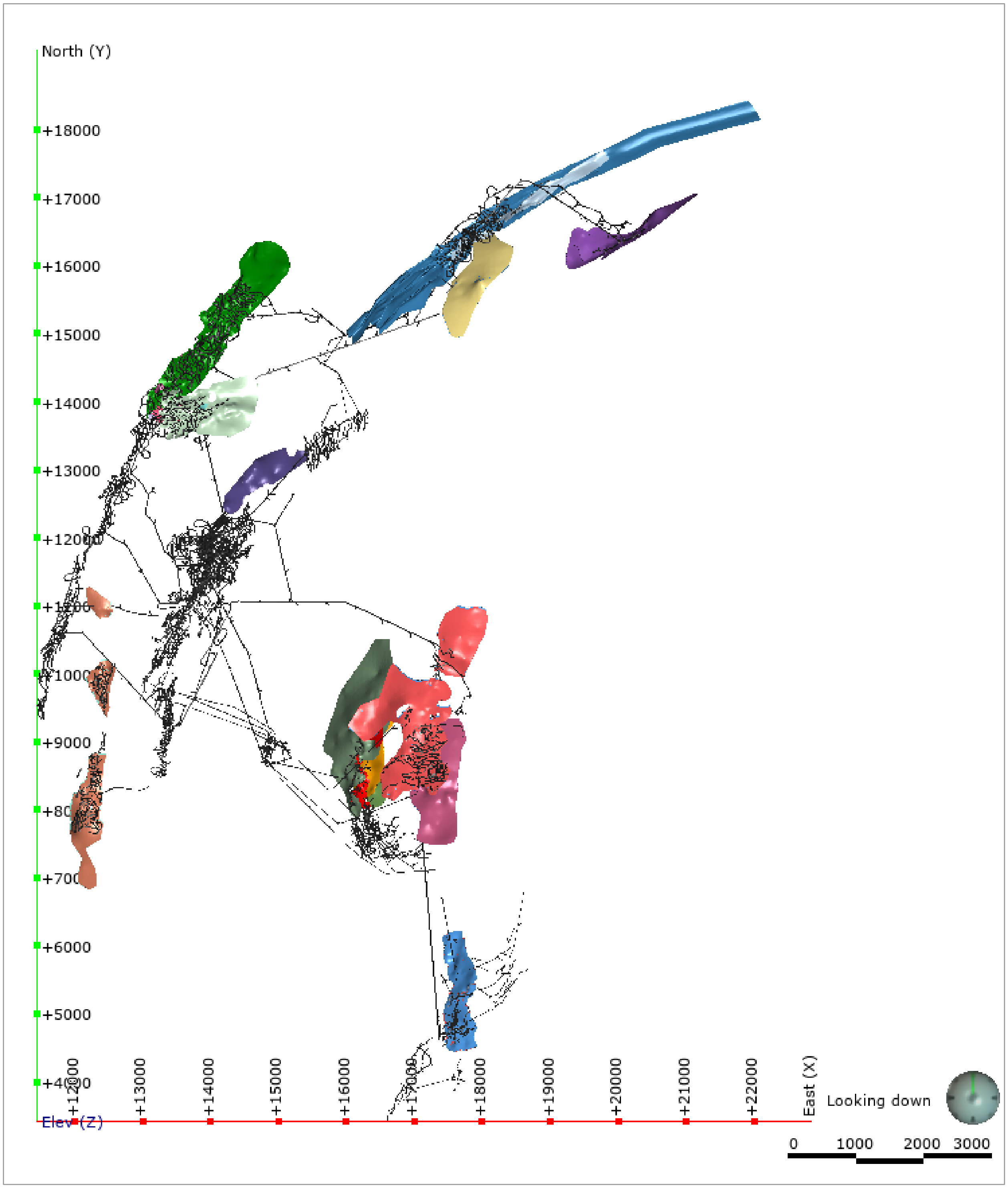
Figure 14.3 All zones shown below topography



Note: Looking East.
Source: SRK 2020.

Historical mine workings, or as-built solids, were used for sub-blocking during model creation and mined blocks contained in these wireframes were removed from the estimated material. A comprehensive as-built wireframe was updated as of 1 October 2020 and utilized to deplete tonnage within the block models. This wireframe is shown in black in Figure 14.4 below.

Figure 14.4 As-built mining wireframes



Source: SRK 2020.

Reporting from the various block models is done via the "Advanced Reserves" feature in Vulcan, using the relevant variables as summarized in Table 14.8. These variables generally represent a depleted tonnage factor (ton/ft^3) and Zn% grade but vary based on the depletion or estimation methodology used for each model.

Table 14.8 Reporting variables for Number 4 Underground complex block models

Block model	Density variable	Grade variable
am_20191023_srk_class.bmf	tf	zn_lva
CM_20190815_srk_class.bmf	tf	zn_lva
da_20191010_srk_class.bmf	tf_dep	zn_lva
FO_20190810_srk_class.bmf	tf	zn_lva
ma_20200810_srk_class.bmf	tf_dep	zn_lva
MP_20200810_srk_class.bmf	tf_dep	zn_lva
n2d_20200810_srk_class.bmf	tf_dep	zn_lva
nef_20191010_srk_class.bmf	tf	zn_lva
nf_20200810_srk_class.bmf	tf_dep	zn
SL_20190810_srk_class.bmf	tf_dep	zn_lva

Source: ESM 2020.

14.7 Estimation methodology

Due to the high variability of the ESM deposits and the lack of robust variography, inverse distance squared estimates were used to estimate grade into parent blocks within the block model. The control of each estimate was based on sample selection criteria such as, minimum and maximum number of composites, minimum number of drillholes and search distances. For each pass, the search distances were either isotropic (spherical) or anisotropic (ellipsoidal) depending on the geometric control and limits in each vein. For isotropic searches, the geometry of the vein was considered adequate to control sample selection. For anisotropic searches, the direction was defined using a variable orientation algorithm in Vulcan called Locally Varying Anisotropy (LVA). This oriented the search ellipse for each block down a plane which paralleled the modelled geologic continuity (i.e., the hangingwall or footwall of the ESM veins). LVA parameters were defined as the mid-point between the vein bounding surfaces, or manually set based on a triangulated surface.

Multiple passes were used, as necessary, to fill the wireframes with estimated grade. The variable constraints for each pass were considered in classification.

Estimation criteria, bypass, is listed below in Table 14.9 for each domain.

Table 14.9 Estimation sample selection parameters

	Area	Pass	Parent block size	Search radius	Rotation (Vulcan ZYX)	Min samples	Max samples	Max samples per DH	Minimum DH
American	ALL	1	50 x 50 x 50	400 x 400 x 400	0/0/0	2	3	2	2
Cal Marble	ALL	1	10 x 10 x 10	300 x 300 x 60	Variable	5	15	2	3
Cal Marble	ALL	2	10 x 10 x 10	600 x 600 x 60	Variable	2	15	2	1
Davis	DAU	1	50 x 50 x 50	650 x 650 x 650	0/0/0	2	30	2	2
Davis	DAM	1	50 x 50 x 50	550 x 550 x 550	0/0/0	2	30	2	2
Davis	DAL	1	50 x 50 x 50	350 x 350 x 350	0/0/0	2	30	2	2
Fowler	ALL	1	10 x 10 x 10	300 x 300 x 50	Variable	5	15	2	3
Fowler	ALL	2	10 x 10 x 10	600 x 600 x 100	Variable	2	15	2	1
Mahler	MAM1	1	20 x 20 x 20	350 x 350 x 350	Variable	2	30	4	2
Mahler	MAM2	1	20 x 20 x 20	60 x 60 x 30	Variable	2	30	4	2

	Area	Pass	Parent block size	Search radius	Rotation (Vulcan ZYX)	Min samples	Max samples	Max samples per DH	Minimum DH
Mahler	MAM2	2	20 x 20 x 20	120 x 120 x 60	Variable	2	30	4	2
Mahler	MAM2	3	20 x 20 x 20	300 x200x 100	Variable	2	30	4	2
Mahler	MAM3	1	50 x 50 x 50	1300 x 1300 x 1300	Variable	2	30	4	2
Mahler	MWD1	1	20 x 20 x 20	100 x 100 x 100	39/-39/-39	2	30	2	2
Mahler	MWD2	1	20 x 20 x 20	100 x 50 x 25	Variable	2	30	2	2
Mahler	MWD2	2	20 x 20 x 20	200 x 200 x 200	Variable	2	30	2	2
Mahler	MWD3	1	20 x 20 x 20	300 x 300 x 300	53/-45/-30	2	30	2	2
Mahler	MWD4	1	50 x 50 x 50	500 x 500 x 500	Variable	2	30	2	2
Mud Pond	10	1	20 x 20 x 20	60 x 60 x 20	Variable	2	15	-	-
Mud Pond	10	2	50 x 50 x 50	600 x 600 x 300	Variable	2	15	2	-
Mud Pond	11	1	50 x 50 x 50	600 x 600 x 600	Variable	2	15	-	-
Mud Pond	20	1	50 x 50 x 50	500 x 500 x 500	0/0/0	2	15	2	-
Mud Pond	20	2	50 x 50 x 50	500 x 500 x 500	0/0/0	2	15	2	-
Mud Pond	22	1	50 x 50 x 50	300 x 300 x 300	Variable	2	15	-	-
Mud Pond	23	1	20 x 20 x 20	300 x 300 x 300	0/0/0	2	15	-	-
Mud Pond	24	1	50 x 50 x 50	150 x 150 x 150	Variable	2	15	-	-
Mud Pond	25	1	20 x 20 x 20	200 x 200 x 50	Variable	2	15	-	-
Mud Pond	26	1	50 x 50 x 50	200 x 200 x 50	Variable	2	15	-	-
Mud Pond	29	1	20 x 20 x 20	150 x 150 x 50	Variable	2	15	-	-
Mud Pond	99	1	20 x 20 x 20	50 x 50 x 50	Variable	2	15	2	-
N2D	V1	1	20 x 20 x 20	500 x 500 x 500	Variable	4	15	4	2
N2D	V2	1	20 x 20 x 20	600 x 600 x 600	Variable	2	15	2	-
N2D	V3	1	20 x 20 x 20	300 x 300 x 100	Variable	2	15	2	-
N2D	V4	1	10 x 10 x 10	300 x 300 x 100	Variable	2	15	2	-
N2D	V6	1	20 x 20 x 20	300 x 300 x 100	Variable	2	15	2	-
N2D	V8	1	10 x 10 x 10	300 x 300 x 100	Variable	2	15	2	-
N2D	V10	1	10 x 10 x 10	300 x 300 x 100	Variable	2	15	2	-
N2D	V99	1	10 x 10 x 10	75 x 75 x 15	Variable	2	15	3	2
N2D	V99	2	10 x 10 x 10	150 x 150 x 15	Variable	2	15	2	2
N2D	V99	3	10 x 10 x 10	300 x 300 x 15	Variable	2	15	2	2
N2D	V99	4	10 x 10 x 10	600 x 600 x 200	Variable	2	15	2	2
N2D	V99	5	10 x 10 x 10	75 x 75 x 15	Variable	2	15	2	2
N2D	V99	6	10 x 10 x 10	300 x 300 x 15	Variable	2	15	2	2
N2D	V99	7	10 x 10 x 10	600 x 600 x 200	Variable	2	15	2	2
New Fold	NF1	1	20 x 20 x 20	450 x 450 x 450	0/0/0	2	4	-	-
New Fold	NF2	1	20 x 20 x 20	200 x 200 x 200	0/0/0	2	4	-	-
New Fold	NF3	1	20 x 20 x 20	250 x 250 x 250	0/0/0	2	4	-	-
New Fold	NF5	1	20 x 20 x 20	250 x 250 x 250	0/0/0	2	4	-	-
New Fold	NF6	1	20 x 20 x 20	150 x 150 x 150	0/0/0	2	4	-	-
New Fold	NF7	1	20 x 20 x 20	150 x 150 x 150	0/0/0	2	4	-	-
NE Fowler	ALL	1	50 x 50 x 50	425 x 425 x 425	Variable	2	30	2	-
Sylvia Lake	SLM1S	1	20 x 20 x 20	300 x 300 x 60	Variable	5	15	2	3
Sylvia Lake	SLM2S	2	20 x 20 x 20	600 x 600 x 120	Variable	2	15	2	2
Sylvia Lake	SLM1N	1	20 x 20 x 20	300 x 300 x 60	Variable	5	15	2	3
Sylvia Lake	SLM2N	2	20 x 20 x 20	600 x 600 x 120	Variable	2	15	2	2
Sylvia Lake	SLN	1	20 x 20 x 20	300 x 300 x 60	Variable	5	15	2	3
Sylvia Lake	SLN	2	20 x 20 x 20	600 x 600 x 120	Variable	2	15	2	2

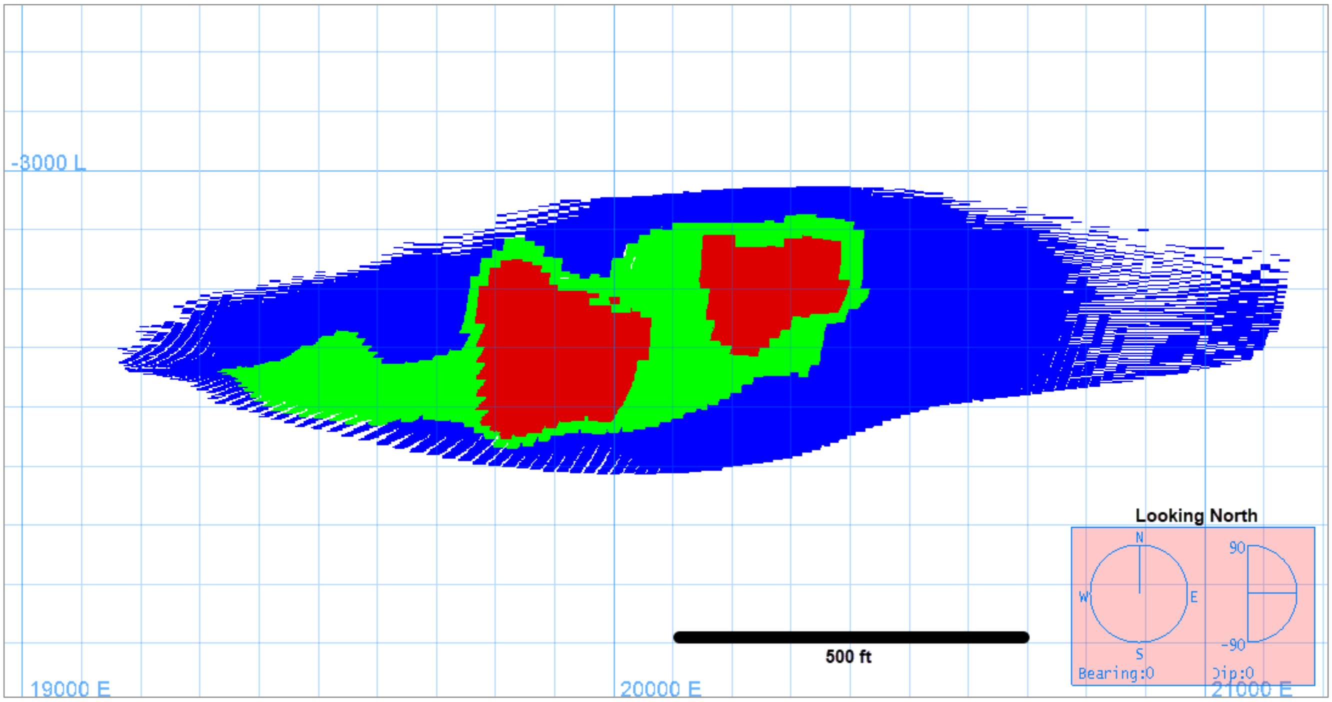
Source: SRK 2020.

14.8 Resource classification

The ESM Number 4 Underground zinc deposits have been classified according to the CIM Definition Standard for Mineral Resources and Mineral Reserves. The resource classification considered the quality, quantity and distance to the data informing blocks in the model, as well as the geological continuity of the mineralized zones. Populated estimation items used to assist the QP in defining classification included, but were not limited to, distance to the closest composite, average distance to the closest composite, number of drillholes informing the estimate and number of samples informing the estimate.

These model items were used as the basis of a calculation within the blocks. The scripted values were used as a guide to the QP in assigning zones of confidence. The results of the calculation were then smoothed and encased in wireframes that facilitated the final model coding for classification. This allowed the QP to remove zones of lower confidence based on additional factors that are not covered in estimation. The parameters of these scripts varied by zone due to changing drilling characteristics, vein geometry and site geologist input. An example vein is shown below in Figure 14.5.

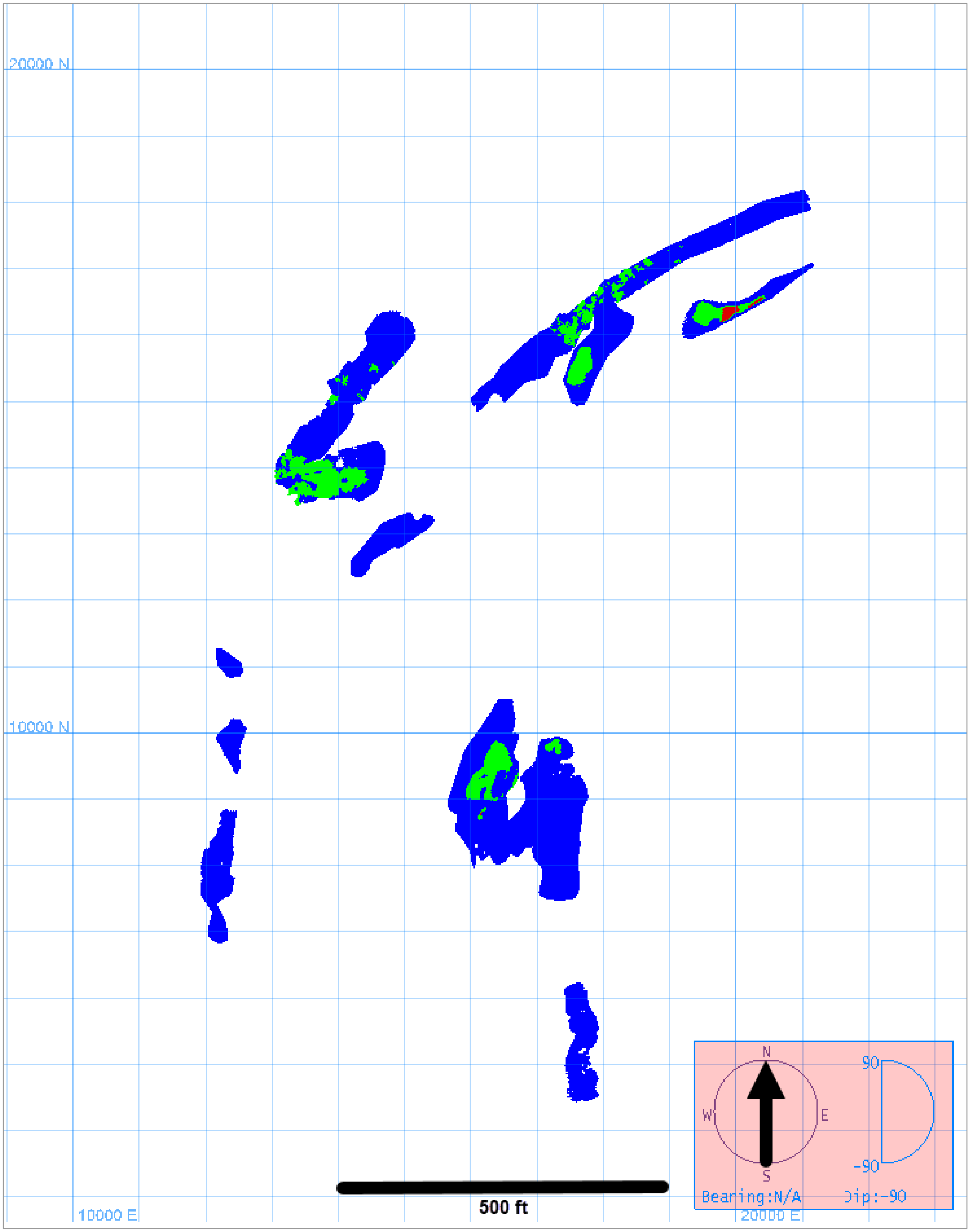
Figure 14.5 Classification for New Fold



Note: Red=Measured, green=Indicated, blue=Inferred.
Source: SRK 2020.

In addition to estimation metadata, the QP spent significant time discussing areas of confidence with regard to geological continuity, mapping, and drilling data with the ESM site geologist prior to assigning classification zones. Classification for all veins is demonstrated below in Figure 14.6.

Figure 14.6 Classification for all ESM Zones



Note: Red=Measured, green=Indicated, blue=Inferred.
Source: SRK 2020.

The zones that were classified as Measured exhibit excellent geological continuity that has been verified at dense sample spacing using reliable testing methods. These blocks were informed by a minimum of two drillholes and satisfied the QP with regard to data quality and quantity. They contained no detrimental factors, such as unreliable spatial data, low data quality, poor validation, or unreliable geological continuity.

The zones that were classified as Indicated exhibit good geological continuity but have sample spacing that is less dense. These areas are considered somewhat less well-understood but still have high quality data informing them including grade data, density, and physical properties. The location of samples and the assay data are sufficiently reliable so support resource estimation and this material can be considered appropriate for mine planning purposes.

Zones that were classified as Inferred are beyond the zone considered to have a reasonable geological continuity, low density sample spacing, or there is concern that the quality of data does not support reliable grade estimation. Geological evidence is sufficient to imply that the material is there, but not sufficient to support an Indicated classification.

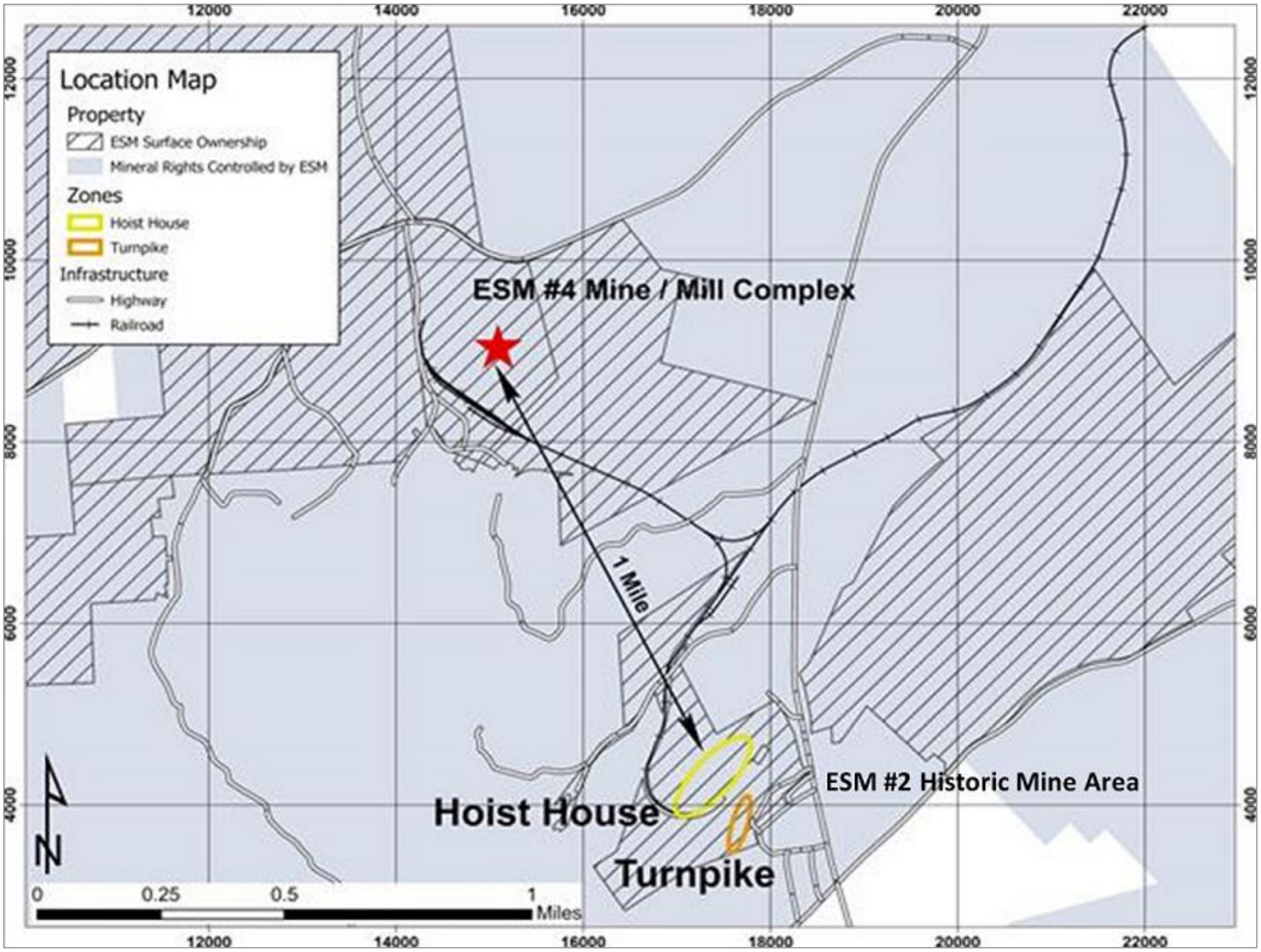
14.9 N2 pit area modelling and estimation

This section describes the estimation of the N2 Pit area, which features significantly different procedures and methodology compared to that described in the previous sections, primarily due to the open-pit possibilities that exist for this area compared to the underground areas.

14.9.1 Geological model

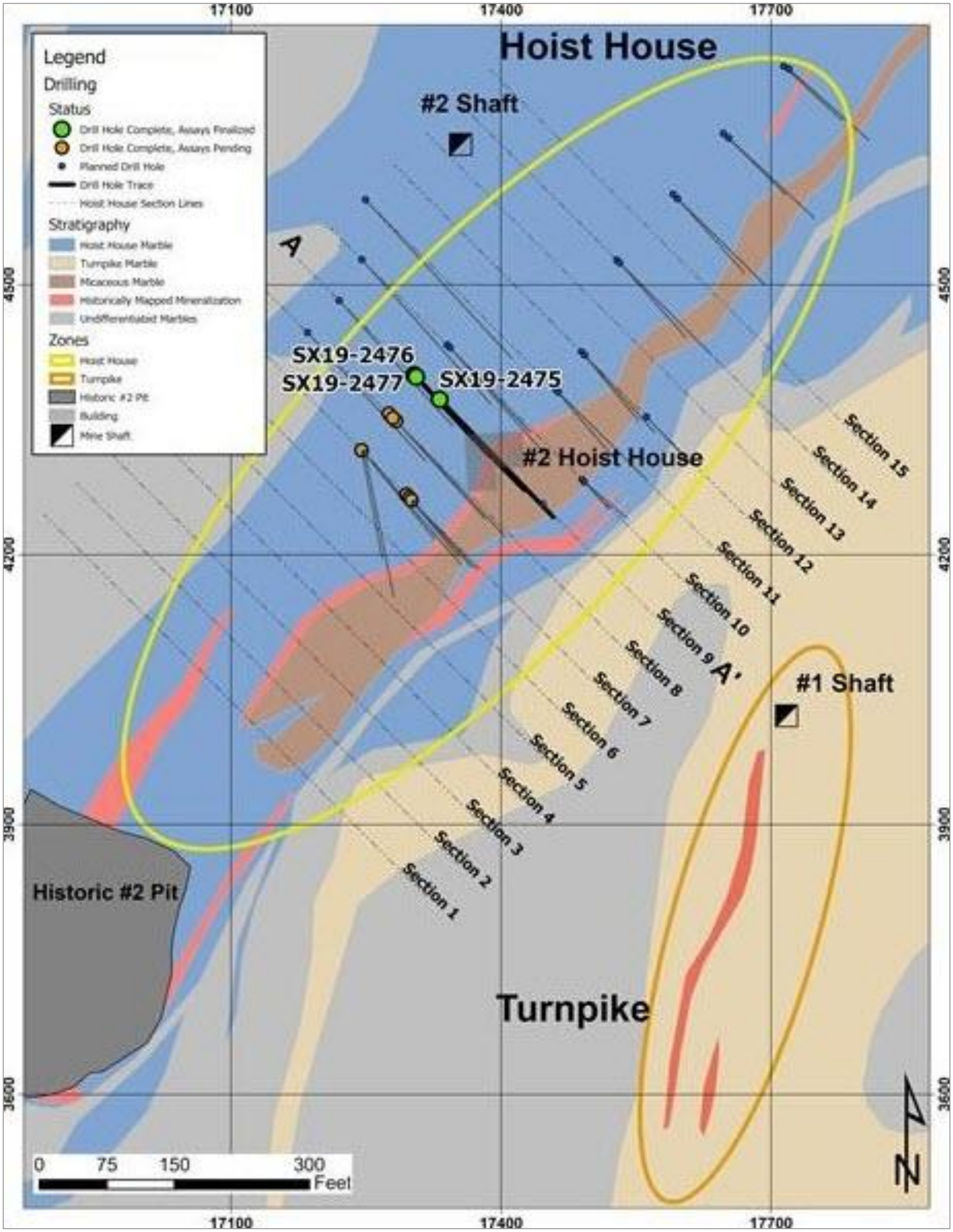
ESM also modelled and estimated Mineral Resources for the Number 2 Open Pit (N2) area, a near-surface target which ESM has identified in the infrastructure area which supported the historical Number 2 mine to the south east of the Number 4 Mine and Mill complex (Figure 14.7). The two primary areas of interest are referred to as Hoist House (HH) and Turnpike (TP), as shown in Figure 14.8. This process of modelling and estimation was markedly different from the underground ESM areas, such as Mahler and Mud Pond. Modelling and estimation of these areas was conducted using Leapfrog Geo and EDGE. Sulphide mineralization is hosted within Hoist House and Turnpike marbles. Hoist House generally sits in multiple massive to disseminated sulphide lenses of mineralization along the limbs of tight isoclinal folds plunging to the NE. Turnpike is constrained to the limbs and hinge of a N-plunging syncline, and is generally modelled as a single lens which is folded at depth, with the upper limb appearing sub-vertical and the lower limb oriented sub-horizontally. Both areas appear open above and below current drilling but are reasonably constrained by drilling along strike, as shown in Figure 14.9.

Figure 14.7 N2 Pits location map



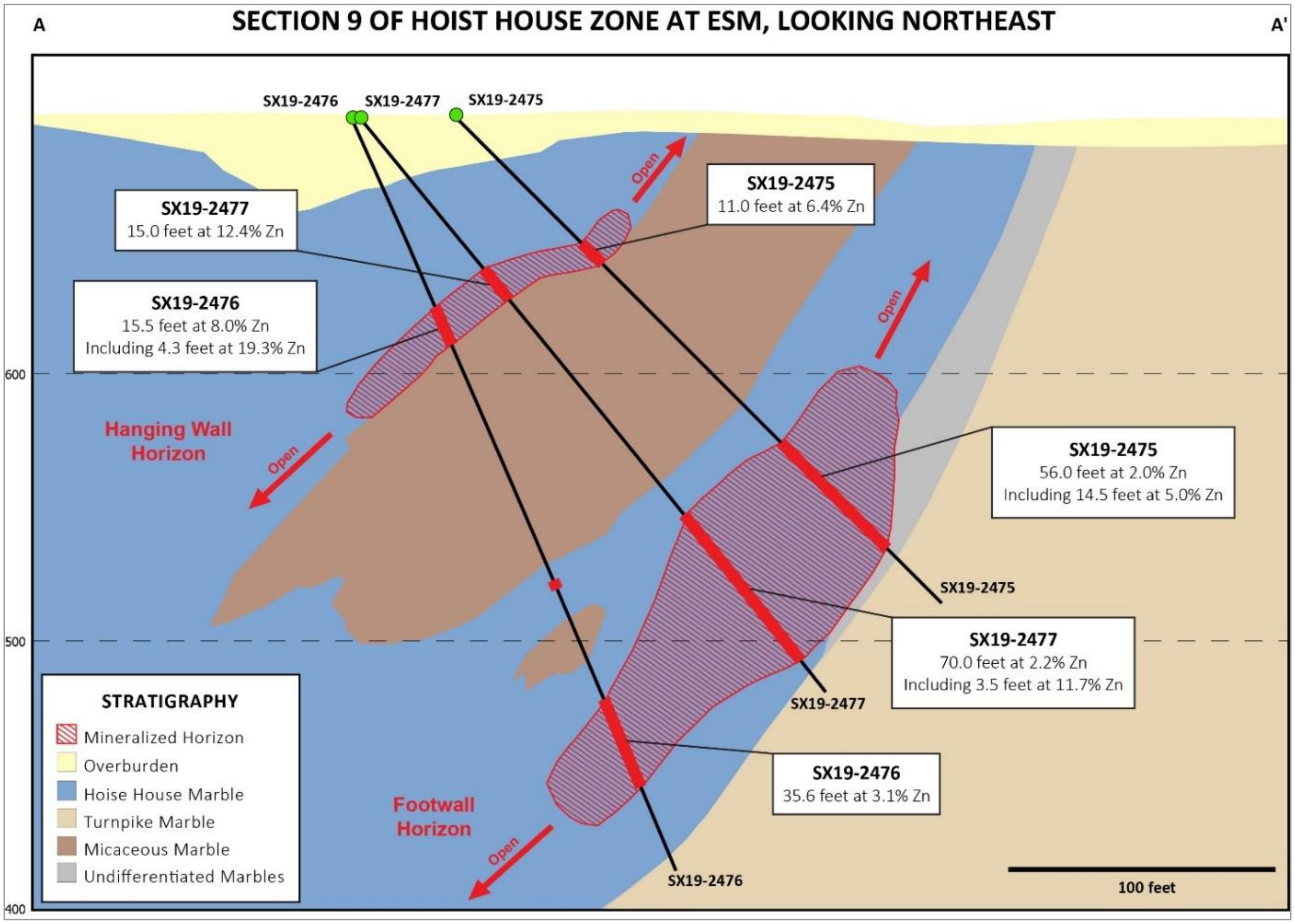
Source: ESM 2020.

Figure 14.8 General geology N2 pit area



Source: Titan Mining 2020.

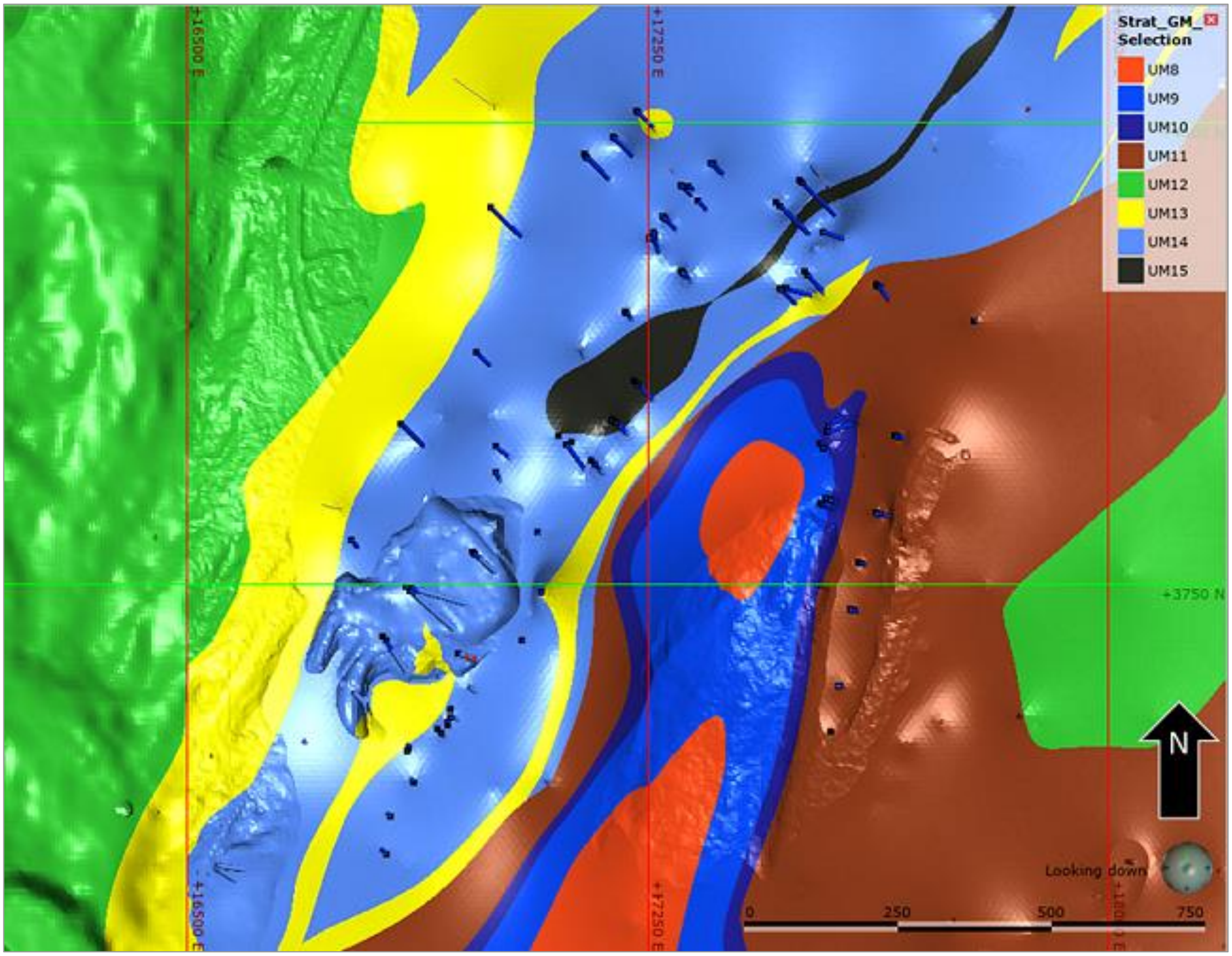
Figure 14.9 Schematic cross section Hoist House area



Source: Titan Mining 2020.

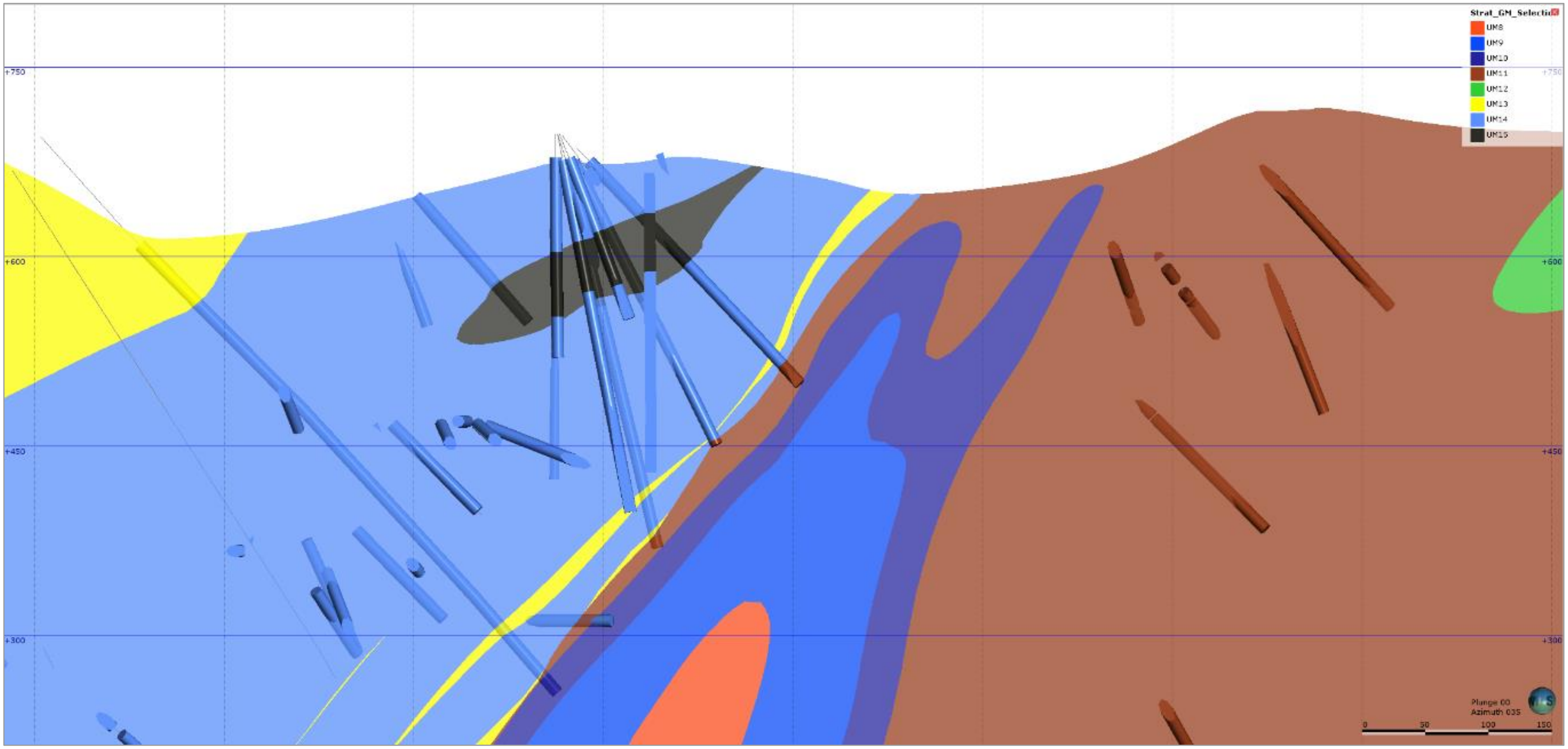
Geological modelling was supported by 75 modern near-surface diamond core drillholes. A selection of historical drillholes was excluded from modelling and estimation due to uncertainty around sampling and logging practices. All modern drillholes were surveyed via Reflex tool, at intervals ranging from 50 to 150 ft downhole. The geology was logged by ESM staff, all of whom are familiar with the style of mineralization and deposit type. Contacts from drilling were utilized to implicitly model the metasedimentary units within the open pit area of interest in Leapfrog Geo. Interpretation was driven between drilling intercepts using polylines digitized in section and 3D by ESM geologists. In general, the model reflects steeply-plunging, north-trending tight folds. The folds are separated into two primary domains to the SE and NW, broadly defined by the UM 8-11 series of anticlines and the UM 12-15 series of synclines, respectively. Mineralization is confined to the UM14 unit in HH area, and UM11 in the TP area.

Figure 14.10 Geological model of N2 pit area



Note: Fill and overburden removed from model to illustrate bedrock geology.
Source: SRK 2020.

Figure 14.11 Detailed cross section of geological model



Note: Looking N35E, +/-100 ft width.
Source: SRK 2020.

14.9.2 Resource domains

To constrain the estimation to the sulphide zones for the Hoist House and Turnpike areas, ESM used indicator interpolants of Zn grades internal to the primary litho-structural domains. Zn grades in the continuous assays were composited to 5 ft lengths and evaluated above a 0.25% cut-off (i.e., the ESM definition of "mineralized") and contoured in 3D to produce resource domain wireframes. Structural trends based on geological surfaces guide the interpolants, which feature ranges of between 100 to 150 ft. Probability factors of 50% are utilized to ensure that the interpolants select intervals which have a better than 50% probability of being above the 0.25% cut-off. Statistics for the indicator volumes indicate reasonable performance of these shapes for excluding isolated samples above the cut-off grade with limited incorporation of samples below the cut-off internal to the wireframes. The resulting indicator interpolants were then edited using sectional and 3D polylines to locally reduce / increase volumes and influence continuity based on geological interpretation. Controls on these domains are driven by the stratigraphy and structural features modelled by ESM.

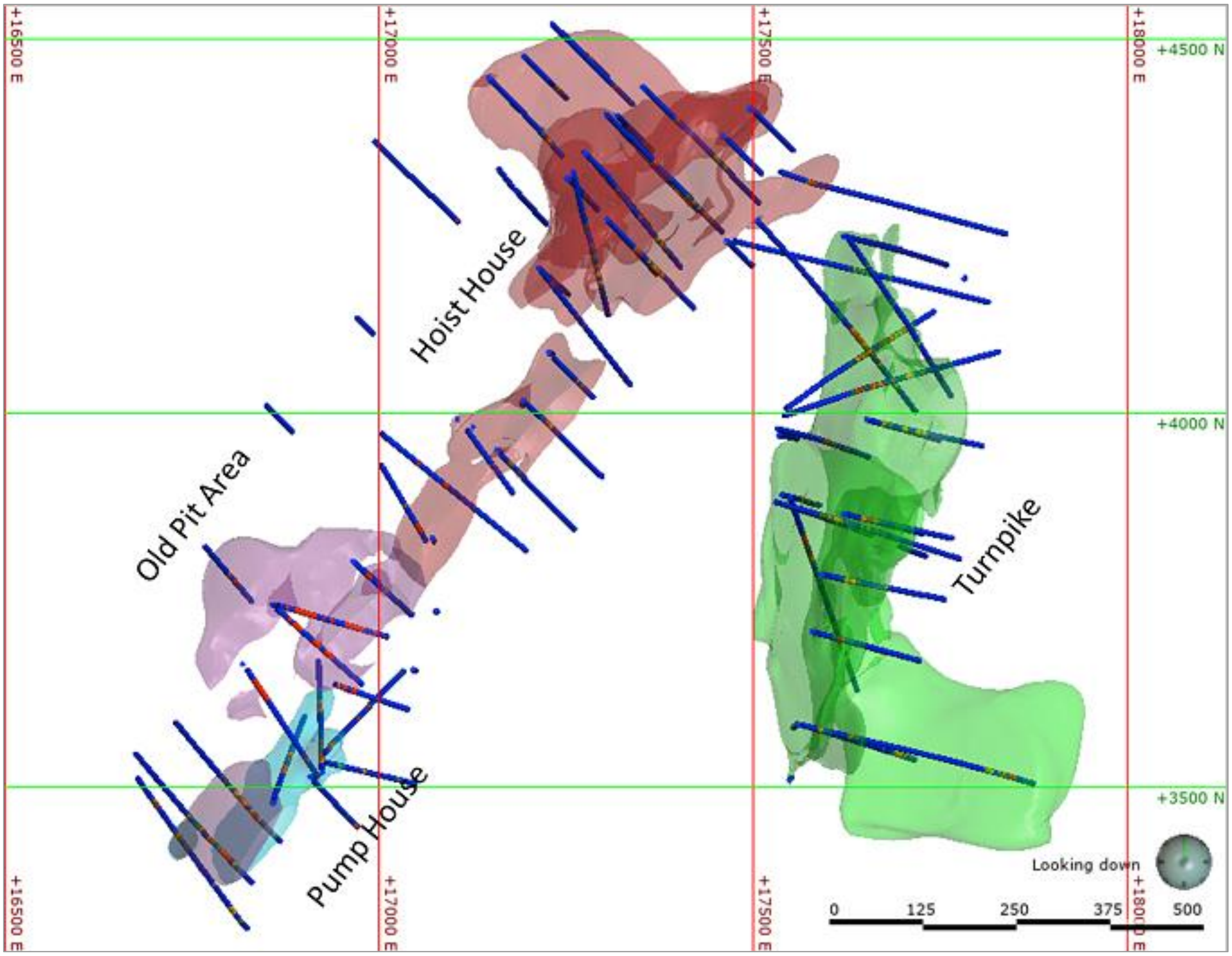
Table 14.10 Indicator interpolant performance metrics

Indicator statistics	**Hoist House**		**Turnpike**	
Total number of samples	2,240		1,004	
Cut-off value	0.25		0.25	
	≥ cut-off	**< cut-off**	**≥ cut-off**	**< cut-off**
Number of points	691	1,549	350	654
Percentage	30.85%	69.15%	34.86%	65.14%
Mean value	4.82	0.01	2.54	0.03
Minimum value	0.25	0	0.25	0
Maximum value	31	0.246	20.5221	0.25
Standard deviation	5.76	0.040	3.20	0.05
Coefficient of variance	1.19	2.76	1.26	2.14
Variance	33.13	0.002	10.23	0.003
Output volume statistics				
Resolution	2		5	
Iso-Value	0.5		0.5	
	Inside	**Outside**	**Inside**	**Outside**
≥ cut-off				
Number of samples	631	60	340	10
Percentage	28.17%	2.68%	33.86%	1.00%
< cut-off				
Number of samples	10	1,539	15	639
Percentage	0.45%	68.71%	1.49%	63.65%
All points				
Mean value	4.86	0.149	2.48	0.039
Minimum value	0	0	0.025	0
Maximum value	31	26.31	20.5221	3.73
Standard deviation	5.83	1.10	3.20	0.18
Coefficient of variance	1.20	7.38	1.29	4.67
Variance	34.0861	1.20	10.2444	0.03
Volume	10,689,834	763,169,360	21,191,045	1,807,836,902
Number of parts	6	8	2	2
Dilution	1.6%		4.2%	
Exclusion	8.7%		2.9%	

Source: SRK 2020.

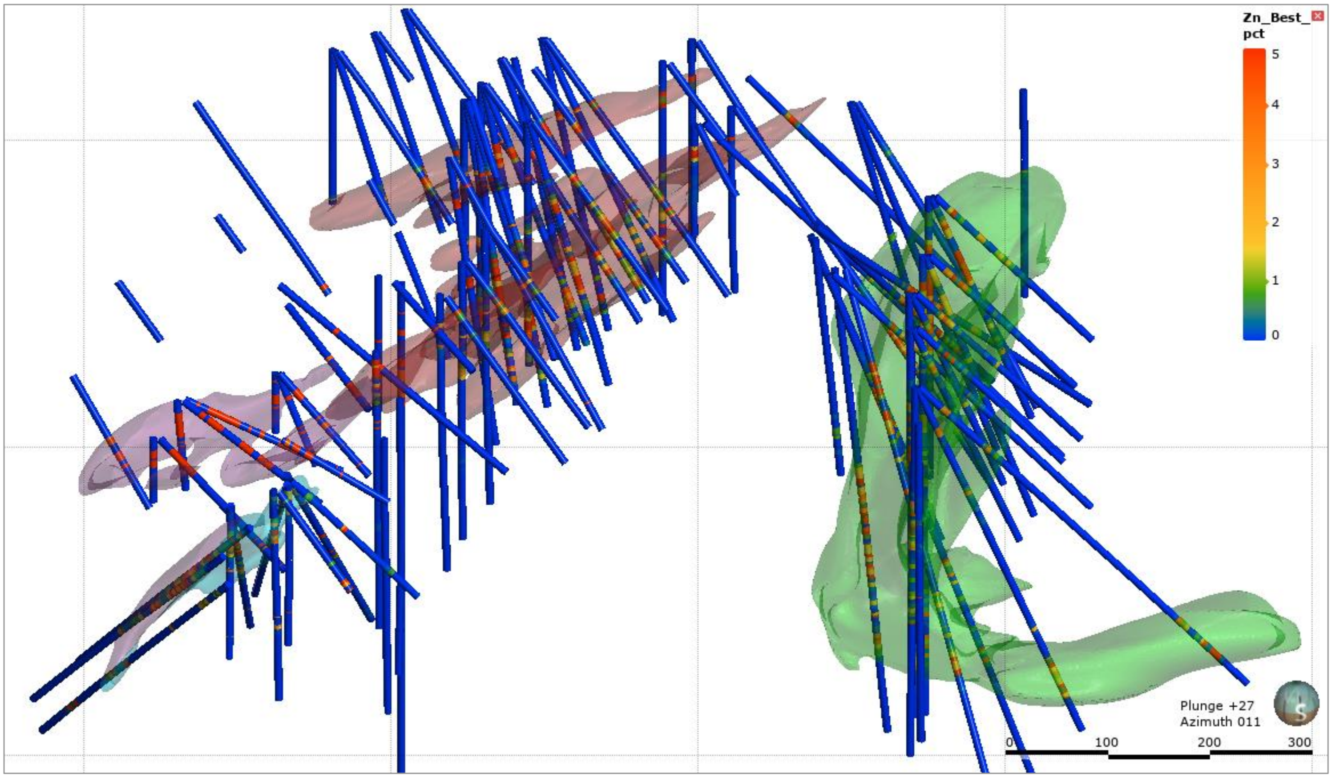
Based on reviews of the statistics internal to the primary domains, ESM split the Hoist House area into sub-domains due to lateral variations in grade distributions, morphology, or mineralization characteristics. These estimation domains are shown in Figure 14.12. In general, the HH domains dip 30 - 50 degrees to the west, while TP dips steeply to the west in the upper limb of a north-plunging syncline and is sub-horizontal in the lower limb. These relationships are shown in Figure 14.13. SRK notes that only the Hoist House, Turnpike, and Pump House domains were developed for estimation purposes due to the Old Pit area being mostly depleted, and drillhole orientations in this area being effectively parallel to the orientation of mineralization.

Figure 14.12 N2 pit area resource domains



Source: SRK 2020.

Figure 14.13 Oblique view of resource domains



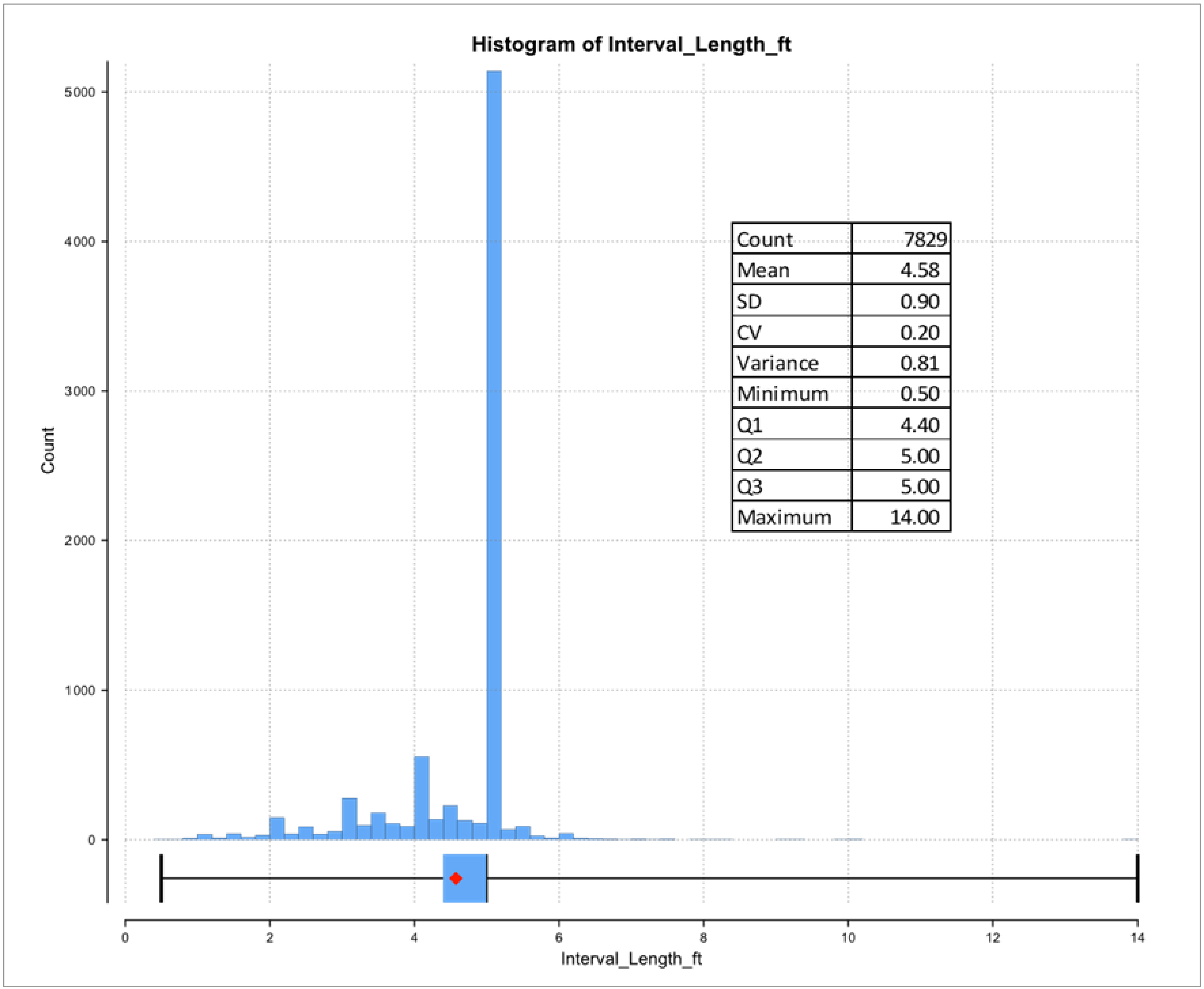
Source: SRK 2020.

14.9.3 EDA, outliers, and compositing

ESM conducted exploratory data analysis (EDA) on the three domains of interest. This started with a review of sample interval lengths within the domains. Samples were collected from the diamond drillholes at 5 ft intervals nominally, although some samples do extend up to 14 ft. SRK noted no bias of sample lengths to Zn grades in review of this work. ESM composited the sample intervals to a 10 ft consistent length within the domains for the purposes of scaling the data up to a volume variance more consistent with the projected block dimension / SMU. Overall, this impacts the mean with a reduction of less than 5% for each domain.

Statistics show that the domains feature discretely different populations. Descriptive statistics for both the raw assays and composited data for each domain are summarized in Figure 14.14 through Figure 14.17. SRK notes that ESM did not elect to cap Zn assays in the samples or the composites. This was based on ESM's review of the grade distributions and the relative consistency of the Zn mineralization in the domains. ESM did limit outliers in the TP domain through the use of outlier restrictions during the estimation, as discussed below.

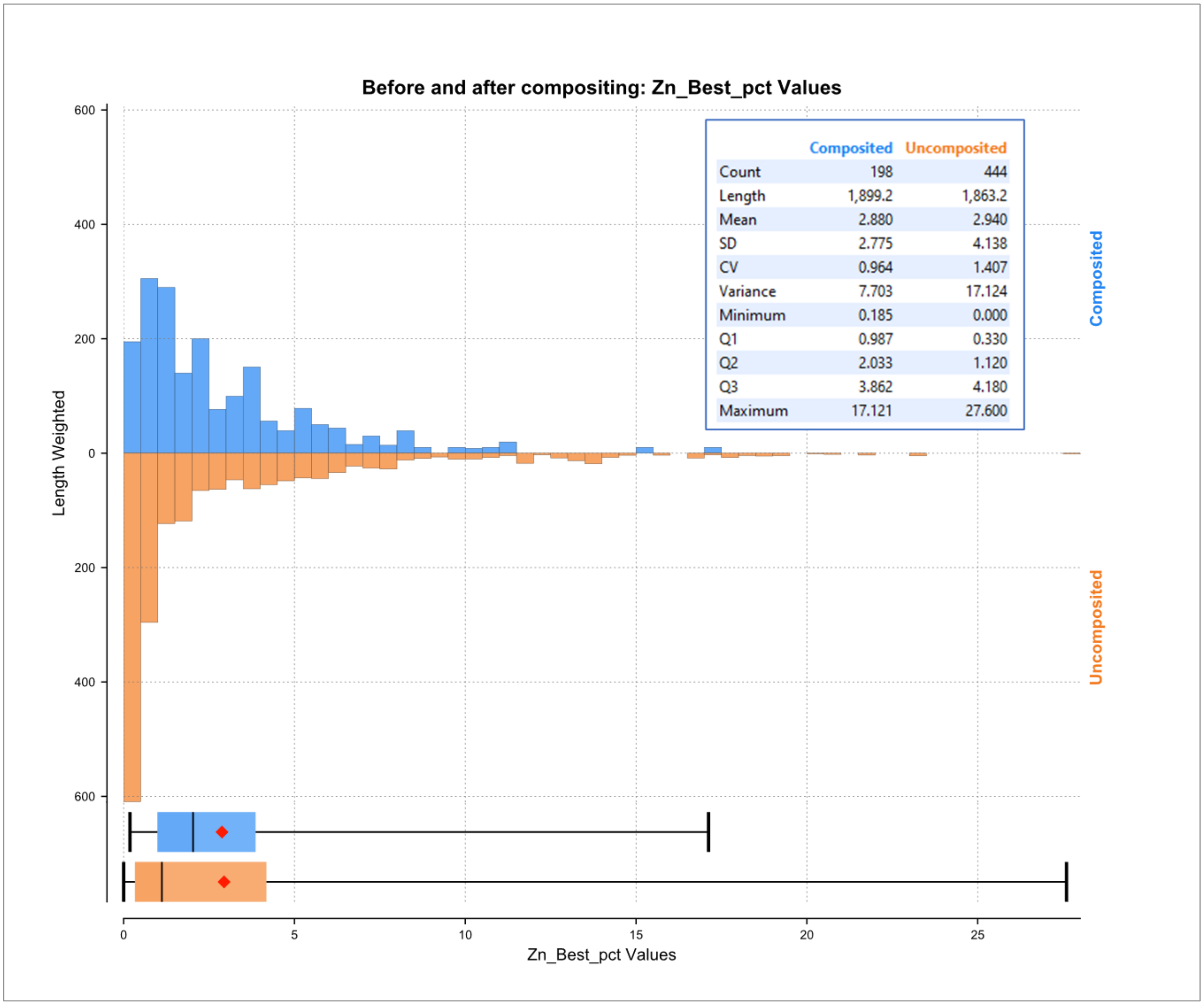
Figure 14.14 Sample length statistics



Count	7829
Mean	4.58
SD	0.90
CV	0.20
Variance	0.81
Minimum	0.50
Q1	4.40
Q2	5.00
Q3	5.00
Maximum	14.00

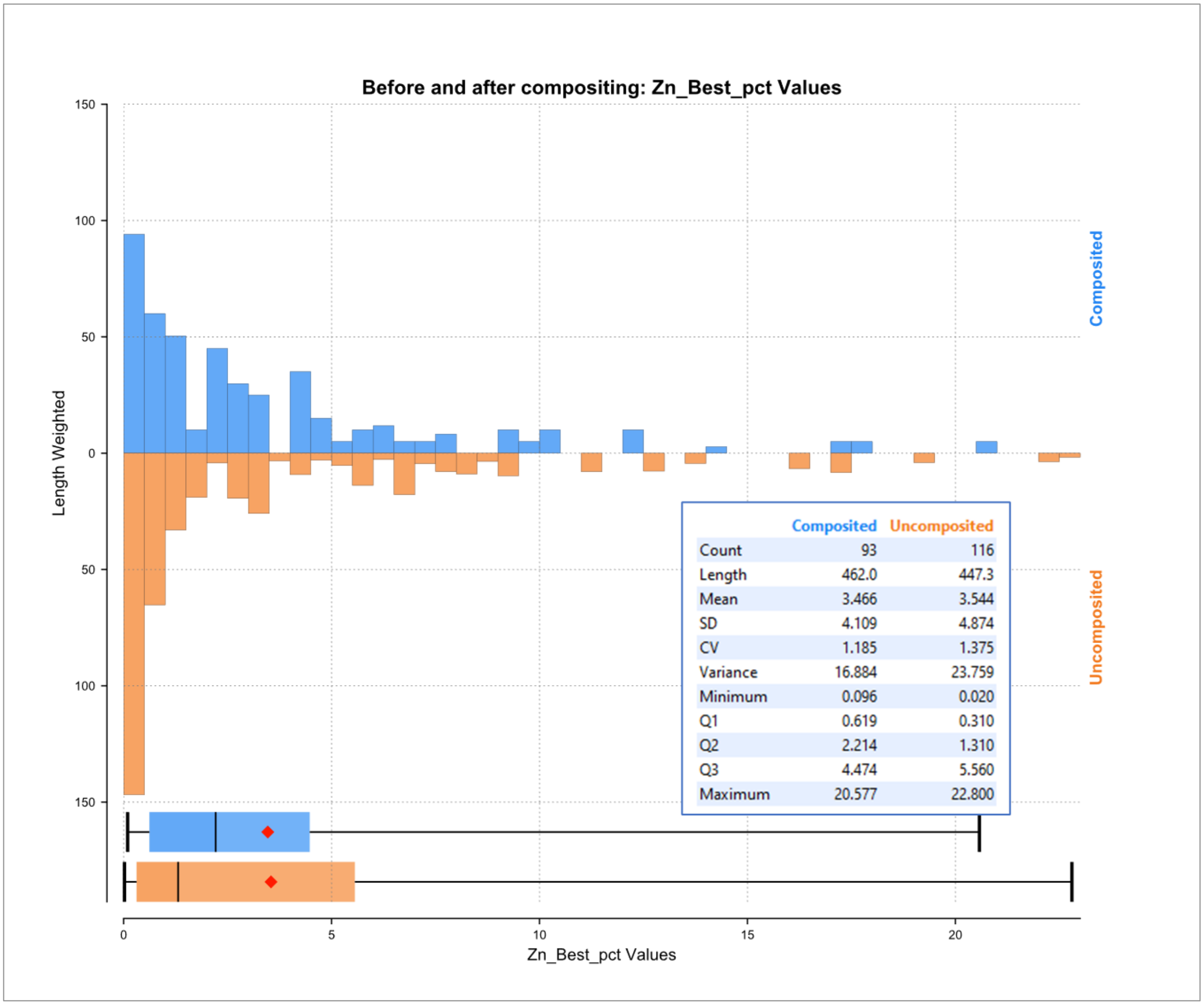
Source: SRK 2020.

Figure 14.15 Hoist House statistics – Zn



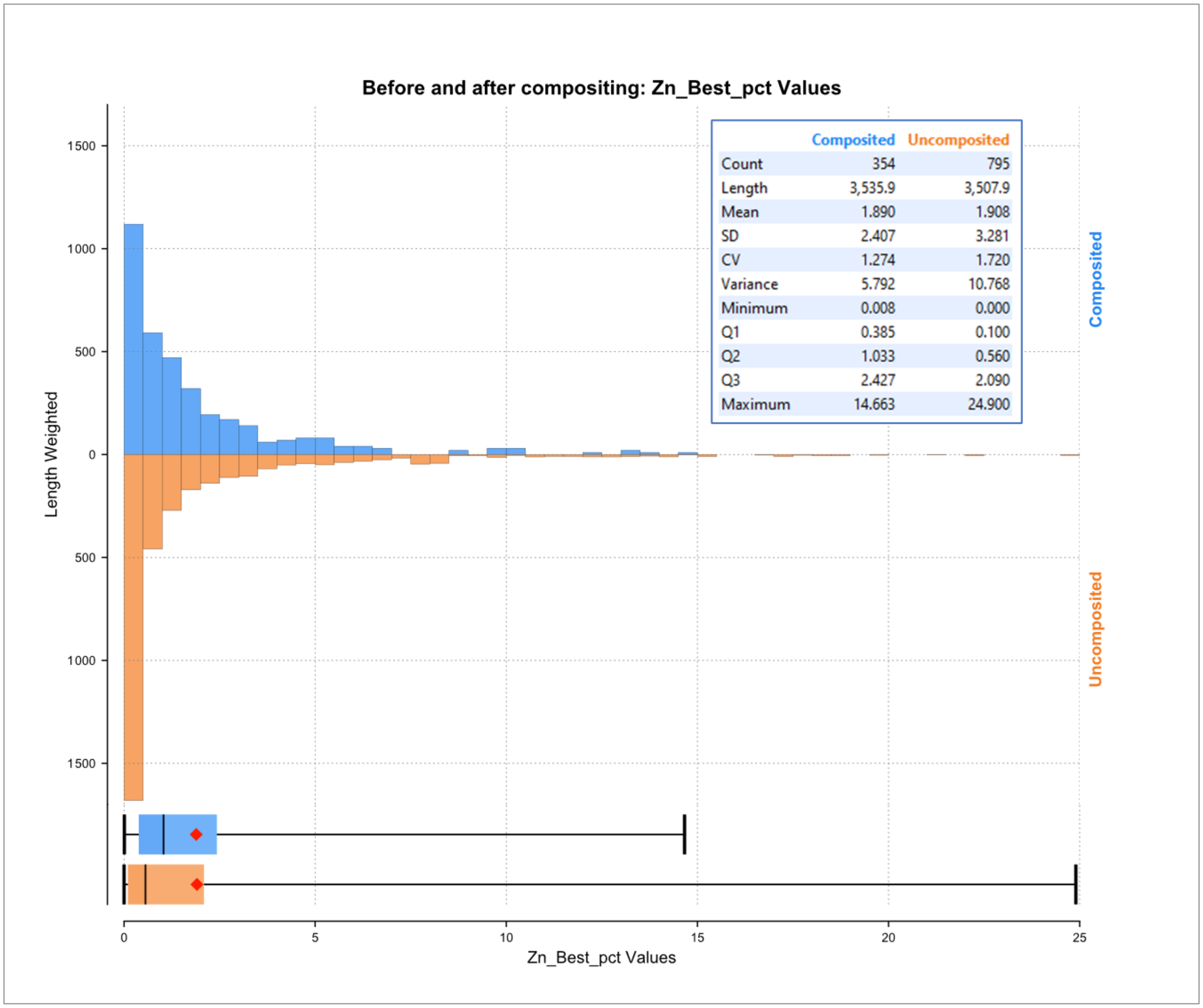
Source: SRK 2020.

Figure 14.16 Pump House statistics - Zn



Source: SRK 2020.

Figure 14.17 Turnpike statistics - Zn



Source: SRK 2020.

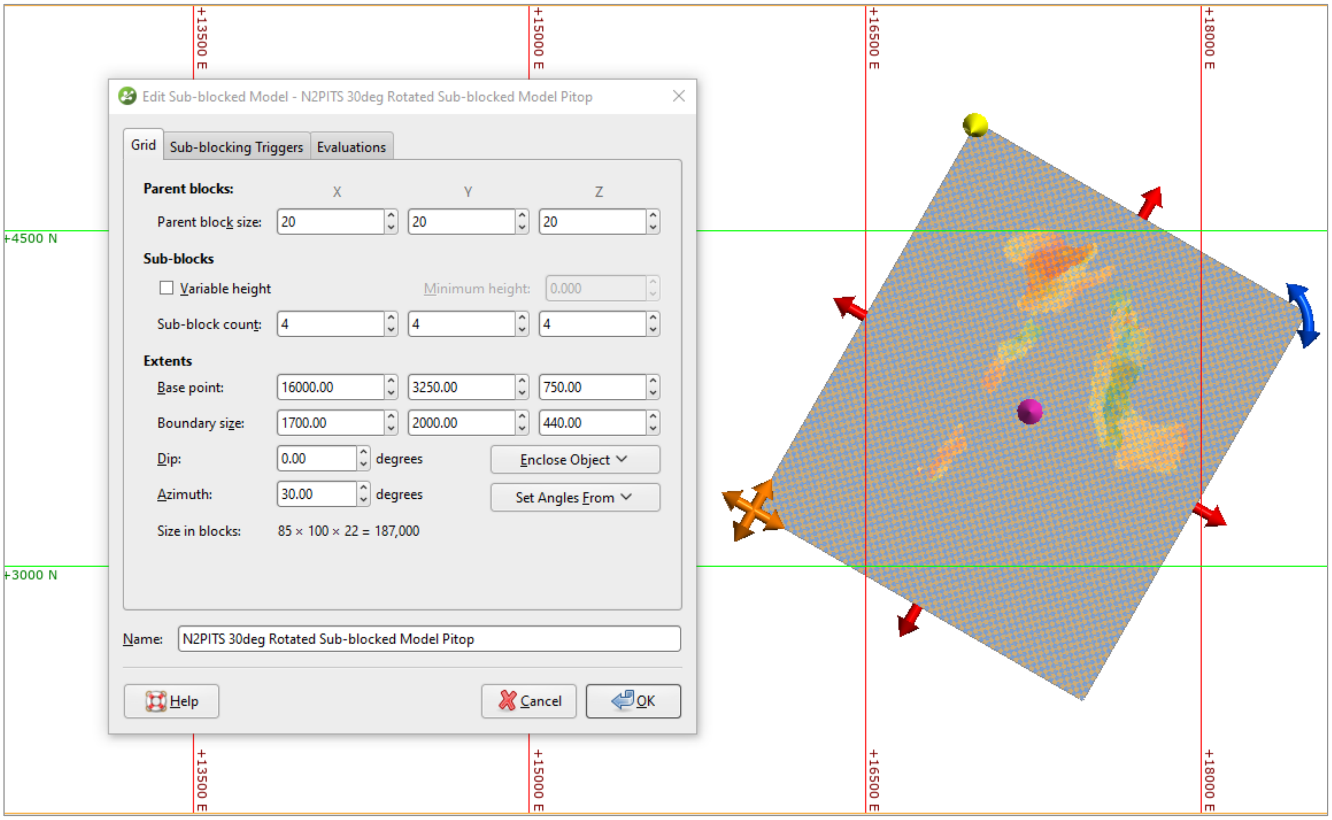
14.9.4 Variography

No variography was produced for the N2 Pits estimation and all estimates utilized an inverse distance weighting for interpolation purposes. SRK did conduct cursory review of variography as a part of this study and notes that the relative paucity of data and relatively high variance of grade does not lend itself to robust variography at the moment. This is likely an area of improvement for the future as more drilling is conducted and should be reviewed considering the multiple orientations present in the model.

14.9.5 Block model and estimation methodology

ESM constructed the block model in Leapfrog EDGE and utilized this model for evaluating the estimations into the relevant domains. The model was sub-blocked along the geological and resource domain boundaries, as well as topography and as-builts for nearby underground workings. The parent block dimension is a 20 by 20 by 20 ft block size, which is broken down into four sub-blocks per parent (i.e. 5 x 5 x 5 ft sub-block). The model is rotated by moving the Y axis to a 30 degree azimuth as shown in Figure 14.18.

Figure 14.18 Block model details



Source: SRK 2020.

Estimates were made using a single-pass inverse distance squared weighting. Search distances were generally based on drill spacing by the individual domain, and orientations were either set to the trend of the mineralized domain (PH) or made variable using the geological surfaces and trends via the variable orientation tool in EDGE. ESM reviewed the sample selection criteria through an iterative process and considered the results through validation. Of note are the estimation parameters for the HH / PH domains, which use a maximum of 20 to 40 samples, respectively. SRK's initial concerns regarding smoothing of these domains were reviewed over the course of this study by adjusting estimation parameters and comparing the validation and reporting of the resources.

SRK reviewed the estimation parameters and tested alternatives such as utilizing different sample selection criteria, interpolation methods, and search distances and noted minimal changes to the resource tonnage (less than 5%) by modifying parameters such as adjusting the ranges to 50% of current, and reducing sample selections to 3 / 15 for min / max. SRK is of the opinion that uncertainty with the estimation parameters, particularly around the PH domain, are also generally mitigated by the fact that there is relatively limited data in the domain and the resource classification considers data spacing and estimation confidence. The estimation parameters for the N2 Pits area are summarized in Table 14.11.

As noted previously, ESM limited the impact of outliers on the estimation of the TP domain using outlier restrictions in EDGE. This was applied as a limit on the distance over which the original composite grade is utilized, beyond which the grade of that composite is reduced to a threshold value. SRK applied the same thresholds and distances to the other domains as a sensitivity check and noted impacts of less than 1% on the estimate considering no other changes.

Table 14.11 N2 pits estimation methodology

	Search ellipsoid						Sample selection			Outliers	
Domain	**Major**	**Semi**	**Minor**	**Dip**	**Dip Azi**	**Pitch**	**Min**	**Max**	**Max samples / hole**	**Threshold**	**Distance**
HH	250	250	100	Variable			6	20	5	NA	NA
PH	150	150	150	51.2	308.5	97.2	10	40	5	NA	NA
TP	300	300	100	Variable			3	10	2	8	10

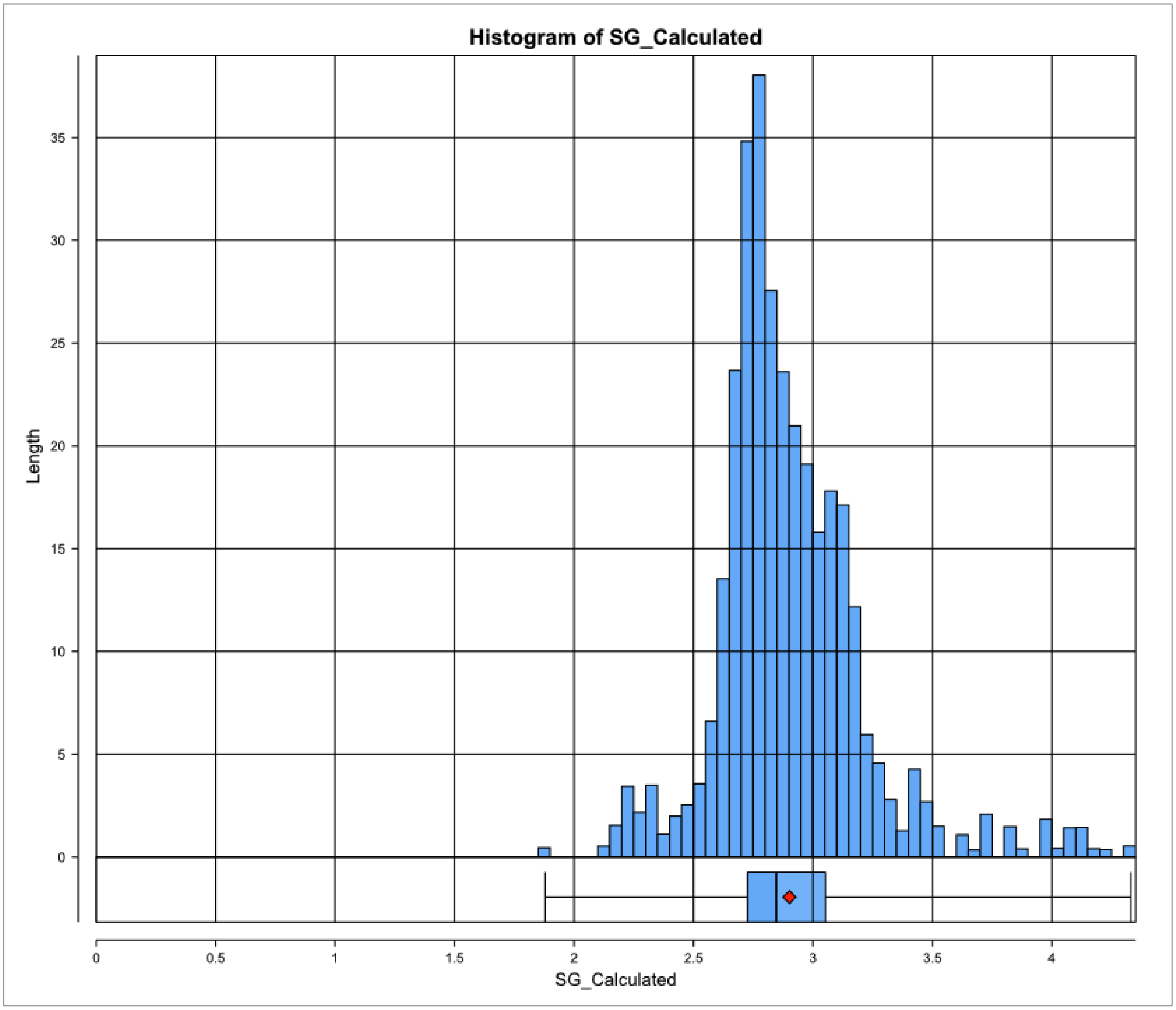
Source: SRK 2020.

14.9.6 Bulk density

Bulk density was estimated for the N2 Pits area from 704 SG samples taken within and external to the resource domains. SG was calculated by ESM geologists using the conventional immersion technique of weighing samples in air and in water and examining the displacement of the water in a controlled environment. SG is variable depending on factors such as lithology and sulphide content, and ranges in the collected data between 1.88 and 4.33 as shown in Figure 14.19. The distribution of the SG sampling is reasonable and covers the entire N2 Pits area, although the degree of sampling does decrease in the PH area to the south (Figure 14.20). Mineralized domains generally feature higher SG than the unmineralized rocks, as shown in Figure 14.21. All SG measurements were converted to bulk density using an assumption of equal relationship of SG to g/cm^3, and a unit conversion to a tf represented in short tons/ft^3.

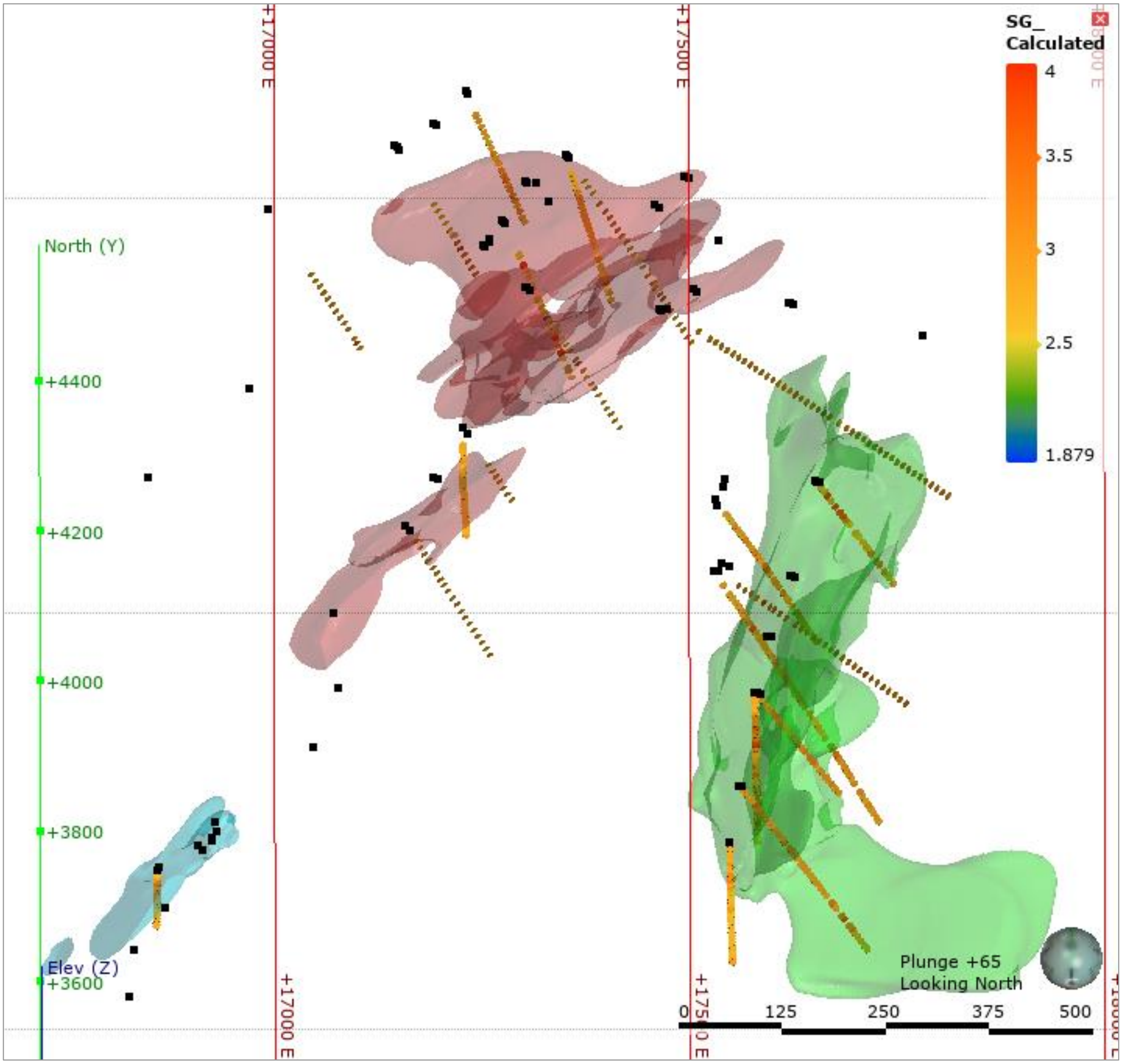
A simple inverse distance weighted interpolation method was used to estimate SG from samples within the domains into the domains themselves and utilized the same basic parameters as the grade estimate, including search orientations and numbers of samples. External to the estimated areas (i.e., distal from sampling or within fill material / overburden) SG was assigned using assumptions about this material from averages of samples collected in the material or ESM experience managing this material. Average bulk densities for the various materials and domains within the model are shown in Table 14.12. All densities were depleted for previous mining prior to reporting.

Figure 14.19 Histogram of N2 pits SG samples



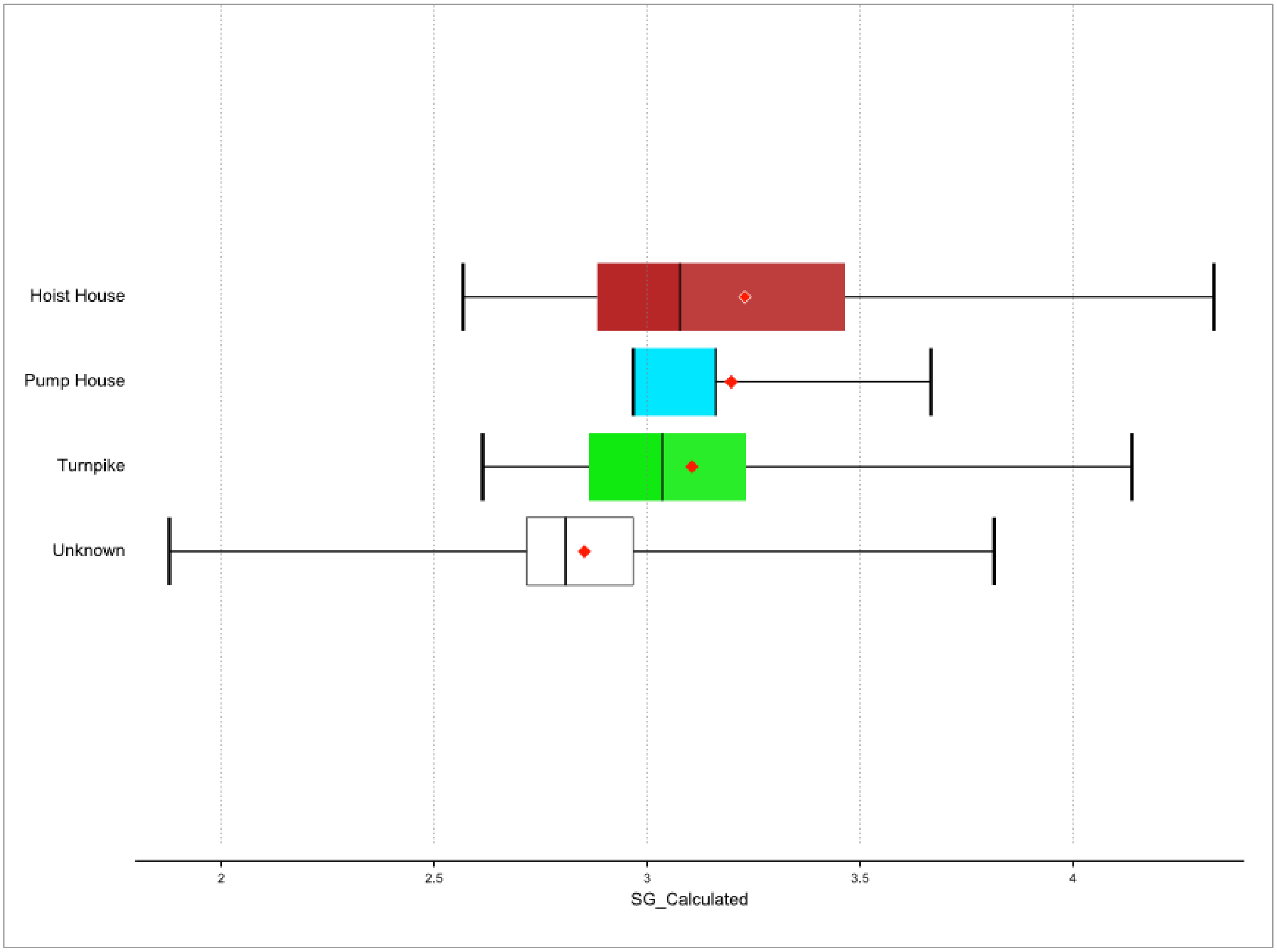
Source: SRK 2020.

Figure 14.20 Oblique view of spatial distribution of SG



Source: SRK 2020.

Figure 14.21 Box plots of SG by resource domain



Source: SRK 2020.

Table 14.12 SG interpolation values vs. calculated tonnage factor by domain

Name		Block count	Mean
Hoist House	SG	44,169	3.24
Hoist House	TF	45,920	0.09
Old pit	SG	0	
Old pit	TF	10,161	0.08
OVB	SG	0	
OVB	TF	287,057	0.06
Pump House	SG	3,397	3.25
Pump House	TF	3,397	0.09
Turnpike	SG	83,806	3.13
Turnpike	TF	93,352	0.09
External	SG	1,336,994	2.85
External	TF	3,083,487	0.09

Note: SG not interpolated into all areas. Those with 0 SG interpolation were assigned nominal tonnage factors based on ESM guidance for these materials.

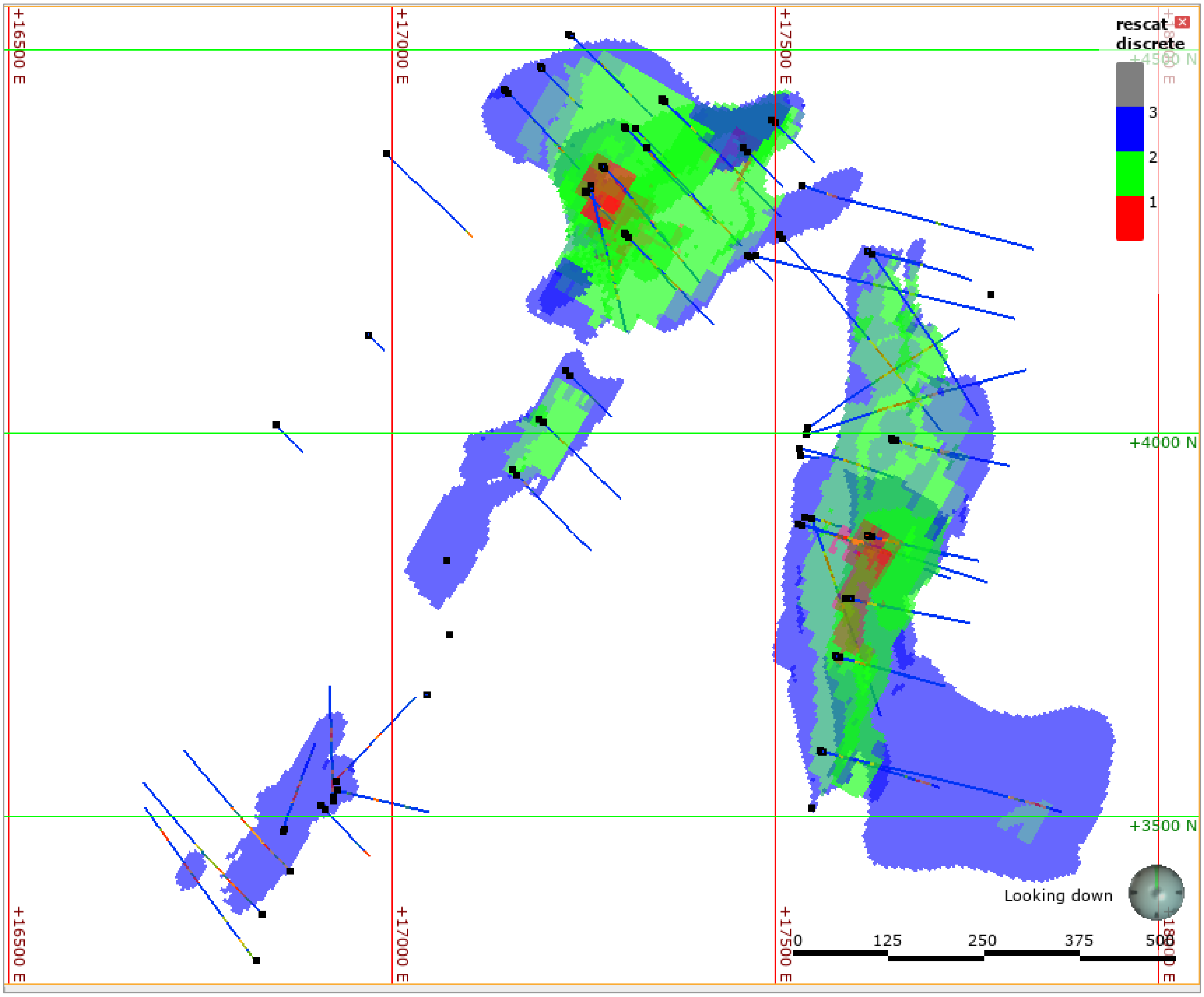
14.9.7 Classification and reporting

The N2 Pits area was classified and reported in a different manner than the other ESM underground resources. Measured, Indicated, and Inferred Mineral Resources are consistent with CIM guidelines for classification, the same overall guidelines as those utilized in the underground resources. Classification considered the following criteria for determining relative confidence in the estimate:

- Measured: Internal to the Hoist House or Turnpike domains, generally using at least three drillholes within an average distance of less than or equal to 50 ft.
- Indicated: Internal to the Hoist House or Turnpike domains, generally using at least two drillholes within an average distance of less than or equal to 100 ft.
- Inferred: Any blocks within the domains where an estimate was made. No material external to the domains was categorized.
  - Blocks south of the 3930 northing in Hoist House were made Inferred due to the nature of drilling and uncertainty with previous mining / stoping in these areas.
  - Blocks below the 420 elevation level in Turnpike, and any blocks within the lower limb were made Inferred due to lack of drilling and uncertainty with geological continuity in these areas.
  - All blocks within the Pump House domain were made Inferred due to the sub-parallel nature of drilling in this domain.
  - Any other blocks within a 20 ft buffer zone around surveyed underground workings were made Inferred due to uncertainty with that surveying.

The classification scheme was designed to outline areas of relatively higher confidence in the estimation and reporting of Mineral Resources and was scripted for all blocks. Distances and sample selection were refined after iterative review of the results and adjusted as needed to reflect ESM and SRK geologists combined view of the mineralization. An oblique view of the classified resource blocks is shown in Figure 14.22. The net impact of this approach to classification is a relatively restricted Measured resource, with comparably more Indicated in areas of reasonable drill support.

Figure 14.22 Plan view of N2 pit area classification



Source: SRK 2020.

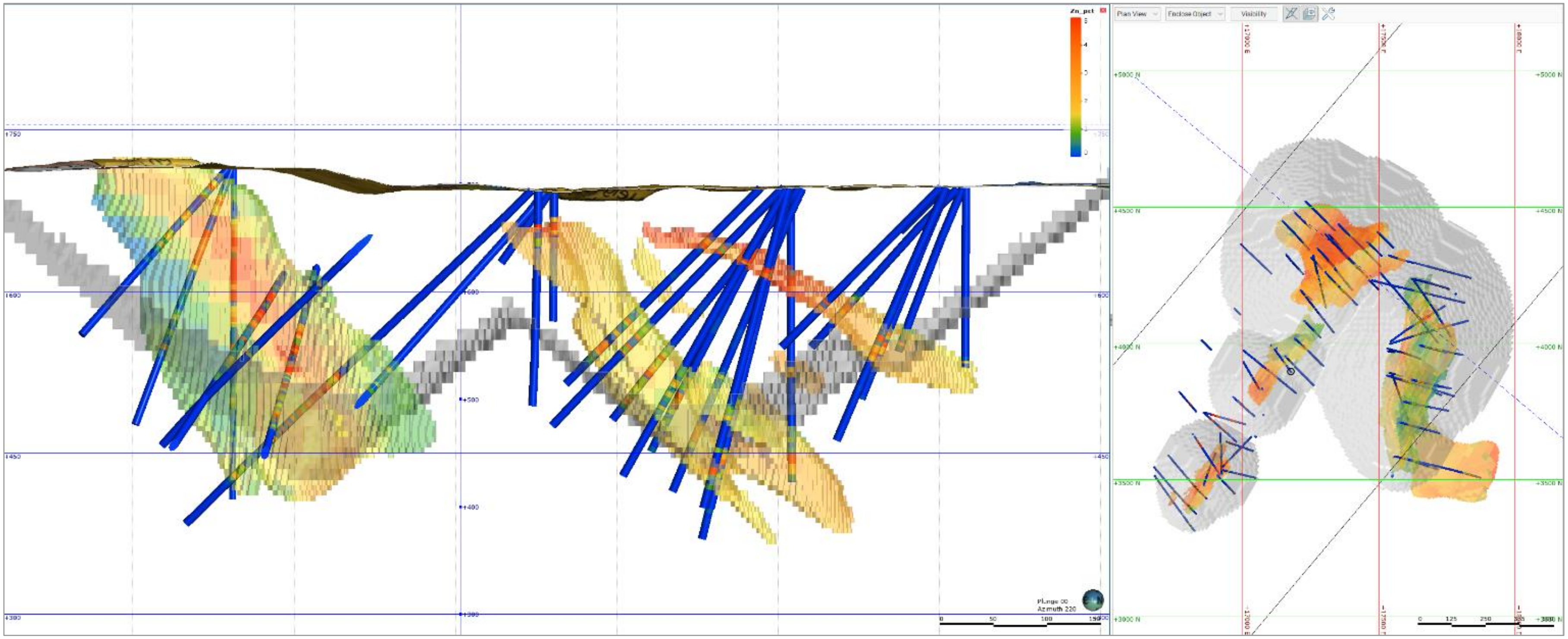
The primary difference in reporting of the N2 Pit area is related to the demonstration of reasonable prospects for eventual economic extraction (RPEEE) and how this varies compared to the underground areas. AMC developed pit optimization scenarios for the N2 Pits resource using parameters for economics and slope stability derived from ESM experience and inputs on geotechnical design parameters from Knight Piésold. Overall pit slope angles are approximately 40 to 45 degrees, and two pit shells are used for reporting purposes. Mineral Resources are reported above a cut-off and within these pit shells to ensure consistency with RPEEE. Grade cut-offs for resource reporting are based on the parameters assumed by AMC and the considerations of break-even cost scenarios within the pit for the non-mining related costs (e.g., processing, G&A, selling). Cost / pricing assumptions are shown below in Table 14.13. Considering the relevant costs and pricing scenarios, SRK has calculated an overall cost within the pit to be $31.50, which equates (considering $1.07/lb Zn and 94% recovery equating to unit value of $20.12/% Zn) to an overall Zn cut-off of 1.57%.

Table 14.13 N2 pit area cost and pricing assumptions

Block model	**nf_20200117_bm_noclass**	
Metal price Zn	1.07	$/lb
Mineralization and waste mining cost	2.50	$/dry short ton (pit optimization only)
Additional mineralization mining cost	2.00	$/dry short ton (pit optimization only)
Processing and G&A cost	25.00	$/dry short ton
Selling cost	6.50	S$/unit Zn metal (20 lbs)
Mining loss	5.0%	Considered for pit optimization only
Mining dilution	5.0%	Considered for pit optimization only
Mill Zn recovery	94.0%	Based on actual ESM recovery

Note: Costs in red are those considered for COG determination within open pit shell.
Source: AMC 2020.

Figure 14.23 AMC Resource pit shells



Note: Pit Shell is described as W4X Export7_14 as provided by AMC.
Source: SRK 2020.

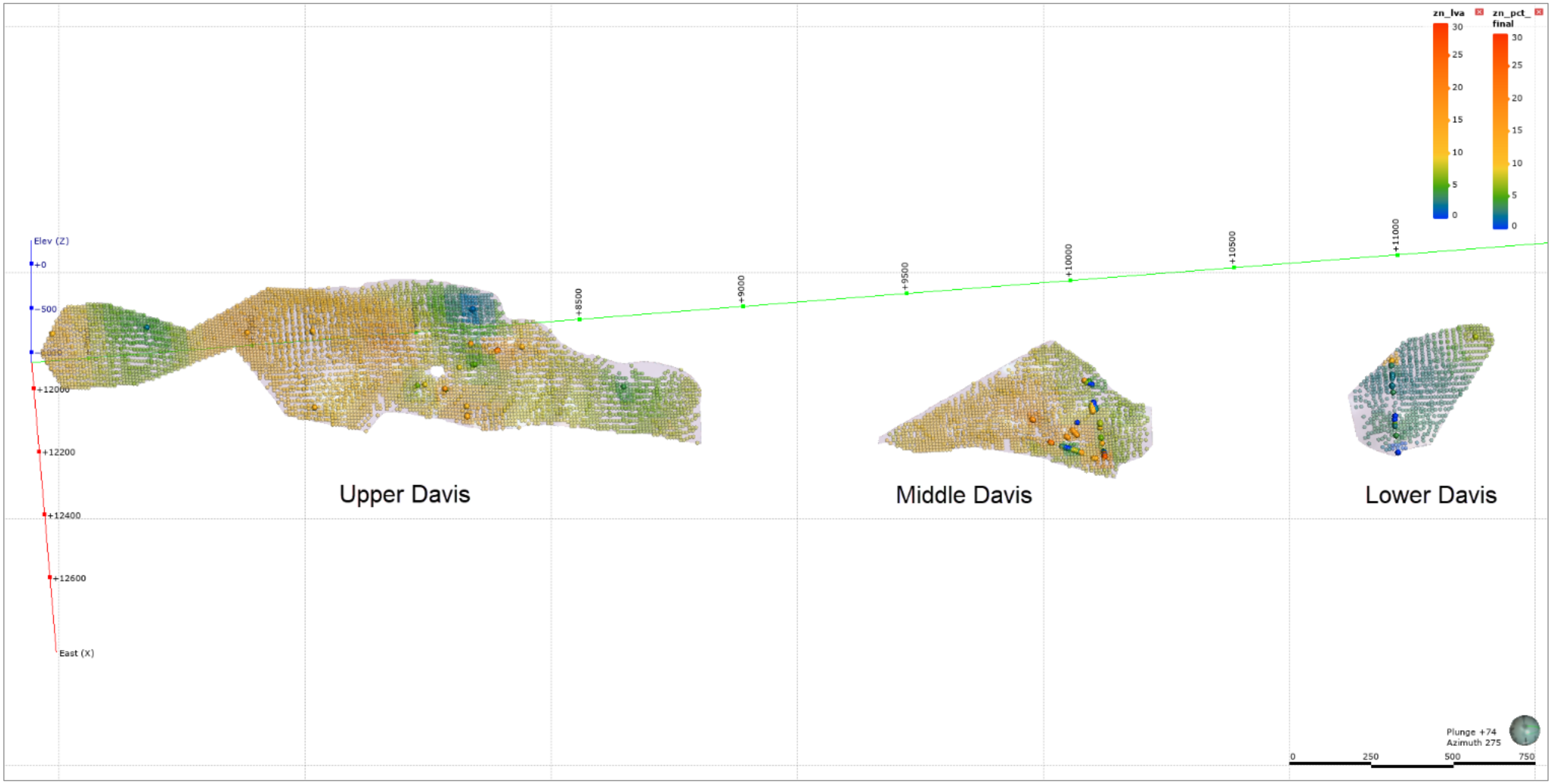
14.10 Model validation

14.10.1 Visual comparison

SRK conducted validation of the block estimates for both the underground and open pit resources. Visual comparison of the estimated grades in the blocks to the informing composites is the first and most important validation step. Within the ESM deposits, most zones compare well, while a few perform less ideally. This is most directly observed where data density within a vein changes dramatically. Poorly performing areas are often unavoidable due to the variability of the composites and complex geometry being estimated. For areas where validation is not ideal, classification was used to address the inherent uncertainty in the estimate.

Davis is provided as an example in Figure 14.24 for the underground. Davis demonstrates both areas of excellent visual representation in the model and less ideal representation. Middle Davis drilling is clustered to the north (right side of the image) while the rest of Davis contains reasonably spaced data.

Figure 14.24 Davis model and composite values for zinc

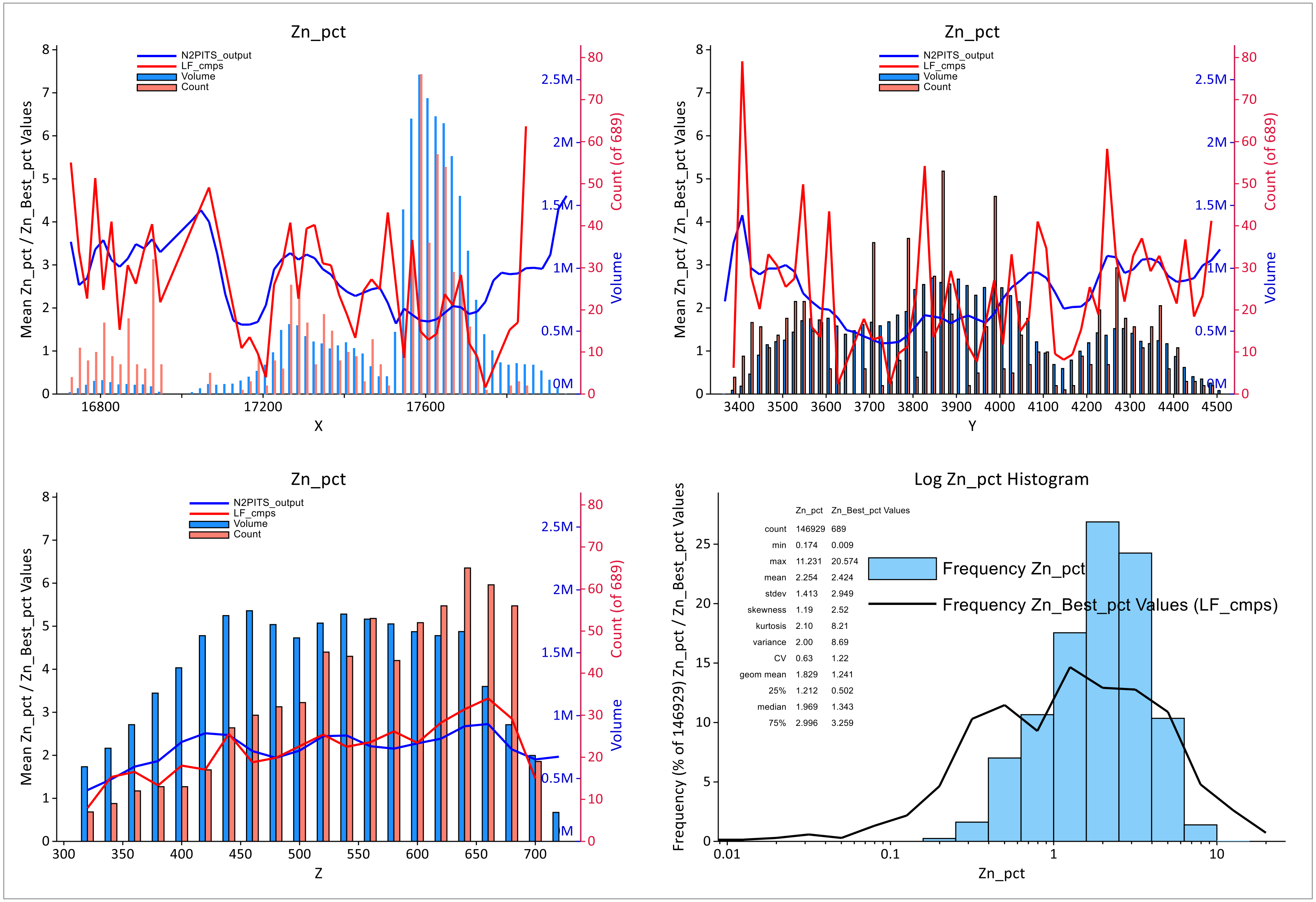


Note: Oblique view looking west.
Source: SRK 2020.

14.10.2 Swath plots

SRK used swath plots to verify that the spatial distribution of grade in the composites is honored in the interpolated model. An example is shown below in Figure 14.25 for the N2 Pits Area. Swath plots generally show agreement of the estimate to the composites, with an appropriate degree of smoothing.

Figure 14.25 Swath plot Zn% - N2 pits area



Source: SRK 2020.

14.11 Mineral Resource statement

Underground Mineral Resources have been modelled (Leapfrog Geo) and estimated (Maptek Vulcan) by ESM geologists and reviewed for consistency with industry standards by SRK. In some cases, SRK participated in classification or refinement of the estimates based on this review. Matthew Hastings of SRK Consulting (U.S.) Inc. is the QP who has reviewed the geological models and estimates and has conducted multiple site inspections. Mineral Resources for the underground Number 4 mine areas have been compiled from ten separate block models including the American, Cal Marble, Davis, Fowler, Mahler, Mud Pond, Number 2 Deeps, North East Fowler, New Fold, and Silvia Lake areas.

Table 14.14 Underground Mineral Resource estimate as of 1 October 2020

Category	Tons (000's US short tons)	Zn (%)	Contained pounds (000's lbs)
Measured	190	13.56	51.6
Indicated	1,524	11.49	350.3
Measured + Indicated	1,714	11.72	401.9
Inferred	6,551	11.11	1,455.1

Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that any part of the Mineral Resources estimated will be converted into a Mineral Reserves estimate. Resources stated as in-situ grade at a Zinc price of $1.07/oz, with an assumed zinc recovery of 96.3% Resources are reported using a 5.3% Zinc cut-off grade, based on actual break-even mining, processing, and G&A costs from the ESM operation. Numbers in the table have been rounded to reflect the accuracy or the estimate and may not sum due to rounding.
Source: SRK 2020.

Open-pit Number 2 Mine Mineral Resources have also been modelled (Leapfrog Geo) and estimated (Leapfrog EDGE) by ESM geologists and reviewed for consistency with industry standards by SRK. In some cases, SRK participated in classification or refinement of the estimates based on this review. Matthew Hastings of SRK is the QP who has reviewed the geological models and estimates, and has conducted one site inspection to the Number 2 Mine surface areas. Mineral Resources for the Number 2 Mine Open Pit area have been taken from a single block model which features the Hoist House, Pump House, and Turnpike areas.

Table 14.15 Open Pit Mineral Resource estimate as of 1 October 2020

Category	Tons (000's US short tons)	Zn (%)	Contained pounds (000's lbs)
Measured	105	3.34	3,190
Indicated	595	3.09	16,675
Measured + Indicated	701	3.13	19,864
Inferred	217	3.37	6,639

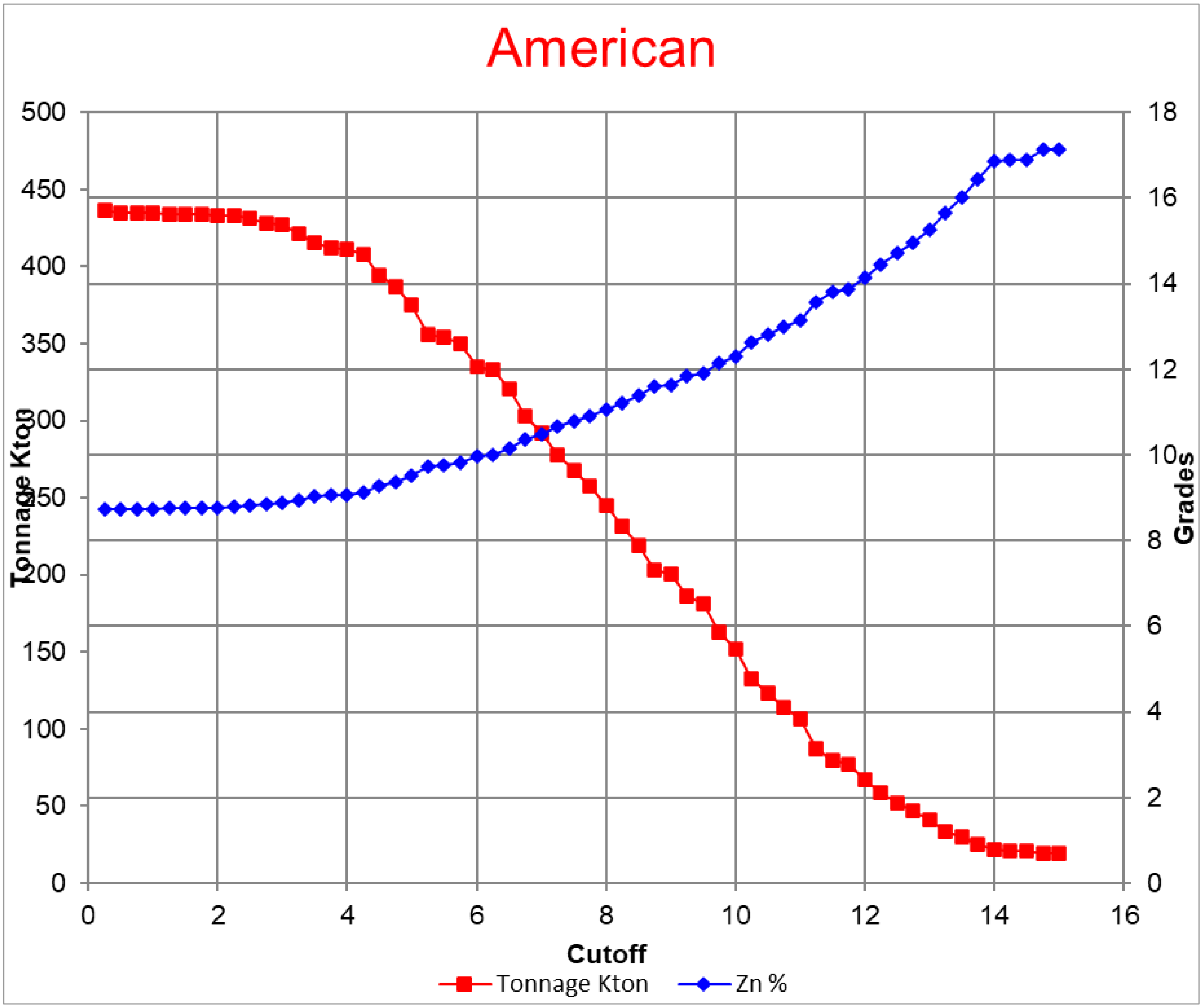
Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that any part of the Mineral Resources estimated will be converted into a Mineral Reserves estimate. Resources stated as internal to an optimized pit shell, above a cut-off grade of 1.57% Zn. Cut-off is based on break-even economics at a Zinc price of $1.07/oz, with an assumed zinc recovery of 94%, and actual processing, and G&A costs from the ESM operation. No mining costs were considered in the calculation of this COG, as the pit optimization incorporates the mining costs to develop the shape for reporting. Numbers in the table have been rounded to reflect the accuracy or the estimate and may not sum due to rounding.
Source: SRK 2020.

14.12 Mineral Resource sensitivity

14.12.1 Number 4 Underground

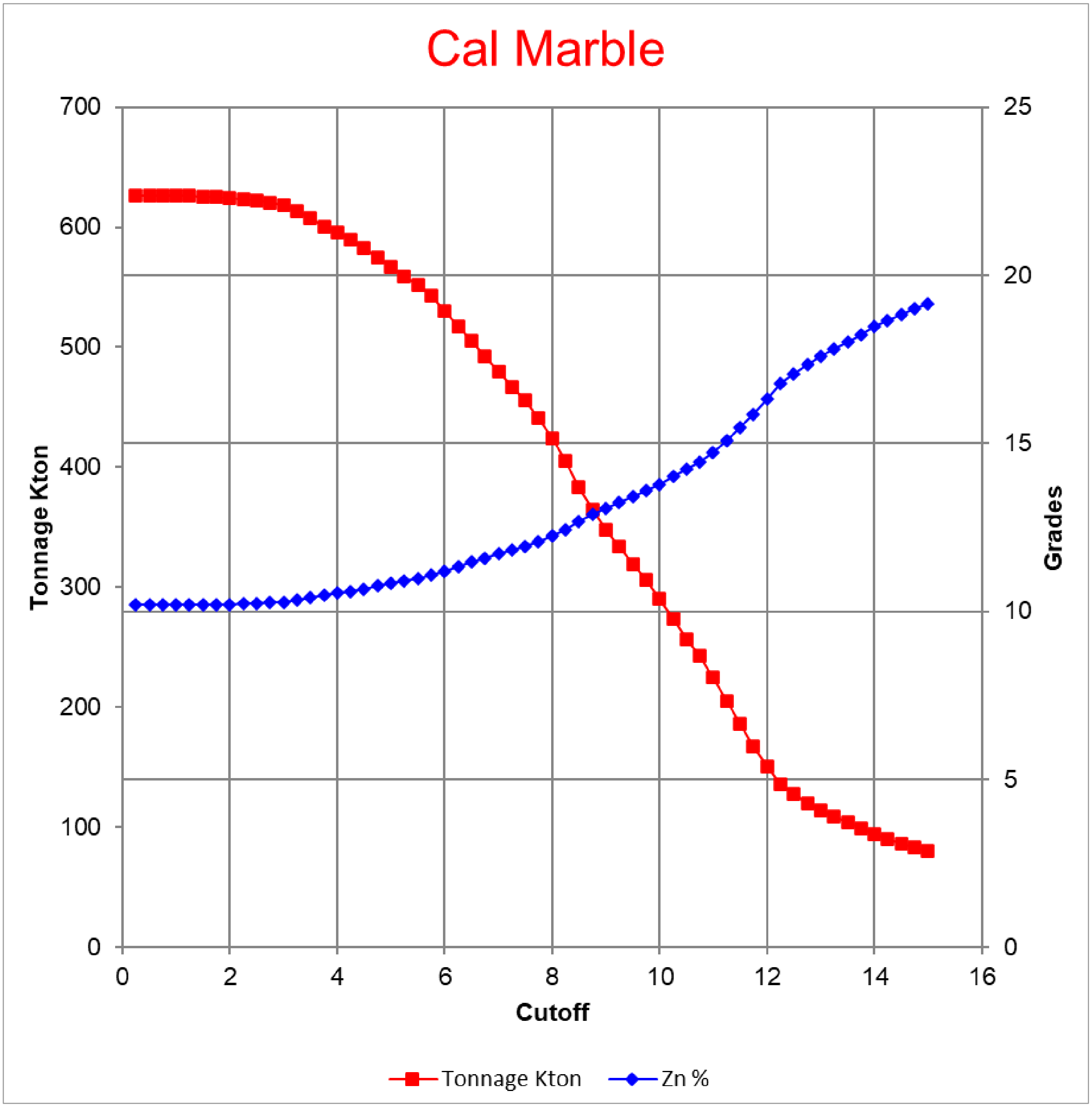
To document the sensitivity of the Mineral Resources to a variety of factors, SRK produced grade / tonnage (GT) graphs for each area as a function of movement in cut-off grade. This reflects the overall sensitivity to anything which would influence the disclosure of resources (independent of geological modelling or additional drilling factors) such as recovery, costs, pricing, etc. All tonnages and grades are reported as totals including Measured, Indicated, and Inferred for these analyses, so do not represent the Mineral Resources as reported, and are not compliant with NI 43-101. These graphs are shown for each area in Figure 14.26 through Figure 14.35. Due to the variances in grade and mineralization within each area, sensitivities to COG differ for each. In general, tons decrease precipitously above a 6% Zn COG.

Figure 14.26 American mine area GT graph



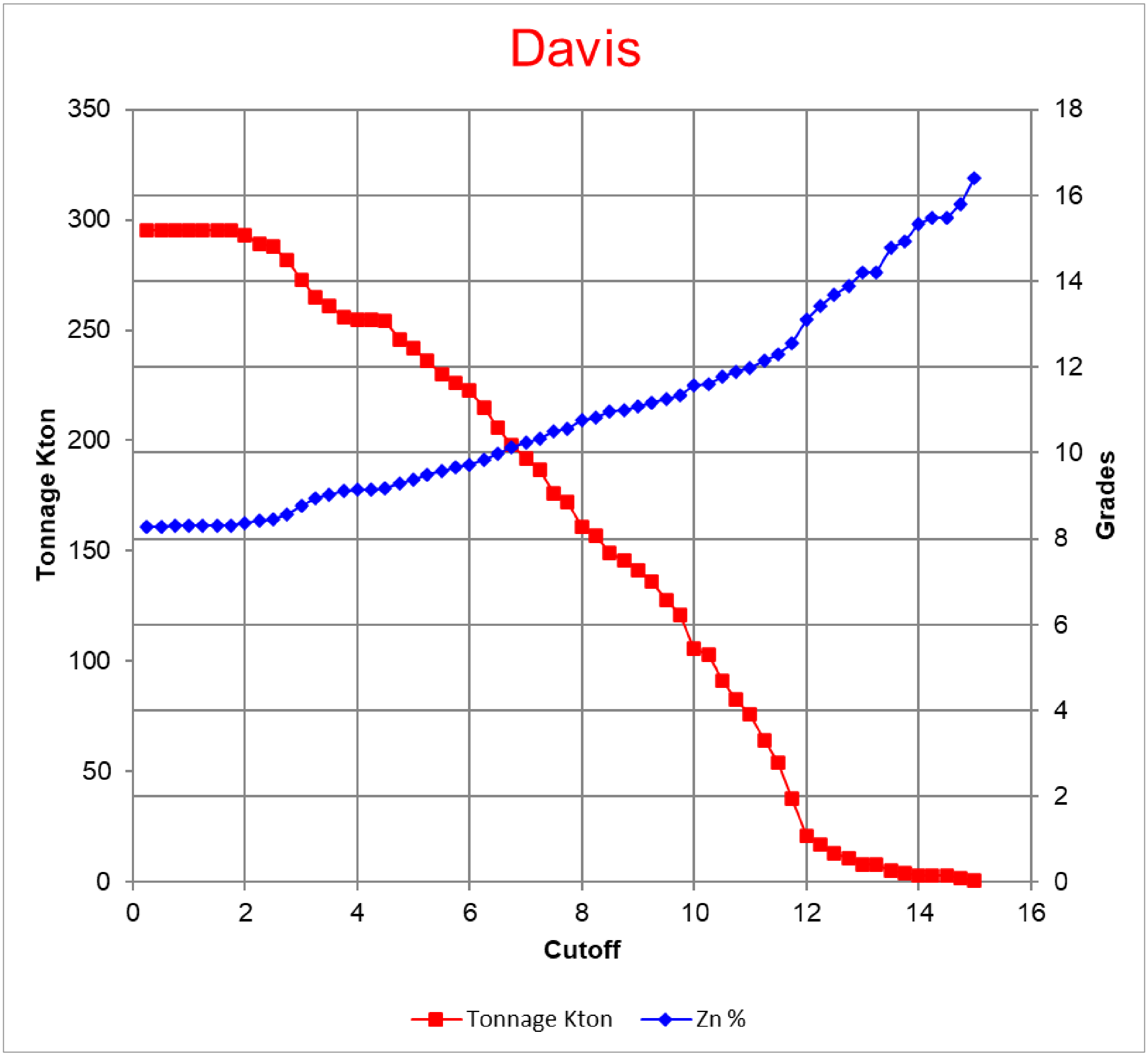
Source: SRK 2020.

Figure 14.27 Cal Marble mine area GT graph



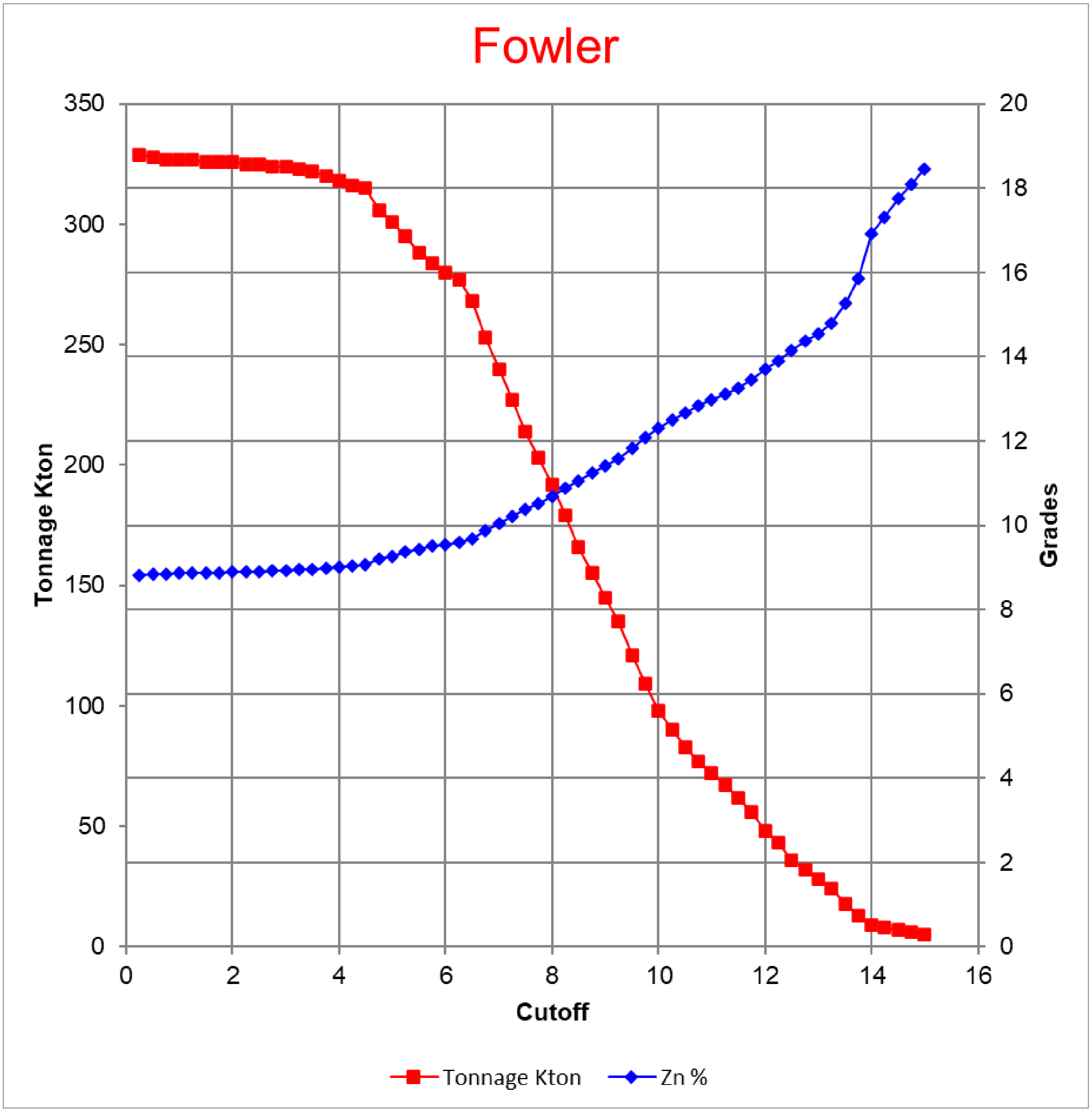
Source: SRK 2020.

Figure 14.28 Davis mine area GT graph



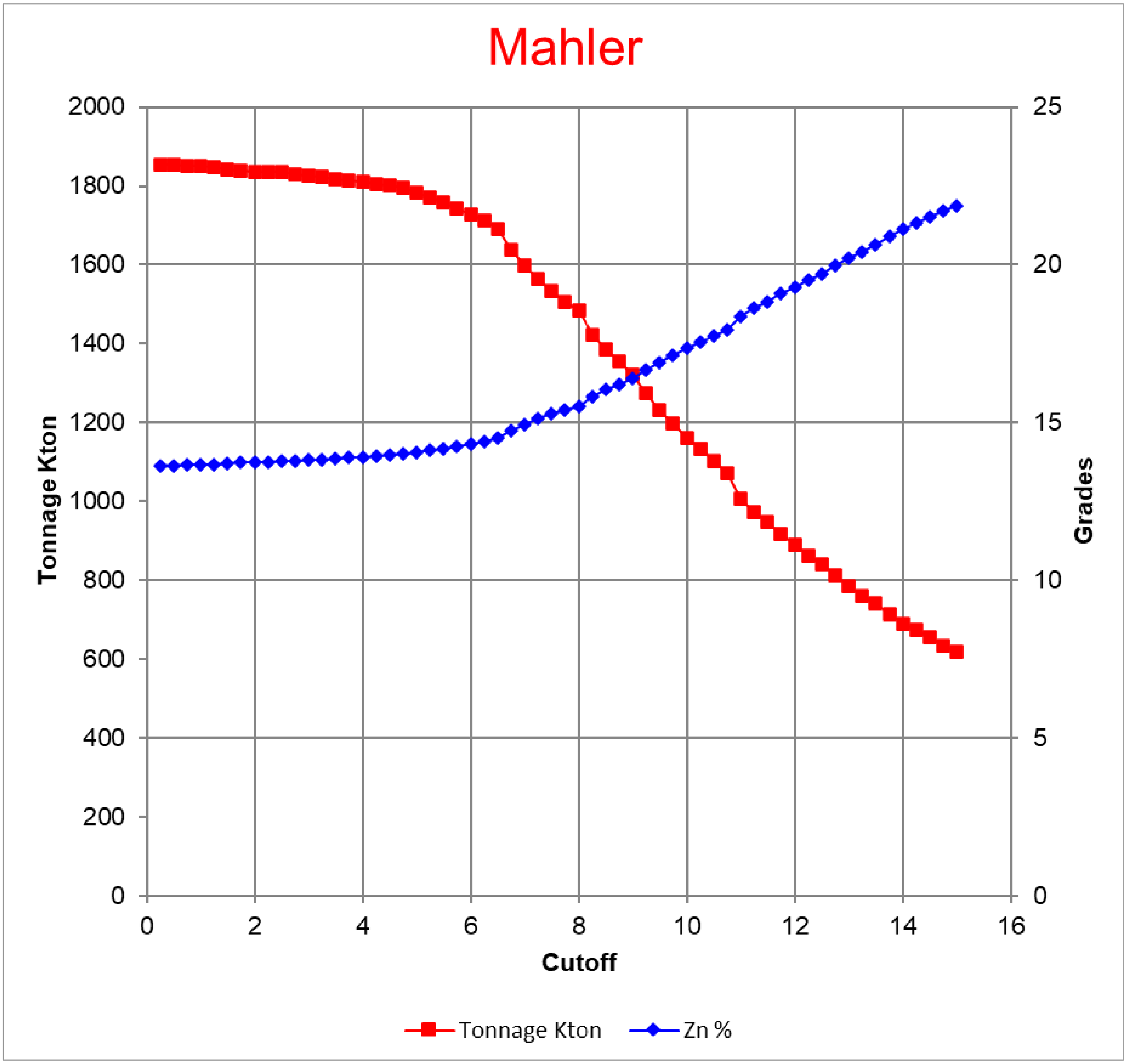
Source: SRK 2020.

Figure 14.29 Fowler mine area GT graph



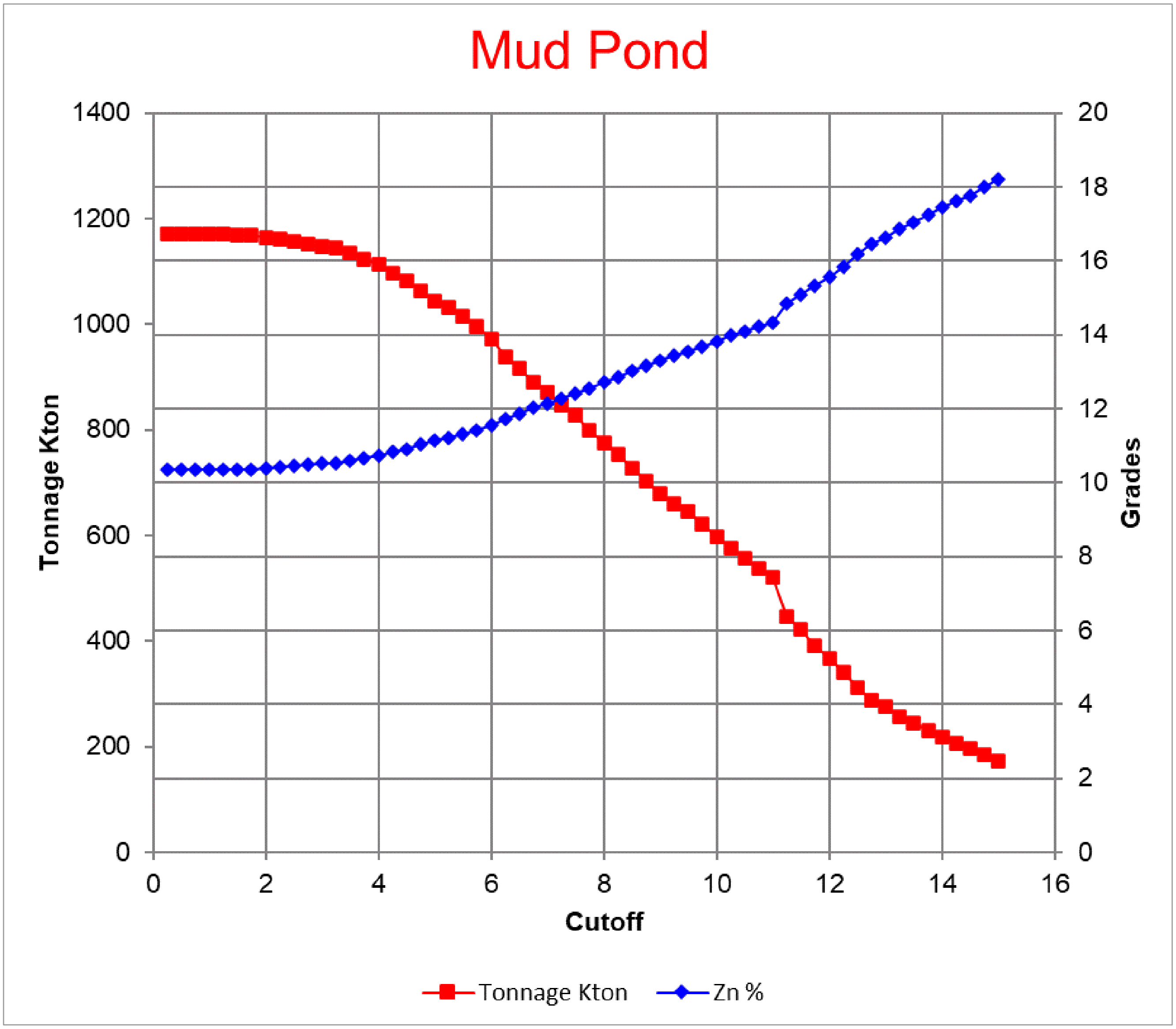
Source: SRK 2020.

Figure 14.30 Mahler mine area GT graph



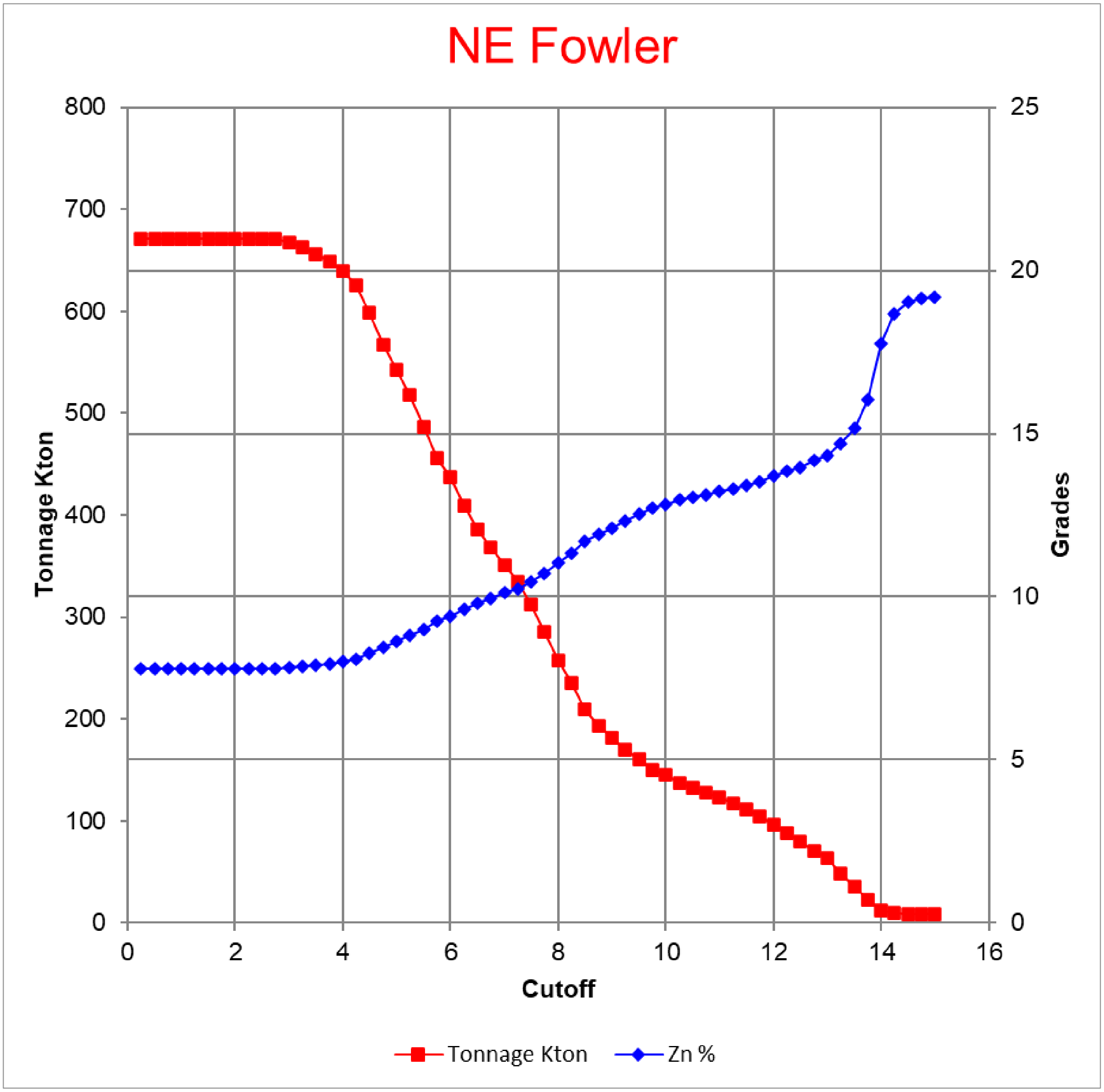
Source: SRK 2020.

Figure 14.31 Mud Pond mine area GT graph



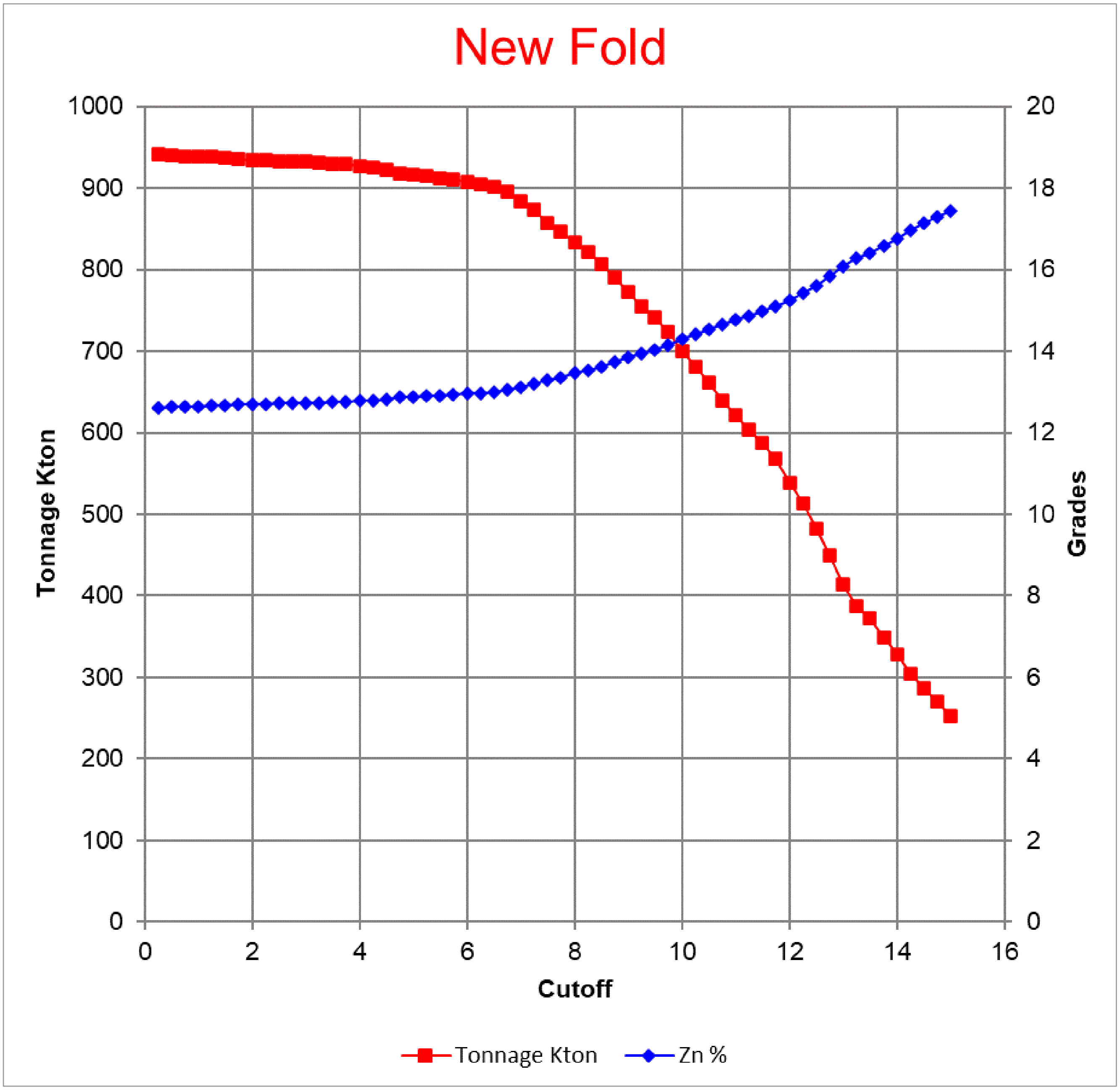
Source: SRK 2020.

Figure 14.32 Northeast Fowler mine area GT graph



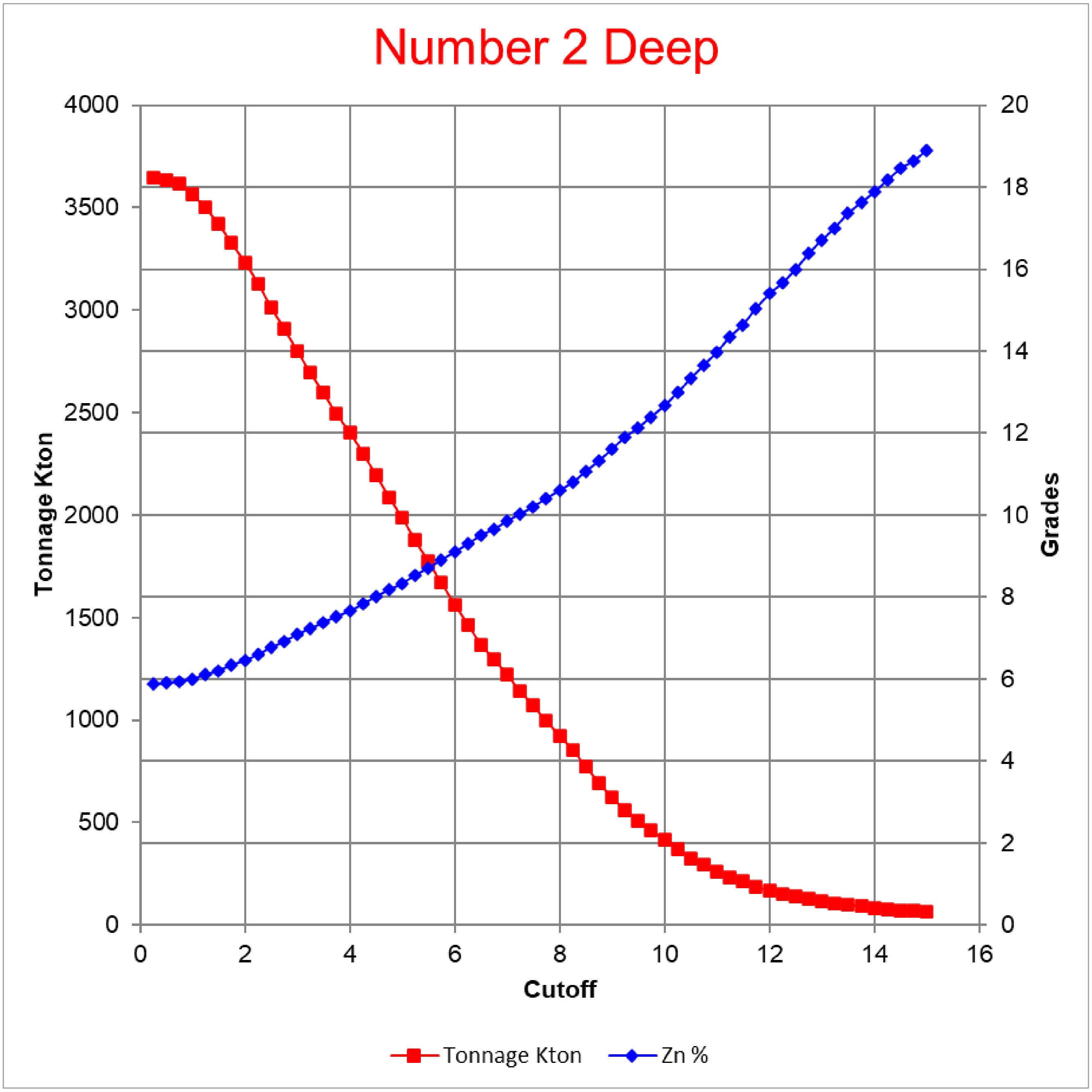
Source: SRK 2020.

Figure 14.33 New Fold mine area GT graph



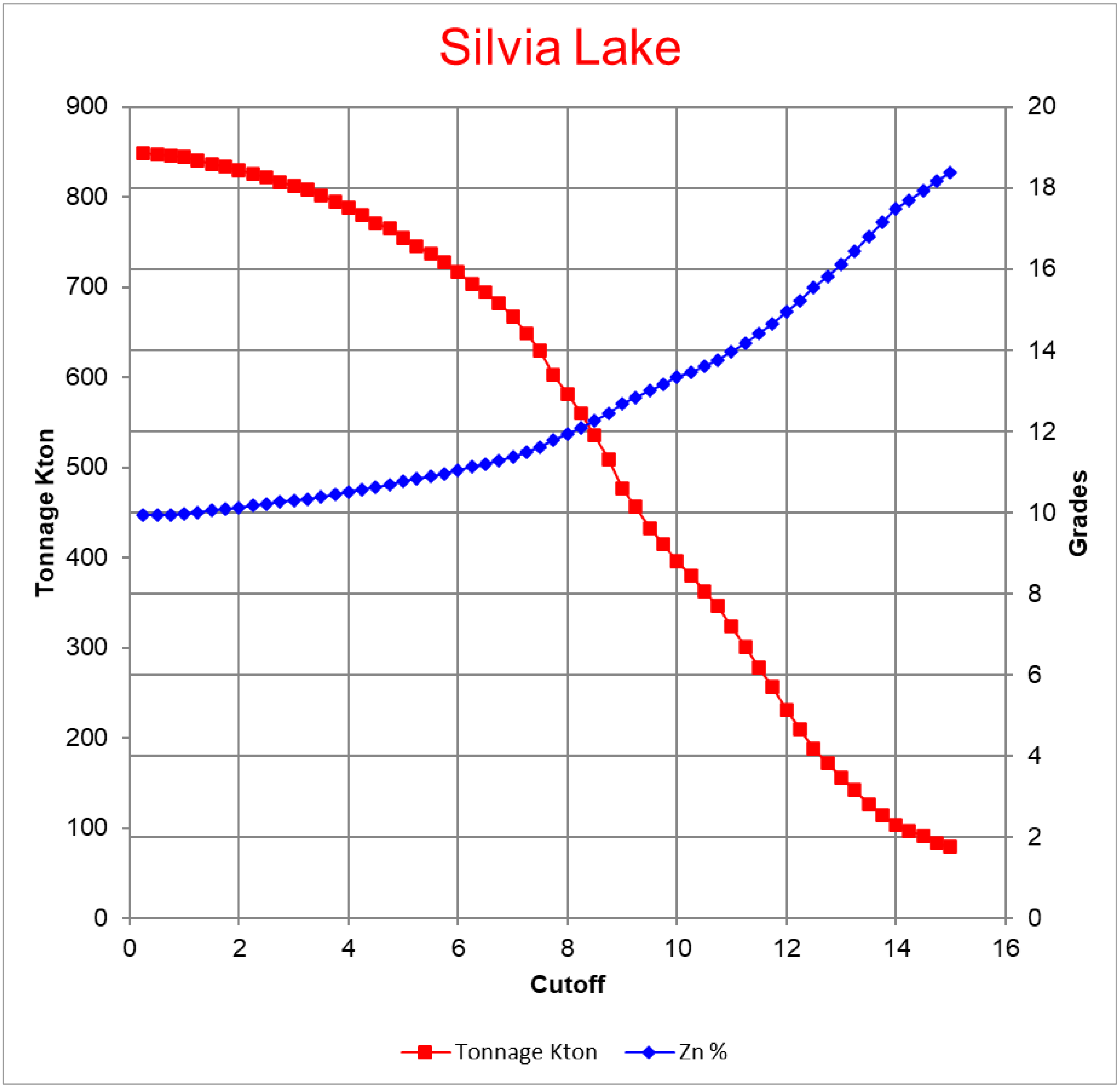
Source: SRK 2020.

Figure 14.34 Number 2 Deeps mine area GT graph



Source: SRK 2020.

Figure 14.35 Silvia Lake mine area GT graph

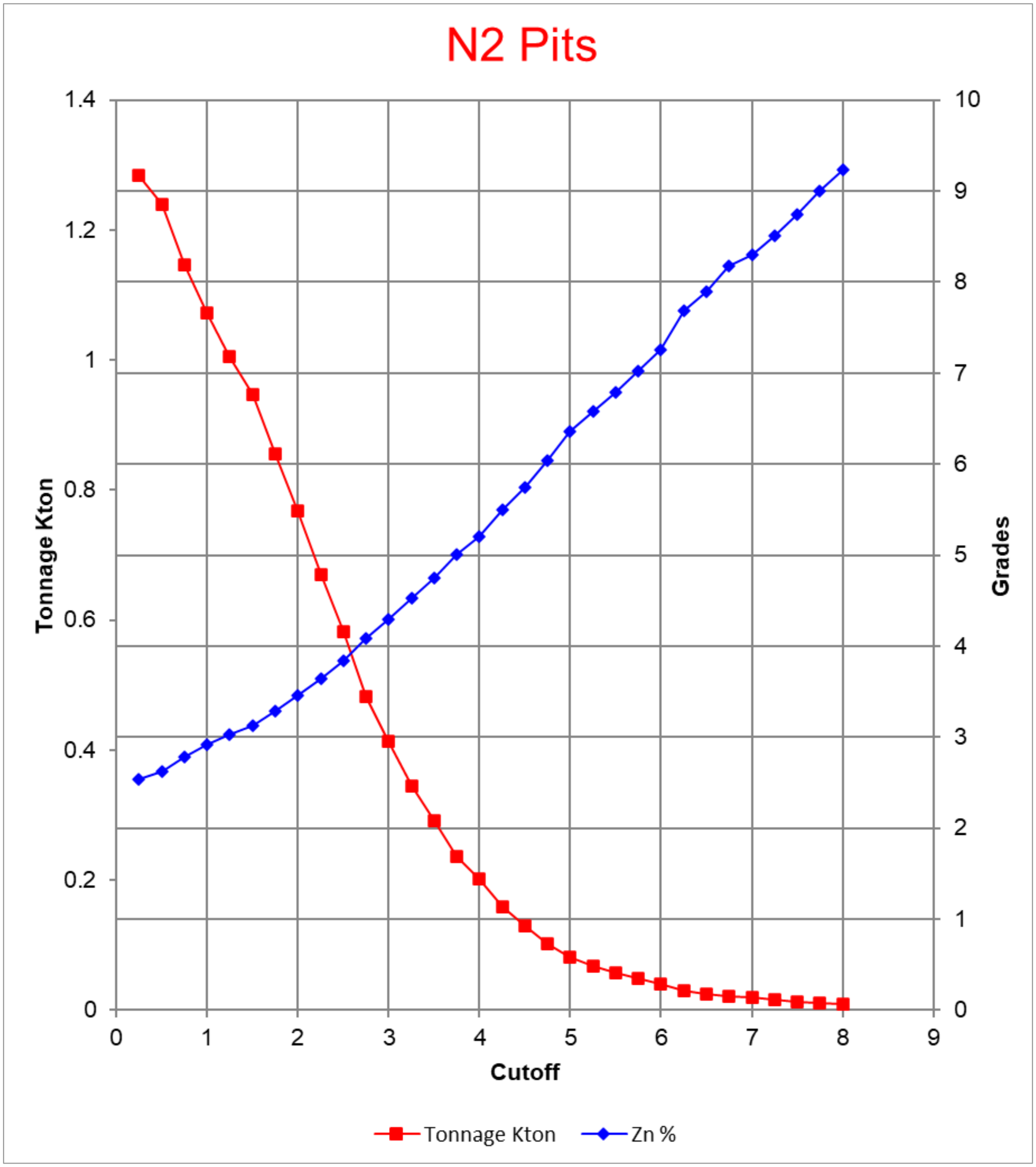


Source: SRK 2020.

14.12.2 Number 2 open pit area

The open pit resource sensitivities have been presented in GT graphs similar to the underground resources but are reported within an optimized pit shell as noted in Section 14.9.7. SRK notes that, as expected, the resources for the pit areas are more sensitive to COG than the underground resources, primarily due to the lower average grades.

Figure 14.36 Number 2 Open Pit mine area GT graphs



Source: SRK 2020.

14.13 Relevant factors

SRK is not aware of any other material factors which may influence the disclosure of Mineral Resources. The Number 2 open pit mining area is subject to permitting and environmental studies to proceed with active mining, but ESM has a history of compliance with all relevant regulatory requirements, has permits in hand for mining in these areas, and there has been previous production in this area from smaller open pits. The Number 4 underground areas are currently being mined.

15 Mineral Reserve estimates

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 Mineral 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.

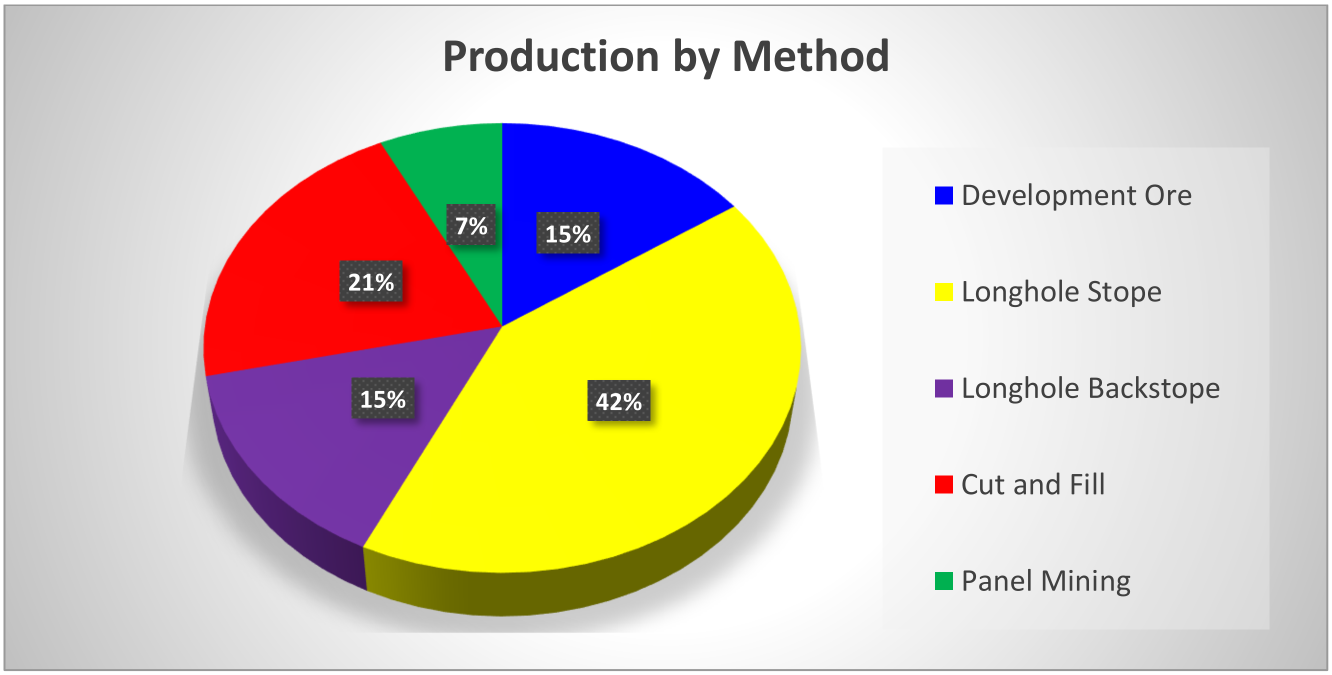
There are no Mineral Reserves reported for ESM.

16 Mining methods

16.1 Underground

The mine plan tons at the ESM deposit will be extracted using a combination of longitudinal retreat stoping (LGS), Cut and Fill (C&F), Panel Mining – Primary and Secondary (PAP & PAS), and development drifting underground mining methods with rock backfill. Longhole backstopes (BCK) are also used in the design where applicable. The proposed combined underground and open pit mine plan is expected to reach an initial target production rate of 1,200 t/d for 2021 and ramp up to 1,800 t/d in 2022. Open pit mining will be completed in Year three (2024). The overall mine life will be seven years. Figure 16.1 below outlines a summary of underground mining methods used at ESM.

Figure 16.1 Mine production by method



Source: Jackleg Consulting 2021.

The ESM deposit will be accessed from surface via the No. 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 proposed Mineral Inventory for the PEA Life of Mine (LOM) by mining method is shown in Table 16.1 below, which includes accessible remnants. The Mineral Inventory is based on the Mineral Resource stated as of February 2020 and is estimated at a 6% Zinc cut-off grade.

Table 16.1 Mineral inventory by mining method

Mining method	Diluted tons (kt)	Percent of LOM plan
Development mineralization	407.7	15
Longhole stope	1,103.9	42
Longhole backstope	383.2	14
Cut and fill	561.2	21
Panel mining	194.4	7
Total	**2,650.4**	**100**

Note: Totals may not compute exactly due to rounding.
Source: Jackleg Consulting 2020.

16.1.1 Deposit characteristics

There are four zinc-rich mineralized zones included in the LOM plan:

- Upper Mahler
- Lower Mahler
- New Fold
- Mud Pond

Vulcan Stope Optimizer shapes and development designs were created for the remaining mining zones:

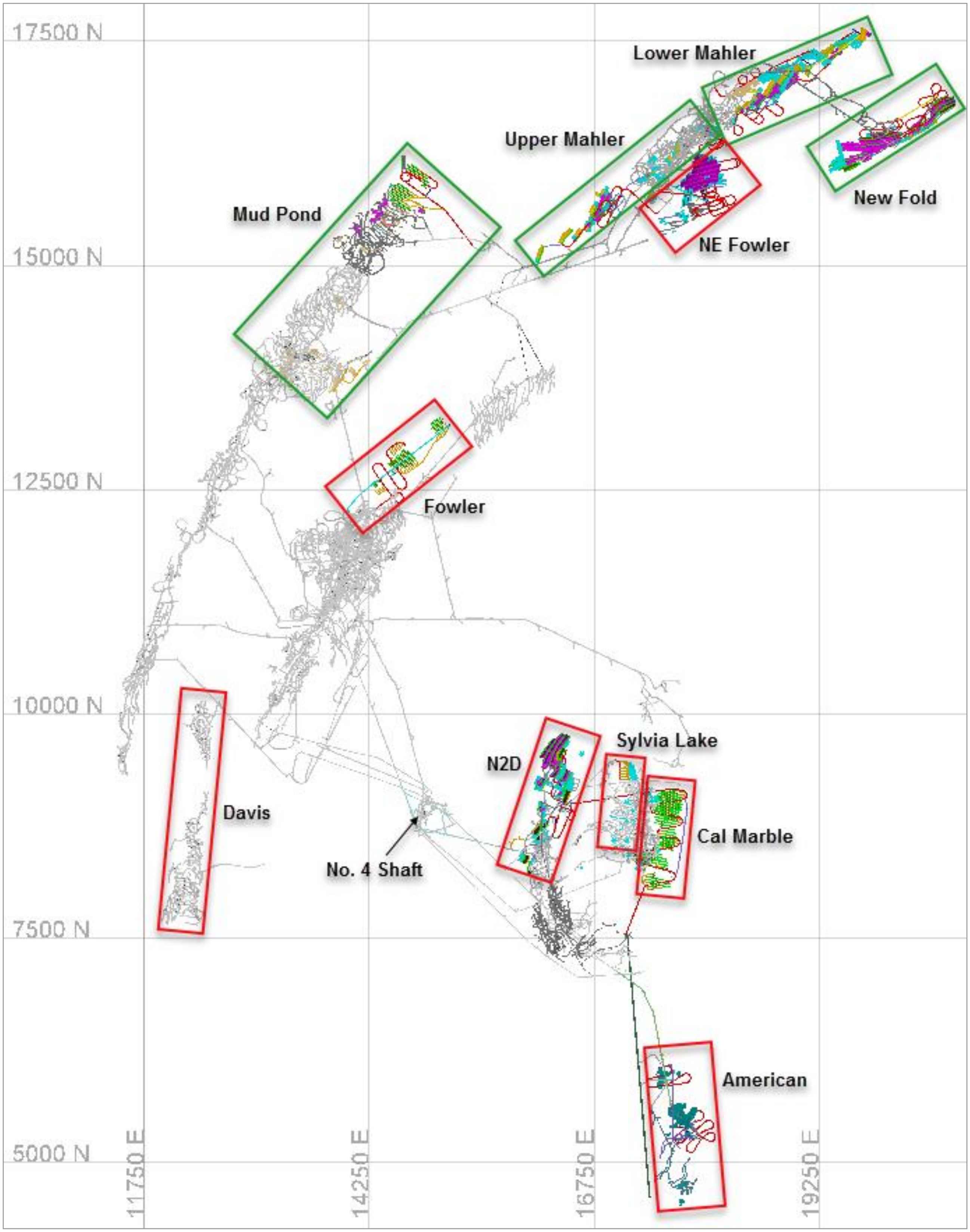
- American
- Cal Marble
- Davis
- Fowler
- North East Fowler
- N2D
- Sylvia Lake

These designs were economically analyzed and due to low NPV values in those zones, they were excluded from the LOM plan. Increases in deposit size, grade or in metal prices may make these deposits economically viable in the future. Figure 16.2 depicts the four zinc-rich mineralized zones included in the LOM plan, outlined in green, along with those zones that were excluded, outlined in red.

Deposits are distributed throughout the property within a 3,800 ft radius and between 2,500 ft and 4,200 ft below surface. Mineralized zones generally strike NE-SW with length ranging from 500 ft to 1,800 ft, width from 50 ft to 350 ft and dip 20° to 60°. At a local mining scale, extreme variations in the dip and orientation are not uncommon.

All zones are connected to existing infrastructure underground and many have not been fully delineated and remain open for further exploration and resource expansion.

Figure 16.2 Selected mining zones (green outline) for LOM plan



Source: Jackleg Consulting 2020.

16.1.2 Mineral Inventory within the PEA mine plan – estimation process

To determine the Mineral Inventory at ESM, the following process was utilized:

- Analyze Mineral Resource model for geometric properties, such as mineralized zone width, depth, length, dip, 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.
- Develop a mine plan around the economically viable production stopes and run economic models on various production scenarios.

It is to be noted that the current Mineral Resource model developed by SRK was used for mine planning purposes.

16.1.3 Mining method selection

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

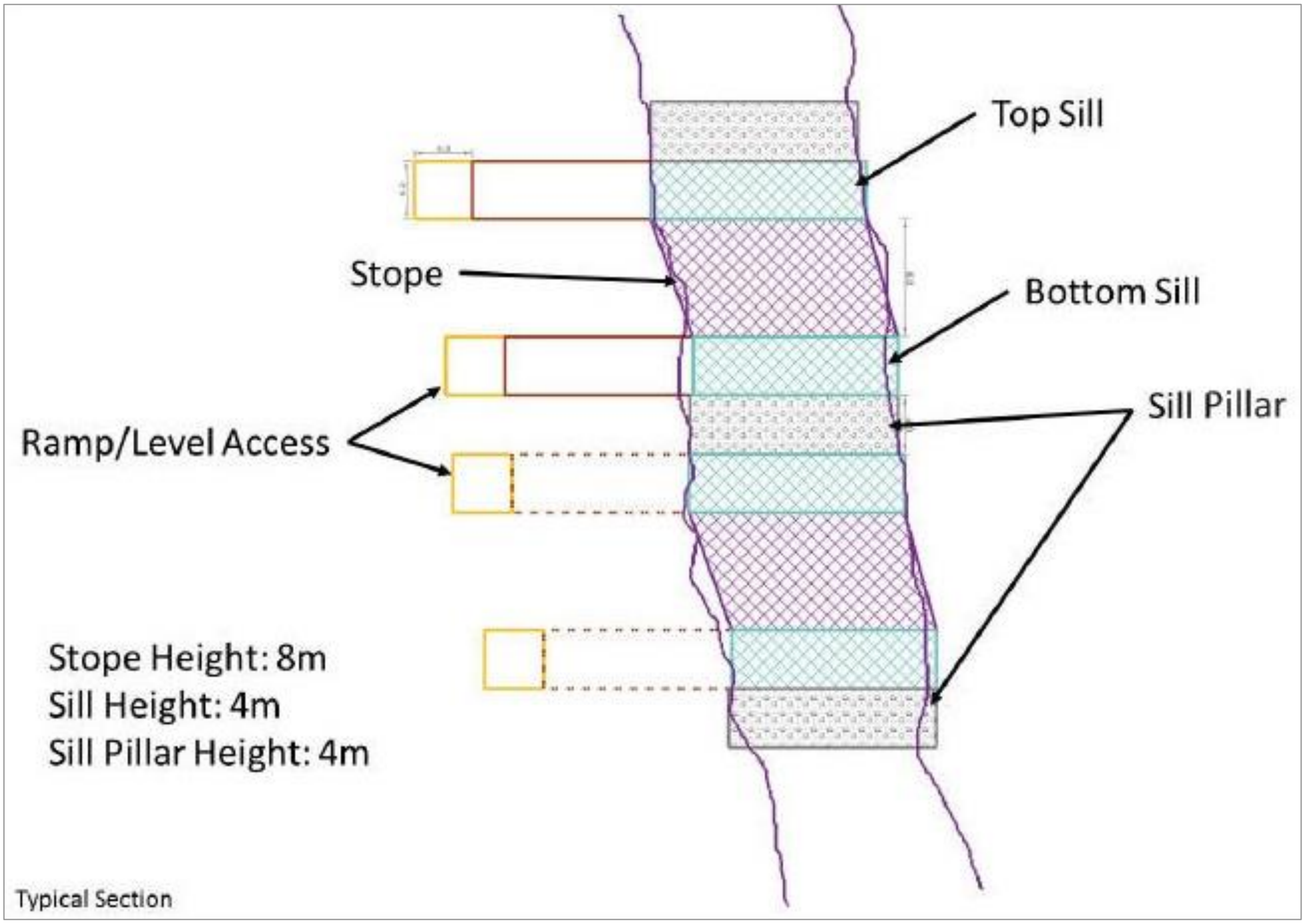
LGS is being used at ESM as the principal mining method due to its high productivity, low cost, selectivity, and successful history of application for deposits of this nature. Alternatively, C&F and modified panel mining with primary and secondary cuts will be used where conditions are not suitable for longitudinal stoping. BCK are also used in the design where applicable to minimize development costs, such as at the deposit extents or areas where previously mined and unsafe areas prevent top cut access.

LGS 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 LGS at ESM, a top and bottom drift delineate the stope and a dedicated longhole drilling machine drills blastholes between the two drifts.

The drillholes are loaded with explosives and the stope is blasted, with broken material falling to the bottom drift for extraction. In longitudinal retreat stopes, remote controlled load haul dump machines (LHD) are required to remove the blasted material from the stope.

One of the limitations with LGS 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 LGS is the stopes must remain open long enough to remove the mineralized material and typically are then filled with an engineered backfill material (where support pillars are not used). This mine plan assumes no backfill plant will be available, so a 10 ft sill pillar is left between the levels when longitudinal stoping is used. These limitations generally restrict level spacing at ESM to 60 ft. This does not apply to backstopes as there is no top cut (or level above), so they are designed at a 60 ft height. A typical cross section of the LGS stope with a sill pillar is shown in Figure 16.3.

Figure 16.3 Typical longitudinal stope with sill pillar



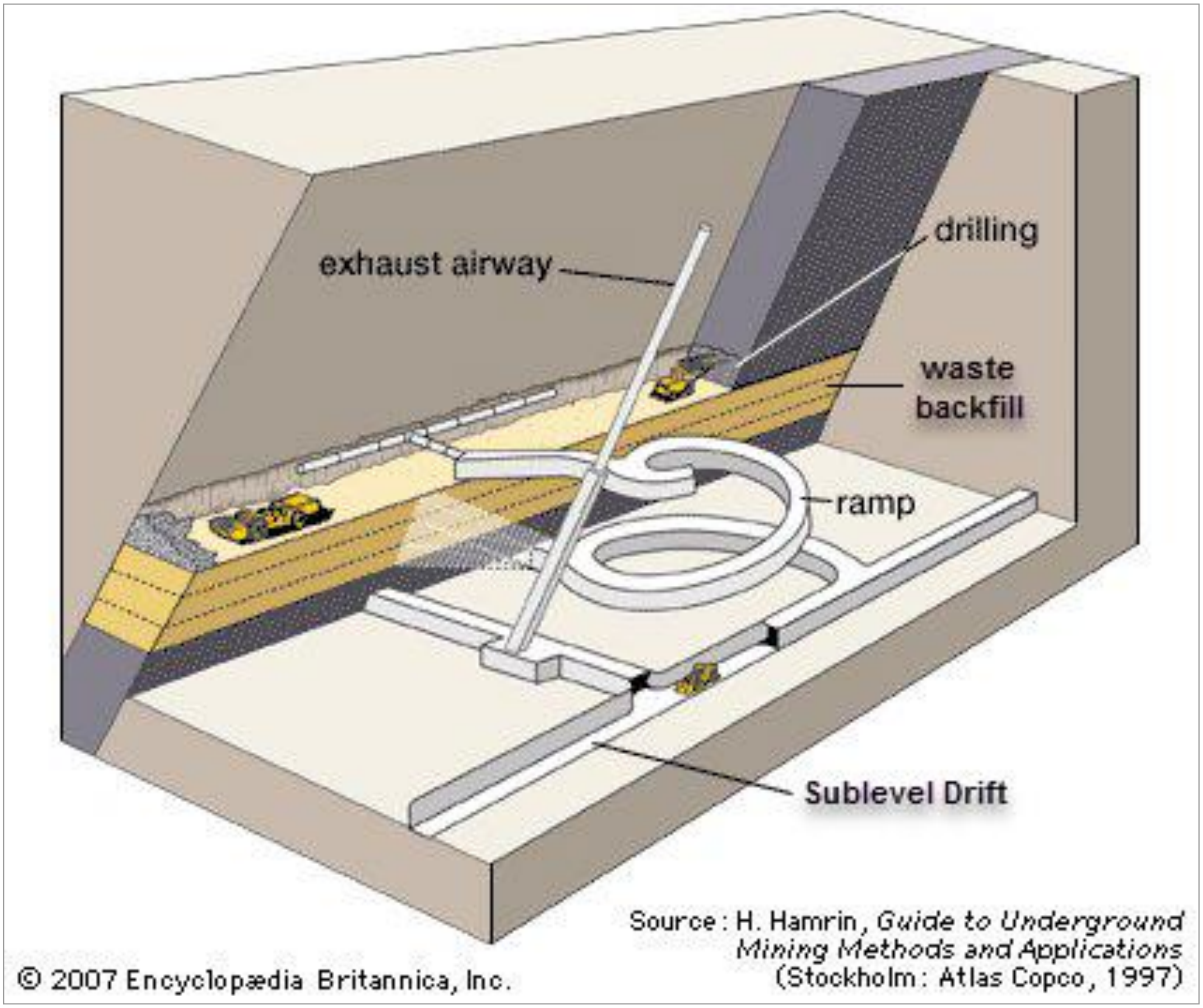
Source: Jackleg Consulting 2020.

LGS is the primary mining method at ESM, whereby a central sub-level is driven along strike through the mineralization to the end of the stope shapes, to provide access for drill and mucking equipment. This method is beneficial for minimizing waste development as the bulk of mining activities stay within the mineralized zones. One constraint of LGS is that production is limited to one stope at a time as the level is mined in retreat to the stope access drives that run perpendicular to the strike.

Longitudinal retreat stoping is used in Mahler, New Fold, and Mud Pond mineralized zones with C&F and Panel Mining accessing the remaining mineralization that does not fit LGS design criteria.

C&F mining is being used at ESM for areas of the deposit which fall below a practical dip for LGS, or where more selective mining is 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. As there will not be a backfill plant, the un-cemented fill will be waste rock from development headings. With the abundance of inactive areas, storage of waste material for C&F mining will not be an issue. This method is well suited for narrow, gently dipping zones. A typical layout for C&F is shown in Figure 16.4.

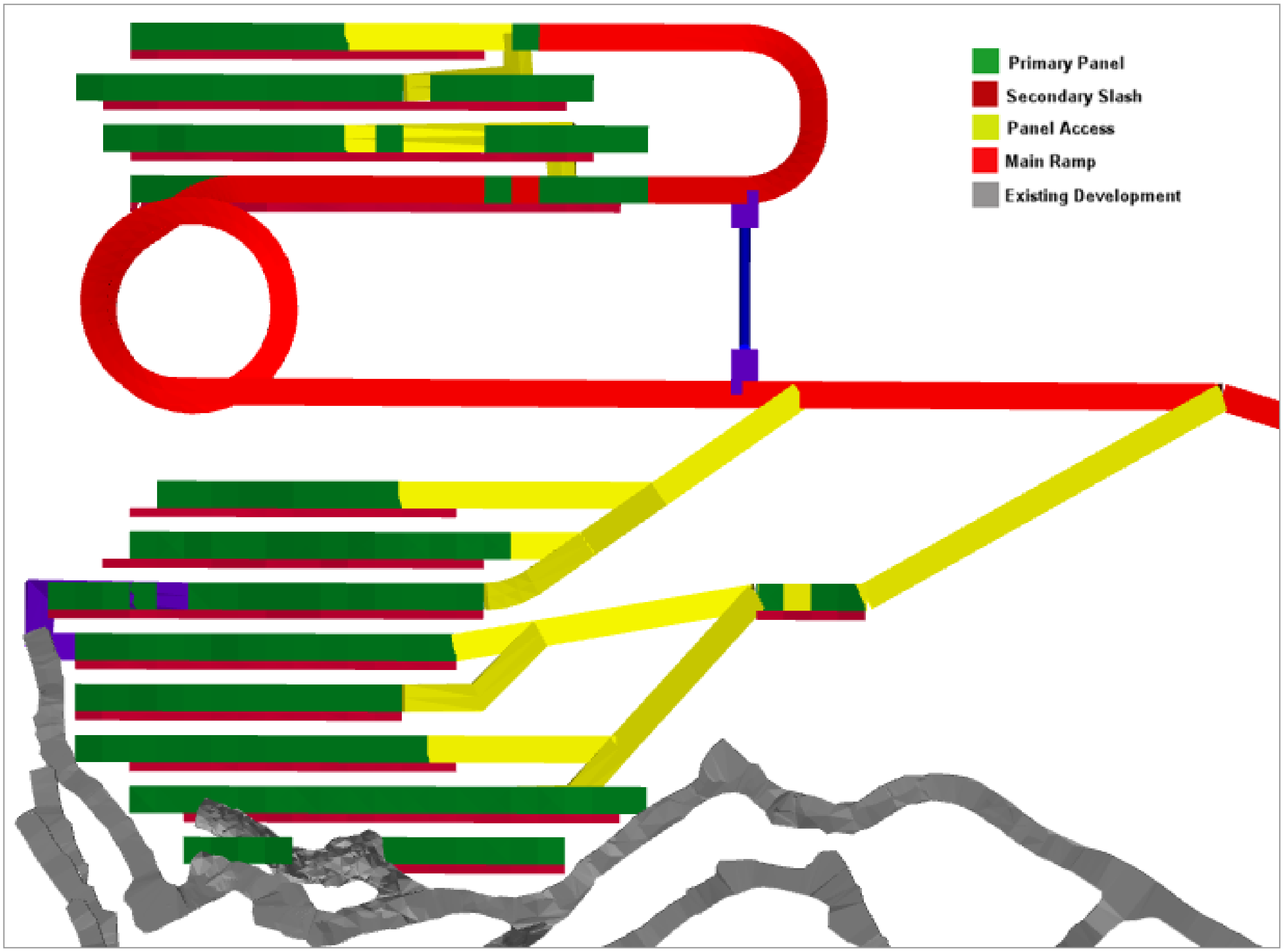
Figure 16.4 Cut and fill (typical layout)



Source: Atlas Copco 1997.

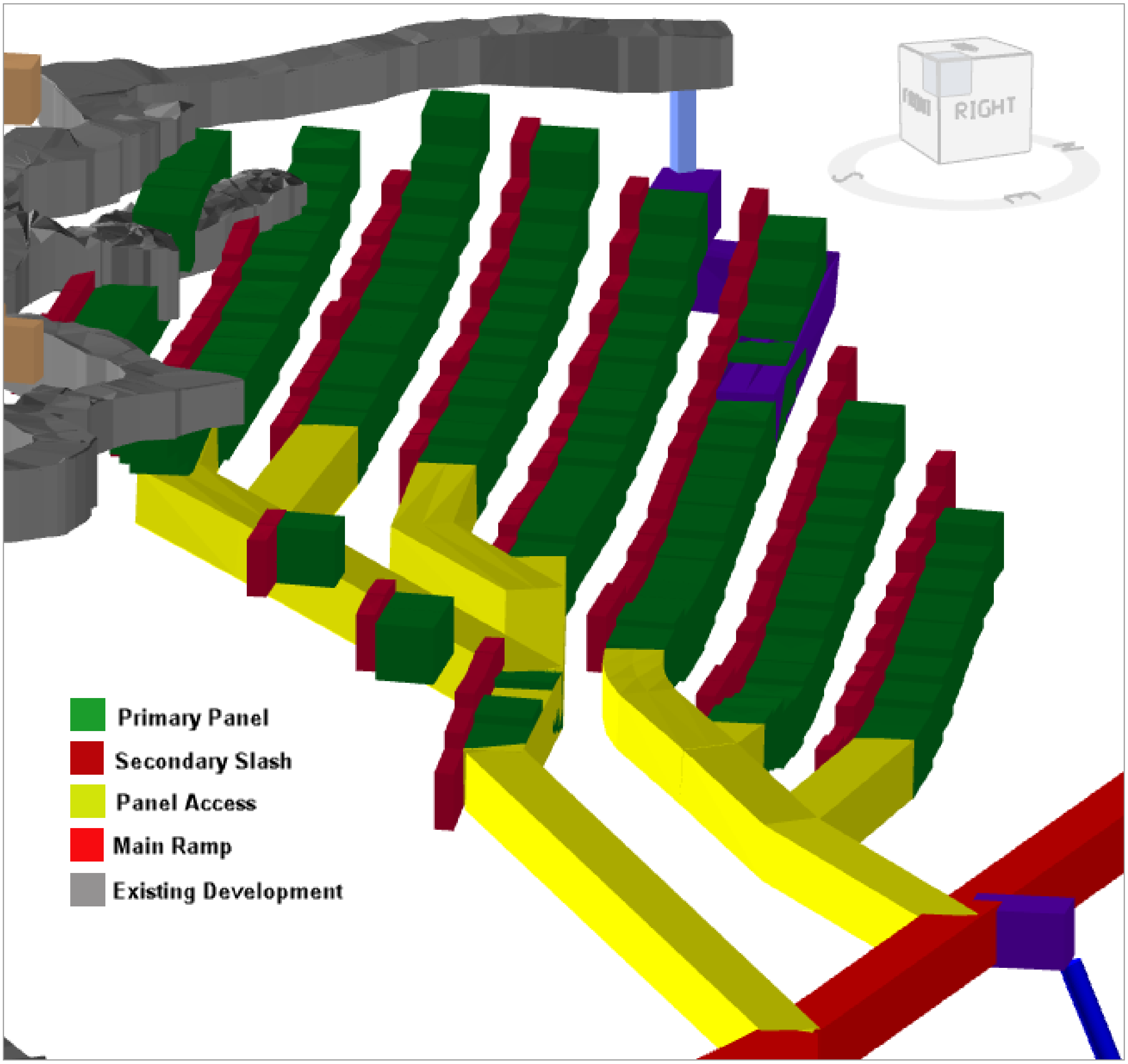
Panel Mining divides a mineralized area into three repeating sections, Primary drives, Secondary slash, and pillars. The Primary drift is mined to the deposit extents and a Secondary slash is mined in a retreating fashion along the up-side of the dip, with a pillar left between the panels. This method works for mineralization with a dip that is too steep for C&F. Figure 16.5 and Figure 16.6 show a plan and isometric view, respectively, of Panel mining.

Figure 16.5 Plan view of panel mining



Source: Jackleg Consultants 2020.

Figure 16.6 Isometric view of panel mining

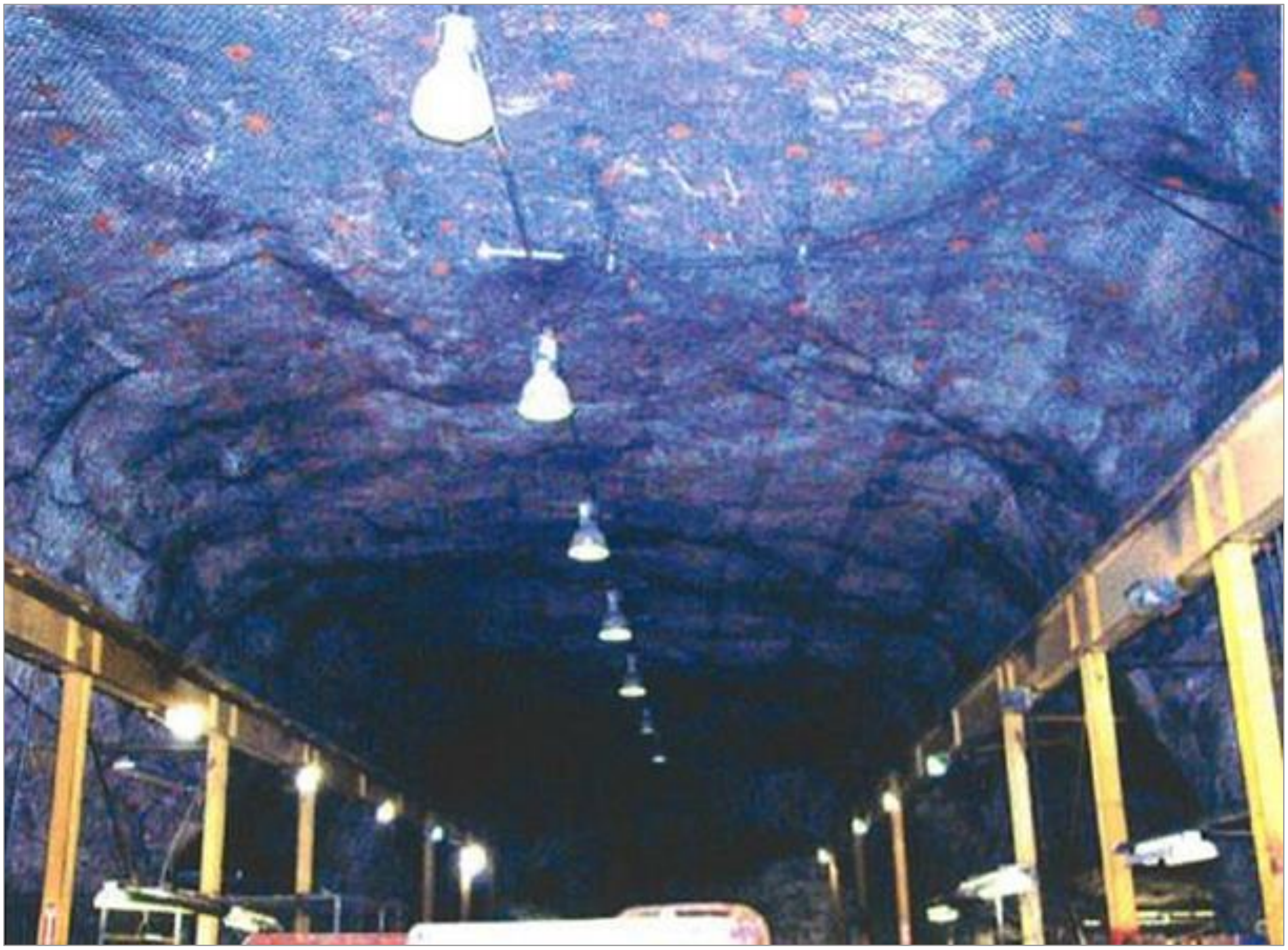


Source: Jackleg Consultants 2020.

16.1.4 Geotechnical parameters

Ground conditions at ESM are considered very good and estimated to have a Rock Mass Rating (RMR) of 80 or greater (Bieniawski 1989). The underground workshop 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 split sets, dywidag resin rebar, and woven chain link mesh. There is no visible loose rock in the mesh or opening joints (Figure 16.7).

Figure 16.7 2,500 Level workshop ground conditions



Source: Itasca 2005.

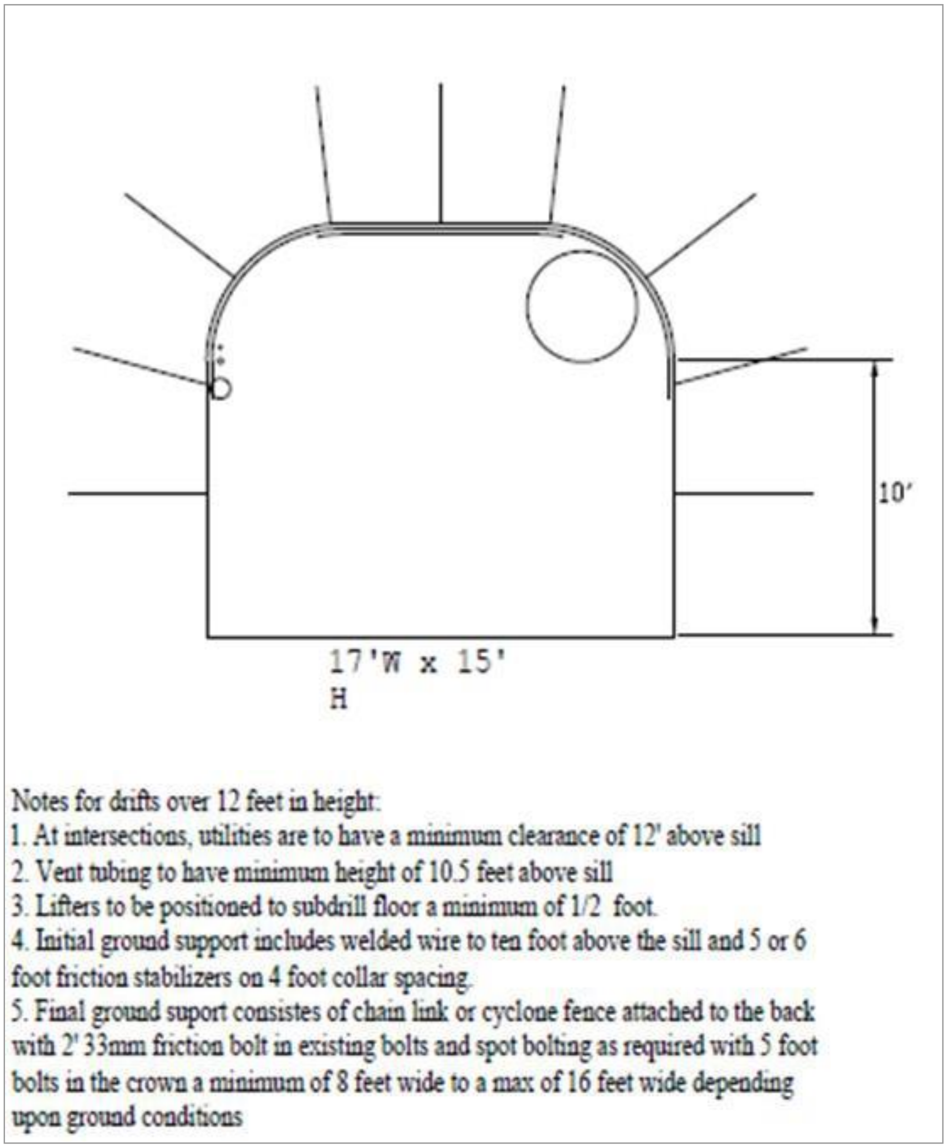
Prior to mine shut down in 2001, the underground workings were supported on an as needed basis using minimal support. Pattern bolting and mesh application was not used, as evident when traveling through historical workings. Fall of ground (FOG) accidents totalled 50 between the years 1994 and 2000; 46 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. All 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 to 6 tons. Annual pull testing of ground support continued once the mine restarted in 2018.

The ground support minimum requirements currently in use at ESM are deemed appropriate for continued use in future lateral development. Figure 16.8 below outlines support requirements for primary heading types used in the LOM design.

Figure 16.8 Minimum ground support profiles



Source: HBMS 2006.

16.1.5 Stope design parameters

Vulcan Stope Optimizer© software was used to create all the mineable stope shapes in the LOM design. Stope design criteria are summarized in Table 16.2.

Table 16.2 Production stope design criteria

Mine method	Minimum stope width (ft)	Stope height (ft)	Stope length (ft)	Dip (°)
Cut and Fill	13	15	N/A	50-90
Panel Primary	15	15	N/A	N/A
Panel Secondary	5	15	N/A	N/A
Longitudinal Stope	10	50	Max 150	50-90
Backstope	10	60	Max 150	50-90

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 LGS stope dimensions.

LGS stope dimensions are variable to accommodate the geometry of the resource. A minimum 10 ft true width was used for stope design, along with a minimum 50° footwall and maximum 50° hangingwall. Level spacing of stopes was set to 60 ft. In areas where there are multiple levels, a 10 ft sill pillar is included in the 60 ft level heights. Backstopes were designed to the full 60 ft sublevel height.

16.1.6 Mine dilution and 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 LGS and BCK stopes, two sources of dilution were considered. Sloughing was estimated to be 2.0 ft on both the hangingwall and footwall of LGS stopes. For C&F, planned over break dilution of 0.5 ft was applied to both walls. A dilution grade of 0% Zn was assumed for all dilution. Planned overbreak dilution parameters are summarized in Table 16.3. An additional 5% of unplanned dilution at a grade 0% Zn is also included in all mining methods.

Table 16.3 Overbreak dilution parameters

Typical profiles	Units	Cut and fill	Panel - Primary	Panel - Secondary	LGS stope w/crown pillar	Backstope
Height	ft	15	15.0	15.0	50.0	60
Width (minimum)	ft	13	15.0	5	10.0	10
Footwall overbreak	ft	0.50	0.50	0	2	2
Hangingwall overbreak	ft	0.50	0	.50	2	2
Unplanned dilution	%	5	5	5	5	5

Mine recovery was calculated under the following mine assumptions:

- C&F and waste development passing incremental cut-off, assume 95% mine recovery after losses.
- Longitudinal retreat and backstopes assume 95% mine recovery.
- Panel mining assumes 71% mine recovery after losses from pillars left behind.

16.1.7 Cut-off grade criteria

Zinc cut-off grade calculation criteria are summarized in Table 16.4.

Table 16.4 Cut-off grade parameters

Parameter	Unit	Value
Zn price	$/lb	1.15
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: ESM 2020.

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 for all mining zones, except N2D. The cut-off grade for N2D was increased to 7% Zn as the zone is, in general, comprised of lower grade material, so raising the grade slightly ensures that a more economic material will be mined and processed. Approximately 10% 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.

16.1.8 Mine plan tons and grade

All stopes were designed based on the applicable stope shapes, geological boundaries, and grade extents, ensuring the final stope shapes meet cut-off grade criteria. Table 16.5, Table 16.6, and Table 16.1 outline the diluted and recoverable mine plan tons used for mine planning purposes by zone, resource class, and mining method, respectively.

Table 16.5 Mine plan tons contained in mine plan

Zone	Diluted tons (kt)	Diluted Zn grade (%)
Mud Pond	250.1	7.8
Mahler – Upper	182.4	8.1
Mahler – Lower	673.9	9.7
New Fold	794.2	9.2
N2D	749.9	7.0
Total	**2,650.4**	**8.5**

Table 16.6 Mine plan tons by Mineral Resource class

Mineral Resource class	Diluted tons (kt)	Diluted Zn grade (%)
Measured	156.1	13.2
Indicated	497.9	12.1
Inferred	1,092.8	13.2
Unclassified	903.6	0
Total	**2,650.4**	**8.5**

16.1.9 Mine design criteria

16.1.9.1 Mine access

The ESM deposit consists of a mining resource extending nearly 4,200 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 mi 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 x 10 ft tall to over 17 ft wide x 15 ft tall. The maximum gradient of the existing development is 20%.

ESM is situated on moderately flat lying terrain.

Where not already completed, existing workings will be rehabilitated to ensure a safe working environment. When accessing new deposits, a ramp will be driven at a maximum grade of 15% at a 15 ft by 15 ft profile. Mineralized zone development will have a 13 ft by 13 ft profile.

16.1.10 Production rate selection

The ESM mine plan has been sized to ramp up to 1,400 t/d in Year 1 and then to a sustained maximum of 1,500 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 Deswik CAD and Sched© software. The scheduling rates used are shown in Table 16.7.

Table 16.7 Scheduling rates used for mine scheduling

Scheduling rates		
Lateral development	**Unit**	**Rate**
Ramp	ft/day	4.5
Auxiliary	ft/day	4.5
Longitudinal access - waste	ft/day	4.5
Longitudinal sill - mineralization	ft/day	4.5
Cut and fill access - waste	ft/day	4.5
Panel access - waste	ft/day	4.5
Vertical		
Drop raise	ft/day	5
Raiseboring	ft/day	9
Stoping		
Longitudinal retreat	t/day	250
Backstope – longhole	t/day	250
Cut and fill	t/day	150
Panel - Primary	t/day	250
Panel - Secondary	t/day	50

16.1.11 Production sequencing

Production in LGS stoping zones is mined with a bottom-up sequence. Where necessary in-situ sill pillars are left to separate mining horizons.

C&F zones are mined in a bottom-up fashion from a main access drift with loose development waste rock used as backfill. From the main ramp, a drift accesses the production area with a +/-15%

attack ramp. Once the production drift is mined out on that level, it is 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.

PAP and PAS zones can be mined from a top-down or bottom-up fashion depending on the direction of development in the zone. Access drifts are driven from the main ramp to the start of each primary panel drift. A primary drift is driven at full size to the end of the deposit. A secondary slash in the hangingwall is then mined in a retreating fashion back to the panel access drift.

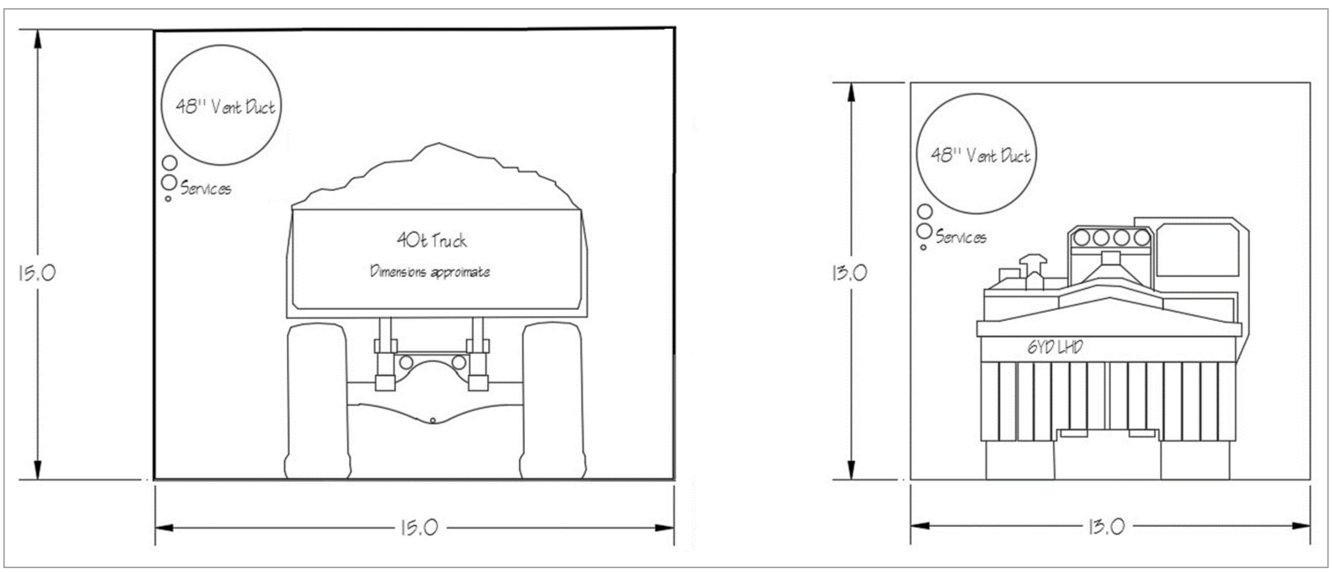
16.1.12 Underground mine development

16.1.12.1 Lateral development

Ramps are driven at a 15 ft x 15 ft square profile to accommodate fully loaded 40 t haul trucks and 48" round vent ducting. Cross-cuts and sub-level development are driven at a 13 ft x 13 ft square profile to accommodate remote LHD entry.

Figure 16.9 depicts a typical development ramp and cross-cut cross-sections.

Figure 16.9 Typical development cross-sections



Source: Jackleg Consulting 2020.

16.1.12.2 Vertical development

Ventilation raises of varying lengths are used in the PEA mine design. For shorter, level to level connections, a 5 ft x 5 ft drop raise is established to provide fresh air for each of the mining zones. For longer raises that cannot be mined with a drop raise, a 6 ft diameter raisebore will be used. Drop raises can be mined by ESM and all raisebore raises will be driven with the use of contractors.

16.1.13 Unit operations

16.1.13.1 Drilling

Development headings are 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 mineralization. The advance per round is assumed to be 10 ft for 12 ft steel. One jumbo has the capacity to drill between two and three rounds per shift, however, cycle productivities are limited to 1.5 rounds per day per jumbo in the schedule.

Production drilling for the longhole stopes is performed by longhole drills. Blastholes with a 3.5” diameter are drilled in a fan pattern from the overcut to the undercut.

16.1.13.2 Blasting

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

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

16.1.13.3 Ground support

After mucking and scaling is complete, ground support is 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 longhole stope firing with an average pattern of six bolts per ring and 10 feet between rings.

16.1.13.4 Mucking

Blasted material from development headings is 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 material-pass. Broken material from longhole stopes is mucked by remote control LHD.

16.1.13.5 Hauling

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

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

16.1.13.6 Backfill

Only the C&F mining method requires the placement of waste rock as backfill. No cemented backfill is currently planned at ESM.

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

16.1.14 Mine services

16.1.14.1 Mine ventilation

In 2016, the ESM ventilation network was modelled using Ventsim® Visual software by Practical Mining LLC (Practical Mining). The ventilation simulation model is routinely calibrated, verified and updated as mine activity changes.

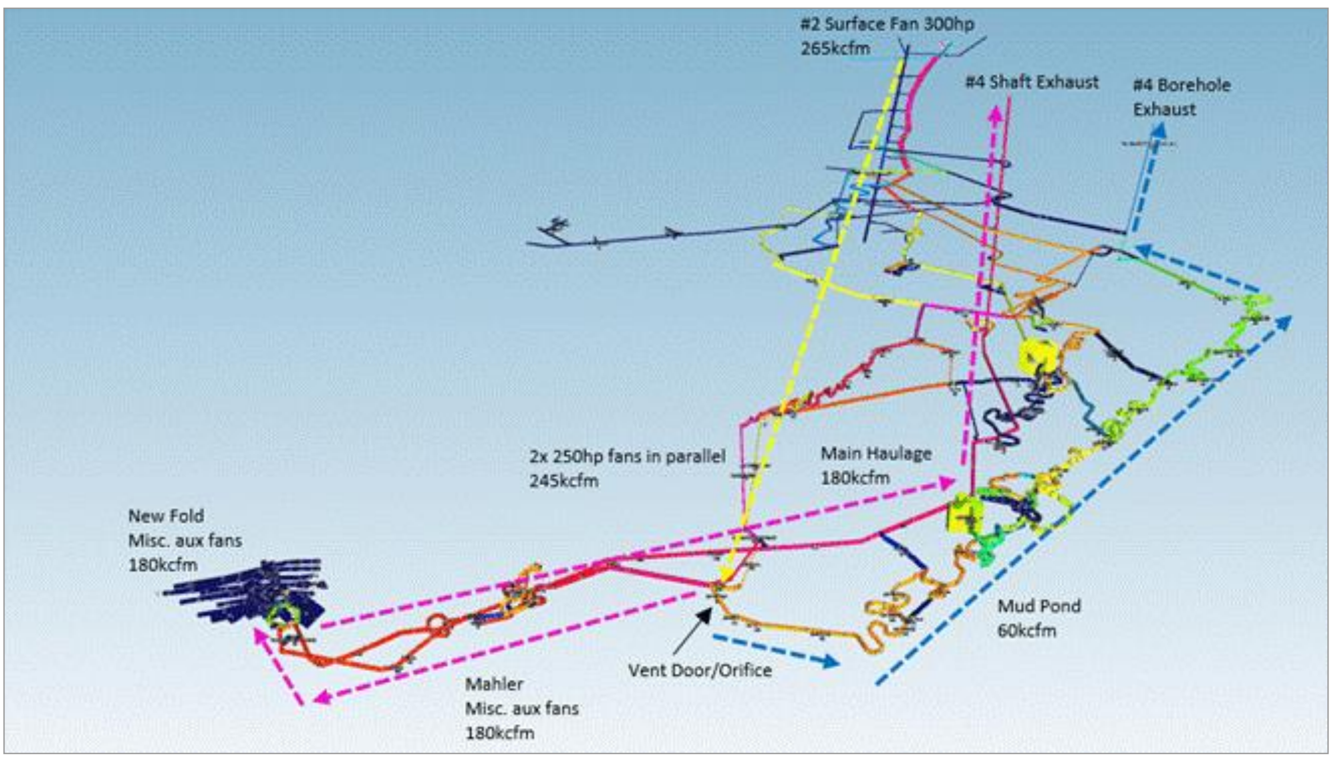
Minimum airflow requirements are based on expected diesel emissions of the underground mining fleet required at peak mine production. Additional airflow is used underground to improve air quality. The power rating of each piece of equipment is determined, and the utilization factors representing the equipment in use at any time, are applied to estimate the amount of air required. Equipment specified for site has undergone testing by Mine Safety and Health Administration (MSHA) to determine the ventilation requirements to dilute the engine emissions to a safe working level. The volume of air determined to ventilate the diesel emissions is 211 kcfm.

The generalized strategy for ventilating the ESM mine is to use the #2 Mine inclined shaft, stopes, and associated workings as intake. Air is exhausted through the #4 Shaft and #4 Borehole. The existing 300 hp ABC centrifugal fan located in the pit west of the #2 hoist house pressurizes the #2 Mine with 265 kcfm. Approximately 5% losses to unknown connections to surface through the #2 Mine are routinely measured.

On the 3500 level, two parallel 250 hp Alphair booster fans draw air from the surface supply and send 245 kcfm to the mine; most of this air is exhausted through the main haulage ramp and up the #4 Shaft while the rest is run through Mud Pond and out the #4 Borehole.

Based on LOM plans, future ventilation upgrades will include the installation of one variable orifice ventilation door within the Mud Pond ramp and additional miscellaneous 75 hp to 150 hp ventilation fans in New Fold and Mahler.

Figure 16.10 LOM ventilation installations



Source: ESM 2020.

16.1.14.2 Mine air heating

There are no identified needs nor plans to introduce heated air to the mine at this time.

16.1.14.3 Electrical power

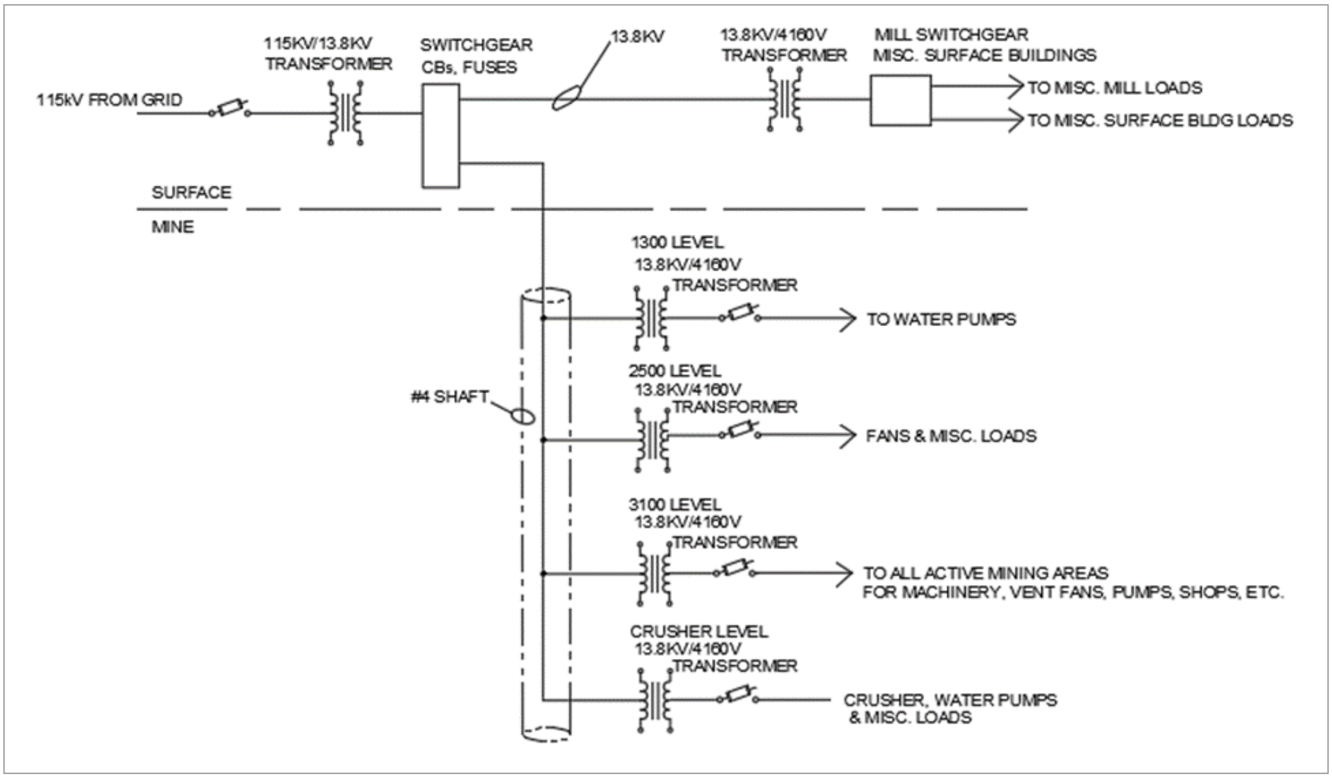
The majority of electrical power consumption at the mine will arise from:

- Main and auxiliary ventilation fans
- Mine air compressors
- Hoisting
- Drilling and ground support equipment
- Dewatering pumps
- 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 is delivered at 4,160 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. The site elementary electrical one-line diagram is shown in Figure 16.11.

Figure 16.11 Site elementary electrical one-line diagram



Source: ESM 2020.

16.1.14.4 Compressed air

Compressed air is currently required for longhole drills, jacklegs, and face pumps. Compressed air is provided by stationary compressors on surface. Reticulation of compressed air through the mine utilizes the existing pipes in addition to new 6” pipes as development advances. To minimize on-going compressed air transportation and leakage costs, it has been determined that all new equipment requiring compressed air shall have its own manufacturer’s air compressor on-board. The StopeMate LH Drill has been provided a dedicated and mobile air compressor for its use.

16.1.14.5 Service water supply

Service water for drilling, dust control, washing and fire suppression is sourced from surface via a 10” stainless steel 314 pipe within the #4 shaft and distributed in 2” diameter steel piping.

16.1.14.6 Dewatering

Water-bearing fracture zones at ESM generally occur above a depth of 900 ft, diminish with depth, and become nearly non-existent 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 (gal/d) to a high of 727,000 gallons per second (gal/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. Additional pumping requirements estimated for the LOM 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 2” to 6” steel and HDPE piping.

16.1.14.7 Explosives storage and handling

Primary explosives storage magazines are located on surface. Secondary magazines are located underground to provide explosives storage for up to seven days. Explosives and detonators are stored in separate magazines on surface and in the underground.

Bulk and bagged ANFO are used as the major explosives for mine development and production. Explosives handling, loading, and detonation are 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 LGS stoping with good fragmentation.

16.1.14.8 Fuel storage and distribution

Mobile equipment is re-fuelled at underground fuelling stations currently in place with delivery by pipeline from a surface storage tank.

16.1.14.9 Underground transport of personnel and materials

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

16.1.15 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 16.8 summarizes the underground mobile fleet.

Table 16.8 Existing mobile mine equipment fleet

Description	On-site
Drill Jumbo – 2 Boom – Sandvik Axera	1
Drill Jumbo – 2 Boom – Gardner Denver MK-65	1
Drill Jumbo – 1 Boom – Gardner Denver MK-35	1
Drill Jumbo - 1 Boom – MTI VR II	2
Longhole – Boart Longyear Stopemate	1
Longhole – Boart Longyear Stopemaster	1
Bolter – Secoma Pluton-17	2
LHD (10 t/6 yd) Epiroc ST 1030	4
LHD (10 t/6 yd) Sandvik LH 410	1
LHD (7.0 t/4 yd) MTI 650	2
LHD (3 t/2.5 yd) MTI 270	1
Haulage Truck – 40 Ton – Tamrock 40 D	3
Haulage Truck – 42 Ton – Epiroc MT 42	1
Powder Tractor – John Deere JD-210C – PT 0003	2
Scissor Lift – Getman A-64	4
Scissor Lift – Walden SLX5000	1
Flatdeck – Walden BTX5000	1
Grader – Champion C80-A27 – GR0002	1
Telehandler – GENI GTH5519	1
Mine Rescue Vehicle – Kubota RTV 900	1
Tractors – Kubota L2500/L2800/L3301	22
Jacklegs / Stopers	43

Source: ESM 2020.

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 material-passes, where it is rehandled and loaded into haul trucks for transportation to the shaft loading station.

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 used for longhole production stoping.

16.1.15.1 Mine equipment maintenance

Mobile underground equipment is maintained at the existing underground mine shop. The 2500 level shop is equipped to handle major rebuilds. The 3100 level shop manages daily maintenance and preventative maintenance. Minor maintenance and repairs are done in the work headings underground with use of a mechanics truck to minimize tramming of equipment to the shop.

16.1.16 Mine personnel

The ESM mine and mine maintenance department employs 77 people at the full production rate for underground of 1,100 t/d. Production occurs on a schedule which provides two 10-hour shifts, five days per week, with no operations on Saturday and Sunday. This allows a two-hour pause between shifts to clear blast gasses from the mine.

Mine personnel reside in nearby towns and are responsible for their own transportation to and from the site on a daily basis.

Table 16.9 outlines the mine labour force quantities, and rotation schedules.

Table 16.9 Mine personnel summary

Position	Roster	Rotation	LOM average
Mining management			
Mine Manager	Salary	5 x 2	1
Mine General Foreman	Salary	5 x 2	1
Subtotal – Mining Management			***2***
Mining Operations			
Shift Supervisor	Hourly	5 x 2	2
Miner 1 (Jumbos, Bolters)	Hourly	5 x 2	7
Miner 2 (Jackleg bolters, LH drillers, Blasters)	Hourly	5 x 2	9
Miner 3 (Loader & Truck operators)	Hourly	5 x 2	12
Miner 4 (Services, equipment operators)	Hourly	5 x 2	6
Diamond Drillers	Hourly	5 x 2	4
Longhole Drillers	Hourly	5 x 2	4
Subtotal – Mining Operations			***44***
Crushing and hoisting			
Hoistman	Hourly	5 x 2	2
Lead Shaft Miner	Hourly	5 x 2	2
Shaft Miner	Hourly	5 x 2	5
Subtotal – Crushing & Hoisting			***9***
Mine maintenance			
Maintenance Manager	Staff	5 x 2	1
Maintenance General Foreman	Staff	5 x 2	1
Electrical General Foreman	Staff	5 x 2	1
Maintenance Clerk	Staff	5 x 2	1
Maintenance Supervisor	Hourly	5 x 2	2
Heavy Duty Mechanic	Hourly	5 x 2	11
Electrician	Hourly	5 x 2	2
Subtotal – Mine Maintenance			***19***
Mining Technical Services			
Technical Services Manager	Staff	5 x 2	1
Senior Mine Engineer	Staff	5 x 2	1
Junior Mine Engineer	Staff	5 x 2	1
Project Engineer	Staff	5 x 2	1
Surveyor	Staff	5 x 2	1
Technician	Staff	5 x 2	1
Chief Geologist	Staff	5 x 2	1
Geologist	Staff	5 x 2	1
Junior Geologist	Staff	5 x 2	1
Subtotal Technical Services			***9***
Grand total			**83**

Source: ESM 2020.

16.1.17 Mine production schedule

Mine scheduling for the ESM project was conducted by Jackleg. The schedule 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. Only the C&F mining areas require the placement of waste rock as backfill. No cemented backfill is currently planned at ESM. As swell factor of 35% is assumed for calculating loose waste rock volumes.

Underground production was considered to have started as soon as first mineralization was 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 earlier given the high indicated content, proximity to existing development, and availability of stopes that were drilled but never fired before the mine shut down in 2008.

Annual mine production statistics are provided in Table 16.10.

Table 16.10 Annual mineralized material by mining zone

Mining zones	Unit	Total	2021	2022	2023	2024	2025	2026	2027
Mud Pond	t	250,111	137,348	112,763					
Mahler – Upper	t	182,351	3,990	-	-	15,619	56,226	55,000	51,516
Mahler – Lower	t	673,868	99,988	72,047	73,071	73,893	116,083	150,000	88,786
New Fold	t	794,164	133,957	205,190	251,773	150,553	52,691	-	-
N2D	t	749,874	-	-	65,156	149,935	165,000	185,000	184,783
Tons total	**t**	**2,650,368**	**375,283**	**390,000**	**390,000**	**390,000**	**390,000**	**390,000**	**325,085**
Zinc grade									
Mud Pond	%	7.8	8.0	7.5					
Mahler – Upper	%	8.1	6.7			7.7	8.0	9.1	7.4
Mahler – Lower	%	9.7	9.0	9.7	10.8	11.3	9.8	8.8	9.7
New Fold	%	9.2	8.9	9.0	9.3	9.6	9.3		
N2D	%	7.0			6.9	6.8	6.9	7.2	7.1
Average mine grade	**%**	**8.5**	**8.6**	**8.7**	**9.2**	**8.8**	**8.3**	**8.1**	**7.8**
Zinc pounds									
Mud Pond	klbs	38,839	21,950	16,889	-	-	-	-	-
Mahler – Upper	klbs	29,653	532	-	-	2,419	9,001	10,029	7,671
Mahler – Lower	klbs	130,679	18,007	13,915	15,843	16,758	22,697	26,258	17,201
New Fold	klbs	146,043	23,856	36,900	46,718	28,775	9,793	-	-
N2D	klbs	105,158	-	-	9,015	20,443	22,908	26,678	26,114
Total zinc pounds	**klbs**	**450,372**	**64,345**	**67,704**	**71,575**	**68,396**	**64,400**	**62,965**	**50,987**

16.1.18 Mine development schedule

The development schedule is based on estimated cycle times for jumbo development. All waste development during pre-production is shown as capital development.

Annual development footages are summarized in Table 16.11.

Table 16.11 Annual development schedule

Development Schedule	**Units**	**Total**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**
Capital Lateral Development	ft	33,010	7,138	5,950	5,966	5,772	4,774	3,132	278
Operating Lateral Development	ft	17,215	1,619	2,652	2,638	2,807	3,674	3,777	47
Total Lateral Development	**ft**	**50,226**	**8,757**	**8,602**	**8,604**	**8,579**	**8,449**	**6,908**	**326**
Raisebore	ft	1,285	-	-	483	292	152	359	-
Drop Raise	ft	1,156	-	135	58	305	353	305	-
Total Vertical Development	**ft**	**2,441**	**-**	**135**	**541**	**597**	**505**	**664**	**-**

16.2 Open pit

16.2.1 Hydrological parameters

AMC has not reviewed any hydrological or hydrogeological information. During the site visit, no water was observed in the existing pit depression nor anywhere on surface in the open pit mining area. Underground workings under parts of the proposed open pit excavations may drain pit inflows. Verbal communication with mine staff affirmed that the existing pit slopes showed minimal signs of seepage and in general water ingress into the pits (from surface and sub surface flows) was small.

AMC therefore does not foresee a major impact from water inflows into the proposed pits. None the less, there will be some water accumulations and pit sumps with de-watering pumps will be required periodically, if not permanently, as the pit reaches lower depths.

16.2.2 Open pit geotechnical considerations

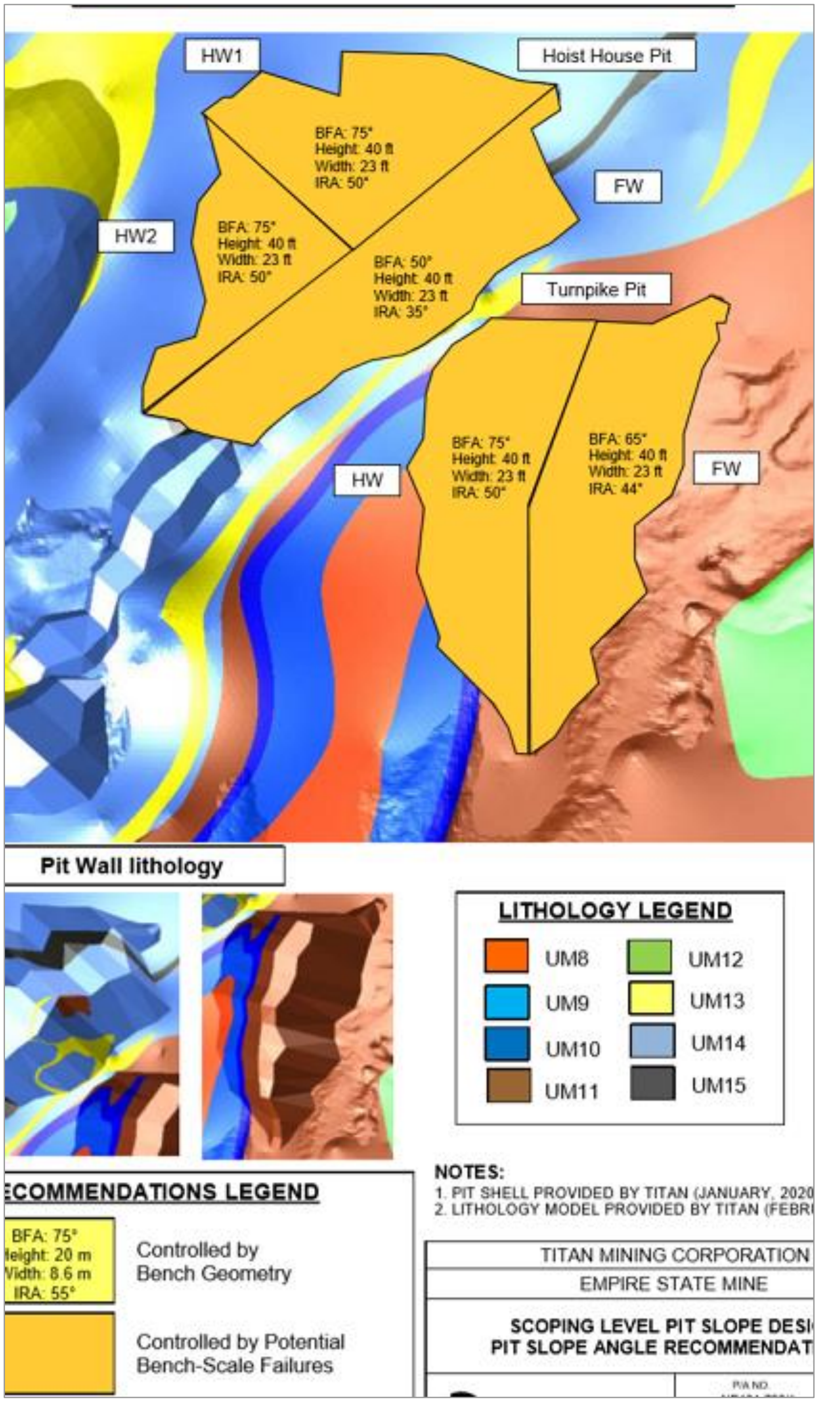
KP provided a study dated 15 May 2020, "Empire State Mine Scoping Level Pit Slope Design" in which the pit slope recommendations were given (Table 16.12 and Figure 16.12).

Table 16.12 KP pit slope recommendations

Open pit	Open pit design sector	Dominant lithology[1]	Nominal pit wall dip direction (°)	Total slope height (ft)[2]	Dominant potential failure mode	Bench configurations			Inter-ramp slope configurations				Overall slope configuration	Comments
									Inter-ramp angle (IRA)					
						Bench face angle (BFA) (°)	Effective bench height (ft)	Bench width (ft)	From bench configuration (°)	Achievable based on kinematics	Achievable based on LE	Max. inter-ramp slope height (ft)	Expected OSA performance based on precedent practice	
Hoist House	HW1	UM14, UM15	155	250	None	75	40	23	50	Yes	Yes	300	FoS > 1.3	Achievable bench and inter-ramp slope performance sensitive to the presence of persistent discontinuities perpendicular to the foliation, striking parallel to the axis of the pit.
	HW2	UM14, UM15	110	240	None	75	40	23	50	Yes	Yes	300	FoS > 1.3	Achievable bench and inter-ramp slope performance sensitive to the presence of persistent discontinuities perpendicular to the foliation, striking parallel to the axis of the pit.
	FW	UM11, UM13, UM14	320	235	Planar	50	40	23	35	Yes	Yes	300	FoS > 1.3	Achievable bench geometry is limited by the potential for planar failure along the foliation. If significant UM13 is present behind the slope, it is recommended that this sector be re-evaluated.
Turnpike	HW	UM8, UM9, UM10, UM11	100	295	None	75	40	23	50	Yes	Yes	300	FoS > 1.3	Potential for local ravelling due to reduced rock mass quality, where the biotite-altered UM10 is encountered in the wall.
	FW	UM11	285	260	Planar	65	40	23	44	Yes	Yes	300	FoS > 1.3	Achievable bench geometry is limited by the potential for planar failure along the foliation.

Notes:
[1] Final pit wall lithology based on lithology models provided by Titan (Feb 2020).
[2] Total slope height and wall orientations based on pit shell provided by Titan (Jan 2020). Reported slope heights are based on the pit shells and are measured from the toe of the walls in the deepest section of the sector.

Figure 16.12 KP pit slope angle recommendations



Source: KP.

KP's report is scoping level and recommends further data collection, particularly on structural features, be undertaken.

16.2.3 Mineral Resource model for mining

AMC used the SRK 2020 estimate block model for the open pit study work.

16.2.4 Cut-off value

The cut-off value is based on NSR value, which accounts for all downstream processing costs. A net payable recovery for each metal was determined that takes into account likely smelter terms and penalties, transport, treatment and refining costs. These smelter terms were supplied by Titan and are based on their current smelter contract. The NSR cut-off value is based on the assumptions shown in Table 16.13.

Table 16.13 NSR cut-off value assumptions

Mining factors	**Unit**	**Open pit**
Mining dilution	%	5
Mining recovery	%	95
Operating costs		
Mining cost for mineralization	C$/t	4.50
Mining cost for waste	C$/t	2.50
Processing cost for mineralization	C$/t	7.00
G&A cost for mineralization	C$/month	100,000
Processing recovery		
Zinc	%	94
Lead	%	85
Silver	%	55
Revenue		
% Payable zinc	%	85
% Payable lead	%	95
% Payable silver	%	95
Zinc price	$/lb	1.07
Lead price	$/lb	0.98
Silver price	$/oz	17.00
Cut-off value[1]	$/ton	25
Cut-off grade	Zn%	1.2

Note: [1] Open pit marginal cut-off excludes mining cost. Rounded down.
Source: AMC.

16.2.5 Dilution and mining recovery factors

The mineralization occurs in lenses as relatively continuous zones with quite sharp contacts against the adjoining waste layers. The contact can be seen visually in most cases. Dilution can be expected along the contact. Any waste bands internal to the lenses have not been modeled selectively and are therefore included in the mineralization block estimation. Dilution and losses along the lens contacts against waste will occur due to blast movement and the ability to identify and selectively mine along the mixing zone after blasting. Provided care is taking during blasting and rigorous mineralization control and monitoring systems are followed, AMC estimates that dilution and mineralization losses can be minimized.

A mining recovery factor of 95% and dilution of 5% has been applied. mineralization loss is assumed to be replaced by an equivalent mass of waste for no net loss in mineralization tons. The dilution material is assumed to have zero value. The mining dilution and recovery were applied as factors during the pit optimization process and to estimate mill feed tons in the schedule.

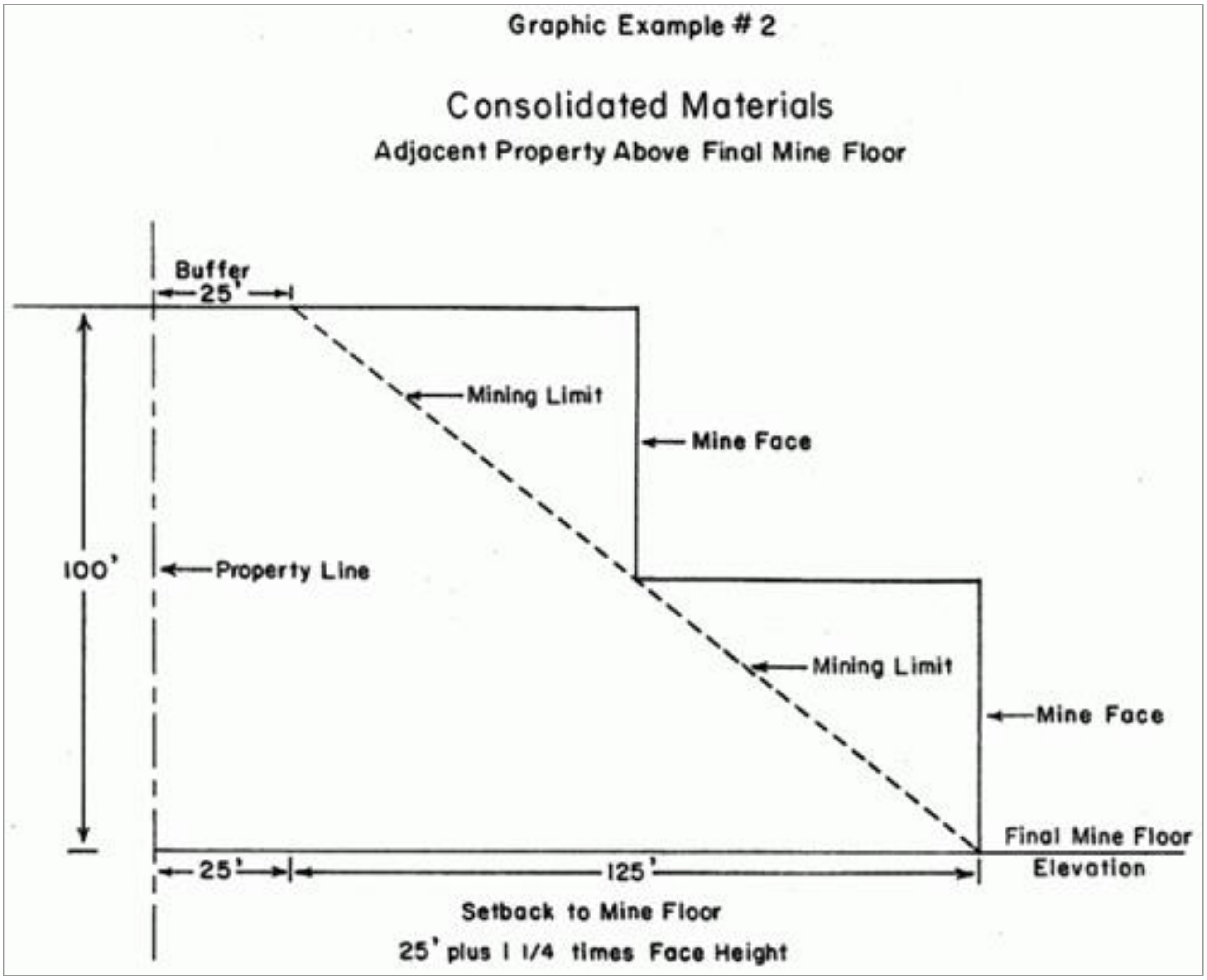
16.2.6 Pit optimization and selection

The Lerchs-Grossmann pit optimization algorithm was used to define the ultimate pit shell for the Pump House, Hoist House, and Turnpike zones. The selected pit shells were then used to produce pit designs and the open pit mining schedule.

Further underground mining is not planned under the open pit zones. The block model was depleted of the existing underground workings. Pit optimization did not therefore consider any further influence from underground mining.

The pit area lies close to houses and residents along the east side of the Turnpike zone and outside Titan's property. As well, New York regulations stipulate an offset and slope cone from the property boundary to the toe of any excavation. The offset is 25' and the slope from the pit crest to toe as 1:1.25 (38.66°). This 'no-go' limit was modeled in the software and all blocks outside the cone were assigned a high density such that mining cost would inhibit the optimization algorithm from mining outside the cone.

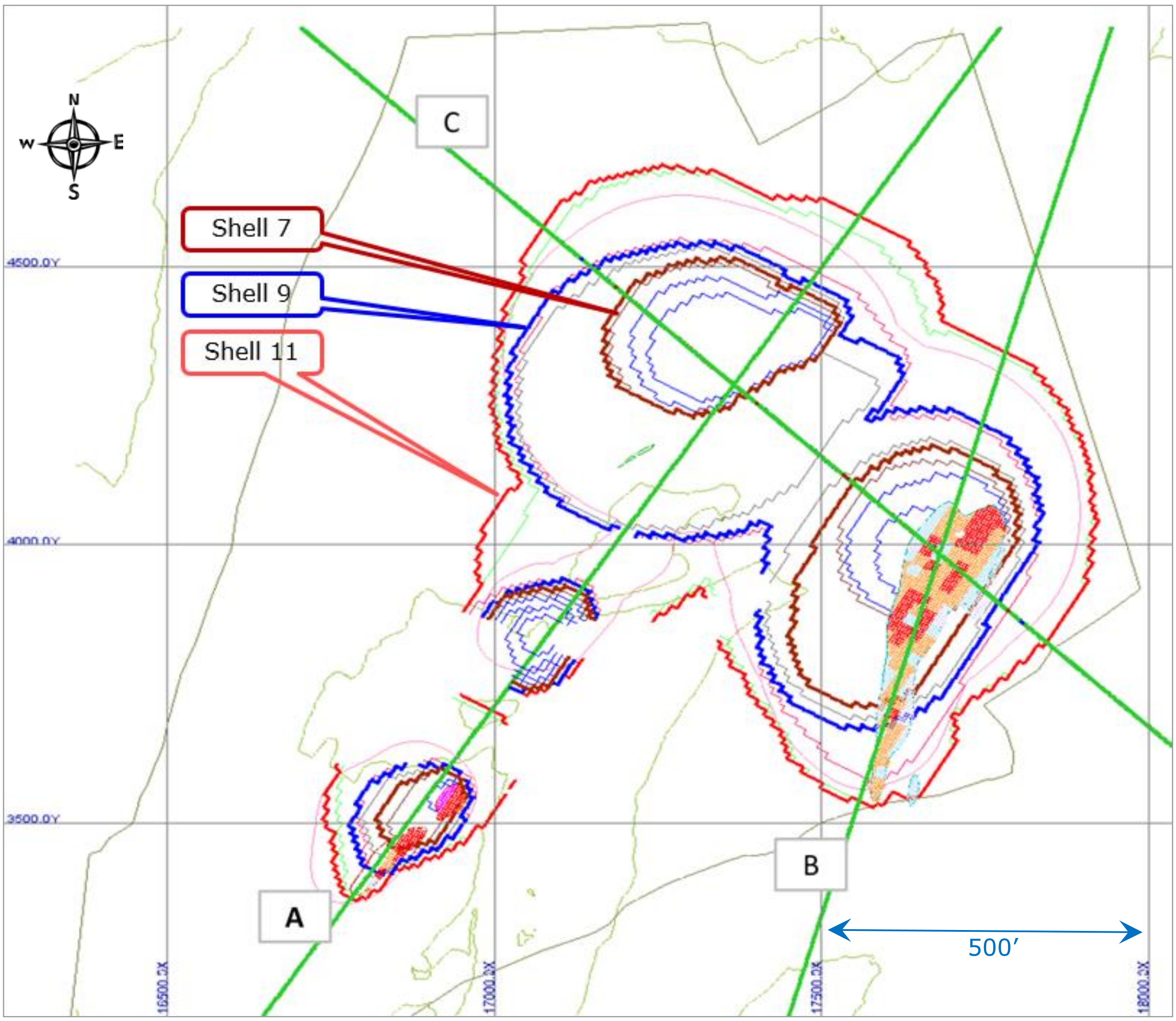
Figure 16.13 Permitting exclusion cone



Source: Titan.

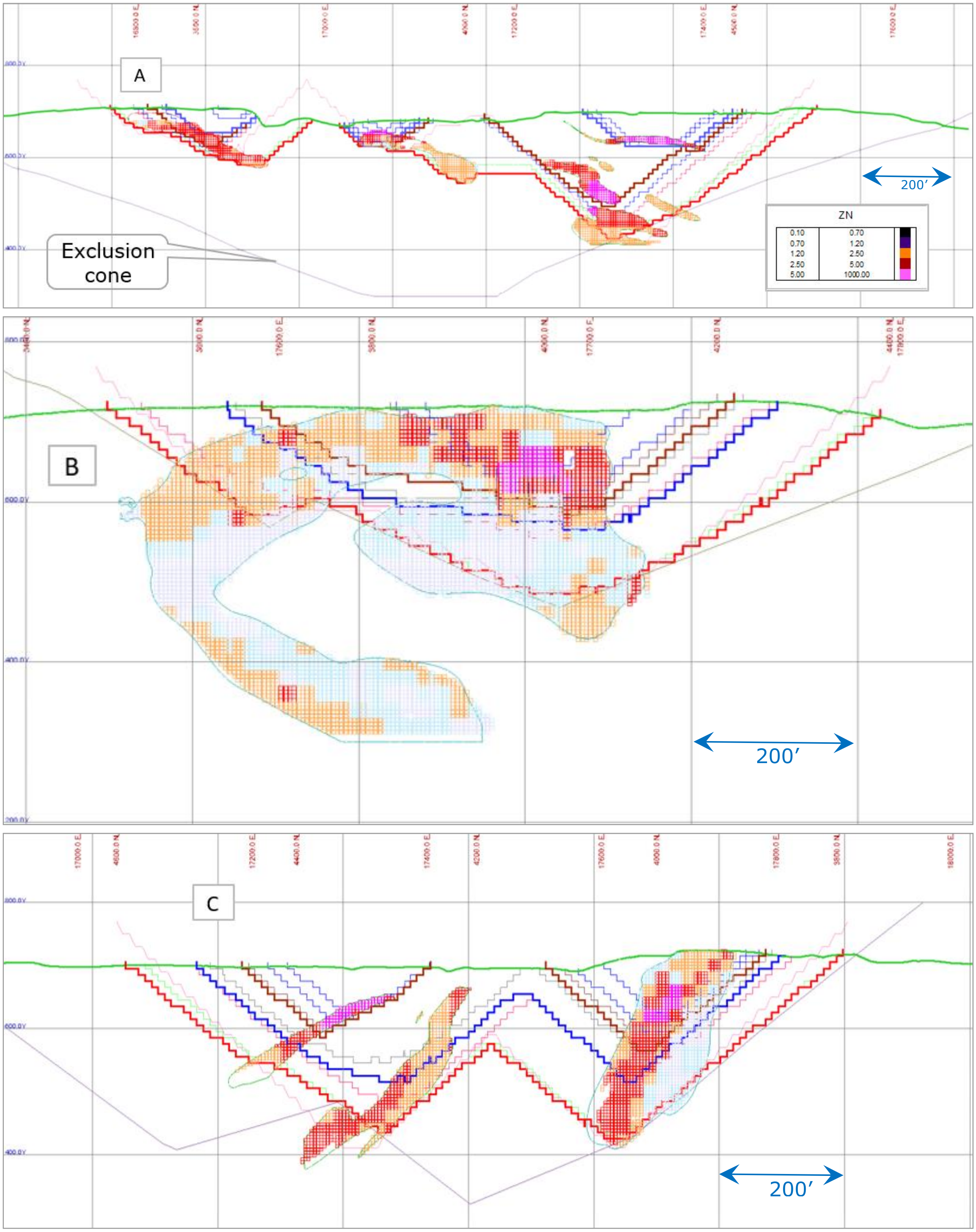
The economic inputs required to run optimization include the costs and revenues of the project and these are classified as mineralization and waste mining costs, mineralization processing costs and selling costs. Revenue is assigned based on mill recoveries and applying the smelter terms. In the case of Titan, various mineralization costs were considered to be covered by the current and future underground operations. There was therefore some ambiguity as to what should be included in mineralization costs for the pit optimization runs. Several runs were performed using differing cost assumptions. In all cases the resulting pit shells had the same general shape. Gaps or jumps between shells occurred at similar locations. All that was different was the position of the gaps within the revenue factor range or sequence of shells. Therefore, the shell at each jump step was analyzed. There are 3 basic jumps corresponding to positions where the mineralized lenses overcome the waste stripping costs as can be seen in Figure 16.14.

Figure 16.14 Plan view optimization shells (with cross-section locations)



Source: AMC 2020.

Figure 16.15 Cross-section views



Source: AMC 2020.

The volumes within each shell were evaluated and input into the Titan economic model. The economic model had underground mineralization zeroed out and mineralization and selling costs adjusted to simulate various cut-offs. The undiscounted NPV of each shell was thus evaluated.

Table 16.14 Shell optimization results

		Tonnage	ZN	PB	AG	SR	UCF ($M)
Cut-off 0.7	Shell 6	247,313	3.32	1.24	11.44	0.60	7.70
	Shell 7	384,869	3.10	1.17	10.88	2.34	10.60
	Shell 8	577,062	3.11	0.95	9.73	3.62	12.50
	Shell 9	702,848	3.06	0.90	9.39	4.03	13.70
	Shell 11	1,231,873	2.69	0.72	8.17	5.52	11.40
Cut-off 1.2	Shell 6	226,562	3.54	1.32	12.17	0.75	7.70
	Shell 7	343,955	3.35	1.26	11.61	2.74	10.50
	Shell 8	526,558	3.32	1.00	10.22	4.06	12.50
	Shell 9	640,767	3.27	0.95	9.85	4.52	13.70
	Shell 11	1,065,310	2.97	0.78	8.76	6.53	11.40
Cut-off 1.75	Shell 6	203,452	3.78	1.40	12.88	0.95	8.60
	Shell 7	301,724	3.61	1.35	12.32	3.26	10.10
	Shell 8	469,695	3.54	1.05	10.69	4.68	11.90
	Shell 9	568,737	3.49	1.00	10.33	5.22	13.10
	Shell 11	876,975	3.28	0.86	9.53	8.15	10.10
Cut-off 2.5	Shell 6	153,290	4.32	1.54	14.26	1.58	6.40
	Shell 7	221,710	4.15	1.49	13.50	4.80	8.30
	Shell 8	344,640	4.05	1.16	11.65	6.74	9.60
	Shell 9	416,020	3.99	1.11	11.28	7.50	10.20
	Shell 11	592,591	3.83	1.03	10.87	12.54	5.50

The results show that in each cut-off scenario, shell 9 shows the highest cash flow (UCF). And the best cut-off occurs around the 1.2% Zn. This cut-off was also resolved to be a good estimate of what the marginal mill cut-off for the open pit mineralization would be (Table 16.13).

Shell 9 was carried forward to design. Design of the small Pump House zone, and its connection with Hoist House, did not follow particular shells precisely. Practical considerations and the block model mostly guided the design in these areas.

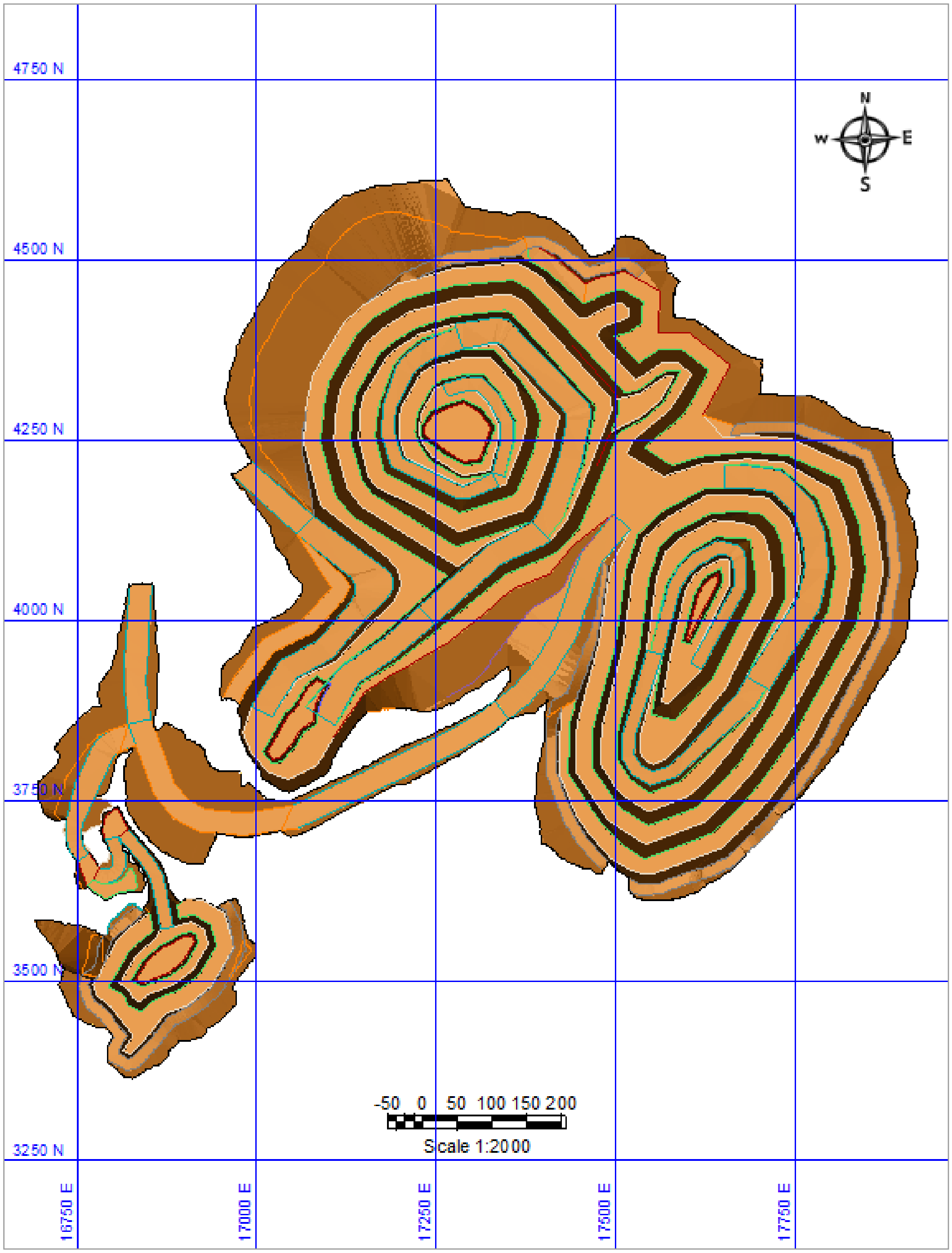
16.2.7 Pit design

Conceptual pits were designed based on the selected pit optimization shell as described above. Design criteria were:

- Double lane ramps = 32 ft wide, 10% grade.
- Single lane 18 ft wide up to 12% grade.
- Pit slopes as per geotechnical guidelines except FW of Hoist House as described below.
- Overburden nominally 2:1 slope with a 15 ft setback along contact.
- Road fills 2:1.
- Bench access maintained on one side of ramp (pits and dumps). i.e., benches not pinched off on both sides.

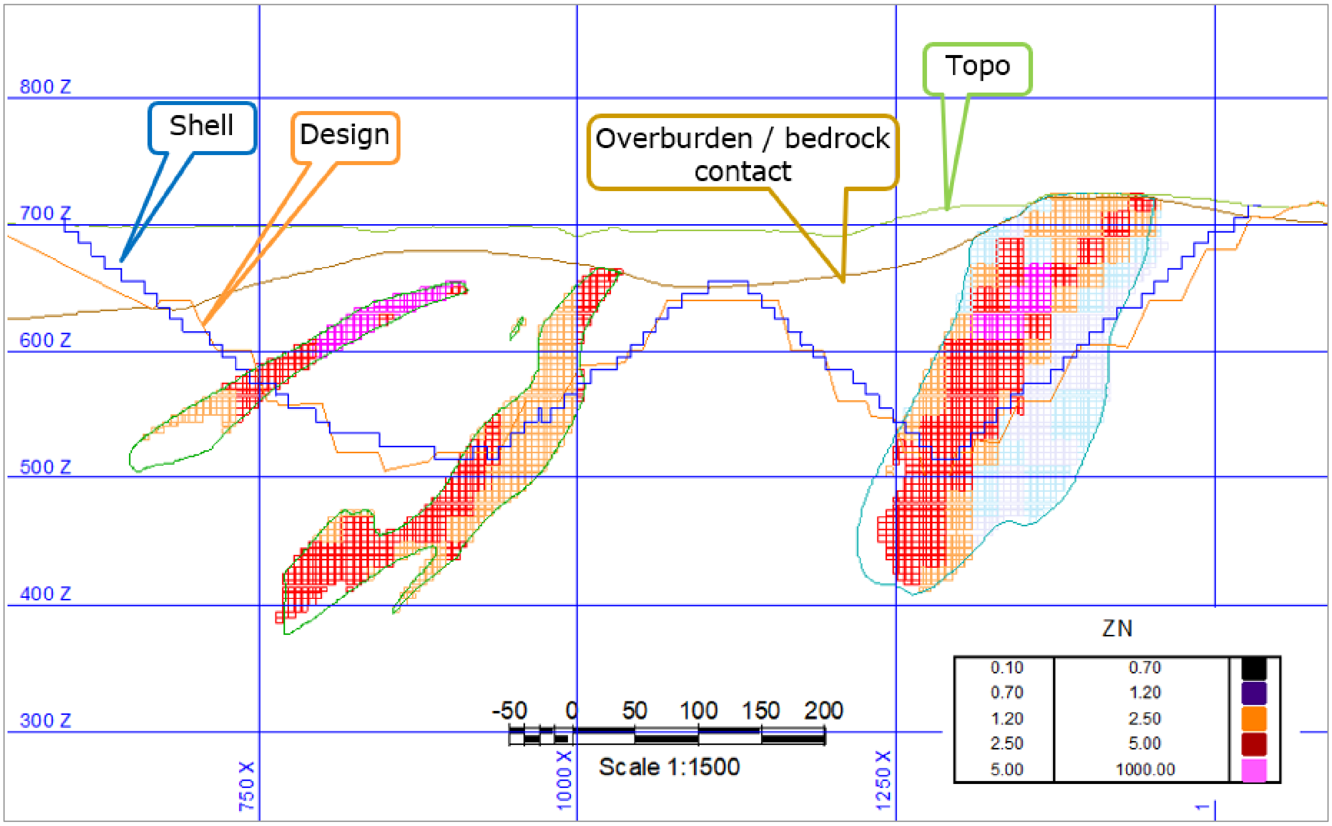
Wall design conformed to the geotechnical recommendations except on the Hoist House footwall. Due to the flat overall slope in this sector caused by the haul road, and the short life and depth of the pit, AMC elected to design a steeper slope in this area. The slope design adopted was the same as for the Turnpike footwall at 44° IRA. "Goodbye" cuts 20′ deep were designed in the floors of each pit to extract the maximum mineralization possible. The overburden wireframe contained some odd undulations in areas and these were smoothed out where they intersected the design pit walls.

Figure 16.16 Titan pit design



Source: AMC 2020.

Figure 16.17 Cross-section C of design, shell, and block model



Source: AMC.

Indicative tons and diluted grades contained within the conceptual pit designs are presented in Table 16.15.

Table 16.15 Open pit projected tons and grades

Zone	Mineralized material short tons (t)	Zn (%)	Pb (%)	Ag (g/t)
Pump House	18,500	3.41	0.43	6.67
Hoist House	387,250	2.89	1.27	10.86
Turnpike	252,750	3.32	0.37	7.15
Total	**658,500**	**3.07**	**0.90**	**9.32**
Total waste	**3,920,500**			

Source: AMC 2020.

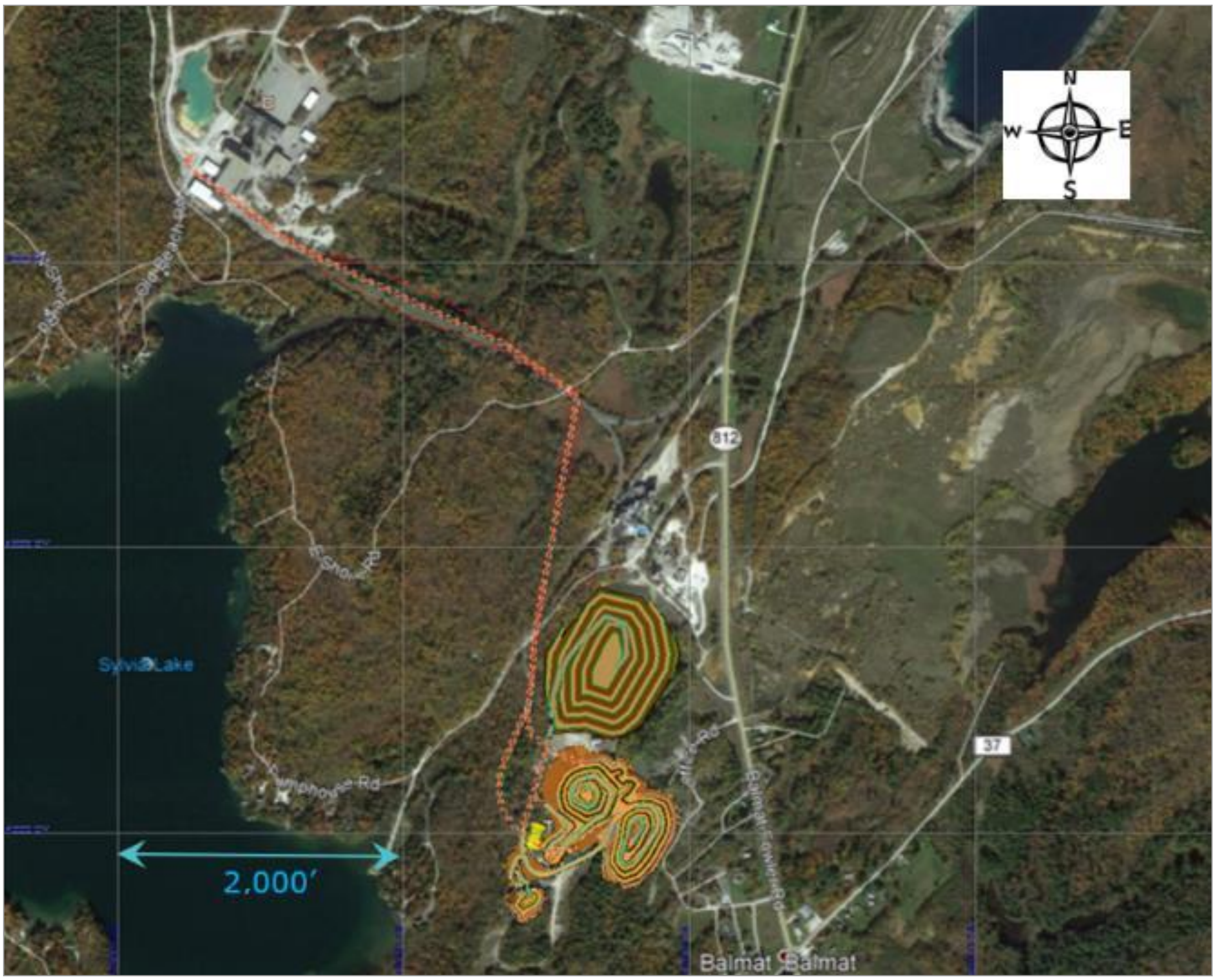
16.2.7.1 Layout of other open pit mining related facilities

A single waste dump has been designed immediately north of the open pits in an existing depression left over from the Vanderbilt open pit mine. The old Vanderbilt pit (a talc mine) is a semi-rehabilitated disturbed site ideally situated for the proposed Titan open pit waste dump. Titan has negotiated land rights with Vanderbilt to dump in this location. A short, direct haul road would connect the pits with the dump.

The haul road is proposed to follow an existing rail line spur right of way. The spur is no longer used for rail cars and is ideally located for hauling mineralization to the Titan mill. The route crosses no public roads and could be readily upgraded for use by standard highway trucks. Certain precautions and permitting issues may be encountered where the route crosses wetland zones.

The existing Titan underground mine uses the No. 2 inclined shaft as a secondary escape egress route for evacuation of personnel in an emergency. The collar of this shaft is located some 40′ from the north edge of Hoist House pit. The head frame and other facilities at that location will be impacted by the pit excavation. While most of the infrastructure at this location is old and no longer in use, the immediate collar and egress system will need to be maintained until a secure, replacement system is in place.

Figure 16.18 Layout of open pit with Google Earth overlay



Source: AMC 2020.

16.2.8 Mining method

It is proposed to mine the open pits using conventional truck and excavator mining methods. A mining contractor operation is presumed. Overburden is assumed to not require blasting. All bedrock will require drill and blast operations. Benches shall be 20 ft high with safety berms every second bench (i.e., double benched to 40′ spacing). The excavator would typically sit on a temporary platform part way up the muck pile and load trucks sitting on the bench level a few feet below. Due to the small pit sizes, none of the pits are phased and mining will be by a simple descending, full bench, top-down sequence. The pits are sequenced in the schedule with Pump House mined first followed by Hoist House and then Turnpike.

16.2.8.1 Drill and blast

The proposed drilling parameters for 20 ft bench heights are presented in Table 16.16. Standard, midsized top hammer or down the hole hammer drill rigs are envisioned. The rigs would be equipped with blasthole sample equipment to collect samples for grade control. Explosives would be straight ANFO, emulsion, or ANFO blends. Drilling and explosive supply including loading and shooting, are assumed to be provided by contractors.

Table 16.16 Open pit drilling parameters

Parameter	Value	Unit
Bench height	20	ft
Burden	11.5	ft
Spacing (equilateral triangle)	13.3	ft
Hole size	5.12	inch
Collar	7.25	ft
Subdrill	2.5	ft
Explosive density	0.8	kg/m^3
Rock density	2.5	kg/m^3
Powder factor	0.23	kg/t
Powder factor	0.46	lb/st

Source: AMC 2020. Kg/m^3 = kilogram/cubic metre, and kg/t = kilogram per tonne.

Due to the projected short life of the open pit mines and the shallow mining depth, AMC has assumed that reduced presplit blasting would be required although buffer rows against final walls would be advised.

Assuming 10% re-drills, 18 metre/operating hour penetration rate, 75% availability and 90% utilization, 50 weeks of 80 h/week scheduled; there would be 2,700 h/yr operating and 48,600 ft drilled. Two drills are required to meet the production demand and would also provide flexibility and redundancy when 1 drill is down.

16.2.8.2 Load and haul

Two hydraulic excavators equipped with 5.9 cubic yards (yards^3 or 4.5 m^3) buckets (similar to CAT 374 machines) would be required to mine waste and mineralized material. They would load into a fleet of 40 st road trucks (such as Mercedes Actros) or articulated dump trucks (ex. CAT 740 ADT). Waste hauls are short (approximately 0.65 mi) while hauls for mineralization are longer (approximately 1.5 mi). Overall, annual excavator productivity is estimated at approximately 320 tons per hour (t/h) and trucks at 100 t/h in mineralization and 190 t/h in waste. Excavators and trucks have been estimated to operate 2,100 h/yr. 5 trucks should be adequate to meet production. One excavator with 2 trucks could stay permanently in waste. The second shovel with 3 trucks could do 3 shifts/week in mineralization and 7 shifts in waste.

Due to the lack of excavator redundancy and the high workload for these two machines, AMC recommends that a front-end loader (FEL) is added to the fleet. The FEL provides flexibility and backup when an excavator goes down. It could also provide backup at the crusher if the mill loader is unavailable. In the equipment estimate, a 4.5 yrd FEL has been included and assigned a maximum of 1,500 h/yr.

16.2.8.3 Stockpile rehandling

Direct dumping of mineralization into the crusher may be possible, but in the current estimate, it has been assumed that 100% of mineralization is re-handled from a ROM stockpile into the crusher. Titan currently has the resources to conduct this re-handle and no extra equipment or cost to the open pit mine operation has been applied.

16.2.9 Open pit equipment

The open pit contractor operations are projected to work on a five day, 8 h/day roster. Two shifts (day and night) are envisioned. Therefore 80 h/week are scheduled over 52 weeks per year for 5,200 h/a. Although labour is assumed to be paid 52 weeks, equipment operating hours have been based on only 50 weeks, thus assuming a 2-week period when the machine is offline.

Based on the production schedule (see Table 16.19), roster schedule, and equipment productivity estimates, the required equipment list is as shown in Table 16.17.

Table 16.17 Equipment estimate

Equipment	Y1	Y2	Y3 (partial)
Excavators	2	2	1
Trucks	5	5	4
Loader	1	1	1
Drills	2	2	1
Grader	1	1	1
Water truck	1	1	1
Dozers	1	1	1
Pickups	3	3	3

Source: AMC.

16.2.9.1 Ancillary equipment

Ancillary mobile equipment includes dozers, graders, water truck and pickups. This standard equipment is used to maintain roads and dumps and transport staff and personnel, respectively.

16.2.10 Open pit labour and staff

The open pit mining contractor is presumed to provide all equipment operators, maintenance workers and shift supervisors. The owner's team is assumed to provide, mine engineers, geologists, survey, and mineralization control staff. Numbers include a small supplement to account for redundancy in case of absenteeism, training etc. AMC has accounted for 3 extra staff over and above the existing underground complement of personnel. These three persons are accounted for in G&A costs.

The open pit contractor labour estimate is provided in Table 16.18.

Table 16.18 Open pit labour and supervision

	Y1	Y2	Y3 (partial)
Mine Foreman	2	2	2
Drill Operator	2	2	1
Drill Helper	3	3	1
Blaster	1	1	1
Blaster helper	2	2	1
Shovel / loader Operator	5	5	3
Haul Truck Operator	12	12	10
Dozer Operator	2	2	2
Water Truck Operator	1	1	
Grader Operator	1	1	1
Mine Labourer	3	3	2
Mine Maintenance Foreman	2	2	1
Mechanic	2	2	2
Mechanic Heavy Equipment	2	2	2
Electrician	1	1	1
Serviceman	2	2	1
Maintenance Labourer	2	2	1
Total	**45**	**45**	**32**

16.2.11 Proposed open pit production schedule

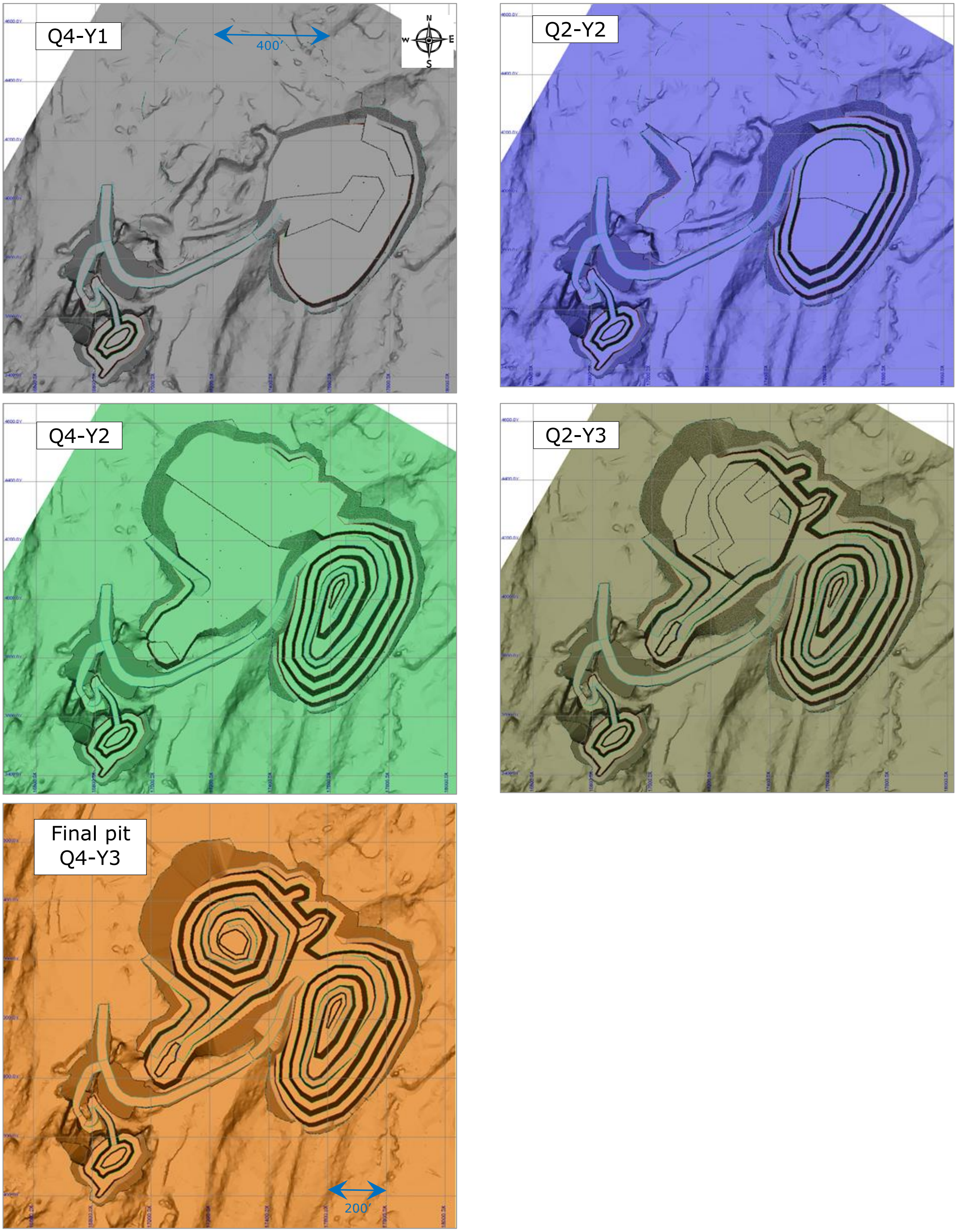
The proposed open pit production schedule extends over a 2½ year period and is summarized in Table 16.19 and Figure 16.19. It is assumed that construction of the process plant and surface infrastructure will take place in Year 0.

Table 16.19 Conceptual open pit production schedule

OP production	YR0	YR1	YR2	YR3	OP totals
Waste (Mt)	-	1,340,000	1,505,000	417,258	3,262,258
Mineralized rock (kt)	-	275,000	266,250	117,085	658,335
Pb (%)	-	1.19	0.88	0.29	0.90
Zn (%)	-	2.71	3.26	3.46	3.07
Ag (oz/st)	-	10.34	9.45	6.61	9.32

Source: AMC.

Figure 16.19 Projected open pit production period maps



Source: AMC 2020.

17 Recovery methods

17.1 Introduction

Mineralized material mined in the ESM deposits is processed at the existing ESM concentrator that was commissioned in 1970 and last shut down in 2008. The concentrator was refurbished in late 2017 and began processing mineralization in 2018. The concentrator flowsheet includes crushing, grinding, sequential lead and zinc flotation circuits, concentrate dewatering circuits, and loadout facilities. The flowsheet for the current operation is shown in Figure 17.1. The flowsheet for the proposed operation, which includes a lead circuit, is shown in Figure 17.2.

The design capacity of the concentrator is 5,000 t/d. Through-out the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines' capacity. The operating strategy is to operate the concentrator at its rated hourly throughput of 200 t/h to 220 t/h, but for only as many hours as necessary to suit mine production. It currently is processing between 6,500 to 7,000 tons per week operating on a schedule of one shift per day, four days per week. The concentrator suffers no notable losses from intermittent operation.

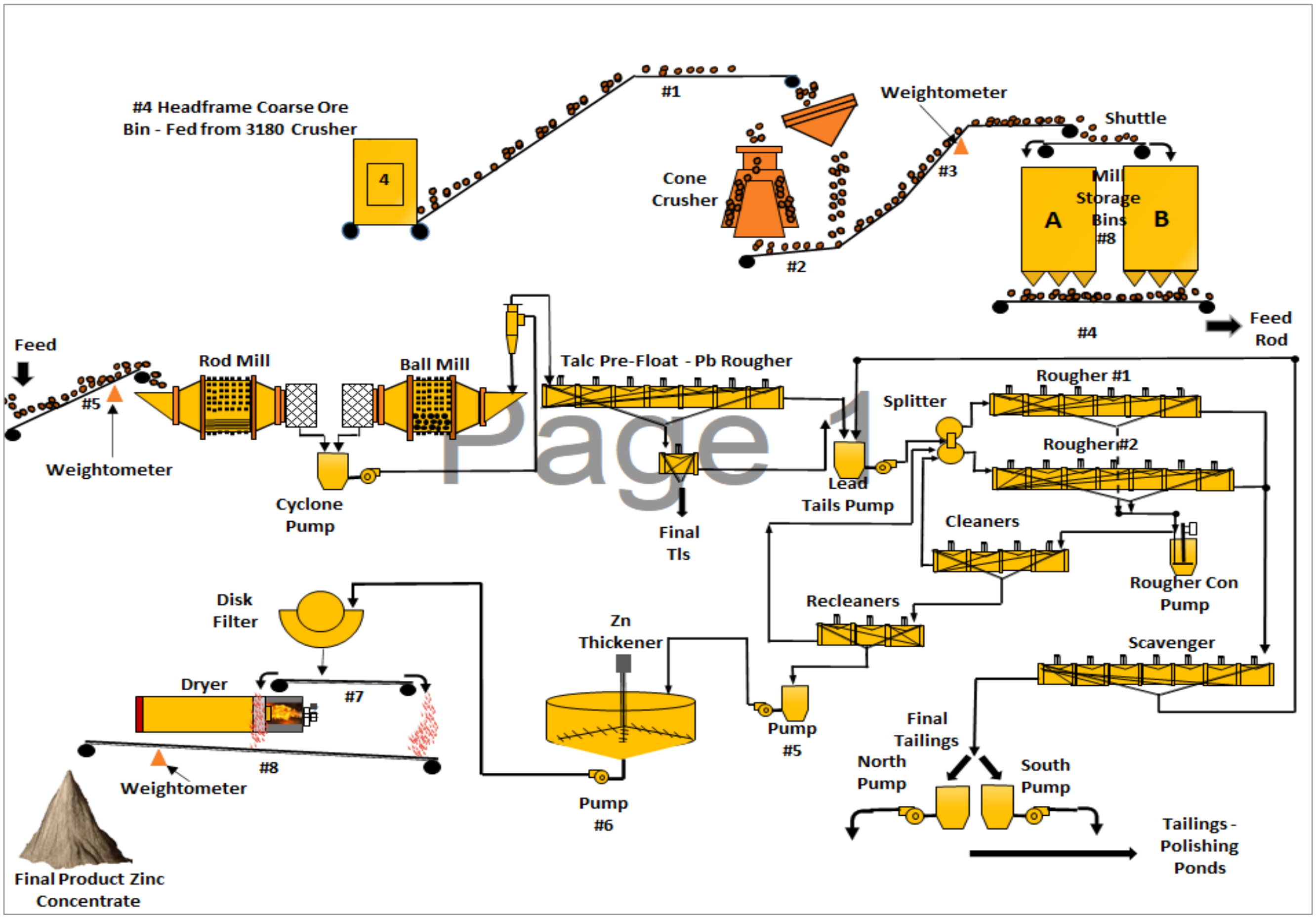
Brief descriptions of the concentrator circuits, equipment condition assessments, design criteria, and recommendations for work prior to re-starting the concentrator follow below.

17.2 Plant design criteria

From a metallurgical perspective, the optimal 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 shutdown.

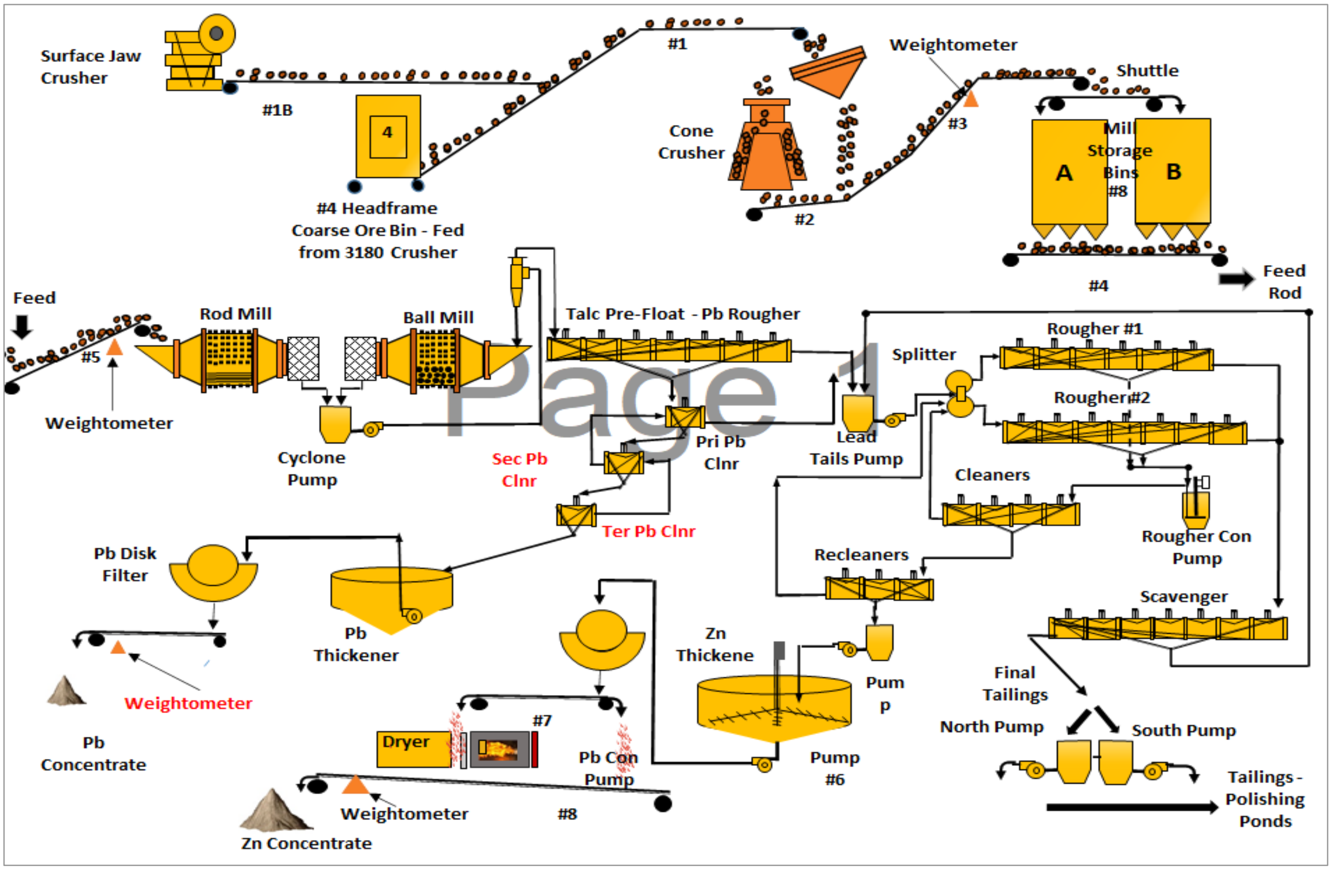
While the mill has a capacity of 5,000 t/d, underground mine production is typically no more than 1,400 t/d. The mill is operated for eight to 10 hours per day. This inherently introduces some amount of instability during start-up and shutdown.

Figure 17.1 Concentrator flowsheet current state



Source: ESM 2020.

Figure 17.2 Concentrator flowsheet with Pb circuit



Source: ESM 2020.

17.2.1 Crushing circuit

Primary crushing is done underground by a 36″ x 48″ jaw crusher, or on surface by a 30″ x 42″ jaw crusher set up outside the concentrator.

Coarse material from the surface crusher or the shaft hoist is 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 is 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. 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 operates 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 t/h 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.

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 ore bins. An automatic sampler is installed on this belt. Discharge from the #3 conveyor is distributed between the two fine ore bins by a shuttle conveyor. Each fine ore bin has a rated capacity of 2,000 t.

While production records show that the operating hours on the crushing plant were approximately the same as that of the grinding circuit, this is more a function of the hoisting rate (200 t/h - 220 t/h) than the actual crusher throughput. The actual capacity of the crusher is higher than indicated by the records, and in any case is more than adequate for future requirements. The crusher cone-mantle 'gap setting' is maintained to deliver ¾" feed to the rod mill. 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
Mineralization 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.	¾"

Source: ESM operating data 2020.

17.2.2 Fine ore bin

There are two bins with a nominal capacity of 2,000 t each. In preparation for start-up, inspections were completed, and the bins have been returned to service. Plugs were drilled and pulled from several points on both ore bins to ascertain a true thickness measurement. The inner surfaces of the bin were scaled to remove any free and loose material. The thickness testing is scheduled to be repeated in 2021.

Each bin is fitted with three slot feeders and DC variable speed drive conveyors. These have been inspected and returned to service as part of start-up.

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 ft by 16 ft 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 ft by 14 ft Allis Chalmers mill with a 1,000 hp 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 t/h to 220 t/h. The final grind size was normally 80% to 85% passing 65 mesh.

The media charges were left in the mills on shutdown, and minimal difficulties were found during mill start- up.

The rod mill was relined in January 2018 by Metso in advance of the recommissioning.

The existing grinding circuit is 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
ESM 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: ESM operating data 2020.

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 seven Wemco 300 ft^3 cells.

All of the air inlet ports on the Wemco cells are wide open as the slide gates are not in use. This is not unusual for Wemco cells. In its current state, the lead flotation cleaning circuit is 1st stage cleaning only. The 2nd, 3rd, and 4th stage cleaners were deemed inoperable and removed during the 2006 re-commissioning by Hudson Bay Mining and Smelting Co.

The underground mine plan suggests that mill feed from underground sources will have lead values in the order of 0.02%. At this low level, it will not be necessary or economic to run the lead circuit. Currently, the lead flotation circuit is used to pre-float talc and magnesium. Excessive talc in the final concentrates results in high magnesium content and will incur penalties.

The open pit mine plan indicates that mill feed from open pit sources will have lead and silver grades that are high enough to produce a saleable lead / silver concentrate.

Various 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. Continue to use lead rougher and 1st stage cleaner as a talc "pre-float" to remove excessive talc.
- Bring lead circuit back to its original design by adding, at a minimum, 2nd and 3rd stage cleaners.
- Install a single vertical cell as final cleaning stage after 1st cleaner.

The second and third options are put forward with the intent of producing a marketable lead concentrate. This may require that mineralization source with higher than normal lead values such as those from the open pits, be handled separately, when feasible, so as not to dilute the lead values by co-mingling with underground mineralization. It is advisable that further benchwork be completed to prove that this approach significantly increases the ability of producing a marketable lead concentrate to justify the additional capital required. Beyond the expansion of the cleaning circuit, a moderate amount of civil work will be required on the lead thickener, cell dividers and center-well to deal with historic corrosion issues and ensure tightness. No issues are anticipated with the lead vacuum pump or disc filter.

17.2.5 Zinc flotation circuit

The zinc rougher circuit consists of two parallel banks of Wemco 300 ft^3 cells. There are six cells in #1 bank and seven cells in #2 bank.

At the end of #1 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.

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 used as a 'talc' pre-float the majority of the time.

The retention times in roughing and scavenging stages are 15 minutes and 8 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 the 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 17.3 and Table 17.5, 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	gal/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	gal/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: ESM operating data 2020. gal/m = gallons/minute.

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	gal/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	gal/m	640

Source: ESM operating data 2020. gal/m = gallons/minute.

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	gal/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	gal/m	326

Source: ESM operating data 2020. gal/m = gallons/minute.

17.2.6 Lead dewatering circuit

The lead thickener is 40’ in diameter and has been modified 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 as a submersible pump is 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 is 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 has been operated since 2009.

17.2.7 Zinc dewatering circuit

The zinc thickener is a 50’ diameter conventional Eimco unit. Thickener underflow is pumped directly to the vacuum filter. Inspection of the main framework indicated need for additional reinforcement. This work was completed during the refurbishment phase in 2017.

The zinc filter is an 8’10” Eimco disc type with seven of eight possible discs installed. The filter is in good condition and has operated without issue since the restart in 2018.

There are two Nash vacuum pumps. One is 100 hp and the other is 125 hp.

Zinc concentrate is conveyed to a 90 ft diameter by 45’ Koppers oil-fired dryer. It is also possible (with a reversible conveyor) to bypass the dryer. The filter cake typically has higher moisture during daily start-up and shut down but averages 8.5% moisture which does not require operation of the dryer. As is noted below, the dryer was operated until March 2019. Since then, it has been by-passed for cost reduction reasons as the reduction in moisture to 7% did not justify its operation. Mechanically, the dryer is in reasonable condition. The inside of the dryer was cleaned out on shutdown.

Dried zinc concentrate is conveyed to the loadout. The front-end loader is used to load trucks.

17.2.8 Ancillary equipment

17.2.8.1 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, 5100, and MIBC). There are three 12 ft diameter copper sulphate storage tanks on the bottom floor of the mill. All copper sulphate tanks have been removed from service.

A collection of diaphragm and peristaltic pumps (variable speed) with magnetic flowmeters are used for reagent distribution.

17.2.8.2 Lime mixing

The design capacity of the lime silo is 100 t. A drag chain conveyor delivers lime from the silo to a 4 ft x 3 ft Denver ball mill for slaking. The lime slaker is fully operational.

17.2.8.3 Process water pumps

There are three water pumps installed on the process water sump inside the mill.

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

17.3 Metallurgical balance

The concentrator mass balance in Table 17.6 shows estimated overall recovery 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 LOM.

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 2018.

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
- Plant not operating, winter

Water flowrates were provided in US gal/d, 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 2018.

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 2018.

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. Currently, the concentrator is producing zinc concentrate at an average of 59.0% zinc with 3% iron and 0.50% magnesium.

The current zinc dewatering equipment consists of a disc filter and rotary dryer. While this arrangement is considered to be largely obsolete, the equipment is in good working order and operates efficiently for its intended use. Since March 2019, the dryer has been bypassed in the interest of cost reduction and the concentrate dewatering has been accomplished by the vacuum disc filter alone. Aided in part by the relative coarseness of the concentrate, a moisture level of 8.5% has been achieved.

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 underground mineralization will be generally very low, and lead concentrate is not planned to be produced. Lead values in the open pit mineralization are expected to be higher and it will be possible to produce a lead concentrate from this mineralization source.
- Since re-commissioning, the recovery of zinc to zinc concentrate is typically over 96%.
- Moisture content of the zinc concentrate is 8.5% based on recent operating data.

17.7 Conclusions

While aged, the concentrator is in good working order and runs efficiently. No modifications are required to continue processing underground mineralization sources and minimal modifications would be required for processing the mineralized material to be mined from the open pits.

Since re-start, specific efforts have been made to modernize when opportunities arise. Examples of such work can be seen in rougher bank level control with the replacement of dart valve / end-box arrangements, replacement of DC motors with obsolete drives by AC motors with up-to-date VFDs and systematic upgrading of electronic controls. The concentrator does benefit from the fact that the operating schedule allows for adequate time for preventative maintenance.

The physical plant refurbishment commenced at the same time in 2017. Significant repairs were required to the steam system in the concentrator after 9 years of inactivity. Improvements were made to increase the capacity and quality of the potable water system. Compressed air is provided by a 7.5 hp IR and 15 hp IR air compressors. The main facility compressed air system provides instantaneous back-up.

The metallurgical laboratory is aged but has shown to be sufficient for the operation. Upgrades are scheduled for 2021. The laboratory maintains a relationship with an outside contract laboratory for the purpose of running comparison and duplicate sample exercises.

18 Project infrastructure

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 2018.

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 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. 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 #4 Mine shaft and main office facilities. The #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.

Minor upgrades to heating and water distribution and communications systems in these structures have been completed as part of the start-up.

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 main lobby 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. Hot water for sanitary purposes is provided by quick recovery propane water heater, eliminating the need to operate a steam boiler through the summer months.

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-ton overhead gantry crane and the machine shop has a 25-ton 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 north-western 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:

- General Manager
- Production Manager
- Mine Manager
- Mine clerk and surveying
- Engineering and geology personnel

- Conference room
- Accounting, purchasing, and human resources

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 ft x 138 ft area and contains a 15-ton overhead gantry crane. An adjoining compressor room houses a 150 hp Gardner Denver and 350 hp Sullair TS-32 air compressor. There is a bundle-type aftercooler in the discharge line. The compressor room has a 10-ton Load Lifter crane. Next to the compressor room is the electrical shop. This is equipped with a 5-ton Shaw Box crane.

18.3.2.1 No. 4 Shaft

Headframe

The 140 ft tall galvanized structural steel headframe was built in 1972 by Northeast Construction. The upper sheave deck supports two 15 ft diameter head sheaves grooved for 2 ¼” wire rope which services the production skip compartment. The lower sheave deck supports two 12 ft 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 ore bin, and a waste crib. The ore bin feeds an inclined mill conveyor over a 48” wide by 14’ 6” long 20 hp Portec apron feeder.

The Headframe has undergone a structural steel inspection as part of start-up activities and is currently in use.

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 2¼” 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 Table 18.1.

Table 18.1 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 – 2500	2
Powder delivery – 3100	4
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 2018.

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 analogue controls were replaced in 2001.

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 ft diameter by 91" wide drums each holding 3,300 ft of 1¾" 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.

18.3.2.2 No. 2 Shaft

Headframe

The hoist building and headframe is a brick and steel structure which supports two head sheaves 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 in acceptable condition and is capable of continued service as an emergency egress.

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 are 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 ft x 42 ft with a 25 ft x 19 ft switch gear room. A mine office / shaft complex (60 ft x 142 ft + 80 ft x 47 ft) is unheated.

18.3.5 Storage and miscellaneous facilities

The following building list in Table 18.2 makes up the rest of the facility.

Table 18.2 Facility building list

Building	Dimensions
Timber storage building	29’ x 118’
Electrical and 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’

Source: SLZ 2018.

Petroleum and chemical storage tanks at ESM are currently registered by the NYSDEC. All tanks and tank farms have containment areas.

18.4 Power

The primary feed for the ESM is 115 kV originating from National Grid's substation at Battle Hill-Balmat #5 circuit. Downstream from the main power supply are two (2) 7,500 kVA General Electric transformers that feed the ESM plant. Secondary voltage of 4,160 volts feeds sub-feeders to mill, mine, the No. 4 ventilation fan, lake pumps and booster pumps.

At the ESM No.4 main ventilation 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 National Grid 23 kV Balmat-Emeryville circuit #24. Downstream from the main power supply are two (2) 3,750 kVA General Electric transformers (23,000-2,200) feeding the surface plant with secondary voltage of 2,300 V for sub-feeders.

The No. 2 ventilation fan feeder is part of the mine feeder ventilation fan transformer 300 kVA in the substation by the ventilation 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 National Grid are:

- Street lighting for the mine entrance.
- South dam pumphouse at the tailings area.
- Environmental sampling station at SPDES permit final outfall designation.

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.

National Grid supplies the transmission and energy, although SLZ has the option to go to other energy suppliers.

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 (gal/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 milligrams (mg) to 1.5 mg of chlorine per litre (L) of water throughput. A Burks 5 hp pump will deliver 65 gal/m at 70 psi to feed a series of three bladder tanks (total drawdown capacity of 94 gal. between 40 psi and 60 psi) to be used for toilets and showers.

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 pump house with (3) Worthington 14-135-2, 75 hp pumps rated at 1,500 gal/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 two 100,000 gal tanks. 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 booster pump house via 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 ft of 14” diameter Sclairpipe. From the tailings area, it will flow north-east 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 is 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 are precipitated out of the water at this point. There is no need to run the dosing system for eight months per year due to the warmer temperatures. The warmer water promotes biomass activity that helps filter metals and other solids. The treated water drains by gravity over the 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 gal/d of fresh water is drawn from Sylvia Lake. ESM underground workings produce 379,000 gal/d of inflow. The mine inflow and process water are collected and pumped through the tailings pipeline to the tailings at a rate of 1,577,000 gal/d. Also, tailings area run-off adds to this volume so that the water treatment plant sees an average discharge at the SPEDES outfall of 2,350,000 gal/d.

During winter months, the water inflows into ESM increase to 491,000 gal/d. Also, during winter, the fresh-water intake from Sylvia Lake increases to 889,000 gal/d average. The tailings line discharge sees an average flow increase of 1,716,000 gal/d over the warmer months. Tailings area run-off adds to this volume so that the water dosing system sees an average discharge at the SPEDES outfall of 2,640,000 gal/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 tpa to 115,000 tpa of concentrate production.

18.6 Waste rock management

The mineralized material 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.

As much as possible, waste rock from the mine will remain in the underground and be used as backfill for drift and fill mining or deposited in completed longhole stopes. If it becomes necessary to hoist waste rock, it 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 ft 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 and Dam Safety. In addition to tailing, mine impacted water is also pumped to the TMF at a rate approximately 500 gal/m. The TMF is permitted as a discharge facility and continuously operates within compliance limits. Slaked lime and / or sodium sulphide is 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 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.

Future embankment raises will be needed to fully contain the current LOM plan tailings. The design of these raises and a future deposition schedule will be determined following the upcoming geotechnical review. This stage of construction will require approximately 445,000 cu yd of fill to be sourced from either mine waste or other local sources. Extrapolating forward from January 2021, the estimated remaining capacity within the active Tailing Pond #1 and without further embankment construction, will approximately be 2.75 years of production at 598,000 tons annually.

While the TMF is classified as a Class D – No Hazard, and there is no visible evidence to suggest otherwise, no 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. Based upon design drawings, it is assumed to be a combination of waste rock and tailings. The impoundment is classified as Low Hazard by Mine Safety and Health Administration.

A geotechnical assessment and engineering design are recommended to establish both of the above capacity estimates along with static and seismic stability. The first stage of this geotechnical assessment is scheduled for the second quarter of 2021.

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
- Polishing Ponds 25 acres

Tailings Pond 1 (TP1) is 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 Tailings Pond 2 (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.

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 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 exits the property boundary at a SPDES discharge point where flow measurements and compliance water quality samples will be taken. To achieve discharge standards, slaked lime is added at the mill to the combined tailing and mine water flow. At times, sodium sulphide may be added to the flow at head of polishing ponds.

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. Traffic on-site can be routed away from the main compound on a dedicated system of haul roads. Delivery of concentrate to the Glencore operated Canadian Electrolytic Zinc refinery in Valleyfield Quebec is undertaken following highways NY-812 N, NY-58 N, US-11 NE, NY-812 N, and in Canada following highways 401 and 201.

19 Market studies and contracts

19.1 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 2018.

19.1.1 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

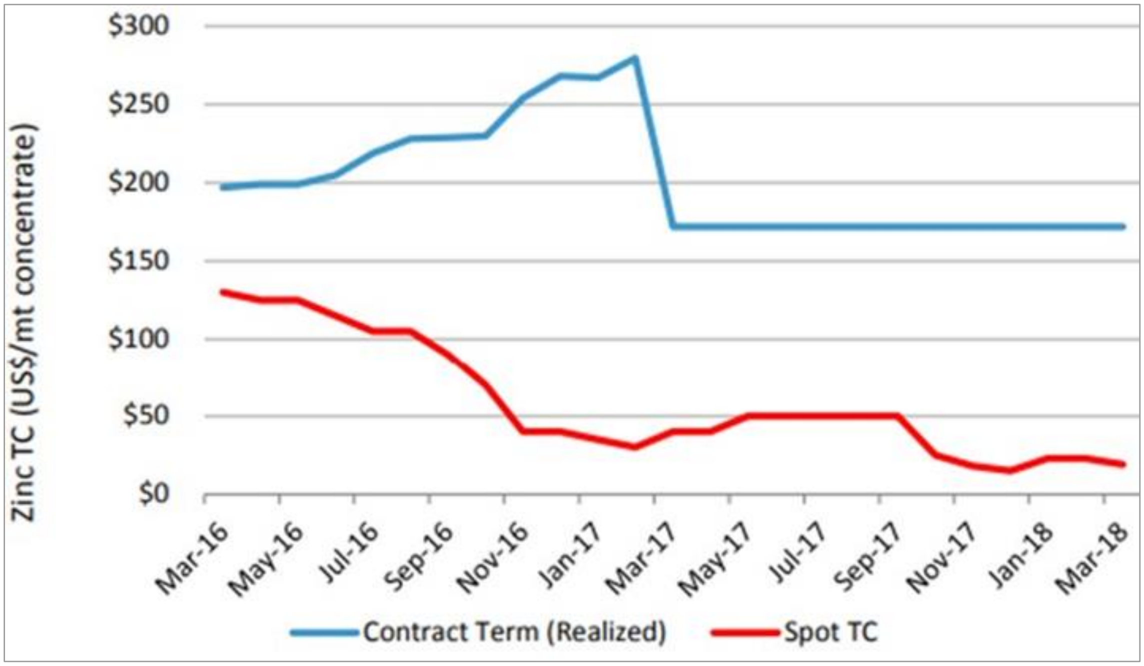
Source: JDS.

19.2 Zinc concentrate terms

Although there have been efforts to adjust the industry standard zinc payable formula to better reflect actual recoveries, zinc smelters generally pay for 85% of the value of contained zinc metal in concentrates typically for 56% zinc. Additional payable by-products may include gold and silver when levels are sufficiently high enough. Penalties may be assessed to concentrates containing impurities such as iron, cadmium, lead, manganese, cobalt, magnesia, and / or mercury above threshold values.

Historical treatment charges for 2016 to 2018 are shown in Figure 19.1. In 2018 treatment charges were set at a 12-year low of $147/dmt. 2019 and 2020 saw steady increases with record highs up to $300/dmt.

Figure 19.1 Zinc smelter treatment charges



Source: Wood Mackenzie, Scotiabank GBM 2018.

The PEA assumptions that are reflected in the project economics and assessment reflect the terms of the confidential agreement in place with Glencore. An offtake agreement is in place with Glencore for 100% of the zinc concentrate from ESM. The long-term contract commenced on the first production of concentrate from ESM. Assumed treatment charges for the zinc concentrates are shown in Table 19.3.

Table 19.3 Zinc concentrate treatment charge assumptions

	2021	2022	2023	2024	2025	2026	2027
Zinc treatment charge ($/dmt)	210	190	170	220	235	250	250

Source: Ocean Partners 2020.

19.3 Lead concentrate terms

The lead feed grade is not high enough in the underground mine to prepare a payable lead concentrate, however the open pit lead grade is suitable. Titan has allowed for capital expenditure to upgrade to the lead circuit in the process plant in order to produce a lead concentrate during open pit mining. Assumptions for payable lead is 95% or a minimum deduction of three units, whichever is less. Lead concentrate grade is estimated to be 55%. It is assumed that 95% of the silver in the lead concentrate will be payable or a minimum deduction of 50 grammes per dry metric tonne (g/dmt), whichever is less.

Assumed treatment charges for lead concentrate are shown in Table 19.4. Refining charges of $1.50/oz are assumed for silver.

Table 19.4 Lead concentrate treatment charge assumptions

	2021	2022	2023	2024	2025	2026	2027
Lead treatment charge ($/dmt)	135	150	170	180	200	190	190

Source: Ocean Partners 2020.

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 monitors the sites routinely as part of their ongoing management practices.

The waste rock and tails are non-acid generating so there are no issues or concerns with material reactivity. 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 SPDES permit. Water quality parameters are in compliance with surface water discharge permits.

20.2 Permitting

All permits required to operate the ESM #4 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 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.

Renewals for SPDES Permit and Water Withdrawal Permit were submitted to the NYSDEC in a timely manner. Both permits are on the Department's schedule for technical review due to length of time elapsed since previous review. Both permits remain in force as written despite listed expiry dates.

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	30 Sep 2024
Water	SPDES Water Discharge Permit	NY0001791	31 May 2019*
Water	Water Withdrawal Permit	6-4038-00024/02001	31 May 2019*
Mining	Mining Permit	6-4038-00024/00006	31 Jul 2025
Storage	NYDEC Chemical Bulk Storage	CBS#6-000122	1 Oct 2021
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770	26 Sep 2023
Radiation	Certificate of Registration for Radiation Installation - XRF	44023174	15 Sep 2022
Public Water Supply	No permit required, but regulated by NYS Dept. of Health Registered ID #NY4430004	Registered ID #NY4430004	None
Hazardous Material Transport	US Department of Transportation Registration – Pipeline and Hazardous Material Safety Administration	072216 550 004Y	30 Jun 2021

Source: ESM 2018.

Tailings storage and management is discussed in detail in Section 18.7 of this report. Tailings are non-acid generating so conventional reclamation methods can be used to rehabilitate the tailings 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 and 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 and No. 3 mines, plus flow from Gouverneur Minerals' 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.

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. The water table elevation is estimated to return to an elevation of approximately 652 ft amsl. Mine openings intersecting the ground surface are all above that elevation with the lowest being the No. 2 Mine ventilation 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 accumulate within this pit.

An August 2012 memo from SRK to Hudbay (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; 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 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 $1,627,341 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 re-establish.

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, 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 five 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
- 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 ditch, North Dam spillway, run-off pond
	Annual	Sylvia Lake, Mine reflood
Years 6 – 10	Quarterly	SPDES permit station, South Dam discharge ditch, interception ditch, North Dam spillway, run-off pond
	Annual	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

Note: 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 / Administration	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 in the area (community hotels, restaurants, grocery stores, retail stores) have 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. ESM 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 change, new owners could be less appreciative of the benefits the mine has historically provided to the community.

There are no known social or community relations issues that would adversely impact the ESM.

21 Capital and operating costs

21.1 Capital cost estimate

21.1.1 Capital cost summary and estimate results

Estimated project capital costs (including closures costs) total $19.1M, consisting of the following distinct areas:

- #2 Mine pre-production
- #4 Mine capital equipment
- #4 infrastructure and process capital

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

Table 21.1 presents the capital estimate summary for each area in Q4 2020 US$ with no escalation.

Table 21.1 Capital cost summary

Area	Cost estimate ($M)
#2 Mine pre-production capital	3.1
#4 Mine capital equipment	5.2
#4 infrastructure and process capital	2.9
Total capital cost	**11.1**
Closure costs	11.9
Salvage value	4.0
Total capital cost (incl. closure costs)	**19.1**

Source: Titan / AMC 2021.

21.1.2 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 1 January 2021. 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$.

21.1.3 Basis of estimate

21.1.3.1 Open pit mine (#2 Mine)

AMC has assumed that, due to the short life of the pits (three years), a contractor will be used to mine the open pits. Mark-ups on the operating costs have been assumed to cover the contractor's mining equipment and infrastructure capital costs.

Capital item allowance for the open pit includes upgrade of the railway right of way into a haul road, land acquisition, process plant upgrade for lead circuit, and site facility preparation. AMC estimates that mine waste rock can be placed, spread, and compacted onto the roadway with the mining fleet in the same way as it would be placed on the waste dump. Therefore, there would be minimal extra cost for the road building per se. However, there are drainage systems to be placed and environmental and engineering studies to be completed. An allowance of $750,000 has been made for the haul road capex.

Titan has an agreement in principle with Valderbilt in the amount of $120,000, which involves acquiring rights to dump in the Valderbilt pit. There has been no mention of costs involved with compensation to the neighbours residing to the east of the pits. AMC is not aware of any additional costs arising from the adjacent property owners due to the onset of open pit mining as proposed.

Other capital costs required for the open pit mining are the permitting, demolition of #2 facilities, replacement of #2 hoist to maintain secondary egress, land acquisition, bonding of approximated 100 acres (at a unit rate of $4,500/acre), and the upgrade of the mill by refurbishing the lead circuit.

Table 21.2 presents the capital cost distribution for the #2 Mine pre-production phase.

Table 21.2 Distribution of #2 Mine pre-production capital costs

Description	**$ (x 1,000)**
Permitting	**85**
#2 facilities demolition	500
Replace #2 hoist to maintain secondary egress	900
Land acquisition	120
Bonding	450
Mill upgrades (Pb circuit refurbishment)	300
Rail route road and surface prep	750
Total	**3,105**

Source: Titan / AMC 2021.

21.1.3.2 Underground mine (#4 Mine)

Underground capital costs are estimated to be $5.2M, which include the lease purchase of one bolter and two 6-yard loaders, mobile equipment rebuilds, replacement of one single-boom Jumbo drill, one bolter, one lift truck and service cage, and purchases of a StopeMaster longhole drill, a 40T haul truck, 750 KW transformer and a leaky feeder head.

Table 21.3 presents the capital cost distribution for the #4 Mine capital equipment.

Table 21.3 Distribution of #4 Mine capital equipment costs

Description	**$ (x 1,000)**
Bolter - Mahler - lease purchase	775
2 x 6 yd loader - lease purchase	896
Mobile equipment rebuilds	800
Replacement service cage	130
Bolter (replacement)	750
Jumbo - single boom (replacement)	750
Lift truck (replacement)	240
750 KW transformer	225
StopeMaster	78
Leaky feeder head	30
40T haul truck	500
Total	**5,174**

Source: Titan 2021.

21.1.3.3 Infrastructure and processing cost estimate

Total infrastructure and processing capital costs are estimated to be $2.9M.

Processing capital costs include some equipment repairs, inspections and relining of the ball and rod mill, mill building roof repairs, and mill sustaining costs.

Infrastructure capital costs include headframe repairs and upgrades, counterweight rail replacement, septic system repairs, headframe apron feeder rebuild, vacuum pump rebuild, tailings storage facility (TSF) lifting, main water tank isolation valves, new ropes for #2 Hoist, XRF replacement for the assay laboratory, and production hoist drum bushings.

All costs are based on quotations. Table 21.4 presents the capital cost distribution for the #4 infrastructure and process capital.

Table 21.4 Distribution of #4 infrastructure and process costs

Description	$ (x 1,000)
Headframe repairs & upgrades (#2 and #4)	150
Counterweight rail replacement	425
Ball mill reline	240
Rod mill reline	432
Roof repair	210
Septic system repairs	25
Headframe apron feeder rebuild	50
Vacuum pump rebuild	25
TSF study for lift	150
TSF lift - initial phase	250
Main water tank isolation valves	50
New ropes - #2	50
Replacement XRF for laboratory	155
Mill sustaining - pumps and motors	400
Production hoist drum bushings	141
Total	**2,853**

Source: Titan 2021.

21.1.3.4 Closure costs and salvage value

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
- Mine openings

Closure costs were estimated based on the SRK cost estimate adjusted for the Consumer Price Index from 2014 to 2018 US$ and now total $11.9M. The majority of the physical closure work will occur over a two-year period. Monitoring and environmental management costs would continue for another 23- years, as estimated by SRK, totalling $1.1M. The details of the closure costs is summarized in Table 21.5.

Table 21.5 Closure cost summary

Closure costs	**Total ($ x 1,000)**	**Closure Year 1 ($ x 1,000)**	**Closure Year 2 ($ x 1,000)**	**Closure Year 3-26 ($ x 1,000)**
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: Titan, from SRK 2018.

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. The salvage value is estimated at $4M.

Closure costs and salvage values were not included in the economic model as the mine has continued for decades with 5 to 8 years of mineable resource in front of it. Titan fully expects that to continue as the mine is running three drills in the underground and one on surface.

21.1.3.5 Indirect, owner’s, and contingency costs

Indirect, owner’s, and contingency costs are all incorporated into the capital cost estimates.

21.1.3.6 Capital estimate exclusions

The following items have been excluded from the capital cost estimate:

- Working capital.
- 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.
- Escalation cost.

21.2 Operating cost estimate

21.2.1 Site operating cost summary

Estimated project operating costs total $231M or $69.85/t milled.

Preparation of the site operating cost estimate is based on current underground operation performance. 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. Open pit operating costs were estimated by AMC and are discussed in detail in Section 21.2.2.

Site operating costs in this section of the report is broken into three major sections, which include mining, processing, and general and administrative (G&A) costs.

Site operating costs are presented in 2020 US$ on a calendar year basis. No escalation or inflation is included.

Table 21.6 presents the operating cost distribution, which is broken down by underground, open pit, and the combination of the underground and open pit. Since #2 pits are considered incremental mineralization, an incremental cost for the processing and G&A have been assigned.

Table 21.6 Breakdown of estimated site operating costs

Site operating costs	Unit cost ($/t milled)	LOM cost ($M)
Underground		
Mining	43.00	114.0
Processing	14.00	37.1
G&A	22.00	58.3
Underground total	**79.00**	**209.4**
Open pit		
Mining	20.07	13.2
Processing	7.00	4.6
G&A	5.92	3.9
Open pit total	**32.99**	**21.7**
Underground and open pit		
Mining	38.44	127.2
Processing	12.61	41.7
G&A	18.80	62.2
Underground and open pit total	**69.85**	**231.1**

Source: Titan / AMC 2021.

21.2.2 Open pit mine operating cost estimate

The operating cost estimate allows for all labour, equipment, supplies, fuel, consumables, and supervision and is supplied entirely by the contractor. Titan has allowed for 3 extra staff to overlook the open pit operations and these three persons are included in site G&A costs.

AMC estimated open pit mining costs assuming a contractor mining operation throughout the LOM of the pit. Estimated costs for the proposed fleet and labour were sourced from AMC's database and benchmarked against knowledge of similar sized and local operations. The LOM average mining cost is approximately $3.37/t mined.

AMC estimated open pit mining costs assuming a contractor mining operation. Estimated costs for the proposed fleet and labour were sourced from AMC's database, and internet searches on New York state Department of Transport rental rates and "Blue Book" dry hire rental rates. AMC believes that in general, the internet based rental rates are higher than what a contractor would quote and therefore hourly rates have been chosen that are more aligned with AMC's database numbers.

A summary of the benchmark cost split and AMC's estimate for the open pits is provided in Table 21.7.

Table 21.7 Summary of estimated open pit operating cost

Category	%	Cost estimate ($/t mined)
Drill and blast	24.0	0.81
Load	8.5	0.29
Haul	8.9	0.30
Ancillary	7.4	0.25
Labour	51.2	1.72
Total	**100.0**	**3.37**

Source: AMC 2020.

21.2.3 Underground operating cost estimate

The operating cost estimate for the underground mine is based on actual operating data from the past year so is considered highly accurate. Mining, milling, and G&A costs for 2020 are considered to be representative of operating costs going forward. Site operating costs for the underground are summarized in Table 21.8.

Table 21.8 Summary of underground operating cost

Underground	Unit cost ($/t milled)	LOM cost ($M)
Mining	43.00	114.0
Processing	14.00	37.1
G&A	22.00	58.3
Total	**79.00**	**209.4**

Source: Titan 2020.

21.2.4 Summary of site personnel

Table 21.9 Summary of site personnel

Position	Staff / hourly	Total
Mining		
Mine management	2/0	2
Mine operations	0/44	44
Mine maintenance	2/17	19
Crush, hoist, shaft	0/9	9
Processing		
Process management	1/0	1
Process operations	0/12	12
Process and surface maintenance	0/6	6
G&A		
General manager	1/0	1
Accounting	3/0	3
Technical services	9/0	9
Warehouse	4/0	4
Human resources	3/0	3
Safety and environment	3/0	3
Site total	**28/88**	**116**

Source: Titan 2020.

Site personnel is based on current staffing levels plus projected requirements for the open pit operations, which are expected to be minimal. The site is currently operating with 110 full time employees.

21.2.5 UG mining operating cost

The underground mine is currently operating and will continue to be operated by company personnel with no contractors. Operating costs are representative of actual mining costs which are currently running at $43 per ton milled. The underground mining cost is summarized in Table 21.10.

Table 21.10 Summary of underground mining cost

UG Mining	Unit cost ($/t milled)	LOM cost ($M)
Labour	25.65	68.0
Supplies	12.50	33.1
Energy	1.85	4.9
Services	2.30	6.1
Admin	0.70	1.9
Total	**43.00**	**114.0**

Source: Titan 2020.

Mining labour includes all production and underground maintenance labour as well as mine administration labour. Supplies include all production related supplies and maintenance related supplies. Energy includes diesel. Services include all external services contracted to the mine department.

The process operating cost is summarized in Table 21.11. Mill labour includes all mill and surface maintenance labour as well as mill administration labour. Supplies include all process reagents and related supplies and maintenance related supplies. Energy includes diesel. All site electrical power is accounted for in the process category. Services include all external services contracted to the mill department.

Table 21.11 Summary of processing operating cost

Process	**Unit cost ($/t milled)**	**LOM cost ($M)**
Labour	5.00	13.3
Supplies	3.27	8.7
Energy	4.85	12.9
Services	0.67	1.8
Admin	0.18	0.5
Total	**14.00**	**37.1**

Source: Titan 2020.

The G&A operating cost is summarized in Table 21.12. G&A labour includes all administration labour as well as engineering and geology. Supplies include all administration and related supplies. Energy includes diesel. Services include all insurance, property and school taxes, and external services contracted to the administration areas.

Table 21.12 Summary of G&A operating cost

G&A	**Unit cost ($/t milled)**	**LOM cost ($M)**
Labour	9.31	24.7
Supplies	0.05	0.1
Energy	0.00	0.0
Services	2.15	5.7
Admin	10.49	27.8
Total	**22.00**	**58.3**

Source: Titan 2020.

22 Economic analysis

22.1 Introduction

An economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and 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 Sections 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.

It must be noted that 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.

22.2 LOM summary and assumptions

Table 22.1 summarizes parameters and assumptions pertinent to the seven-year mine life that were used in the economic analysis.

Table 22.1 LOM plan summary

Parameter	Unit	Value
Mine life	Years	7.0
Underground waste	kt	504
Open pit waste	kt	3,262
Total waste	kt	3,766
Underground mineralization	kt	2,650
Open pit mineralization	kt	658
Total plant feed material	kt	3.309
Throughput rate	t/d	1,294
Average zinc price	$/lb	1.15
Average lead price	$/lb	0.87
Average head zinc grade	% Zn	6.6
Average head lead grade	% Pb	0.4

Source: Titan / AMC 2021.

Other economic factors include the following:

- Discount rate of 8%.
- Nominal 2021 dollars.
- Revenues, costs, taxes are calculated for each period in which they occur.
- All costs and time prior to 1 January 2021 are considered sunk costs.
- Results are presented on 100% ownership basis.

22.3 Revenues and net revenue parameters

Mine revenue is derived from the sale of zinc concentrate and lead concentrate into the international marketplace. No silver revenue is generated from the lead concentrate since the low silver grade does not pay for the deductions. Details regarding the terms used for the economic analysis can be found in the Market Studies (Section 19) of this report.

Table 22.2 indicates the Net Revenue (NR) parameters that were used in the economic analysis.

Table 22.2 Net revenue parameters

Parameter	Unit	Value
Mine operating days	Days/a	365
Lead recovery from process plant (#2 Mine)	%	85
Zinc recovery from process plant (#2 Mine)	%	94
Zinc recovery from process plant (#4 Mine)	%	96

Source: Titan 2020.

22.4 Taxes

The project has been evaluated on an after-tax basis to provide an indicative value of the potential project economics. A preliminary tax model was prepared by AMC and Titan. The tax model contains the following assumptions:

- 21% federal income tax rate.
- 6.5% New York state income tax.
- Total taxes for the LOM $8.4M.

22.5 Royalties

The economic analysis incorporates royalties. A royalty of 0.3% is applied to the NSR for the zinc concentrate. However, it is assumed that there are no royalties for the sale of the lead concentrate.

22.6 Results

The economics for the open pit alone have been analyzed and the results shown are economically feasible. The project, combined with the underground mine and open pit mine, is economically viable with an after-tax NPV of $81M at an 8% discount rate.

Table 22.3 summarizes the economic results. Table 22.4 shows the pre-tax and post-tax projected cash flows for the project.

Table 22.3 Summary of results

Summary of results	**Unit**	**Value**
Mine life	Years	7.0
Resource mined	kt	3,309
LOM throughput rate	t/d	1,294
Average head zinc grade	% Zn	6.6
Average head lead grade	% Pb	0.4
LOM recovered zinc	Mlbs	470
LOM recovered lead	Mlbs	10
LOM payable zinc	Mlbs	400
LOM payable lead	Mlbs	9.5
Revenue by commodity (zinc)	%	98
Revenue by commodity (lead)	%	2
Zinc revenue	$M	460
Lead revenue	$M	8
Total revenue	$M	468
Total offsite charges	$M	113
Royalties	$M	1
NSR (net of royalties)	$M	349
Capital costs (including sustaining)	$M	11
Operating costs	$M	231
Operating costs	$/t processed	69.85
Pre-tax cash flow	$M	107
Taxes	$M	8
After-tax cash flow	$M	98
Pre-tax NPV (8% discount)	**$M**	**88**
After-tax NPV (8% discount)	**$M**	**81**

Source: AMC 2021.

Table 22.4 Cash flow model for ESM

Item	**Unit**	**LOM**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**
Zinc price	$/lb	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.15
Lead price	$/lb	0.87	0.82	0.82	0.86	0.91	0.91	0.91	0.91
Tons mined UG	000s tons	2,650	375	390	390	390	390	390	325
Zinc grade	%	8.5	8.6	8.7	9.2	8.8	8.3	8.1	7.8
Contained zinc	000s lbs	450,371	64,345	67,704	71,575	68,396	64,400	62,965	50,987
Mineralization mined OP	000s tons	658	69	275	275	40	-	-	-
Total waste OP	000s tons	3,262	325	1,450	1,331	156	-	-	-
Stripping ratio OP		5.0	4.7	5.3	4.8	3.9	-	-	-
Total material moved OP	000s tons	3,921	394	1,725	1,606	196	-	-	-
Zinc grade	%	3.1	2.5	2.9	3.3	3.3	-	-	-
Lead grade	%	0.9	0.8	1.3	0.6	0.4	-	-	-
Contained zinc	000s lbs	40,364	3,454	15,968	18,321	2,621	-	-	-
Contained lead	000s lbs	11,875	1,146	7,308	3,129	293	-	-	-

Item	Unit	LOM	2021	2022	2023	2024	2025	2026	2027
Mineralization processed	000s tons	3,309	444	665	665	430	390	390	325
Zinc grade	%	6.6	7.6	6.3	6.8	8.3	8.3	8.1	7.8
Lead grade	%	0.4	0.1	0.5	0.2	0.0	0.0	0.0	0.0
Contained zinc	000s lbs	490,735	67,799	83,672	89,896	71,016	64,400	62,965	50,987
Contained lead	000s lbs	11,875	1,146	7,308	3,129	293	-	-	-
Zinc concentrate produced	000s dst	405	56	69	74	59	53	52	42
Shipping weight	000s wst	441	61	75	81	64	58	57	46
Zinc in concentrate	000s lbs	470,298	65,018	80,006	85,934	68,123	61,824	60,446	48,947
Lead concentrate produced	000s dst	9	1	6	2	0	-	-	-
Shipping weight	000s wst	10	1	6	3	0	-	-	-
Lead in concentrate	000s lbs	10,094	974	6,212	2,660	249	-	-	-
Payable zinc	000s lbs	399,754	55,266	68,005	73,044	57,905	52,550	51,379	41,605
Gross metal value - zinc	000s $	540,843	74,771	92,007	98,824	78,342	71,097	69,513	56,289
Payable zinc value	000s $	459,717	63,555	78,206	84,000	66,590	60,433	59,086	47,846
Less treatment charges	000s $	87,657	11,949	13,452	13,105	13,053	12,571	13,000	10,527
Less penalties	000s $	12,285	1,698	2,090	2,245	1,779	1,615	1,579	1,279
Less transportation costs	000s $	15,424	2,132	2,624	2,818	2,234	2,028	1,982	1,605
NSR value	000s $	344,351	47,775	60,040	65,832	49,524	44,219	42,525	34,435
Less royalties	000s $	1,033	143	180	197	149	133	128	103
Revenue - zinc	000s $	343,318	47,632	59,860	65,635	49,376	44,087	42,397	34,332
Payable lead	000s lbs	9,543	921	5,873	2,514	235	-	-	-
Gross metal value - lead	000s $	8,406	799	5,093	2,287	227	-	-	-
Payable lead value	000s $	7,947	755	4,816	2,162	214	-	-	-
Less treatment charges	000s $	1,287	108	768	373	37	-	-	-
Less penalties	000s $	416	40	256	110	10	-	-	-
Less transportation costs	000s $	798	77	491	210	20	-	-	-
NSR value	000s $	5,446	529	3,300	1,470	147	-	-	-
Revenue - lead	000s $	5,446	529	3,300	1,470	147	-	-	-
Zinc concentrate	000s $	343,318	47,632	59,860	65,635	49,376	44,087	42,397	34,332
Lead concentrate	000s $	5,446	529	3,300	1,470	147	-	-	-
Total revenue	000s $	348,765	48,162	63,160	67,105	49,523	44,087	42,397	34,332
#4 Infrastructure & process capital	000s $	2,853	1,516	701	100	220	316	-	-
#4 Mining capital equipment	000s $	5,174	1,576	2,323	1,175	100	-	-	-
#2 Mine pre-production capital	000s $	3,105	2,605	500	-	-	-	-	-
Total capital costs	000s $	11,132	5,697	3,524	1,275	320	316	-	-
Mining	000s $	113,966	16,137	16,770	16,770	16,770	16,770	16,770	13,979
Processing	000s $	37,105	5,254	5,460	5,460	5,460	5,460	5,460	4,551
G&A	000s $	58,308	8,256	8,580	8,580	8,580	8,580	8,580	7,152
Total #4 Mine operating costs	000s $	209,379	29,647	30,810	30,810	30,810	30,810	30,810	25,682
Mining	000s $	13,212	1,327	5,813	5,413	659	-	-	-
Processing	000s $	4,608	481	1,925	1,925	277	-	-	-
G&A	000s $	3,900	300	1,200	1,200	1,200	-	-	-
Total #2 Mine operating costs	000s $	21,721	2,108	8,938	8,538	2,136	-	-	-
Total operating costs	000s $	231,100	31,756	39,748	39,348	32,946	30,810	30,810	25,682

Item	Unit	LOM	2021	2022	2023	2024	2025	2026	2027
Revenue	000s $	348,765	48,162	63,160	67,105	49,523	44,087	42,397	34,332
Capital costs	000s $	11,132	5,697	3,524	1,275	320	316	-	-
Operating costs	000s $	231,100	31,756	39,748	39,348	32,946	30,810	30,810	25,682
Pre-tax net cash flow	000s $	106,533	10,709	19,888	26,482	16,257	12,961	11,587	8,650
Cumulative pre-tax net cash flow	000s $	-	10,709	30,597	57,078	73,335	86,296	97,883	106,533
Pre-tax net present value (8%)	000s $	87,596	10,709	18,415	22,704	12,905	9,526	7,886	5,451
Net income before tax	000s $	106,533	10,709	19,888	26,482	16,257	12,961	11,587	8,650
Corporate tax	000s $	8,383	592	1,866	2,868	1,202	879	687	290
Post-tax net cash flow	000s $	98,150	10,117	18,022	23,614	15,054	12,082	10,900	8,360
Cumulative post-tax net cash flow	000s $	-	10,117	28,140	51,753	66,808	78,890	89,790	98,150
Post-tax net present value (8%)	000s $	80,568	10,117	16,687	20,245	11,951	8,881	7,418	5,268

Source: AMC 2021.

22.7 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 price, then zinc grade, followed by operating costs and capital costs. Table 22.5 outlines the results of the sensitivity tests performed on pre-tax and after-tax NPV at 8%.

The project was also tested under various discount rates. The results of these tests are demonstrated in Table 22.6.

Table 22.5 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
Zinc price	13	88	162	13	81	144
Zinc grade	31	88	144	31	81	128
CAPEX	90	88	85	83	81	78
OPEX	125	88	50	112	81	48

Source: AMC 2021.

Table 22.6 Discount rate sensitivities

Discount rate (%)	Pre-tax NPV ($M)	After-tax NPV ($M)
0	107	98
5	94	86
8	88	81
10	84	77
12	80	74

Source: AMC 2021.

23 Adjacent properties

There are no adjacent properties relevant to the scope of this report.

24 Other relevant data and information

There is no other relevant data or information relative to the scope of this report.

25 Interpretation and conclusions

ESM began operating over 100 years ago (from 1915) and has a proven track record of replacing Mineral Reserves with continued exploration efforts; it is also a past producer with demonstrated production rates and metal recoveries well within the LOM plan. The mine is fully developed with shaft access and mobile equipment on-site. The mine and its facilities were maintained to good standards during the period of care and maintenance.

ESM is comprised of multiple deposits in and around Fowler, NY. There are ten deposits currently considered as viable economic targets. Historic mining at these locations has provided a good geological understanding of each, with supporting mapping, sampling, and drilling data.

This Mineral Resource report has been created through a collaboration between ESM and SRK and has been prepared under the Canadian NI 43-101 guidelines. A comprehensive re-modelling effort was undertaken by ESM in 2018 using Leapfrog Geo for all geological models. Mining and grade control experience by ESM geologists have supported that the implicit modelling of the mineralized zones as veins in Leapfrog Geo, results in more accurate geological wireframes.

The ten deposit zones were defined and modelled by ESM geologists. Each one is comprised of multiple veins designating variably oriented and spatially-distinct mineralized zones which were modelled using combinations of explicit and implicit methods. Input data for these models are based on drilling intercepts and years of surface and underground mapping.

Underground Mineral Resources have been modelled (Leapfrog Geo) and estimated (Maptek Vulcan) by ESM geologists and reviewed for consistency with industry standards by SRK. Mineral Resources for the underground Number 4 mine areas have been compiled from ten separate block models including the American, Cal Marble, Davis, Fowler, Mahler, Mud Pond, Number 2 Deeps, North East Fowler, New Fold, and Silvia Lake areas.

Open-pit Number 2 Mine Mineral Resources have also been modelled (Leapfrog Geo) and estimated (Leapfrog EDGE) by ESM geologists and reviewed for consistency with industry standards by SRK. Mineral Resources for the Number 2 Mine Open Pit area have been taken from a single block model which features the Hoist House, Pump House, and Turnpike areas.

The ESM deposit will be extracted using a combination of LGS, C&F, Panel Mining – Primary and Secondary and development drifting underground mining methods with rock backfill. Longhole backstopes are also used in the design where applicable. The proposed combined underground and open pit mine plan is expected to reach an initial target production rate of 1,200 t/d for 2021 and ramp up to 1,800 t/d in 2022. Open pit mining will be completed in Year three (2023). The overall mine life will be five years.

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.

Mineralized material mined in the ESM deposits is processed at the existing ESM concentrator that was commissioned in 1970 and last shut down in 2008. The concentrator was refurbished in late 2017 and began processing mineralization in 2018. The concentrator flowsheet includes crushing, grinding, sequential lead and zinc flotation circuits, concentrate dewatering circuits, and loadout facilities. The design capacity of the concentrator is 5,000 t/d. Through-out the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines' capacity.

The operating strategy is to operate the concentrator at its rated hourly throughput of 200 t/h to 220 t/h, but for only as many hours as necessary to suit mine production.

The recently discovered open pit prospects can be processed using sequential flotation process to produce separate lead and zinc concentrate. The mineralization from Turnpike and Hoist House prospects are slightly harder than the current mineralization being processed in the plant. The lead recovery and concentrate grade are dependent on the feed grade. The higher the feed grade, the higher the final concentrate recovery and grade.

Lead values in the underground mineralization will be generally very low, and lead concentrate is not planned to be produced. Lead values in the open pit mineralization are expected to be higher and it will be possible to produce a lead concentrate from this mineralization source. The cleaner flotation circuit will recover ±95% of lead recovered in the rougher flotation stage. Hence, the overall recovery of lead is projected to be 80% to 85%. The zinc recovery will be similar to that obtained with the underground mineralization.

While aged, the concentrator is in good working order and runs efficiently. No modifications are required to continue processing underground mineralization sources and minimal modifications would be required for processing the mineralized material to be mined from the open pits.

All permits required to operate the ESM #4 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.

Tailings are non-acid generating so conventional reclamation methods can be used to rehabilitate the tailings 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.

The results of the economic evaluation indicate that the project is economic under the current assumptions. The pre-tax cash flow is estimated to be $107M, with a Pre-tax and post-tax NPV at a discount rate of 8% of $88M and $81M respectively. A sensitivity analysis revealed that the project is most sensitive to zinc price, then zinc grade, followed by operating costs and capital costs.

The most significant risks associated with the project are commodity prices, uncontrolled dilution, mineral recovery, operating and sustaining capital cost escalation, ventilation limitations and Inferred Mineral Resource confidence.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

25.1 Risks

The main risks to the project are summarized in Table 25.1.

Table 25.1 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. Mine designs need to be customized to the mineralization geometry to minimize external dilution. On shift grade control geologists to follow the mining. Focused grade control efforts have been successful, and results of current work appear to be achieving desired results.
Resource modelling	All Mineral Resource estimates carry some risk and are one of the most common issues with project success. The majority of the Mineral 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 is required to convert Inferred Mineral Resources to Measured and Indicated.
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, mill reagents, liners and explosives could be higher than planned. This will negatively affect operating costs.	Consider long term contracts for major 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 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 are important elements of project success.	Improvement of cost estimation accuracy with the next level of study, and the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.

Source: AMC 2020.

25.2 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 25.2 Identified project opportunities

Opportunity	Explanation	Potential Benefit
Resource expansion	The mineralized zones have not been fully delineated and there is an opportunity to expand the Mineral Resource.	Increased mine life and increased project Net Present Value.
Mine plan expansion	Resource zones added may add significant mineable tons to the LOM plan.	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.

Source: AMC 2020.

26 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, channel sampling, and re-assay of existing drillholes to gain resolution and accuracy of the resource and to upgrade the resource classification of Inferred Mineral Resource.
- Evaluate geotechnical conditions of longhole stoping to support the stope and pillar dimensions used in this PEA, and to provide guidance on ground support requirements.
- Confirm the geotechnical conditions in the vicinity of the planned open pits, particularly the orientation of the discontinuities, in order to support the slope geometry used in this PEA.
- Conduct optical sorting test work to test the ability to separate mineral from waste before entering the mill facility. Perform an integration study to assess the impact of the system on the mine and the logistics of application.
- Obtain contractor quotes for open pit mining to improve estimate accuracy in the next level of study.

Table 26.1 shows the cost of the recommended additional definition drilling and engineering field and test programs.

Table 26.1 Project recommendations and cost

Item	**Cost ($)**
Infill drilling and conversion of Inferred Mineral Resources	1,500,000
Geotechnical review	50,000
Sorting test work and integration study	100,000
Contractor quotes for open pit cost assumptions	15,000
Total estimate	**1,665,000**

Source: AMC 2020.

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28 QP certificates

CERTIFICATE OF AUTHOR

I, David Warren, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.
2. This certificate applies to the technical report titled "Empire State Mines 2021 NI 43-101 Technical Report", with an effective date of 24 February 2021, (the "Technical Report") prepared for Titan Mining Corporation ("the Issuer").
3. I am a graduate of University of British Columbia (B.A.Sc. 1978) and Helsinki University of Technology in Helsinki, Finland (M.Sc. 1997). I am a registered member in good standing with Engineers and Geoscientists British Columbia (License #15053) and a member of the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM). I have practiced my profession continuously since 1978, and have been involved in open pit mine operations engineering, mine optimization, design and planning, due diligence and technical reviews, feasibility studies, operational reviews and improvement, project management, and mine financial analysis.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Project site on 18 February 2020, for one day.
6. I am responsible for parts of Sections 1, 16, 21, and 25-27 of the Technical Report;
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
8. I have not had prior involvement with the property that is the subject of the Technical Report;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 24 February 2021
Signing Date: 23 March 2021

Original Signed and Sealed by

David A. Warren, P.Eng.

CERTIFICATE OF AUTHOR

I, Gary Methven, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Mining Engineer and Underground Manager with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.

2 This certificate applies to the technical report titled "Empire State Mines 2021 NI 43-101 Technical Report", with an effective date of 24 February 2021, (the "Technical Report") prepared for Titan Mining Corporation ("the Issuer").

3 I graduated from the University of Witwatersrand in Johannesburg, South Africa with a Bachelor of Science degree in Mining Engineering in 1993. I am a registered member in good standing with Engineers and Geoscientists British Columbia (License #180019), a member of Registered Professional Engineers of Queensland (License #06839), and a member of the Australian Institute of Mining and Metallurgy (CP). I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Project site.

5 I am responsible for Sections 2-5, 18-20, and 22-24 and parts of 1, 21 and 25-27 of the Technical Report;

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7 I have not had prior involvement with the property that is the subject of the Technical Report;

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 24 February 2021
Signing Date: 23 March 2021

Original Signed and Sealed by

Gary Methven, P.Eng.

CERTIFICATE OF AUTHOR

I, Deepak Malhotra, SME, of Wheat Ridge, Colorado, do hereby certify that:

1 I am currently employed as Principal / Director of RDi Minerals Inc. with an office located at 11475 West I-70 Frontage Road North, Wheat Ridge, Colorado, 80033, USA.

2 This certificate applies to the technical report titled "Empire State Mines 2021 NI 43-101 Technical Report", with an effective date of 24 February 2021, (the "Technical Report") prepared for Titan Mining Corporation ("the Issuer").

3 I am a graduate of Colorado School of Mines with a M.Sc. degree in Metallurgical Engineering (1974), and PhD in Mineral Economics (1978). I am a registered member of the Socity of Mining, Metallurgy and Exploration, Inc. (SME), member No. 2006420RM. I have worked as a Metallurgist / Mineral economist for over 40 years, since my gratuation from university; as an employee of several mining companies, an engineering company, a mine development and mine construction compnay, an exploration company, and as a consulting engineer.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Project site in 2016, for 2 days.

5 I am responsible for Sections 13, 17, and parts of 1 and 25-27 of the Technical Report;

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7 I have had prior involvement with the property that is the subject of the Technical Report; Participation in due diligence examination for purchase of the property in 2017.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 24 February 2021
Signing Date: 23 March 2021

Original Signed and Sealed by

Deepak Malhotra, SME

CERTIFICATE OF AUTHOR

I, David Vatterodt, Registered Member of SME, of Coeur D Alene, Idaho, do hereby certify that:

1 I am currently employed as an Underground Mine Engineer with Jackleg Consulting, LLC with an office located at 3220 W. Fairway Drive, Coeur D Alene, Idaho, 83815, USA.

2 This certificate applies to the technical report titled "Empire State Mines 2021 NI 43-101 Technical Report", with an effective date of 24 February 2021, (the "Technical Report") prepared for Titan Mining Corporation ("the Issuer").

  I am a graduate of University of Arizona in Tucson, AZ, USA (Bachelors of Mining in 2004). I am a member in good standing of the Society for Mining, Metallurgy and Exploration (SME) – Registered Member (QP) #4123583, and a member of the American Exploration and Mining Association. I have experience in underground mine operations, underground Life-of-Mine design, scheduling and economic evaluations.

  I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

3 I have visited the Project site from 3 – 7 November 2019, for four days.

4 I am responsible for parts of Sections 1, 16, 21, and 25-27 of the Technical Report;

5 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6 I have not had prior involvement with the property that is the subject of the Technical Report;

7 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

8 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 24 February 2021
Signing Date: 23 March 2021

Original Signed and Sealed by

David Vatterodt, SME

CERTIFICATE OF AUTHOR

I, Ben Peacock, P.Eng., of North Bay, Ontario, do hereby certify that:

1 I am currently employed as a Senior Engineer with Knight Piésold Ltd..with an office located at 1650 Main Street West, North Bay, Ontario, P1B 8G5.

2 This certificate applies to the technical report titled "Empire State Mines 2021 NI 43-101 Technical Report", with an effective date of 24 February 2021, (the "Technical Report") prepared for Titan Mining Corporation ("the Issuer").

3 I am a graduate University of Waterloo in Waterloo, Canada (Bachelors of Applied Science in 2008). I am a member in good standing of Professional Engineers Ontario (License #100141409), and a member of Professional Engineers and Geoscientists Newfoundland & Labrador and the Nunavut and Northwest Territories Association of Professional Engineers and Geoscientists. I have more than 10 years of experience in the mining industry. Areas of specialty include geomechanical and hydrogeological site investigations, rock mass characterization, and stability analyses for open pit and underground mines worldwide.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Project site.

5 I am responsible for parts of Sections 1, 16, and 25-27 of the Technical Report;

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7 I have not had prior involvement with the property that is the subject of the Technical Report;

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 24 February 2021
Signing Date: 23 March 2021

Original Signed and Sealed by

Ben Peacock, P.Eng.

CERTIFICATE OF AUTHOR

I, Matthew Hastings, MSc Geology, MAusIMM (CP), of Denver, Colorado, do hereby certify that:

1 I am currently employed as a Principal Consultant Resource Geologist with SRK Consulting (U.S.), Inc., with an office located at 1125 Seventeenth Street, Suite 600, Denver, Colorado, USA, 80202.

2 This certificate applies to the technical report titled "Empire State Mines 2021 NI 43-101 Technical Report", with an effective date of 24 February 2021, (the "Technical Report") prepared for Titan Mining Corporation ("the Issuer").

I am a graduate of The University of Georgia in Athens, Georgia, USA (B.Sc. of Geology in 2005), as well as The Univeristy of Nevada in Reno, Nevada, USA (M.Sc. of Geology in 2008). I have completed the Citation Program in Applied Geostatistics (University of Alberta, 2012). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (AusIMM), and a Chartered Professional member of the Idaho registered professional geologist in the United States. I have worked as a Geologist for a total of 15 years since my graduation from university. My relevant experience includes exploration, development, and estimation of mineral resources in a variety of geological settings and deposit types.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

3 I have visited the Project site from 9 – 13 March 2020, for four days.

4 I am responsible for Sections 6-12, 14 and parts of 1, and 25-27 of the Technical Report;

5 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6 I have not had prior involvement with the property that is the subject of the Technical Report;

7 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

8 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 24 February 2021
Signing Date: 23 March 2021

Original Signed and Sealed by

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